PREFACE

The inception of every worthy project, I am told, can be traced to the influence of **a** person or an event or both. The beginnings of this book go back over 30 years to Lute Parkinson at the Colorado School of Mines and his assignment of the greenest, brashest assistant professor on the faculty to team-teach with him the introductory mining engineering course, It was quite an experience . . . for both of us! He and my other mentors, Eugene Pfleider at the University of Minnesota and Merritt Williamson at the Pennsylvania State University, now deceased, together inspired this effort, because they encouraged me to address the perennial shortage of textbooks in our field.

If there is a uniqueness to this book, it lies in the quantitative and numerical approach adopted at the beginning level. Every introductory text must to some extent be descriptive and a survey of the field. From the outset, 1 tried to make this book something more, to provide in-depth treatment for the first professional course in mining engineering—to me, the most important course in the curriculum.

There is a model course for which this text was designed; it has evolved over 40 years at four institutions. Here are its objectives, taken from the syllabus:

The purpose of this course is to educate beginning students in mining engineering, and to bring them to an understanding of the industry and the challenges that it faces-economic, environmental, political, societal, as well as technological. Emphasis is placed on encouraging students to learn mining engineering by observing and doing: they conduct case studies, visit mines, see films, solve problems, compile mining method notebooks, and design basic mining systems.

The course falls in the sophomore on junior year of the curriculum, and its computer programming.

I have followed a simple and, I hope, logical outline in writing this book. The first four chapters deal with fundamentals, explaining the four stages in the life of mine and the unit operations of mining. The next nine chapters use divided between surface and underground mining, with emphasis on mining methods. The final two chapters deal with novel mining methods, coil comparisons, method selection, and a summary.

Although I rely on descriptive material to explain and differentiate the stages, operation, and methods of mining, a quantitative emphasis is provided by numerous examples and problems. Special topics, selected from the myriad ones representative of a diverse field like mining engineering, inject calculations in nearly every chapters: the instructor may assign them discretionally. Appropriate computer applications are suggested throughout, and students are encouraged to write and use programs to solve problems. \

There are several features of the book of which the course instructor should be aware. Case studies, which I consider an invaluable teaching tool, are identified with each mining method. Most were chosen because of geographic proximity or personal familiarity and can be modified by instructor’s choice. They should be supplemented with real-life case studies, carried out at the mine site by teams of students. Likewise, class field trips and films lend realism to assignments. A

national focus will be apparent in the text, but the international scene is not neglected, especially in descriptions of mining practice. References cited comprise an exhaustive bibliography of the subject matter: they are intended for more advanced as well as introductory students. Dual units (English and S.I.) are employed throughout the text, although numerical examples are solved only in English units, with S.I. units supplied in the statement and answer.

Special and warm acknowledgment is due Jan Mutmansky, Penn State faculty member, colleague, and former student, who generously assisted in the preparation and editing of the manuscript, Further, The University of Alabama allotted me time and clerical support, and the Society of Mining Engineers was especially helpful in permitting me unlimited use of illustrations and extensive bibliographical material from their publications.

Howard L. Hartman

CONTENTS

Introduction to Mining **1**

1.1 Mining's Contributions to Civilization / 1

1.2 Elements of Mining / 2

1.3 Development of Mining Technology / 4

1.4 Stages in the Life of a Mine / 6

1.5 Unit Operations of Mining / 13

1.6 Economics of the Mineral Industries / 15

1.7 Mining and Its Consequences / 20

1.8 Government Influence and Regulation / 21

1.9 Special Topic: Economic Analysis of a Mineral Commodity / 22

Stages Of Mining: Prospecting and Exploration **26**

2.1 Precursors to Mining / 26

2.2 Prospecting: General / 29

2.3 Prospecting: Geologic / 32

2.4 Prospecting: Geophysical and Geochemical *I* 34

2.5 Exploration: General / 45

2.6 Exploration: Methods / 48

2.7 Exploration: Reserve Estimates / 55

2.8 Exploration: Case Studies and Costs / 63

2.9 Feasibility Studies / 71

2.10 Special Topic: Calculation of Ore Reserve Estimate / 74

CONTENTS

3 Stages of Mining: Development and Exploitation 85

3.1 Mining Proper / 85
3.2 Development: General / 86
3.3 Development: Land Acquisitions and Environmental Protection / 90
3.4 Development: Financing and Implementation / 94
3.5 Development: Taxation and Costs / 98
3.6 Exploitation: General / 100
3.7 Exploitation: Mining Methods / 101
3.8 Exploitation: Organizations and Costs / 106
3.9 Special Topic: Mining Methods Notebook / 110

4 Unit Operations of Mining 100

4.1 Fundamental Operations and Cycles / 111
4.2 Drilling and Rock Penetration /112
4.3 Blasting and Rock Fragmentation / 124
4.4 Loading and Excavation / 134
4.5 Haulage and Hoisting / 139
4.6 Auxiliary Operations / 142
4.7 Cycles and Systems / 143
4.8 Special Topic: Chemical Design of Explosives / 146

5 Surface Mine Development 149

5.1 Nature and Scope of the Task / 149
5.2 Pit Planning and Design / 154
5.3 Equipment and System Selection / 160
5.4 Stripping Ratios and Pit Limits / 167
5.5 Special Topic: Calculation o f Stripping Ratios and Pit Limits / 173

6 Surface Mining: Mechanical Extraction Methods 177

6.1 Classification of Methods / 177
6.2 Open Pit Mining / 178
6.3 Quarrying / 187
6.4 Open Cast Mining */ 194*
6.5 Auger Mining / 207
6.6 Special Topic A: Surface Blasting /212
6.7 Special Topic B : Slope Stability /216

7 Surface Mining: Aqueous Extraction Methods 231

7.1 Classification of Methods / 231
7.2 Placer Mining: Hydraulicking / 232
7.3 Placer Mining: Dredging / 236
7.4 Solution Mining: Borehole Extraction *I* 241
7.5 Solution Mining: Leaching / 247
7.6 Special Topic: Selection of Materials-Handling Equipment / 251

8 Surface Mining: Method Comparison and Summary 260

8.1 Method Recapitulation / 260
8.2 Sequence of Development / 261
8.3 Cycle of Operations / 263
8.4 Geologic and Natural Conditions / 265
8.5 Advantages and Disadvantages / 267
8.6 Importance and Summary / 269
8.7 Special Topic: Cost Estimation of Materials-Handling Equipment / 270

9 Underground Mine Development 281

9.1 Nature and Scope of Task / 281
9.2 Mine Development and Design / 286
9.3 Mine Plant Layout / 296
9.4 Construction o f Development Openings / 299
9.5 Shaft Hoisting Systems / 317
9.6 Special Topic: Design of Hoisting System / 321

10 Underground Mining: Unsupported Methods 336

10 .1 Classification o f Methods / 336
10 .2 Room and Pillar Mining / 338
10.3 Stope and Pillar Mining / 351
10.4 Shrinkage Stoping / 360
10.5 Sublevel Stoping / 366
10 .6 Special Topic A : Calculation o f Percentage Recovery / 374
10.7 Special T op ic B : Design of Mine Openings / 375

CONTENTS

11 Undergound Mining: Supported Methods 388

11.1 Classification of Methods / 388
11.2 Cut and Fill Sloping / 389
11.3 Stull Stoping / 395
11.4 Square Set Sloping / 400
11.5 Special Topic A: Selection and Cost Estimate of Drilling Equipment / 405
11.6 Special Topic B: Underground Blasting /411

12 Underground Mining: Caving Methods 428

12.1 Classification of Methods / 428
12.2 Longwall Mining / 429
12.3 Sublevel Caving / 441
12.4 Block Caving / 450
12.5 Special Topic: Mine Ventilation / 462

13 Underground Mining: Method Comparison and Summary 477

13.1 Method Recapitulation / 477
13.2 Sequence of Development / 478
13.3 Cycle of Operations / 479
13.4 Geologic and Natural Conditions / 481
13.5 Advantages and Disadvantages / 487
13.6 Importance and Summary / 487
13.7 Special Topic: Determination of Mining Costs and Profits / 488

14 Novel Methods of Mining 496

14.1 Classification of Methods / 4%
14.2 Rapid Excavation / 498
14.3 Automation and Robotics / 505
14.4 Hydraulic Mining / 512
14.5 Methane Drainage / 521
14.6 Underground Gasification / 526
14.7 Underground Retorting / 529
14.8 Ocean Mining / 535
14.9 Other Novel Methods / 543
14.10 Importance and Summary / 546

15 Mining Method Selection and Summary **548**

15.1 Comparison: Surface vs. Underground Mining / 548
15.2 Comparison: Novel vs. Traditional Methods / 550
15.3 Mining Costs / 553
15.4 Method Selection / 569
15.5 Summary / 574
15.6 Special Topic: Mine Design Problem / 580

Appendix: Conversion Table, English to Metric Units (S.I. or International System) Useful In Mining Engineering **583**

Answers to Selected Problems **586**

References **591**

Index **619**

Introductory Mining Engineering

1

INTRODUCTION TO MINING

1.1 MINING'S CONTRIBUTIONS TO CIVILIZATION

Mining may well have been the second of man's earliest endeavors—granted that agriculture was his first. The two certainly rank together as the primary or basic industries of human civilization.

Another way of expressing the fundamental importance of mining in both, ancient and modern culture is to recall that nature provides only limited sources or ways to generate new wealth (Beall, 1973). In addition to parent-age, they consist of mining and agriculture (which includes farming, hunting, fishing, husbandry, and forestry).

Since prehistoric times, mining has been integral and essential to man's existence (Madigan, 1981). Mere the term *mining* is used in its broadest context as encompassing the extraction of all naturally occurring mineral substances—solid, liquid, and gas—from the earth for utilitarian purposes. By *utilitarian* we mean those essential human needs and desires that have been uniquely met by minerals down through the ages. The most prominent are identified in Table 1.1.

In fact, most of the cultural ages of man are associated with and identified by minerals or their derivatives. They include the Stone Age (prior to 4000 B.C.), the Bronze Age (4000 to 1500 **B.C.),** the Iron Age (1500 B.C. to A.D. 1780). the Steel Age (1780 to 1945), and the Nuclear Age (since 1945). Nor is it coincidental that many milestones in human history — Marco Polo's journey to China. Vasco da Gama's voyages to Africa and India, Columbus's discovery of the New World, and the modern gold rushes that led to the settlement of California. South Africa, Australia, the Canadian Klondike, and Alaska—were achieved with minerals as the prime incentive or goal (Rickard, 1932).

It is also demonstrable that minerals and mining have been associated with the ascendancies of the great civilizations of history. Indeed, the far- flung expansions of the Roman Empire into Spain and Britain, the conquest of North and South America by the Spanish, French, and English, and the colonization of Africa and parts of Asia by a variety of European powers were fueled by the lust for mineral wealth. A different sort of modern empire—an economic cartel, the Organization of Petroleum Exporting Countries (OPEC)—exists to control petroleum prices and supplies in an awesome exercise of mineral might.

TABLE 1.1 Man's Uses of Minerals

Need or Use	Purpose	Age
Tools and utensils	Food, shelter	Prehistoric
Weapons	Hunting, defense, warfare	Prehistoric
Ornaments and decoration	Jewelry, cosmetics, dye	Ancient
Currency	Monetary exchange	Early
Structures and devices	Shelter, transport	Early
Energy	Heat, power	Medieval
Machinery	Industry	Modern
Nuclear fission	Power, warfare	Modern

1.2 ELEMENTS OF MINING

There are various terms and expressions unique to mining that characterize it as a technical field. The novice is advised to "master the language" early, to become familiar with it without resorting to rote memorization. Most terms are introduced selectively in the text that follows, but some are general and best defined now. For a complete glossary of mining terminology, see a standard reference (e.g.. Thrush, 1968; Gregory. I983).

Three basic definitions are closely related:

Mine: an excavation made in the earth to extract minerals
Mining: the activity, occupation, and industry concerned with the extraction of minerals

Mining engineering: the art and science applied to the processes of mining and the operation of mines

Some terms distinguish various materials mined. Geologically, one distinguishes between the following:

Mineral; a naturally occurring substance, usually inorganic, having a definite chemical composition and distinctive physical characteristics
Rock: an assemblage of minerals
Economically, the distinction made between minerals is the following:

Ore: mineral that has sufficient utility and value to be extracted at a profit
Waste or *gangue:* mineral that lacks utility and value when mined (gangue is more intimately associated with ore than is waste)

When both geologic and economic relationships are involved, the distinguishing terms arc the following:

Mineral deposit: geologic occurrence of minerals in relatively concentrated form
Ore deposit: economic occurrence of minerals that can be extracted at a profit

A further convenient subdivision of commercial minerals, into three main categories, is made on the basis of primary constituent and usage. *Metallic ores* include ores of the ferrous metals (iron, manganese, molybdenum, and tungsten); base metals (copper, lead, zinc, and tin); precious metals (gold, silver, and platinum); and radioactive metals (uranium, thorium, and radium). *Nonmetallic ores* consist of industrial minerals such as phosphate, potash, stone, sand, gravel, sulfur, salt, and industrial diamonds. The *mineral fuels*, sometimes called fossil fuels, include coal, petroleum, natural gas, uranium, and several less common, marginal sources (lignite, oil shale, tar sand, and coalbed methane). It should be noted that, while an offshoot of mining, the activity associated with the extraction of petroleum and natural gas has now evolved into a separate industry with a specialized technology of its own.

The essence of mining in extracting mineral from the earth is to drive or construct an excavation, a means of entry, from the existing surface to the mineral deposit. (When the value of the mineral is established with some confidence, usage of the terms *ore* and *ore deposit* is preferred, except that coal and stone deposits are usually so designated.) If the excavation is entirely open to or operated from the surface, it is called a *surface mine.* If the excavation consists of openings for human entry driven below the surface, then it is an *underground mine.* The specific details of the procedure, layout, and equipment used distinguish the *mining method*, uniquely determined by the physical, geologic, environmental, economic, and legal circumstances that prevail.

Mining is not done in isolation, nor is it an entity unto itself. It is preceded by geologic investigations that locate the deposit and economic analyses that prove it financially. Following extraction of the coal, stone, or ore, the run-of-mine material processing. The products of those processes may then undergo further concentration, refinement, or fabrication during conversion, smelting, or refining to provide consumer products. The end step in converting a useful mineral into a usable product is marketing.

Occasionally, excavation in the earth is employed for purposes other than mineral extraction. These include civil and military works in which the objects is to produce a stable opening of desired size, orientation, and permanence. Examples are vehicular tunnels, storage reservoirs, waste disposal chambers, and military installations. They are excavated using methods that are borrowed from mining. Since the objective is the excavation or opening itself rather than the mineral

extracted, however, other kinds of conditions or circumstances may govern, such as time, shape, or life.

Professionally, the fields of endeavor associated with the mineral industries are linked to the phase or stage in which the activity occurs. Locating and exploring a mineral deposit fall in the general province of geology and the earth sciences. Mining engineering, already defined, encompasses the proving (along with geology), planning, developing, and exploiting of an ore deposit. The fields of processing, refining, and fabricating properly are assigned to metallurgy, although there is overlap in the mineral processing or extractive area with mining engineering.

1.3 DEVELOPMENT OF MINING TECHNOLOGY

As one of the earliest of man's endeavors—and certainly among his first organized endeavors—mining has had an ancient and venerable history (Gregory. 1983)- To understand modern practices in thc mineral industries, it is useful to trace the evolution of mining technology, which, as pointed out at the beginning of this chapter, has paralleled man's own evolution and the advance of civilization.

Mining began with Paleolithic man some 450.000 years ago. There are, of course, no records to substantiate the fact, but flint implements have been found with the bones of early man from the Old Stone Age (Lewis and Clark, 1964). He extracted these from the earth and learned to shape them by crude fabricating techniques. At first he was satisfied to recover stone raw materials from surface excavations, but by the beginning of the New Stone Age he had progressed (o underground mining in systematic openings 2-3 ft (0.60.9 m) in height and over 30 ft (9 m) in depth (Stoces, 1954). However, the oldest known underground mine is from the Old Stone Age and believed to be 40.000 years old; it is a hematite mine at Bomvu Ridge. Swaziland (Gregory, 1981). Early deep miners employed crude methods of ground control, ventilation, hoisting, lighting, and rock breakage. Underground mines attained depths of X00 ft (250 m) by Egyptian times.

Metallic minerals attracted the attention of prehistoric man. Initially, metals had to be used in their native form, probably obtained by washing river gravel in surface placer deposits. With the advent of the Bronze and Iron Ages, however, man discovered smelting and learned to reduce ores to native metal or alloy form.

The first notable technological feat that challenged miners in the excavation of stone or ore in place was to break it loose from the constraining rock mass While they could readily excavate soil or loose rock, their crude tools of bone, wood, and stone were no match for solid rock, unless crevices existed or cracks could be formed and opened by wedging. Soon they devised a revolutionary technique with heat to expand rock and water to cool, contract, and break it. termed *fire setting.* Among the great discoveries of humankind and the first technological breakthrough in mining, the art and science of rock breakage are of fundamental importance. No other technological advance in mining had an equal impact, until black powder was first utilized to blast rock in the seventeenth century.

As social systems and culture evolved, mining became more organized (Beall. 1973). Because of the arduous, hazardous nature of the work, slaves and convicts were often committed to the mines. Supervisors and engineers, however, commanded premium wages. The Egyptians were the first organizers and recorders of mining activity, but the most successful mine operators were the Romans. Using their famed colonizing skills, the Romans established a mineral industry that thrived and prospered across the empire.

Mining technology, like that in all industry, languished during the Dark Ages. Notably, a political development in A.D. 1185 improved the standing o! mining and the status of miners, when the bishop of Trent granted a charter to miners in his domain. It gave miners legal as well as social rights, including the right to stake mineral claims. A milestone for the industry, the edict had long-term consequences that persist to this day.

The greatest impact on the need for and use of minerals, however, was registered by the Industrial Revolution, coming at the close of the eighteenth century. Along with the soaring demands for minerals came spectacular improvements in mining technology, especially in scientific concepts and mechanization, that have continued to this day. They form the backbone of the methodology with which we will subsequently deal.

It is not possible to chronicle all the developments of mining technology, but some of the most significant of those that have impacted the industry or civilization in general arc enumerated in Table 1.2. They culminate in the launching of the era of modem mining at the beginning of the twentieth century, with the advent of mechanization, mass production, and other cost- cutting techniques (most recently, computerization) that have made possible the current exploitation of massive, low-grade ore deposits. A more complete chronology of events appears in the chart prepared by John Myers

Table 1.2 Chronological Development of Mining Technology

Date	**Event**
450 000 b.c.	First mining (at surface), by Paleolithic man for stone implements
40 000	Surface mining progresses underground, in Swaziland, Africa
30 000	Fired city pots used in Czechoslovakia

Marketing (Anon. 1979a), For authoritative, recent references on mining history), see the works by Gregory (1981). Stack (1982) and Molloy (1986).

1.4 STAGES IN THE LIFE OF A MINE'

The overall sequence of activities involved in modem mining is often compared to the stages in (he life of a mine There arc four: *prospecting*, *exploration, development,* and *exploitation.* Prospecting and exploration, precursors to actual mining, are linked and sometimes combined. geologists and mining engineers share responsibility for these two stages, geologists more for the former and engineers more for the latter Likewise. development and exploitation

arc closely related stages: they arc usually considered to constitute mining proper and are the main province of the mining engineer.

The four stages in the life of a mine are summarized in Table 1.3 and discussed below. There is more extensive coverage in Chapters 2 and 3 but we introduce the subject now. In addition to the stages and steps involved in advancing the mineral deposit to mine status. Table 1.3 specifies usual ranges for the time and cost of each stage

TABLE 1.3 Stages in the Lite of a Mine

Stage/ (Project Name)	Procedure	Time	Cost/Unit Cost
Precursors to Mining			
1. Prospecting (Minaret deposit)	Search for one a. Prospecting methods Direct: physical, geologic Indirect: geophysical, geochemical b. Locate favorable loci (maps, literature old mine) c. Air, aerial photography, airborne geophysics. d. Surface ground geophysics.. geology e. Spot anomaly, analyze, evaluate	1-3yr	$01-5 million or 2-50 e/ton
2. Exploration (Ore body)	Defining extent and value of ore (examination/evaluation a Sample (drilling or excavation), assay test b Estimate tonnage and grade c. Vaiuata deposit (Hoskold formula or discount method) **present value – income - coat** Feasibility study make decision to abandon or develop	2-5 yr	$0,5-10 million or 10e-$1/ton

3. Development (Prospect)	Opening up ore deposit for production a. Acquire mining right (purchase or lease), if not done in stage 2 d. File environmental impact statement, technology assessment permit c. Contruct access roads, transport system d. Locate surface plant, construct facilities e. Excavate deposit (strip or sink shaft)	2-5 yr	$10-250 million or 25c-$5/ton (23c-$4.50/ton)
4. Exploitation (Mine)	Large-scale production of ore a Factors In choice of method geologic, geographic, economic, environmental, societal, safety b. Types of mining methods Surface: open pit, open cast etc Underground: room and pillar, block caving, etc c. Monitor coats and economic payback (3-10 yr)	2-5 yr	$10-250 million

Prospecting

Prospecting, the first stage, is the search for metallic ores or other valuable minerals (coal or nonmetallics). Because mineral deposits are found at or beneath the surface of the earth, both direct and indirect prospecting techniques are employed.

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

A valuable scientific tool being employed in the ***indirect search*** for hidden ore bodies is geophysics, a method that detects anomalies caused by the presence of mineral deposits, through the analysis of gravitational, seismic, magnetic, electrical, electromagnetic, and radiometric measurements. It is suitable for airborne, surface, and subsurface use. Geophysics applied from the air or space through remote sensing enables vast areas to be prospected and explored. On the ground and in logging boreholes, it provides more definitive information. Geochemistry, the microquantitative analysis of soil and water samples, and geobotany, the study of vegetational and plant growth patterns, also are being employed as prospecting tools.

Exploration

The second stage in the life of a mine, exploration determines as accurately as possible the size and value of a mineral deposit, utilizing techniques similar to but more refined than those used in prospecting. The line of demarcation between the two is not sharp; in fact, a distinction may not be made between the two stages. The locale in exploration shifts from the air more to the surface and subsurface. In addition, more positive information of the extent and richness of the deposit is obtained by subjecting representative samples to chemical. X-ray, spectrographic, or radiometric analyses. Samples are obtained by chipping outcrops and by trenching, tunneling, and drilling: additionally, borehole logs may be taken by geophysics. There are several common drilling methods; diamond drills provide core samples and rotary or percussion drills produce chips or cuttings.

An evaluation of chip or core samples or logs enables the geologist or mining engineer to calculate the tonnage and grade, or richness, of the deposit. He or she establishes the financial worth of the ore, estimates mining costs, and assesses all other foreseeable factors in an effort to reach an accurate conclusion concerning the merits of a given deposit and the profits likely to be realized. This entire procedure is called ***reserve estimation*** or examination and valuation of the ore deposit. At the conclusion of this stage, a thorough ***feasibility study*** is conducted to determine the potential of developing the mineral deposit into a producing mine. The outcome of the study is ***a*** decision to abandon or develop the deposit.

Development

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

In either case, certain preliminary development work, such as acquiring mineral rights and financing and providing access roads and other transportation,

power sources, ore-handling and -treatment facilities, and waste disposal areas, - must precede the actual mining. Stripping of the waste material overlying the ore body then commences if the mine is to be a surface one. The cycle of operations to break and handle the overburden is the same as that employed in exploitation of the ore. Economic considerations deter-mine the stripping ratio (ratio of waste removed to ore recovered); it may range from as high as 45-to-1 yd^3/ton (38-10-1 m^3/tonne) in coal mining to as low as 1-to-I in metal and approach 0-to-l nonmetal.

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

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

Exploitation

Exploitation, the fourth and final stage of mining, is associated with the actual recover of mineral from the earth in quantity. While some development work necessarily continues throughout the life of a mine, the emphasis in the exploitation stage is on production. Only enough development is done prior to exploitation to ensure that production, once started, can continue uninterrupted throughout the life of the mine.

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

Typical of the large-scale operations and interchangeability of equipment used in modern mining is the scene in Figure 1.1. While the locale is underground. the equipment was adapted from surface mining.

Surface Mining. Surface mining is the predominant exploitation method worldwide, producing in the United States nearly 85% of all minerals, excluding petroleum and natural gas. Almost all nonmetallic minerals (96%), 87% of the metallic ores, and 60% of the coal in the United States are mined from (he surface, and most of these by the open pit or open cast methods. In *open pit mining,* a *mechanical extraction method,* a thick deposit is generally mined in benches or steps, although a relatively thin deposit may be mined from a single face, as in *quarrying, augering,* or *open cast mining.* Any overburden must be removed by a stripping process before or during mining, except in open cast mining, where overburden is removed and mineral (usually coal) recovered in successive operations. Open pit or open cast mining is usually used to exploit a deposit near the earth's surface or in those cases where a deposit exists that has a low stripping ratio, is preferably large in extent, and is reasonably uniform in value. It necessitates a large capital investment but generally results in high productivity, low operating cost, and good safety conditions.



Figure 1.1. Exploitation by underground mining methods, using equipment developed for surface mining. Unit operations: loading and haulage. Exploitation method: unsupported class, slope and pillar mining. (By permission from Georgia Marble Co , Atlanta.)

The aqueous extraction methods are uniquely reliant on water or another liquid (e.g. ferric sulfate, anthracene oil) during mining and processing. *Placer mining* is used to exploit mineral deposits that are loosely consolidated such as sand or gravel, and contain a valuable heavy mineral in a free state. Gold, diamonds, platinum, and tin commonly occur in placer form. *Hydraulicking* utilizes a high-pressure stream of water that is directed against an exposed placer

bank, thereby undercutting it and causing it to crumble. *Dredging* accomplishes extraction of the ore minerals mechanically or hydraulically from floating vessels. In both placer methods, the valuable mineral constituent, generally heavier than the waste material, is removed from a water-base slurry by concentration. *Solution mining* includes both *borehole* methods, such as salt wells or the Frasch process to melt sulfur, and solvent *leaching,* either in situ or heap. Placer and solution mining methods arc the most economical of all exploitation methods but can be used only for large deposits of minerals that are easily excavated and susceptible to aqueous (solution) attack.

Underground Mining. Underground methods— unsupported, supported, and caving— arc differentiated by the type of wall and roof supports used, the configuration of production openings, and the direction in which mining operations progress. The *unsupported methods* of mining are used to extract mineral deposits that are roughly tabular, flat or steeply dipping, and generally in contact with strong wall rock. *Room and pillar mining* is adaptable for regular, flat-lying deposits, with the advance horizontal; support of the roof is provided by natural pillars of coal or ore that arc left standing in a systematic pattern, and rooms are cut from access entries to provide working faces. When necessary, additional support is supplied by roof bolts or timbers. *Slope and pillar mining* (a stope is a large production opening) is a similar method used in noncoal mines where thicker, more irregular ore bodies occur; the pillars arc usually spaced randomly and consist o f relatively lowgrade ore, since the richer ore is extracted in the slopes. The two methods account for some 75% of all underground mining in the United States. In *shrinkage stoping.* mining progresses upward, with slabs of ore being broken along the length of the stope. The broken ore is allowed to accumulate in the stope to provide a working platform for the miners and is thereafter withdrawn through chutes into haulage drifts on the level below. *Sublevel stop*- Modifiers from shrinkage by providing several working benches, aligned vertically or staggered, and mining progresses horizontally. Long blastholes are drilled into the ore in a parallel of fanlike design to fracture the rock. Shrinkage stoping is more suitable than sublevel stoping for stronger ore and weaker wall rock.

Since they provide support to the wall rock, supported stoping methods are often used in mines having weak structure. Cut and fill stoping is employed for dipping, tabular deposits. As the mining progresses upward, waste backfill is placed in the stope to provide support for the watts. The ore is moved to chutes or orepasses mechanically, and the waste is usually distributed hydraulically. *Square set stoping* also involves backfilling; however, it relies mainly on timber sets to support the walls during mining. These timber sets are assembled in a continuous supporting structure to form hollow prisms that is subsequently filled with waste material. Since it is (he costliest method, it is generally used only in rich mines of very weak structure. *Stull sloping* is a supported method using limber or rock bolts in narrow, tabular, pitching ore bodies.

Caving methods are varied and versatile; and involve caving the ore and/ or overlying rock. Subsidence of the surface eventually follows. *Longwall mining*

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

In addition to these traditional methods, new *novel methods* of mining are continually evolving. They are applicable to unusual deposits or employ unusual techniques or equipment. Examples are *automation, rapid excavation* in hard rock, *underground gasification,* and *ocean mining.*

1.5 UNIT OPERATIONS OF MINING

During the development and exploitation stages of all mining when natural materials-—rock or soil, ore or waste—are extracted from the earth, remarkably similar unit operations are employed. The *unit operations* of mining are the basic steps employed to produce mineral from the deposit, together with the auxiliary steps involved. Those steps contributing directly to mineral extraction arc *production operations,* comprising the production cycle of operations. Those ancillary steps which support the production cycle are called *auxiliary operations.*

The production cycle employs unit operations which are normally grouped in two functions: ***rock breakage*** and ***materials handling.*** Breakage includes a variety of mechanisms but in rock is usually accomplished **by** drilling and blasting. Handling generally encompasses loading or excavation and haulage (horizontal transport), with optional hoisting (vertical or inclined). Thus the basic production cycle in mining consists of these unit operations:

Production cycle = drill + blast + load + haul

While production operations tend to be separate and cyclic in nature, the modem and future trend in mining and tunneling is to eliminate or combine functions and to increase continuity. For example, soil may be excavated by a machine which requires no drilling or blasting. If loosening is necessary, it may be accomplished without explosives by ripping prior to loading. In coal or soft ores,

continuous miners break and excavate mechanically and thus eliminate drilling and blasting; boring machines perform the same tasks in medium-hard rock.

The cycle of operations in surface and underground mining is distinguished mainly by the scale of the equipment. Specialized machines have evolved to meet the unique needs and conditions of the two regimes.

In modem surface mining, blastholes several inches (mm) in diameter are bored by mobile rotary or percussion drills for the placement of explosives when consolidated rock must be excavated. The charge is then inserted and detonated to reduce the ore or waste to fragments. The broken material is loaded by power excavators of the shovel, dragline, or wheel type into haulage units—railroad cars, belt conveyers, or, most usually, trucks or cast on a waste (spoil) bank. Soil and coal are mined in a similar way, although blasting is often unnecessary. In quarrying dimension stone, the blocks are freed without blasting by channeling machines or saws.

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

In addition to the productive phases of the actual mining cycle, certain auxiliary operations must be performed. Underground, these auxiliary opera ions consist of providing and maintaining adequate roof support, ventilation conditioning, power supply, pumping, maintenance. lighting, communications, and delivery of supplies. In surface mining, the first two functions are not necessary, but slope control and waste disposal must be practiced instead.

1.6 ECONOMICS OF THE MINERAL INDUSTRIES

Mineral Production

It is estimated that only a fraction of 1% of the earth's surface is underlain with mineral deposits of commercial value. Yet the annual nonfuel mineral production in I9K4 of the United States alone amounted to over 2 billion tons valued at $23 billion (Fig. 1.2); including the fuels, the total value of raw mineral production exceeded $200 billion (Anon., 1984b). Note that with value added in processing, the figure for nonfuels increases tenfold to $253 billion, about 7% of the $3.7 trillion U.S. gross national product (in less developed countries, LDCs, the figure may reach 25%). Per capita consumption of minerals is equally impressive (Fig. 1.3); for the nonfuels in 1983, it cumulated 9 tons, and for all minerals, nearly 20 tons (Anon., 1979b).

World consumption of minerals increased to such extent in modem times that more minerals have been used in the twentieth century than were used in all

the previous centuries combined. Since the beginning of the Industrial Revolution, the annual rate of increase in mineral consumption has averaged

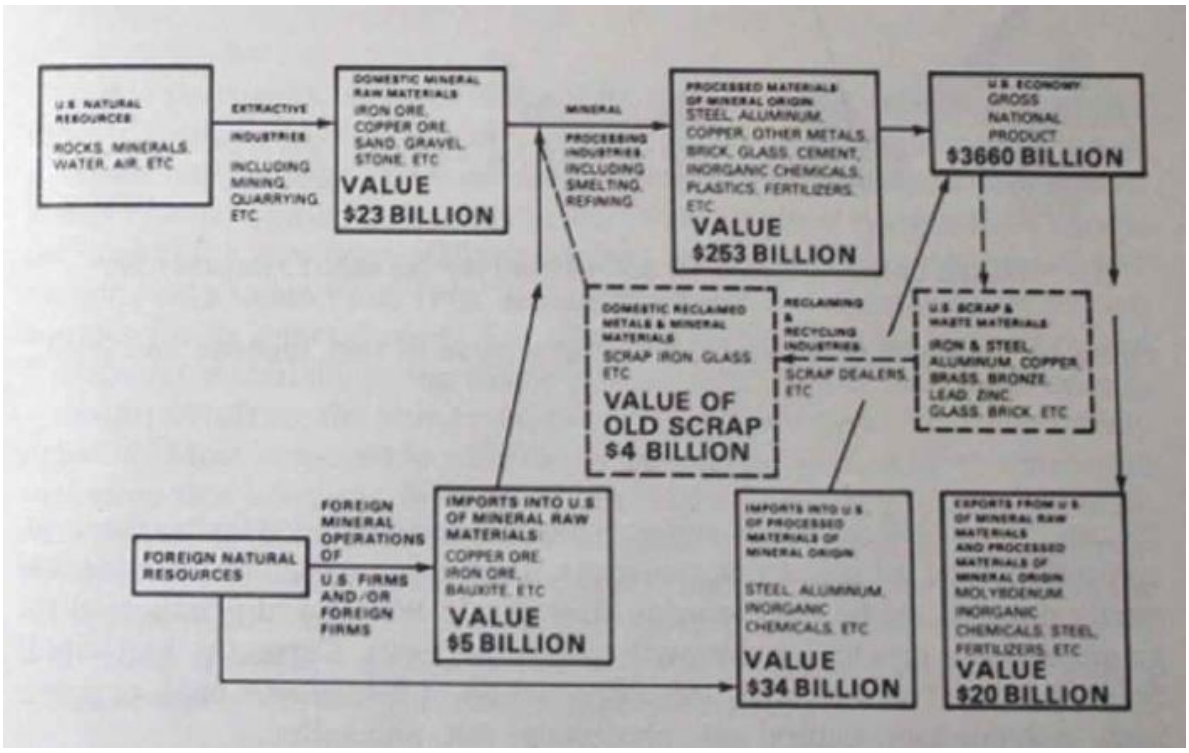
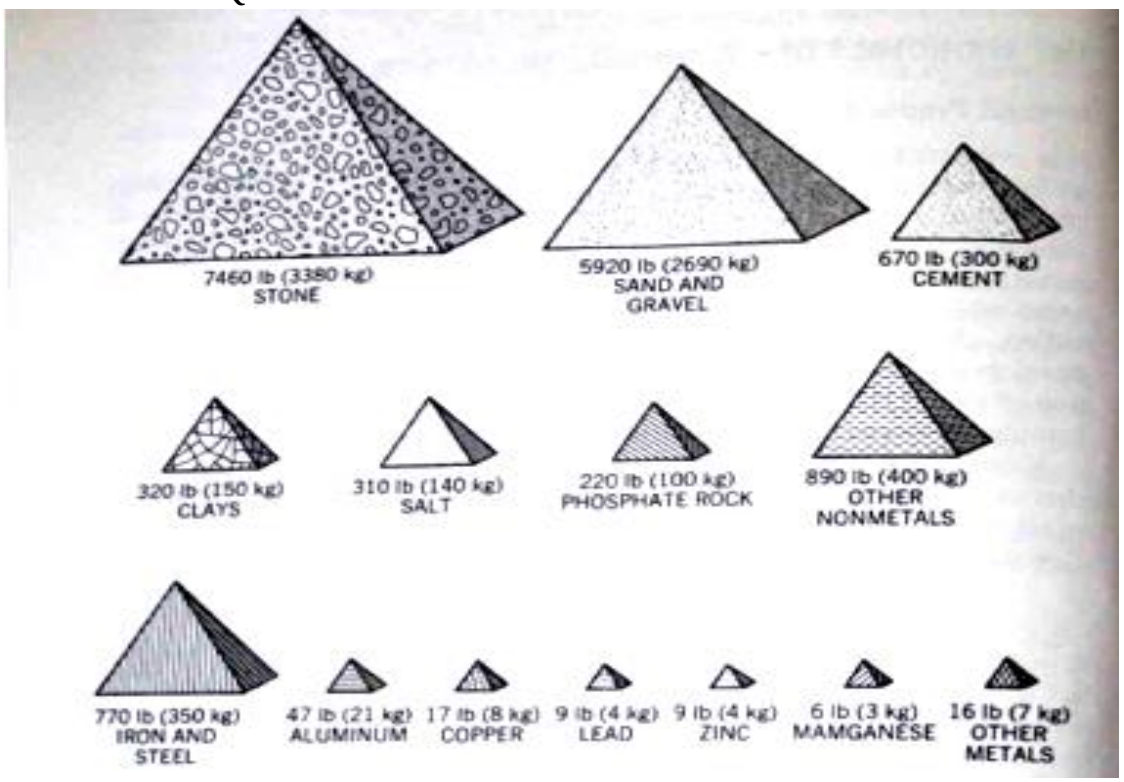


Figure 1.2. Value of nonfuel minerals produced and processed in the United States in 1984 Based on incomplete statistics for the year. (After Anon.. 1984b.)

OVER 18 000 lb (8160 kg) NONFUEL MINERAL MATERIALS ARE NOW REQUIRED ANNUALLY FOR EACH U.S. CITIZEN



U.S. TOTAL USE OF NONFUEL MINERAL SUPPLIES IN 1983 WAS ABOUT 2 BILLION TONS (18 x 109 tonnes)

Figure 1.3 Per capita consumption of nonfuel minerals in 1983. (Updated from Anon.. 1979b.)

5%, and since 1950 the use of minerals has increased twice as fast as the total consumption of all other raw materials. Domestic production o f minerals nearly doubled in the two decades after World War II and is expected to maintain about one-half that growth rate in the 1980s. Currently the United States leads the world in the mine production of bituminous coal, copper, lead, molybdenum, natural gas, phosphate, salt, and sulfur.

Although the United States produces a significant portion o f the total world output, it nevertheless has become a major and growing importer of minerals. (Fig. 1.4). The United States now imports substantial quantities (50-70%) of its antimony, asbestos, barium, cadmium, silver, vanadium , and zinc and practically all (over 70%) o f its aluminum ore, chromium, cobalt, industrial diamonds, manganese, nickel, potash, and tin.

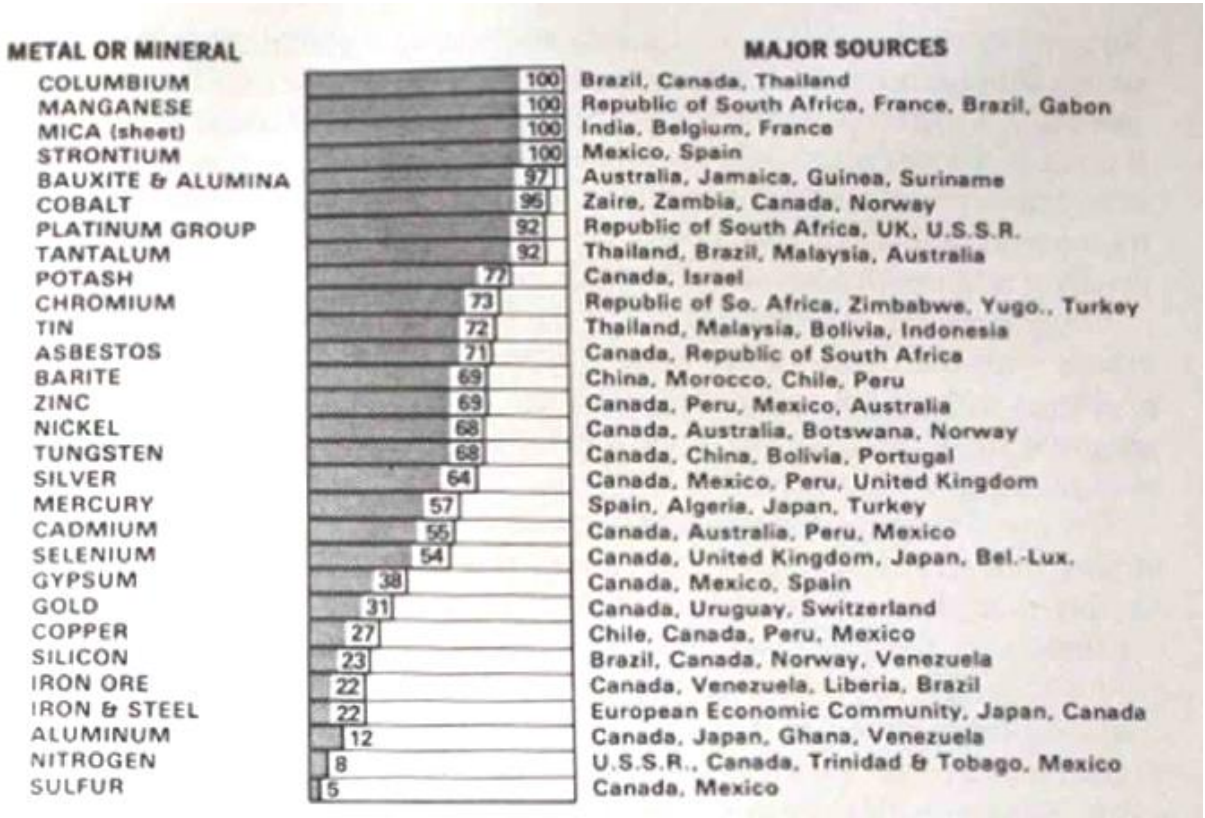


Figure 1.4. Net import reliance of selected minerals and metals as a percentage of consumption for the United States in 1985. (Updated from Anon., 1979b.)

All these latter minerals are critical ones needed for national defense or food production. Further, it has ceased to be an exporter of raw mineral commodities except for coal and a few nonmetallics: imports now exceed exports by a 2-to-l margin, on a value basis (Fig. 1.2). Growing U .S. dependence on foreign sources

for its mineral needs has created a troubling defense concern and contributes materially to the nation's worsening international trade deficit.

Issues involving the conservation of mineral resources are often widely debated. Most responsible officials of mining companies favor extraction measures that conserve their resources while maximizing profits. Controversy arises sometimes between those in the industry and in the conservation movement over an extremist "preservationist" position, advocating the protection o f all mineral resources. Increasingly, voices of reason prevail on both sides, allowing compromises to be reached without expensive litigation when new projects undergo environmental review.

Mineral Economics

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

Other peculiar features are operating ones. Production costs tend to increase with the depth of the mine and the declining grade of ore, creating problems with which every mine eventually is confronted. Financial hazards are great since the estimate of the ore supply, market price, or other factors may prove to be incorrect.

The law *of supply* and demand likewise complicates the economics of the mining industry, because the price and production of minerals vary more sharply than the price and production of commodities manufactured by the consumers of the raw materials. The output of minerals from by-product producers and foreign sources can create an oversupply that w ill depress the market. Some minerals, such as precious metals, iron, and certain base metals, arc recycled and in a sense never expended because of their utility as scrap. Reservoirs of scrap— lead is the extreme case at 50% o f consumption— may depress the market. Certain minerals arc exceptions to economic laws because their prices are fixed by government decree or cartels. Official prices of gold, silver, and uranium have been regulated by governments (although the market prices of gold and silver now fluctuate in free world markets), and cartels strongly influence the prices of industrial diamonds, mercury, oil, and tin. Substitutes for a particular mineral may be developed, especially if the price of the mineral remains at a high level (e.g., aluminum for copper, plastic for metal). Market speculation affects the prices o f minerals as it does most other commodities.

A country follows a fairly predictable pattern in its economic and social development, reflecting the periods of discovery, exploitation, and exhaustion of its mineral resources and the rise and decline of its mineral industries (Lovering,

1943). The characteristic, successive periods o f mineral industrial development are shown in Figure 1.5; each is described below:

1. *Period of mine development:* exploration, discovery of new districts, many small mines working; first recognition of large deposits and development of large mines; rapidly increasing production of metal.
2. *Period of smelter development:* few new discoveries; small mines becoming exhausted, but increasing output from large mines; many smelters competing for the ore.
3. *Period of industrial development:* decreasing costs, increasing standard of living; rapid accumulation of wealth; expanding internal and external markets, approaching zenith of commercial power.

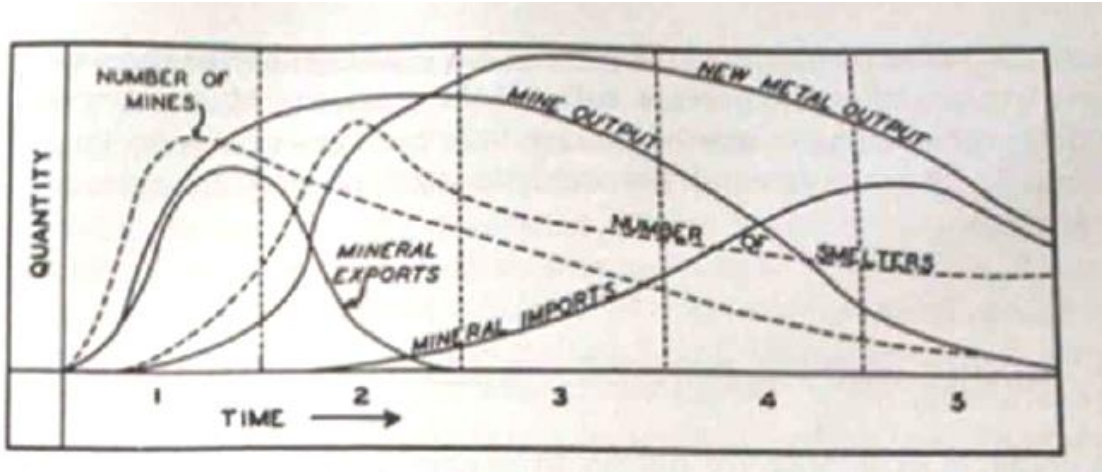


Figure 1.5. Periods in Ihe growth and decline of a country's mineral industries. (Alter Hewett. 1929: Lowering, 1943. By permission from Society of Mining Engineers. Inc., Littleton, CO)

4. ***Period of rapid depletion of cheap raw materials at home:*** involves ever-increasing costs of mining and of minerals produced; more and more energy is required to get the same amount of raw material. Mines and smelters continue decline. Some foreign markets arc lost; foreign imports, both **raw** materials and manufactured products, invade the home market.
5. ***Period of decreasing internal and external markets:*** increasing dependence on foreign sources of raw materials brings increasing costs to manufacturers. This period is characterized by a decreasing standard of living with its accompanying social problems and political unrest; quotas, tariffs, subsidies, cartels, and other artificial expedients arc used in the effort to maintain a competitive price in the domestic and world market. It is a period of decreasing commercial power. A drive is generally made to acquire cheap foreign sources of raw materials when domestic sources become low.

As current-day examples of the cycle in Figure 1.5, many LDCs fall in period I. Australia in period 2, the Soviet Union in period 3, the United States in period 4,

and the United Kingdom in period 5. Some economists maintain that the cycle can be modified if not reversed.

To exercise some management over its mineral destiny, a country is wise lo enact a national mineral policy. While proponents have long advocated the adoption of such a policy in the United States, there exists only a de facto one at this time (Dorr, 1984). Manifestations of the policy are evident in Section 1.8.

A final aspect of mineral economics concerns financing and marketing of mines and mineral properties. Mining enterprises are financed in much the same manner as arc other businesses (Wanless. 1984; Tinsley et al., 1985). Because of greater financial risks, however, the expected return on an investment is higher and the payback period shorter in a mining enterprise. Mineral properties as well as mines are marketable. The selling price is determined generally by a valuation based on the report of a qualified engineer; the value of future earnings is usually discounted to the dale of purchase in computing the present value of the property. Mineral deposits may be determined to have worth because they are very rich, very large, easily accessible, in great demand, favorably located. cheaply minable. or militarily strategic.

1.7 MINING AND ITS CONSEQUENCES

The impacts of mining arc not all favorable. Few of man s productive or industrial endeavors generate the controversy that mining has. The benefits of mining are well documented; we need now to examine the deleterious effects and their containment.

The less favorable and more controversial aspects of mining are generally side effects—secondary or tertiary—some of which are hard to measure and all of which are difficult to predict in advance. They may be grouped in the following categories:

1. Environmental and land use impacts
2. Accidents and health hazards
3. Economic-political-social-psychological impacts

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

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

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

The anticipation of unwanted, indirect consequences is the most important and difficult challenge that mining confronts in managing these effects. Two procedures are employed by mining companies to assist them: (1) technology assessments (TAs) and (2) environmental impact statements (EISs). Federal agencies and the Congress pioneered TAs, which examine the effects of primarily technological change and attempt to forecast all the side effects; they are now used internally by a variety of organizations involved in industrial development, including mines. Mining companies are required by government regulation, the National Environmental Policy Act (NEPA) of 1969, to file an EIS on any mineral project that involves "federal action," that is, approval of a lease, permit, right of way, or mining plan (Parr, 1982). Both TAs and EISs attempt to measure the impacts of a proposed project by means of cost/benefit analyses or similar procedures; generally, mining engineers are responsible for preparing both statements.

1.8 GOVERNMENT INFLUENCE AND REGULATION

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

The law governing the acquisition of mineral rights in the United States has developed from the common law of England, the laws and statutes of the federal government, and the laws of the various states. Although the federal Mining Law of 1872 has been somewhat modified by later legislation, it remains the recognized and pertinent statute. It provides for the location of claims for mineral deposits located in the public domain, the performance of

annual assessment work to retain rights to a claim, and the acquisition of title to claims. Certain nonmetallic minerals such as coal, gas, oil, phosphates, sodium deposits, and sulfur are exempted from this act and are governed by a leasing law, the Mineral Leasing Act of 1920. Uranium is Object to a leasing arrangement also, under the Atomic Energy Act of 1954. Many states have also enacted legislation to provide mineral rights within their boundaries. Regulations governing the conduct of mining activities, particularly safety measures, arc enforced by federal and state governments.

A mining company is subject to the same forms of taxation upon income as any other business and. in certain states, to production royalty, and severance taxes as well (Anon.. 1983c). In the federal corporate income tax law however, the Internal Revenue Service wrote in two provisions that were advantageous to mining companies. The first was a depletion allowance. similar in effect to a depreciation charge, that permuted a deduction from taxable income in recognition of the gradual exhaustion of ore. The second allowed the deduction of exploration and development costs as a current expense (although depletion is deferred if exploration is successful).

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

Stockpiling of strategic minerals by the government became a common practice after 1939, and the practice was sharply increased after 1946. The Federal Emergency Management Agency is responsible for procuring certain minerals as part of the program of national preparedness and entered into purchase agreements with individual producers, at negotiated prices, to meet its stockpiling objectives. In recent years, U.S. government stockpile purchases have declined under provisions of the Strategic and Critical Materials Stockpiling Revision Act of 1979 (Dorr. 19X4). The mineral industry has often been critical of the government's stockpiling policy, since sudden large purchases or sales from the stockpile can have drastic artificial effects on the price and demand for a commodity. Economists tend to favor private sector management of inventories.

1.9 SPECIAL TOPIC: ECONOMIC ANALYSIS OF A MINERAL COMMODITY

Problem. Prepare a summary analysis of pertinent production and economic data for several important mineral commodities. Obtain statistics for mine production (world and domestic) and domestic imports, exports, scrap, stockpile changes (private and government), and net consumption (total and by use). Determine for the most recent year for which data are preferably on an as-mined basis ($/ton, or $/tone, of mineral, F.O.B. mine). ***(Note:*** Commodities selected may correspond to those in the mining method case studies, and the analyses may be conducted with the case studies.)

Procedure. Select the commodities chosen in this text for case studies. Consult annual statistical references for each mineral commodity; several general ones follow.

NONFUELS

1. *Minerals Yearbook.* 3 volumes, U .S. Bureau of Mines, U .S. Government Printing Office, Washington. DC (annual).
2. *Mineral Commodity Summaries,* U .S. Bureau of Mines, U .S. Government Printing Office, Washington. DC (annual).
3. *Minerals and Materials,* U .S. Bureau of Mines, U .S. Government Printing Office, Washington, DC (bimonthly).
4. " EMJ Markets," *Eng. Mng. J.,* McGraw-Hill (monthly).

COAL

5. *Coal Production,* U .S. Department of Energy, Energy Information Administration, U .S. Government Printing Office, Washington, DC (annual).
6. *Annual Energy Review* , U .S. Department of Energy, Energy Information Administration. U .S. Government Printing Office, Washington, DC (annual).

To present the data in concise form, use Table 1.4 or a similar format. It summarizes all pertinent statistics about a mineral commodity and at the bottom shows the calculation of apparent consumption (the actual consumption is also entered, if reported). Note that it is the change in stockpiles, where + represents withdrawal from and - an addition to stocks, that is entered. Principal producers and consumers are also identified, as are the sources of information.

Example 1.1. Complete an economic analysis for bituminous coal and lignite.

Solution . See Table 1.5.

2

STAGES OF MINING: PROSPECTING AND EXPLORATION

2.1 PRECURSORS TO MINING

Particularly al the beginning of this chapter and the two that follow on the stages and unit operations of mining, the reader will find it useful to review the corresponding introductory material of Chapter I. Section 1.4 provides an overview of the stages in the life of a mine and Section 1.5 of the unit operations. Table 1.3., for example, is especially helpful in distinguishing and relating the four stages of mining.

A detailed study of the stages in the life of a mine begins quite properly with prospecting and exploration, forerunners or precursors to mining. By that we mean they precede the two main stages of mining proper, development and exploitation. Concerned with locating and defining an ore body, they are closely related and transitional. While some classifications of mining activity draw no distinction between prospecting and exploration, it is useful to separate them in an introductory study.

It will become apparent in this book that a working knowledge of geology is presumed; it is one of our foundation sciences. In a beginning text on mining engineering, prior study of physical geology, minerals and rocks, structural geology, and mineral deposits is assumed. It is certainly prerequisite to a discussion of prospecting and exploration.

The discovery of an ore deposit has been likened to the search for the proverbial needle in the haystack. A mineral deposit is a geologic anomaly; an ore deposit is a freak of nature. The odds against a mineral deposit in western Canada evolving into a mine—of progressing successfully from stage 1 to stage 4-have been estimated as 1000 to 1, at an associated cost of prospecting and exploration for each viable discovery of the order of $30 million (Anon., 1980a). Kruger (1969) estimates the chances of discovery during a continuing prospecting program as 1 in 10 in any one year, while Cook (1983) states that only 1 property in 100 explored results in a discovery, of which only 1 in 15 results in a highly profitable mine. Using approximate, total prospecting-exploration costs of S665 million in 1982 for the 24 largest mineral-producing companies in the world, Cook further estimates that the average cost of finding a “world-class" ore (noncoal) deposit is $290 million, based on 2.3 such discoveries a year!

The degree to which the forces of nature must coalesce in forming an ore deposit is a reflection of these staggering odds. Using the crustal abundance of an element, one can compute the *enrichment factor*, the multiplier by which the concentration of the element must be increased to become commercially minable. Based on the prevailing *cutoff grade*, or minimum economic tenor of ore, enrichment factors for several common metals appear in Table 2.1. Rather surprisingly, gold, although the most scarce, does not rank at the top of the list—mercury docs—and not surprisingly, aluminum and iron require the least enrichment. Gregory (1983) also lists some cutoff grades, but limited to surface mining.

With the opposing odds 1000 or 1500 to I and costs $30 million to $300 million for a modest to major discovery, it is easy to see why prospecting and exploration have become highly specialized, sophisticated endeavors. They must be equally well-financed business ventures. Is it any wonder that the prospector and his burro have faded into obscurity—if, in fact, they ever were a viable force in exploration—or that mining is a risky business, not to be undertaken by amateurs? Because of the high risks and high costs involved, major mining companies today employ an alternative to prospecting and exploration in obtaining ore reserves: acquisition by purchase.

TABLE 2.1 Enrichment Factors for Ore Bodies

Metal	Crustal Abundance (%)	Minimum (Cutoff) Grade (%)	Enrichment Factor
Mercury	0.0000089	0.2	22,500
Lead	0.0013	4	3,100
Tin	0.00017	0.5	2,900
Tungsten	0.00011	0.2	1,800
Gold	0.00000035	0.0003	900
Molybdenum	0.00013	0.1	800
Uranium	0.00017	0.1	600
Zinc	0.0094	3	300
Copper	0.0063	0.3	50
Nickel	0.0089	0.3	35
Iron	5.80	30	5
Aluminum	8.30	30	4

Source: Peters, 1980. (By permission from Rocky Mountain Mineral Law Foundation, Boulder, CO.)

This may be accomplished by acquiring an undeveloped prospect, a producing mine, or an entire operating company. Since 1975, acquisition has been the principal means of growth exercised by major corporations in the mineral industries. In part, in the United States, this trend has been a reflection of the general corporate pattern of growth, not restricted to mineral firms. However, there is a good deal of evidence *to suggest* that the most cost-effective business strategy in mining is a combination *of both* exploration and acquisition (Cook, 1983). *All ore bodies, o f* course, whether acquired or discovered by the owner, were found as

a result of prospecting and exploration, and practically all mining companies engage *in* some activity of this type. The manner in which prospecting *and exploration* are organized and relate to each other is shown in the *accompanying flowchart (Fig.2.1).* Although the diagram is intended to portray a regional effort, it is useful for any full-scale mineral search (Payne, 1973). Notice that stage *I*, prospecting, stresses *reconnaissance, while* stage 2, exploration, focuses on *target investigation*. Each stage in turn consists of two phases, proceeding from regional appraisal to deposit evaluation. Following alternative paths leads to favorable, untimely, or unfavorable decisions, as indicated. In a major, complex project, data handling may be computerized, and operations research techniques like the critical path method (CPM) may be employed.

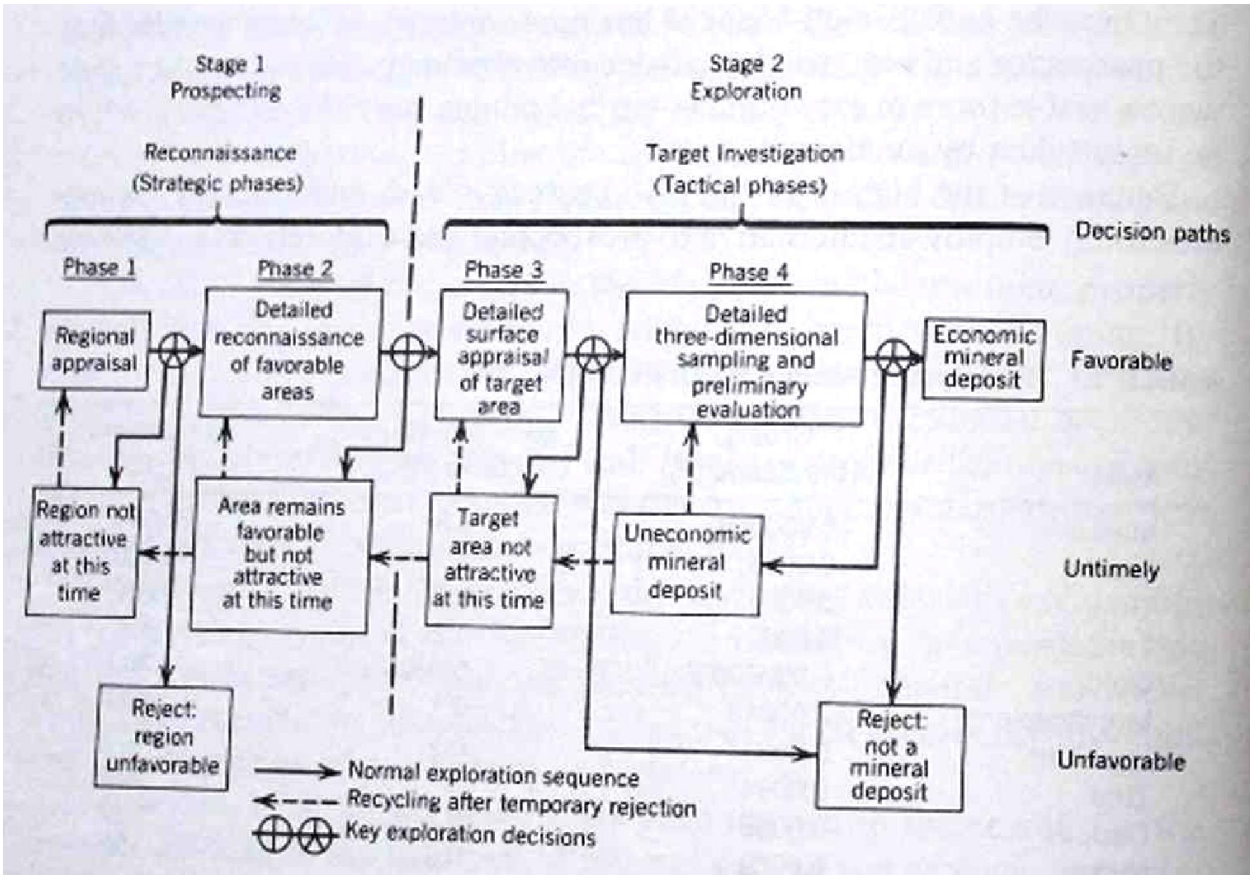


Figure 2.1. Phases in regional mineral prospecting and exploration. Sequential paths and decision points are arranged in a flowchart. (After Bailly, 1968; Payne, 1973. By permission from Society of Mining Engineers. Inc., Littleton. CO.)

It is evident in progressing from left to right through the diagram that the first objective of both prospecting and exploration is to *narrow the search,* to reduce the area under consideration from regional to areal in scope and then from target area to mineral deposit. Typically, areas decrease from 1000100,000 mi^2 (2500-250,000 km^3) in phase 1 to 1-50 mi^2 (2.5 to 125 km^2) in phases 2 and 3, narrowing finally to 0.1-20 mi^1 (0.25-50 km^2) in phase 4 (Bailly, 1966). The second objective is to *increase the favorability of the area being appraised.* In achieving these objectives, the third, *reducing the risk factor*, is automatically

attained. The progression of steps and decisions in the flow sheet dramatizes the events that ensure the attainment of all three objectives.

2.2 PROSPECTING: GENERAL

We have defined *prospecting* as the search for metallic ore deposits or commercial mineral deposits of coal or nonmetals, It is stage I, the discovery stage, in what hopefully becomes the life of a mine. Prospecting and exploration together accomplish their prime goal when they find and acquire a maximum number of mineral deposits with the potential to be ore deposits at a minimum cost and with a minimum time (Payne, 1973). More specifically, however, prospecting has as its goal the location of an *anomaly,* a geologic incongruity with the characteristics of a mineral deposit.

Prospecting may be commodity- or site-specific; that is, the search may be limited to a specified mineral (or type of mineral) or to a particular area (or country or region). Today that is seldom the case. The mineral industries are populated with multinational corporations, often energy-oriented as well. While some companies engage only in prospecting and exploration, most are vertically integrated through development and exploitation. Some process, refine, and fabricate mineral products as well. Thus it is common for the targets in prospecting to encompass multiple sites and a wide suite of minerals, sometimes spanning metals, nonmetals, and fuels.

If a company plans to initiate a regional prospecting effort, Kruger (1969) recommends the use of a quantitative indicator such as value of mineral production per unit area (e.g., $/mi^2, or S/km^2) to target promising countries. His reasoning is that countries, often LDCs, with the least developed mineral industries are the best candidates for a regional search. In rank order of preference, selected countries in this category arc Brazil, Australia, Zaire, Canada, Mexico, and China (in the United States, the most promising state is Alaska). Large-scale regional studies are rarely undertaken without government or consortium involvement.

Decisions as to commodity and site arc part of a general plan for prospecting-exploration and are dependent on a variety of factors. They include market conditions, both price and supply and demand projections; commodity usage as to competition and substitutes; corporate objectives for exploration, production, and growth: favorable geologic and geographic conditions: and suitable business and political climates. These factors can only tenuously be understood at this preliminary stage, but they will recur in subsequent appraisals throughout the life of the mine.

In populous, developed countries like the United States, ore bodies with surface expressions (outcrops, gossan, float) have, by and large, already been discovered. The task of prospecting in such countries then is to locate so-called hidden ore bodies, an incredibly difficult assignment. In these instances, all conceivable sources of information, such as geologic maps, satellite photographs, old reports and documents, and technical literature, should be assembled. A study of this information along with knowledge of the existence o known mineral

deposits and old mines in the area help predict favorable loci that can be tested by prospecting.

Because the usual targets today are concealed, direct search techniques (physical or visual examination, geologic study and mapping, sampling) must be supplemented by *indirect methods* (geophysics, geochemistry, geobotany). The locale o f their application may be the atmosphere (or space), the surface o f the earth, or the subsurface, as determined by range and definition Direct techniques are still successful in prospecting for coal and nonmetallic deposits (hat often outcrop or occur under shallow overburden, but for metallic deposits, indirect techniques are almost always required.

Generally, prospecting procedure follow s these steps:

1. Search reports and technical and published literature.
2. Study available geologic and surface maps.
3. Study aerial and satellite photographs.
4. Prepare photogeologic map from available information and new aerial data.
5. Conduct airborne geophysical survey of area under study.
6. Establish base of operation, set up mapping control, and organize ground prospecting parties.
7. Conduct preliminary ground geologic, geophysical, and/or geochemical surveys.
8. Assemble and analyze findings.

Phase I proceeds through step 4 or 5; phase 2 encompasses the rest. Certain steps may be omitted or performed out of sequence. Further, overlap between prospecting and exploration may occur in steps 7 and 8 . as shown in Figure 2.1.

A useful summary of all the methods that may be employed in both prospecting and exploration is given in Table 2.2. The phase(s) in which they normally find use. their class as to direct or indirect, and their detection capability (for base metals) are indicated. Many of these methods will be discussed further in the articles that follow. Continue to bear in mind that most methods have application for both prospecting and exploration.

TABLE 2.2 Compilation of Prospecting and Exploration Methods

Methods and Techniques	Usable at Phase: 1 Regional Appraisal	2 Detailed Reconn.	3 Detailed Surface Study	4 Detailed 3-D Study	Detection Capability for Nonferrous Metallic Deposits[a]: Direct Detection: Good	Direct Detection: Questionable	Indirect Detection: High Discrimination Capability[b]	Indirect Detection: Low Discrimination Capability
Geologic								
Office compilation	X							———
Photogeologic study	X	X						———
Aerial examination	X	X						———
Outcrop examination	X	X	X		———		———	
Geologic mapping and investigations	X	X	X	X	———		———	
Geologic logging				X	———			
Boulder tracking		X	X		———			
Geochemical								
Stream sediment sampling	X	X					———	
Water sampling	X	X						———
Rock sampling	X	X	X				———	
Specialized sampling	X	X	X					———
Assaying	X	X	X	X	———			
Geophysics—airborne								
Aeromagnetic surveys	X	X						———
Electromagnetic	X	X						———
Radiometric surveys	X						———	
Remote sensing surveys	X							———
Geophysics—ground								
Gravity	X	X	X					———
Magnetic	X	X	X					———
Radiometric		X	X				———	
Seismic		X	X					———
Resistivity		X	X					———
Self-potential		X	X					———
Induced polarization		X	X					———
Down-hole electrical				X				———
Sampling and evaluation								
Trenching			X	X	———			
Rotary drilling				X	———	———		
Core drilling				X	———			
Tunnel–shaft work				X	———			
Mineral dressing tests				X	Not a detection method			
Economic evaluation				X	Not a detection method			

* *Detection* refers to the ability to detect a deposit if it is there *Indirect detection* refers to a geological, chemical, or physical response, showing a deposit may be the cause of the response; this is in opposition to direct evidence of the presence of a deposit
* *Discrimination* with regard to indirect methods refers to the ability to determine if a certain response (anomaly) is due to a deposit or to another cause.

Source Bailly. 1966; Payne. 1973. (By permission from Society of Mining Engineers. Inc.. Littleton. (CO)

2.3 PROSPECTING: GEOLOGIC

Geologic prospecting applies knowledge of the genesis and occurrence of mineral deposits, structural mapping , and mineralogic and petrographic analyses to discover, define, and appraise mineral prospect. It remains the keystone of both prospecting and exploration; all other methods depend on it and utilize if for *interpretation.*

The applications of geology to mineral search include the derivation of *target* concepts, the collection of geologic data obtained during any phase of the prospecting - exploration endeavors (mapping, alteration and zoning studies core and borehole logging, etc.), the interpretation of geologic data collected, the integration of all other data (geophysical, geochemical), and *the formulation* of judgments to guide the search and evaluation (Payne,1973).

As an example, the geologist through photo interpretation, surface mapping, and detailed geologic examinations produces diagrammatic plans of the portion of the earth being studied, showing the composition, distribution, age, and relationship of the various rock units (Anon., 1980a). From these data, he interprets the structures and sequence of geologic events. By mapping, he notes evidences of mineralization, alteration, and other geologic features that might indicate an ore body. With care, he can often extrapolate observed geologic features through the overburden or adjacent formations to hypothesize the existence of hidden ore bodies within the earth. Finally, he conceptually develops a geologic model of the ore body, useful for reserve estimate and mine planning purposes (Dowis, 1982).

The use of a classification of mineral deposits is imperative in the application of geologic prospecting. Figure 2.2 depicts one time-honored scheme. It is useful to the mining engineer because it is based on both genetic and economic principles (Lewis and Clark, 1964). The classification assumes a magmatic origin of minerals. Deposits are either *prim are* (formed directly from magmas), *secondary* (altered through chemical or mechanical weathering), or *metamorphic* (formed from other minerals and rocks when subjected to great heat and pressure). Further, those formed contemporaneously with the host rock are *syngenetic; if* formed subsequently, they are *epigenetic* in origin. Another helpful feature of the classification is that it includes all minerals, metals, nonmetals, and fuels) and conforms to mineral economics terminology. For a comprehensive treatment of mineral deposits, see Jensen and Bateman.

A knowledge of the processes forming mineral deposits is valuable to the prospector in seeking favorable loci of occurrence. Callahan (1987) notes the following as promising sites or environments: erosional unconformities, tonic intrusions, enrichment and alteration zones, overthrust zones where tectonic plates collide, stratiform deposits in volcanics and sediments, and the interface between the ocean and its floor.

Data compilation and display are crucial steps in geological prospecting. Essential to both is the computer, which has revolutionized information handling and promises to reshape prospecting-exploration strategy equally. Mapping capabilities have already been vastly extended and improved; maps remain basic

format of geologists to assemble and display their data. The same maps may provide the mining engineer with base drawings for development and exploitation of the ore body.

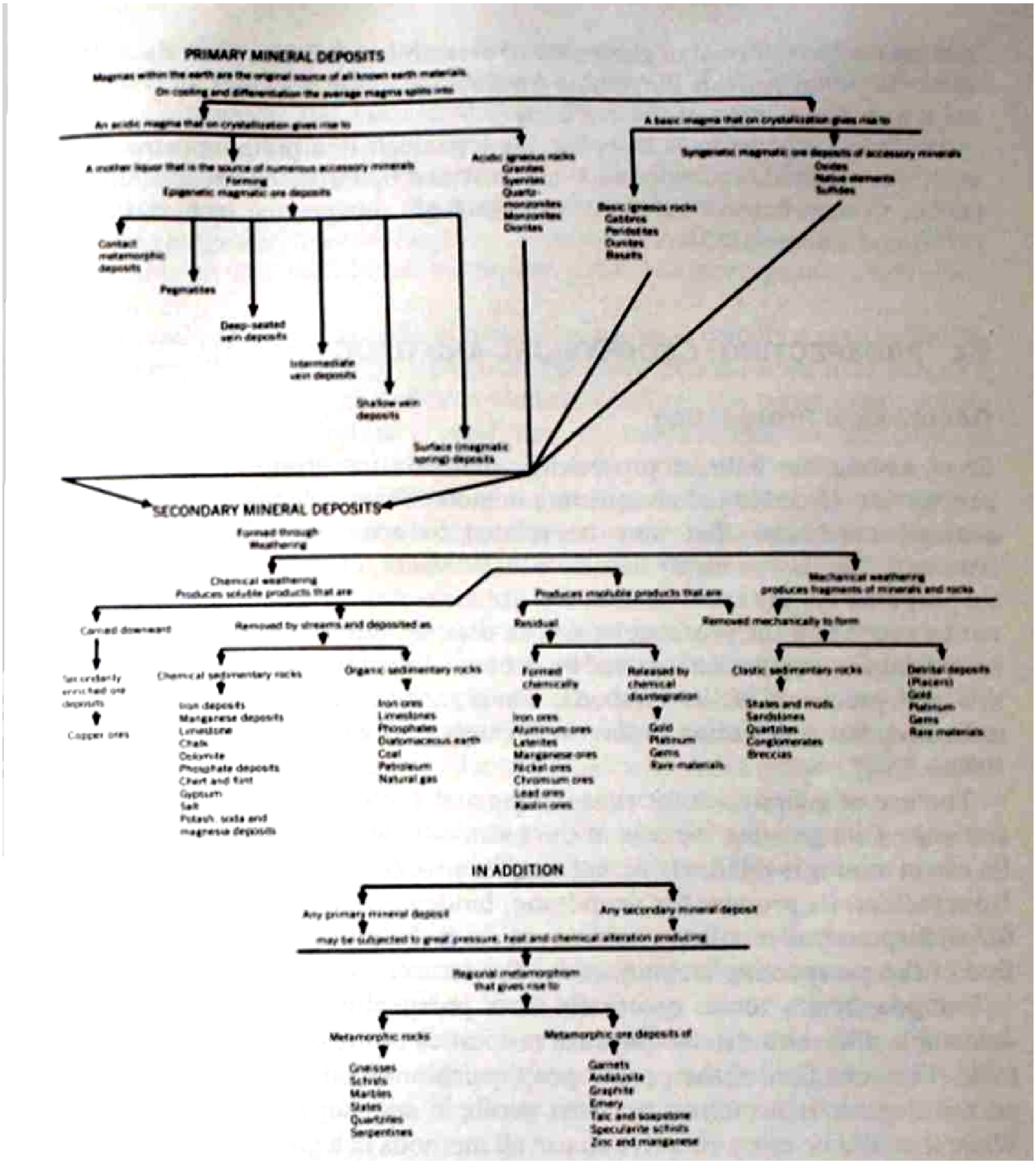


Figure 2.2. A classification of mineral deposits. (After Lew is and Clark, 1964. Copyright 1968, John Wiley & Sons. Inc., New York.)

Procedures for geologic mapping, photogeologic interpretation, structural analysis, and field reconnaissance are provided by Payne (1973) and Ahrens (1983). Comprehensive references on geologic prospecting include Peters (1978) and Lacey (1983).

2.4 PROSPECTING GEOPHYSICAL AND GEOCHEMICAL

Geophylsical Prospecting

Chief among the indirect prospecting methods (or group of methods) is geophysies. *Geophylsical Prospecting* detect ''from a distance" changes in geologic conditions that may be related to economic mineral deposits (Payne, 1973). Using highly sensitive instruments, change* in the geophysical properties of (he earth (anomalies) are measured Anomalies may or may not be caused by the presence of an ore deposit. but sophisticated, computerized data interpretation assisted by geologic inference improve* the resolution and precision of the methods. Their range in air or space is almost unlimited, but penetration in the earth rarely exceeds 300 ft (90 m) (Anon., 1980a**)**

The use of geophysics for prospecting and exploration had its beginning and enjoys its greatest success in the petroleum and natural gas industries, lit u*e in mining i* relatively recent, and its success somewhat more modest. Nonetheless, its promise for deep-lying, hidden ore bodies and its potential for widespread, diversified searches makes it the most versatile and attractive of the prospecting techniques for the future.

For geophysics to be successful as a prospecting tool, there must be detectable differences in the physical properties of the ore body and the host rock. The selection of the proper geophysical methods(s) to use for a suspected deposit is based on the best geologic and physical data available. While it would be most effective to use all methods in a given search, it is too costly and time-consuming The usual compromise is to try methods in the order of their assumed appropriateness (singly or jointly), based on field knowledge or if possible laboratory samples of the host rock and alleged deposit. In some cases, geophysics may be employed for indirect detection, using the principle of association; that is, a gangue mineral associated with the deposit may be more detectable than the ore mineral itself. Also valuable information for geologic mapping that defines deposit spatial relations (depth, shape, etc.) or structural features (folds, faults, etc.) may be supplied by geophysics.

The selection and sequence of use of geophysical methods is determined in part by their suitability for different locales (i.e., airborne or space, ground surface, or subsurface). For (his reason, some are preferred for prospecting (air or ground) and others for exploration (ground of subsurface) Commonly, the most cost-effective scheme is to package several methods for airborne surveys and then to repeat the process with other methods suited to a surface or subsurface locale. One method is limited to air or space (remote sensing), three are applicable cither in the air or on the ground ***(magnetic, electromagnetic,*** and ***radiometric),*** and the others are applicable to the surface or subsurface only ***(seismic, gravity,*** and ***electrical).***

The end product of geophysical measurements is usually a map or profile portraying the variation of a physical property or measured field across a suspected deposit. In an airborne survey, readings are taken continuously along flight lines: but on the ground, they are made at stations located along profile or grid lines. Locations must

be determined with some accuracy to permit meaningful interpretation of the geophysical data. Examples of profiles resulting from ground surveys over a copper sulfide ore body are shown in Figure 2.3: notice the correlation of the anomaly in the data for three geophysical methods.

Geophysical methods arc usually classified on the basis of the principle or property measured. Table 2.3 lists the seven main methods and numerous variations currently in use for prospecting and exploration of mineral deposits. the parameter and characteristic measured, causes of anomalies, applications, coverage, cost, and interpretation (Van Blaricom, 1980). While all methods may be utilized in mining, several have had more extensive application than the others. Four of the more useful arc pictured in Figures ***2*** 4 through 2.7 and discussed briefly below (Anon., 1980a).

Electrical: Self-Potential (Fig. 2.4). Groundwater can act on a massive sulfide ore body to produce a weak electric charge (battery action). Systematic measurements of voltages at the surface may show a significant change when massive sulfide mineralization is present beneath the surface. Electrical conductivity is the property of the earth measured. The self-potential method has also been used in the search for magnetite, cobalt minerals, graphite, and anthracite.

Electrical: Induced Polarization (Fig. 2.5). A field of electricity can be created in the ground by passing a measured amount of electric current through it using two electrodes and a generator. By measuring the voltage caused by this field with a second pair of electrodes a known distance away, the geophysicist can calculate the electrical property of the ground known as resistivity. Where metallic minerals are present, even in concentrations as low as 0.5%, the ground can also become charged by the electric field. This charging phenomenon is called induced polarization (IP) and can be measured in several ways. Used mainly with sulfide and oxide ore bodies, IP can also locate water and deposits of sand, gravel, and petroleum.

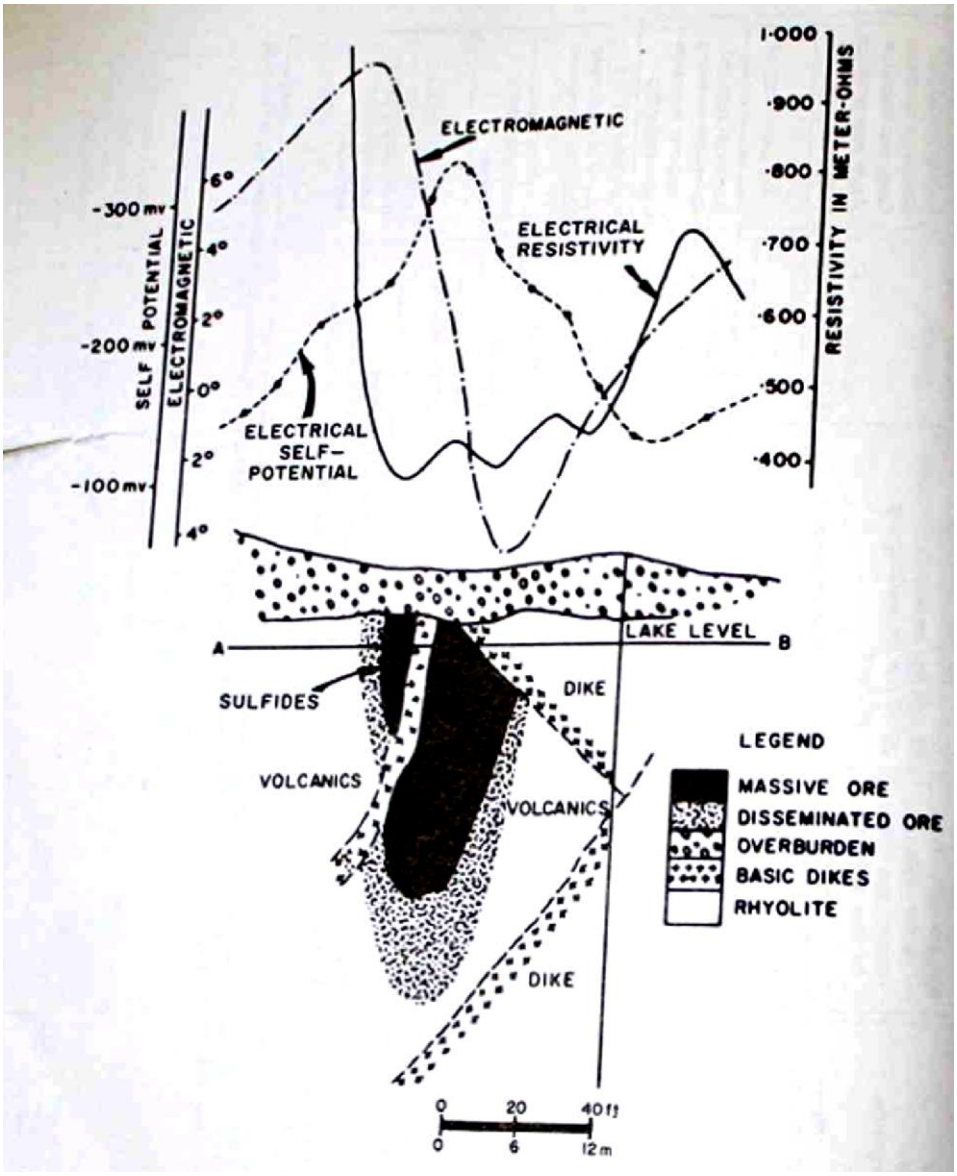


Figure 2.3. Geophysical measurements made by three different methods over a copper sulfide ore body. (After Bruce, 1982. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

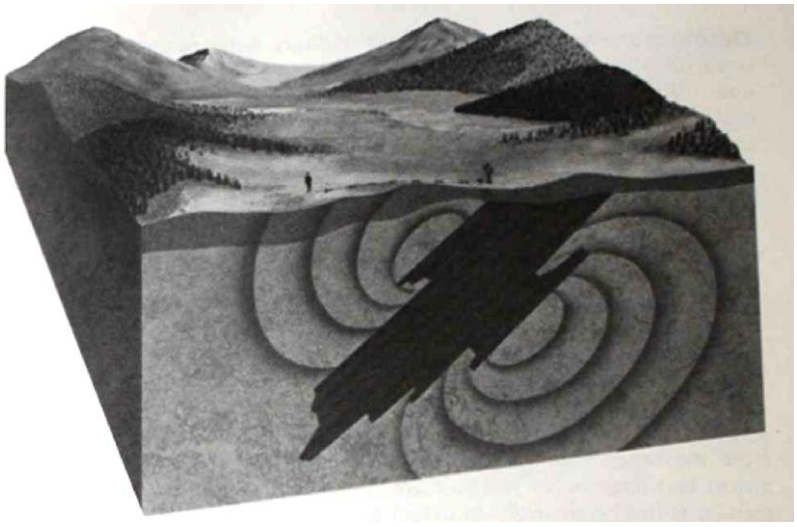


Figure 2.4. Electrical self-potential method of geophysics. (After Anon., 1980a. By permission from Placer Development Ltd.. Vancouver. BC. Canada.)

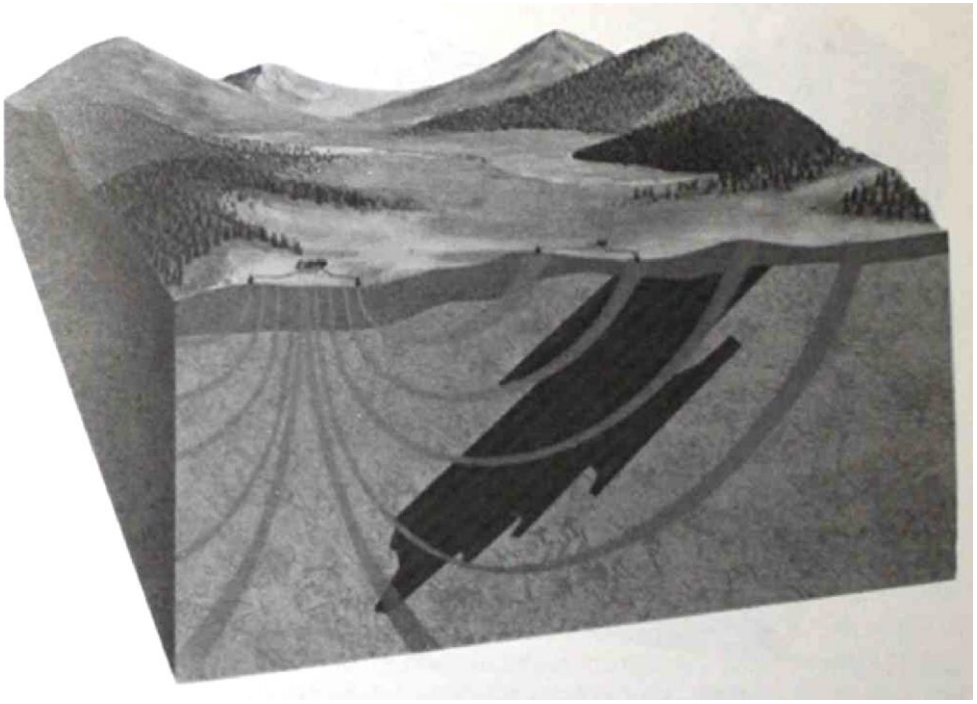


Figure 2.5. Electrical induced-polarization method of geophysics (After Anon.. 1980a. By permission from Placer Development Ltd.. Vancouver. BC. Canada.)

Electromagnetic (Fig.2.6.). Powerful military radio stations around the world use VLF (very low frequency) electromagnetic waves for communication with distant submarine fleets. In Figure 2.6. the VLF signal is represented by the primary waves. Certain geologic structures, such as fault zones,which may be mineralized, will produce measurable secondary waves. The electromagnetic method utilizes induction coupling suppressed by resistivity methods. A transmitter coil energizes the ground, and a receiver coil detects the signal. The target sought, either from the air or ground, is a sulfide ore body, graphite, or associated pyrite or pyrrhotite.

Gravity *(Fig.2.7).* The earth s gravitational pull changes by very small amounts depending on altitude, latitude, and the presence or absence of dense rock formations under the point of measurement. The instrument which detects changes in gravity- a gravimeter-can register a variation in gravity on the order of one part per hundred million, which means that a measurable change can be noted when the instrument is raised or lowered a mere two inches. Figure 2.7 indicates how a dense area of mineralization will distort and measurably increase gravitational pull. In mining, the gravity method *is* used principally to detect and help identify geologic structures such as faults, anticlines, salt domes, intrusions, and buried channels.

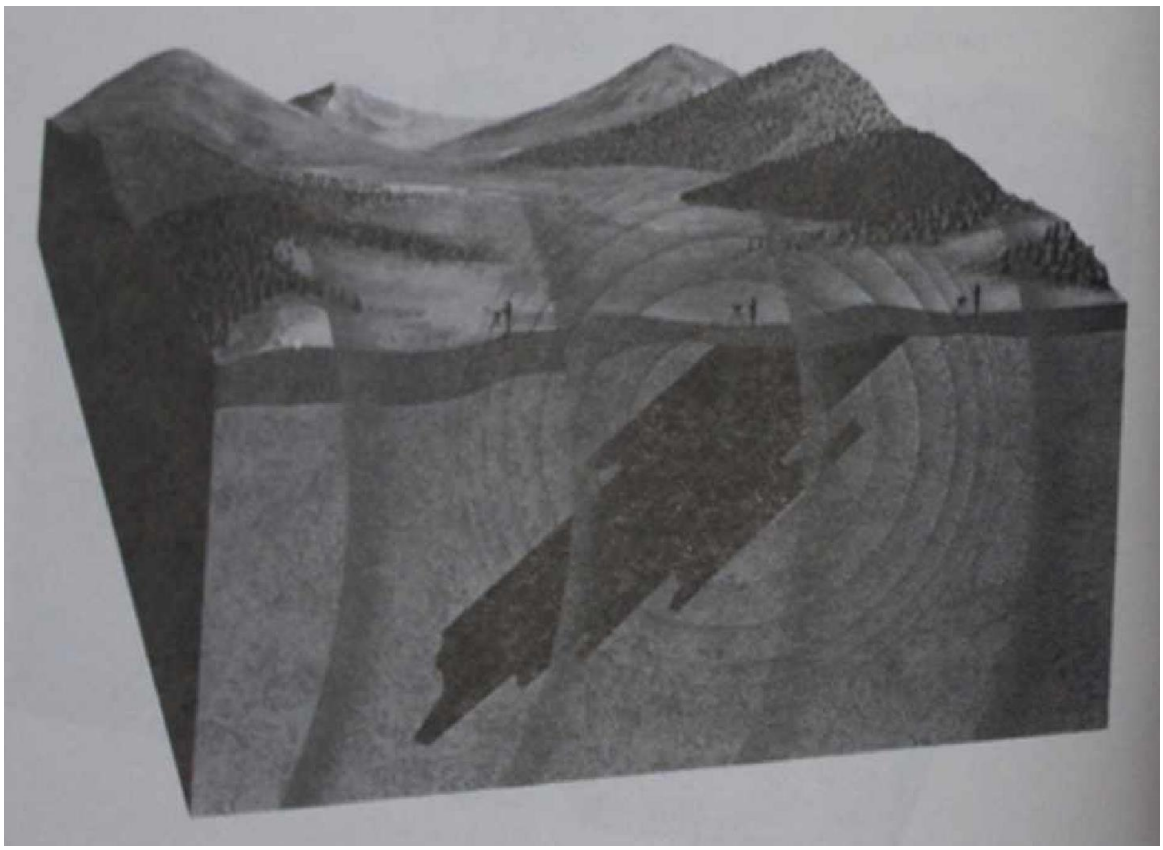


Figure 2.6. Electromagnetic method of geophysics. (After Anon.. 1980a. By permission from Placer Development Ltd.. Vancouver, BC. Canada.)

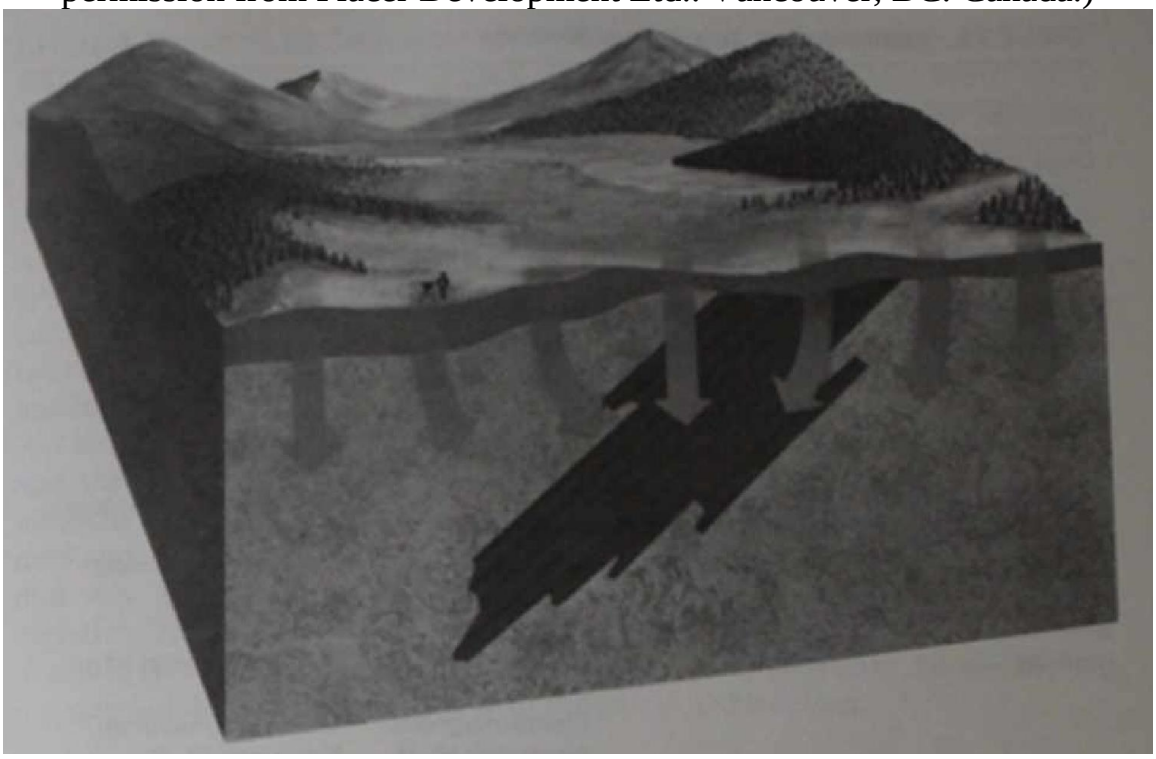


Figure 2.7. Gravity method of geophysics. (After Anon.. 1980a. By permission from Placer Development Ltd , Vancouver, BC. Canada)

Geophysical Applications. Mining engineers ordinarily have no need to understand the intricacies of geophysical theory, although they may help conduct prospecting or exploration surveys with geophysical instruments. They clearly must be able to interpret and use the results of geophysical surveys, however, and should be knowledgably of the application of different methods and the reasons for their choice. Table 2.4 summarizes the applications of geophysical methods that are suited for direct and indirect detection of mineral deposits and related geologic studies.

For more detailed discussions of geophysics applied to mineral prospecting and exploration, see accounts by Payne (1973), Smith (1977), Thomas **(1978),** Brant (1980), Van Blaricom (1980), and Bruce (1982).

Geochemical Prospecting

A newer earth science than geophysics, geochemistry has enjoyed only recent popularity (since the 1950s) in the search for mineral deposits. ***Geochemical prospecting*** detects minute changes caused by the near presence of ore bodies, usually metallic, in the chemical composition of samples of air, groundwater, soil, or botanical specimens (Thomas, 1978). While few ore bodies have been discovered by geochemistry alone, it is a valuable adjunct

TABLE 2.4 Application of Geophysical Methods

Deposits	Direct Detection	Indirect Detection	Geologic Studies, Structures, Etc.
Disseminated sulfide	Electrical: IP, R, SP	Magnetic Radiometric (alteration) Gravity	Gravity (basin and range studies) Magnetic (detection of intrusive, mapping) Electrical: MT (basement studies)
Massive sulfide	Electrical: IP, R, SP, MT Electromagnetic Magnetic Gravity	Gravity Magnetic Electromagnetic	Magnetic Electromagnetic
Lead–zinc (Mississippi Valley type)	Electrical: IP, R, MT Electromagnetic	Gravity Magnetic Seismic	Gravity (basement topography) Magnetic (basement topography) Seismic
Uranium	Radiometric	Radiometric Electromagnetic Electrical: IP, R Magnetic	Electromagnetic (structure) Magnetic (mapping) Electrical: IP, R Magnetic
Coal	—	—	Seismic Electrical: R Electromagnetic
Laterite (iron, aluminum)	Electrical: MT, R Seismic	Gravity Magnetic	Magnetic (mapping)
Placer (steam channel)		Magnetic (black sand) Electrical: IP, R (black sand)	Seismic (stream channel location) Electrical: R (clay detection)
Gravel deposits	Seismic Electrical: R		
Water	Electrical: R, MT Electromagnetic Seismic	Seismic (aquifer location) Gravity (basement studies)	Magnetic
Overburden depth studies	—	—	Seismic Electrical: R Gravity
Location of buried features	—	—	Electrical: AP Magnetic Electromagnetic Gravity (void, cavern)

Symbols: AP, applied potential; IP, induced polarization; MT, magnetotelluric; R, resistivity; SP, self potential.

Source: Modified after Van Blaricom, 1980. (By permission from Northwest Mining Association, Spokane, WA.) (Data source: Geoterrex Ltd. of Canada.)

to direct search by geological prospecting and to indirect search by geophysical means. An indirect method itself, geochemical prospecting is carried out on the ground. Geochemistry has often been most rewarding where geology and geophysics have been relatively ineffective, particularly for primary reconnaissance of very large areas of remote terrain (Hawkes, 1973).

Through systematic collection and precise trace analysis of appropriate samples, geochemical anomalies can be detected. When interpreted properly and integrated with geologic data, they arc useful in indicating or helping to confirm promising target areas. Sample analysis must be done micro quantitatively (in parts per million) because of the high background levels of many indicator elements.

The different applications of geochemical prospecting and exploration and the phases in which they are used appear in Table 2.2. Target ore minerals are mainly the sulfides of copper, lead, zinc, nickel, and molybdenum and. to a lesser extent, uranium, tungsten, tin, mercury, gold, and silver minerals In general, geochemistry is not used in the search for coal, nonmetallic**,** gemstones, bauxite, or ores of iron, manganese, chromium, and titanium (Hawkes, 1973). Useful for reconnaissance purposes, however, it is not dependable for pinpointing drill targets in exploration.

Other Prospecting Tools

Finally, mention should be made of two ancillary search techniques employed for qualitative, reconnaissance purposes. With some similarities to geochemistry, *geobotanical prospecting* employs changes in the patterns of vegetation growth and foliage of an area as a visual or analytical guide to mineralization (Brooks, 1983). Inexact and only an indicator unless geochemical methods are also used, geobotany can still be a useful adjunct to the prospector's kit of tools. Reportedly, it has helped reveal the presence of uranium deposits.

Geothermal prospecting is sometimes classified as a geophysical method, measuring thermal gradients within the earth. As such, it can detect the presence of mineral deposits having a geothermal anomaly (e.g., volcanics. hot springs, recent magmatics, alteration zones, etc.), but it is more likely to be used in the search for geothermal energy sources.

2.5 EXPLORATION: GENERAL

If the goal of prospecting is to locate anomalies due to mineral deposits, then the goal of exploration is to define and evaluate them. We said in Section 1.4 that ***exploration*** determines the geometry, extent, and worth of an ore deposit using techniques similar to but more precise than those in prospecting. As stage 2 in the life of a mine, exploration continues the search process through the tactical phases of detailed appraisal and evaluation, culminating in preparation of a feasibility report that accepts or rejects the deposit(s) under study (Fig. 2.1).

Exploration employs some unique techniques of its own, but for the initial field work of phase 3 continues to rely on the two main approaches utilized for prospecting, geology and geophysics. (Table 3.2). Geochemistry and geobotany have little application beyond phase ***2.*** The specific methods described in the previous m sections are applicable here as well.

There are however, some distinct differences between prospecting and exploration:

1. Locale***s*** As the search area diminishes and the favorability improves, the locale shifts from air (or space) to the ground surface and subsurface. Airborne geophysics is replaced by ground geophysics, geology is increasingly subsurface oriented, and additional subsurface exploration techniques are utilized.
2. ***Physical samples.*** As the site shifts from surface to underground, indirect search methods give way to direct methods that provide physical evidence. Because most ore bodies sought today are hidden, subsurface excavation methods to obtain actual mineral samples must be employed. The most commonly used is drilling.
3. ***Data.*** To diminish risk during the exploration stage, much more substantial information about the target deposit is required. Data are characterized by greater precision, specificity, and certainty.

Generally, the progression of steps in exploration is as follows. First, the favorable area identified by prospecting must be delineated by exploration techniques, Second, once located, the mineral deposit is sampled thoroughly and impartially and the samples analyzed. Third, the sampling data are utilized to prepare an estimate of tonnage and grade (extent and value) from which the present worth can be calculated and recommendations made regarding the feasibility of mining.

In dealing with economic evaluation of mineral deposits, certain terminology has acquired accepted usage. For the discussion here, we will use the definitions adopted jointly by the U.S. Bureau of Mines (USBM) and U S Geological Survey (USGS) (Anon., 1984b). The main concept to grasp is the distinction between resource and reserve. A *resource* is a concentration of naturally occurring material that is potentially economic to extract. A *reserve* IS a portion of a resource that can be economically extracted at the to present time. Notice that the two terms occupy the same relationship to one another as a similar pair of terms, mineral deposit and ore deposit; that is, all reserves are resources, just as all ore deposits are mineral deposits.

Figure 2.8 is a helpful classification to visualize the relationship between resources and reserves. The diagram is arranged such the degree of certainty of existence of given resources increases horizontally to the left,

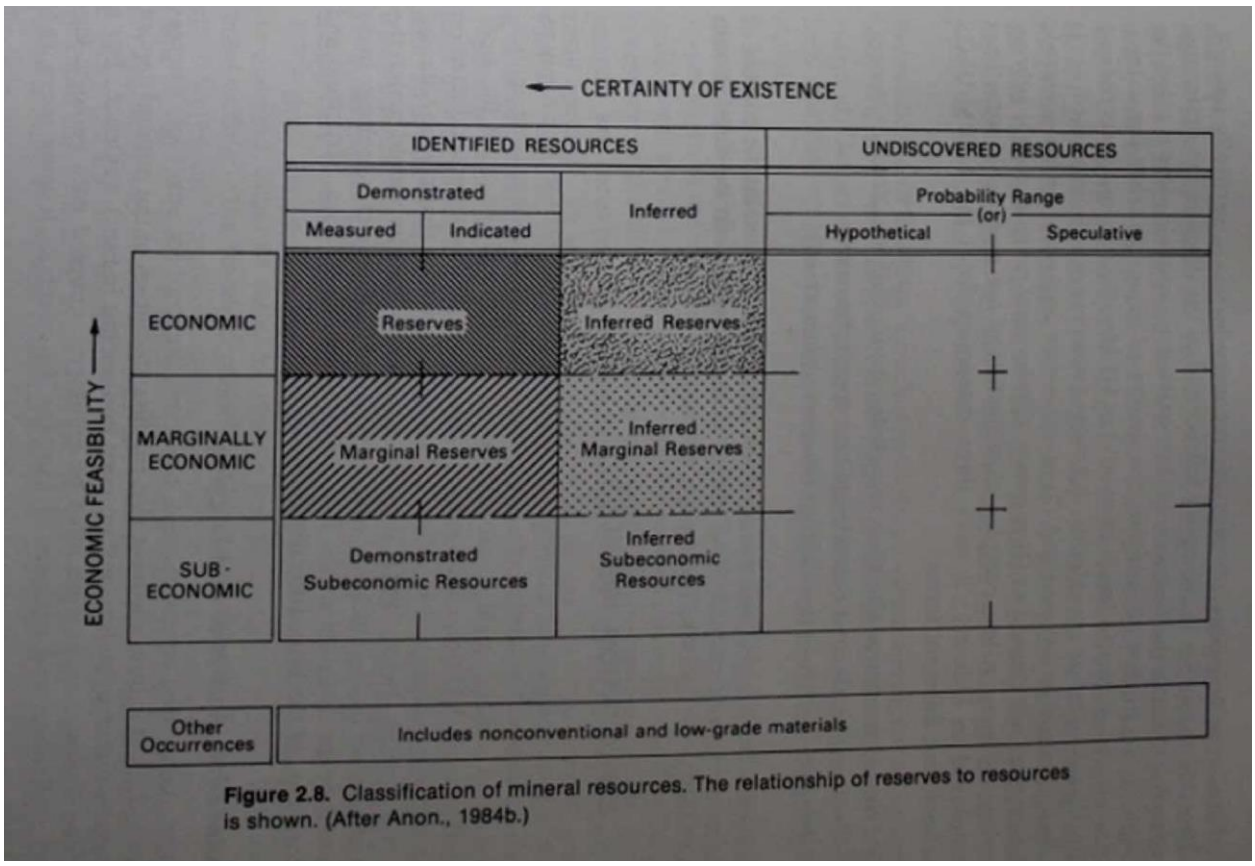


Figure 2.8. Classification of mineral resources. The relationship of reserves to resources is shown. (After Anon., 1984b.)

whereas the feasibility of economic recovery increases vertically upward. Thus the resources in the upper left portion of the diagram are designated reserves because the certainty of their existence and their value is rated the highest. To further differentiate the quality of resources, they are categorized (from greater to lesser certainly) as (1) identified or undiscovered and (2) hypothetical or speculative. Likewise, reserves are distinguished as demonstrated or inferred and measured or indicated; on an economic basis, they are classed as reserves, marginal reserves, or subeconomic reserves. A less complex classification of ore reserves is sometimes employed in industry; it uses three categories from the USBM-USGS scheme but different terms;

Proven: measured (detailed knowledge from complete direct evidence)
Probable: indicated (knowledge less comprehensive)
Possible: inferred (knowledge involves assumptions)

Sometimes the distinction among these categories is based on the number of sides of the ore deposit (two, three, or four) which can be defined with certainty (e g., by outcrop, trench, drillhole).

2.6 EXPLORATION METHODS

While the geologic and geophysical methods used for prospecting apply as well for exploration during phase 3, modifications are made in their application to fit shifting locales, more focused targets, and the need for more reliable data.

Additional field surveys using both methods are conducted on the surface as necessary to refine and complete the data that can be obtained in this fashion (Beck, 1981). By now, exploration results should match the expectations and confidence requirements of the project. Geologic maps should begin to establish the first approximate boundaries, structural and economic, of the hoped-for ore body.

Excavation, Analysis, and Logging

When exploration enters phase 4. however, the need for new tools emerges (Table 2.2). It is now apparent that the final definition of the mineral deposit requires direct evidence and/or very convincing indirect evidence. For a hidden deposit, excavation must be employed, coupled with geologic logging, collection and analysis of samples, or borehole logging using geophysics.

Geologic Logging. Geologie logging in the field is largely a visual process carried out on drill core, drill cuttings. or excavation debris. Contamination and loss are major threats to guard against. If drill core is collected, it is placed carefully in a core bo* in 5-ft (1.5-m) lengths and exactly in the sequence it is removed from the hole. The core may then he split longitudinally, one-half being preserved for geologic study and one-half used for analysis. A pick and magnifying glass may be sufficient accessories for a skilled geologist to make petrologic and mineralogic identifications and rough distribution estimates. More accurate results require laboratory analyses and microscopic studies of thin and polished sections. Soil and rock mechanics information for pit or mine design and permitting generally depends on laboratory testing also.

Excavation and Drilling. Conclusive evidence of the existence of an ore body requires that samples representative of the whole be collected and analyzed. Details of the sampling program are held for later; suffice it to say here that the samples must be extensive, reliable, and statistically meaningful.

Mineral samples are obtained by excavation or drilling. If the deposit is exposed at the surface or occurs under shallow depth of burial, an opening is *excavated* from the surface in the form of a channel, pit, or trench. For large- scale work, a bulldozer or similar machine is used, preceded by blasting if necessary. If (he deposit occurs at moderate or considerable depth, then samples arc recovered by drilling holes from the surface or excavating a small opening—an adit, tunnel, or shaft—to the ore body (which may serve later for mine development purposes). Underground openings arc sunk by specialized mining equipment; see stage 3, development (Section 9.4).

Drilling small circular holes from the surface into the deposit is by far the most expeditious and economical means of sampling an ore body that extends to depth. Over 95% of mineral sampling today is performed by drilling, which has become highly specialized and virtually all contracted (exceptions are underground

exploration and large companies, situations in which the drilling may be done by in-house crews).

The three common drilling methods employed for exploration purposes arc diamond, auger or roller-bit rotary, and percussion (Section 4.2). ***Diamond drilling*** is the standard procedure for consolidated rock, moderate to great depths, and conditions requiring an intact core sample (as opposed to chips, sludge, or cuttings) for geologic logging and analytical determinations. A valuable adjunct to core drilling is the information it provides on structural geology and for rock mechanics. The cutting action is rotary (today often hydraulically actuated), accomplished by many diamonds mounted or impregnated in the tungsten carbide bit matrix. Drill cuttings are flushed from the hole by water, air, or mud. Variations on the standard method are a wireline core barrel for retrieving the core without pulling the bit and drill rods and a retractable bit that can be changed without pulling the rods. Hole sizes range from 1-6 in (38-165mm), with small sizes standardized as follows:

Series X Standard (Q Wireline Barrel, T Thin-Wall Rods)

CORE SIZES	BIT SIZES	HOLE SIZES
EX $\frac{7}{8}$ in. (22.2 mm)	$1\frac{7}{16}$ in. (36.5 mm)	$1\frac{1}{2}$ in. (38.1 mm)
AX $1\frac{5}{16}$ (29.4)	$1\frac{13}{16}$ (46.8)	$1\frac{7}{8}$ (47.6)
BX $1\frac{5}{8}$ (41.3)	$2\frac{5}{16}$ (58.7)	$2\frac{3}{8}$ (60.3)
NX $2\frac{1}{8}$ (54.0)	$2\frac{15}{16}$ (74.6)	3 (76.2)

The larger sizes are more expensive hut yield better core recovery, of prime concern in exploration (80-95% is desirable); NQ is the most popular size and series. A typical surface drill rig installation is shown in Figure 2.9; the hypothetical, hidden deposit being explored is the same one depicted in figures 2.4 to 2.7 during stage I. prospecting. Components of a surface diamond drill rig arc shown in Figure 2.10.

Rotary drilling is employed for noncoring exploration in shallow holes in soft to medium-hard ground, cither rock or soil. The usual maximum depth is a few hundred feet (about 100 m). but holes to 3000 ft (900 m) have been drilled (oil wells have reached 25.000 ft. or 7500 m. using the rotary method)

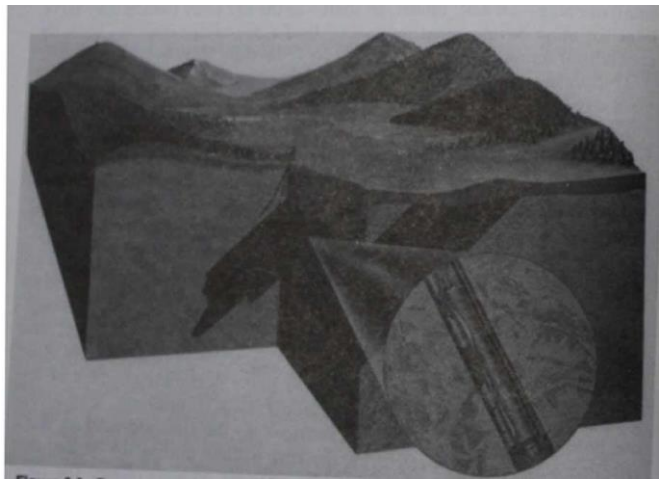


Figure 2.9 Exploration drilling. Inset shows details of a diamond drill bit and core barrel. (After Anon, 1980a. By permission from Placer Development Ltd, Vancouver, BC, Canada)

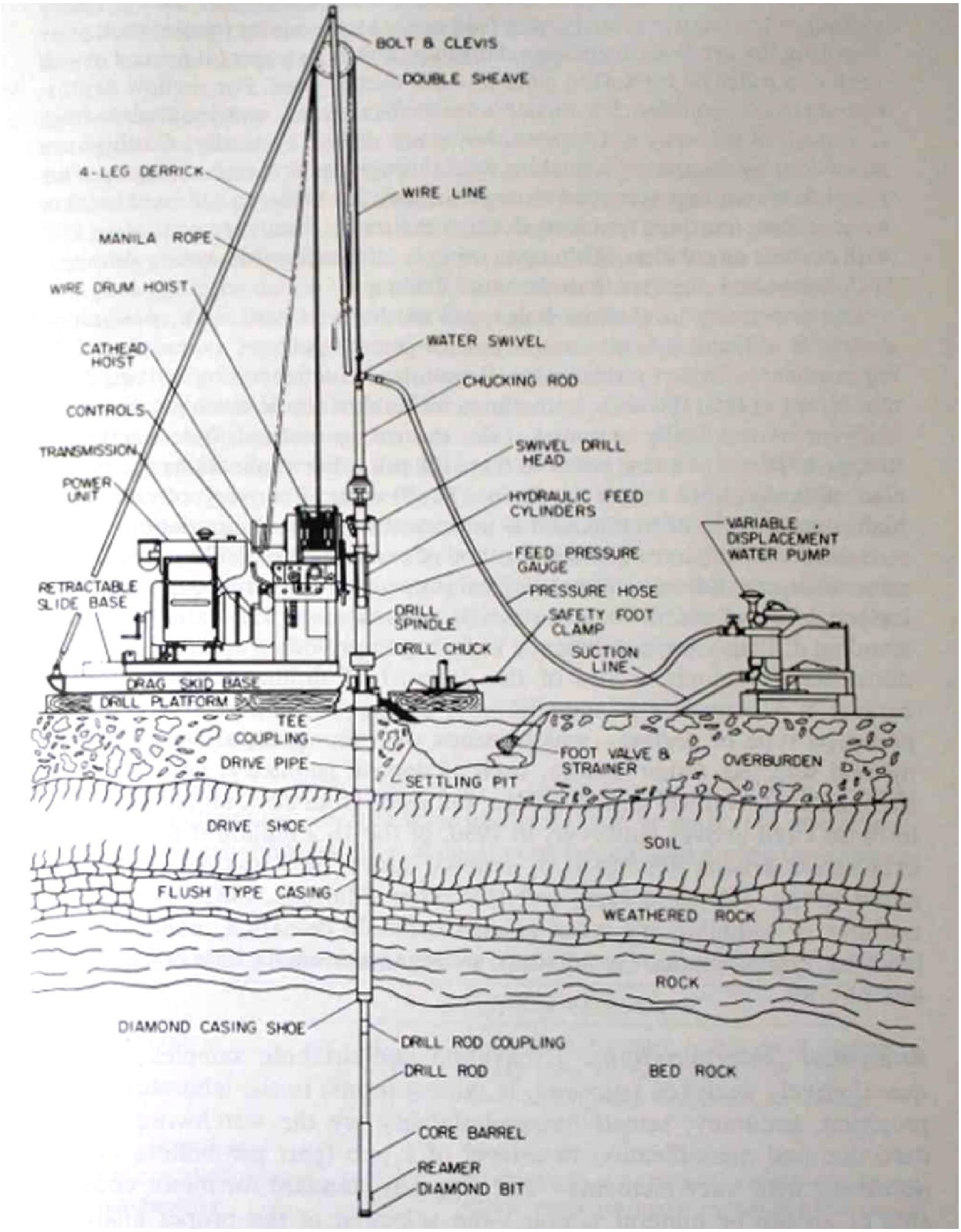


Figure 2.10. Diamond drill rig with mechanical drive, used for exploration from the surface. (By permission from Acker Drill Co. Inc.. Scranton, PA.)

Hole size is 3-6 in. (89-152 mm). One of the uses of rotary drilling is to penetrate overburden (soil or rock) or country (waste) rock prior to coring the ore body by diamond drilling. Either an auger bit for soil or soft rock or a roller bit for soft to medium – hard rock is used. For shallow depths, rigs are truck – mounted; for deeper holes, self – powered, crawler – mounted rigs are used. With some

exceptions, holes are drilled vertically. Cutting are recovered by circulating a flushing fluid through the drill rods, using conventional flow (cuttings removed through the annulus between rods and hole) or reverse flow (cutting removed through the rods). Recovery can attain 95 % with reverse circulation. Although coring is seldom feasible, rotary drilling is both faster and cheaper than diamond drilling.

For noncoring in shallow holes and medium to hard rock, *percussion drilling* is utilized; it is also employed for general-purpose exploration drilling because of its favorable costs. Penetration is achieved by sinking hammer blows against the rock, sometimes with a down-hole machine, pneumatically or hydraulically actuated. Like the rotary method, hole depths arc usually confined to a few hundred feet (100 m); other applications arc similar also, although hole size is 1-3 in. (38-89 mm). Recovery of cuttings is high, usually 90% or better, and is improved by reverse circulation.

Selection of the most suitable method of excavation or drilling to sample a mineral deposit follows a direct, logical procedure. For a thorough, convincing job of sampling, drilling is strongly favored over excavation (notwith – standing drilling's basic deficiency in failing to provide a continuous, three dimensional geologic record of the deposit). II drilling is used, and the deposit is a consolidated metallic one. coring will almost always be the preferred type of drilling—which means that diamond drilling will be the method selected. Other factors, such as deposit geometry, hole depth, rig site, and cost, warrant consideration; but aside from cost, all arc secondary to those cited above. However, in 1980, of the 16.2 million ft (4940 km) of exploration drilling done in U.S. noncoal mines. 69% was rotary, 9% diamond coring, 7% percussion, and 15% other (Martens. 1982). Special precautions in sampling are taken against drillhole deviation, concentration, leaching, contamination (accidental), salting (intentional), hole collapse, and loss of core.

Analytical Determinations. Excavation and drillhole samples must be quantitatively analyzed ***(assayed,*** in mining terms) in the laboratory. Here precision, accuracy, sensitivity, and stability arc the watchwords. When detection and quantification to a level of 1 ppb (part per billion) may be necessary with trace elements-and 1 ppm is standard for minor constituents of an ore or mineral sample - the selection of the proper analytical method is of critical importance (Cardwell, 1984). Traditional methods like chemical analysis (for inorganic compounds) and fire assaying (for sold and silver) are seldom used today, because they are slow, expensive, limited, and insensitive. Many sophisticated, versatile instruments have lately found application for the kind of exact determinations so essential in mineral exploration, of which atomic absorption is the most popular (used by *90%* of the laboratories). In order of availability, Table 2.5 lists the analytical method;» in current use and their application for the common elements in ores.

Borehole Logging. In addition to providing physical samples (core or cuttings) of a mineral deposit, drillholes permit the acquisition of other useful exploration information. First, engineering or directional surveys of the hole confirm its position and inclination, revealing the extent of deviation and permitting accurate correlation of geologic data. They are conducted by magnetic compass, gyroscope, camera, or television; TV also provides visual geologic information (Payne, 1973). Second, borehole logging other than geologic or directional is provided by geophysics, in which a sonde, a downhole sensor package, transmits signals to a surface recorder. While not as extensively used as in oil and gas exploration, geophysical logging is gaining acceptance in coal and uranium (Chironis, 1982b). Logging generally supplements coring; to save money, the number of cored holes is limited, and the gaps in exploration knowledge are filled in with logs of noncored holes. Several geophysical methods lend themselves to logging, especially electromagnetic, electric, radiometric, and gravimetric (Elkinglon et al., 1983). In coal exploration, borehole logging has partially supplanted core sampling, employing up to six geophysical methods to detect a coal seam and measure its thickness and some of its properties (Table 2.6). Ash. sulfur, and washability analyses, however, still require core sampling.

Metallurgical Testing

Since most mineral deposits (ore, coal, or stone) require processing—preparation or beneficiation—before their product(s) can be marketed, metallurgical tests (or washability studies, if coal) must be initiated during the exploration stage.

TABLE 2.5 Analytical Laboratory Methods

Method	Elements, Applications
Atomic absorption	Gold, silver, mercury, molybdenum, copper, lead, zinc, tin, nearly all others
Colorimetry	Arsenic, tungsten, molybdenum, titanium
Fluorimetry	Uranium
Emission spectrometry	70 elements
Inductively coupled plasma	50 elements, e.g., barium, manganese, boron
X-ray fluorescence	Minor elements, major oxides in geochemical samples
Neutron activation analysis	Gold (nondestructive)
Electron microprobe	Many elements, useful for small quantities
Radiometric	Uranium, thorium
Fire assaying	Gold, silver, platinum

Source: Cardwell, 1984. (By permission from Society of Mining Engineers, Inc., Littleton, CO.)

TABLE 2.6 Borehole Logging Methods for Analyzing Coalbeds

Logging Method	Response to Coal	Unit Scale	Log is Invalid When:
Natural gamma ray (natural radioactivity)	Low gamma ray	API units	Very clean sand adjacent to coalbed Coalbed uranium acceptor
Bulk density (gamma–gamma density)	Low density	g/cm³	Caving shale adjacent to coalbed Hole size erratic Bed thinner than tool spacing Rock around drillhole badly fractured
Neutron density	Low density (high apparent porosity)	Porosity index	Caving shale or wet clay adjacent to coalbed Hole size erratic Bed thinner than tool spacing Rock around drillhole badly fractured
Sonic (acoustic) velocity	Very low velocity	μsec	Very loose clean sand adjacent to coalbed Hole size erratic Bed thinner than tool spacing Rock around drillhole badly fractured
Resistance	High resistance	(ohm-m)²	Highly resistant bed adjacent to coalbed
Laterolog (electrical conductivity)	Low conductivity	millimhos/m	Highly resistant bed adjacent to coalbed

Source: Chironis, 1982b. (Copyright © 1982, McGraw-Hill, Inc., New York.)

Their purpose is to determine mineral amenability to processing and the approximate recovery and to develop a preliminary flow sheet for treatment. Tests required include additional geologic examinations, principally mineralogic and petrographic, and analytical determinations, screen analyses, crushing and grinding studies, and gravity and heavy-media separation tests (McQuiston and Bechaud, 1968). More specialized investigations of suitable concentration processes, such as magnetic, electrostatic, .ind flotation, follow if appropriate. Tests are conducted at bench scale at first (50-500 lb, or 25 – 225 kg) and then usually in a pilot plant (25-1000 tons, or tones). A consulting firm with specialized equipment is often engaged to perform the investigation. While probably not conclusive for design and financing purposes during subsequent mine development, testing initiated now and continued later suffices for the feasibility study.

2.7 EXPLORATION: RESERVE ESTIM ATES

In another era, this section would have been entitled "Examination and Valuation of Ore Deposits." So as not to lose touch entirely with a fitting and descriptive term, we retain it here as an implied subtitle. Generally, it means the determination of the extent and value of a mineral deposit.

Sampling Grids

Certainly one of the major challenges facing the exploration geologist - engineer in laying out a sampling program for an ore body is the pattern and spacing of sampling locations. These geometric parameters define the ***grid,*** a three-dimensional array. If surface excavations arc employed—shallow channels or deeper pits or trenches—then the deposit geometry generally dictates the pattern and spacing (Lewis and Clark, 1964; Readdy et al., 1982). Sampling by underground excavation is usually a joint function with development and tailored to the circumstances (shown in a subsequent illustration, Fig. 2.12). If drilling, the more common sampling means, is employed, then choosing the optimal array and number of holes is a crucial, complex decision requiring geologic, economic, and statistical input. That decision is rendered more difficult because the most important factors of |the location, shape, and attitude of the mineral deposit—are established only sketchily at this point in the exploration program.

The optimal sampling grid is one that provides the maximum amount of information from the least expenditure for drillholes. Often that outcome is attained by a drilling program that progresses in steps, from a few holes widely spaced to many at close spacing (Bailly, 1968). Figure 2.11 illustrates the concept with an actual ore body. Drilling here is performed in three steps, each based on the knowledge generated by the preceding step. Thus five holes are drilled initially, 15 arc added in the second step, and SI in the third. Notice how the grid emerges: irregular in steps 1 and 2 and regular (square) in step 3. Each step was justified by the results of the previous one; in this case, the results continued to be favorable. Had they been negative, the drilling and exploration program could—and should—have been terminated without further expenditure. The decision making depended as much on statistical input as on analytical determinations and geologic interpretation. an example of the application of geostatistics.

For different geologic conditions—steeply pitching lenses—the drilling pattern assumes a different form (Fig. 2.12). Note also that the grade of the ore must be higher in this example than in Figure 2.11 because underground exploration is being utilized.

As illustrated above, basic sampling grids take two geometric forms: ***regular*** and ***irregular.*** Although geologic conditions may favor an irregular grid initially, a regular grid should be adopted as early as feasible. The reasons are (hat the latter provides a uniform level of knowledge of all parts of the deposit and enhances visual interpretation of the results.

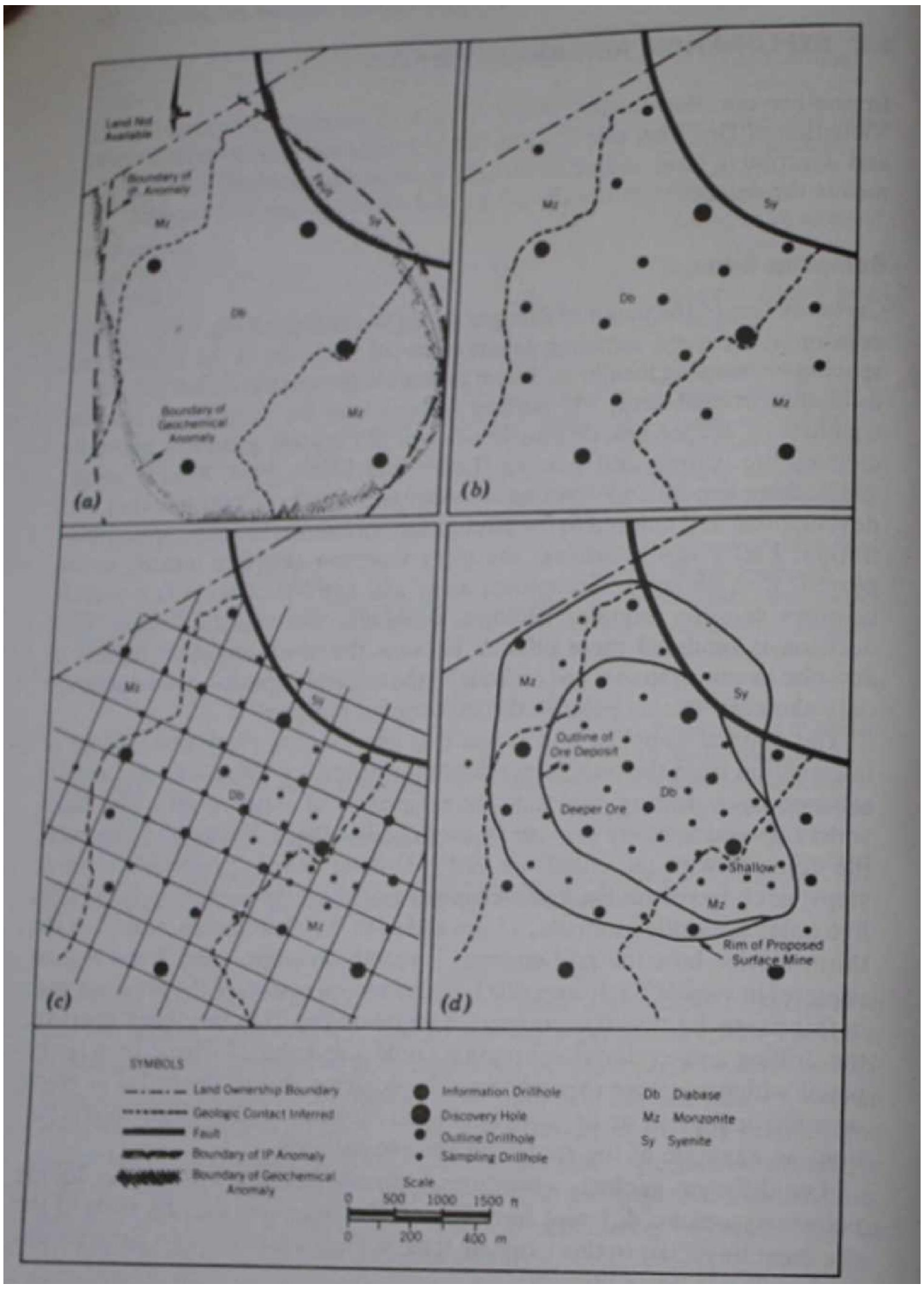


Figure 2.11. Successive steps in an exploration drilling program to detect, outline and sample a m ass,* suif.de deposit The result is an outline of the ore body and a tentative mine layout (a) information drilling (b) Outline drilling. (c) Sampling grid (d) Ore deposit and mine Outline. (After Bailly, 1968 By permission from Society of mining Engineers, Inc., Littieton CO)

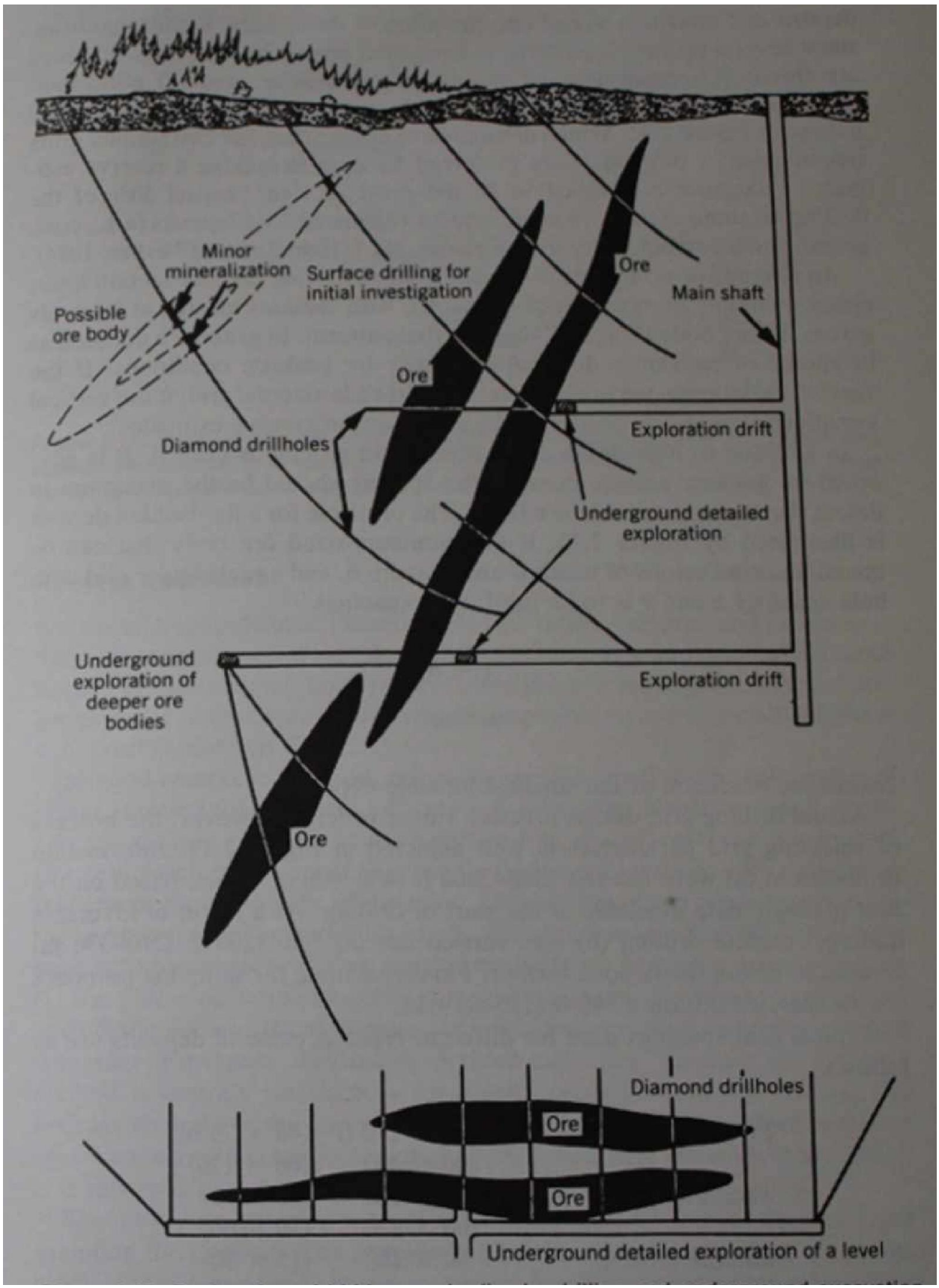


Figure 2.12. Exploration of hidden ore bodies by drilling and underground excavation (shaft and drifts). Diamond drillholes originate initially at the surface but later are continued from drifts as the ore bodies deepen. (After Hamrin, 1982. By permission from Society of Mining Engineers. Inc.. Littleton. CO.)

Regular grids assume several geometric patterns in horizontal projection, but basically there are three: (1) rectangular or square, (2) triangular, and (3) polygonal. Drillhole centerlines form the grid outline, such as the two simpler grids (rectangular or triangular) are preferred for ease in making a reserve estimate.
Triangular is claimed to be the most efficient saving 30% of the drilling (at some sacrifice in accuracy) for regular tabular deposits (e.g. coal, potash stone. sedimentary iron deposits, etc.) (Hamilton and Tasker. 1984).

An alternative arrangement locates drillholes along vertical section lines, either randomly or systematically spaced, with sections located at intervals across the ore body (Fig. 2.12 suggests that pattern). In general, a decision as to choice of pattern is determined largely by geologic conditions: If the vertical variation in the ore body is small, use a horizontal grid; if the vertical variation is great, use sections (see discussion of reserve estimates).

In addition to hole depth and pattern, grid spacing is critical. It is governed by geology and economics: the spacing should be the maximum to detect the smallest minable ore body. The principle for a flat-bedded deposit is illustrated by Figure 2.13. If the minimum-sized ore body that can be mined has dimensions of width *w* and breadth *h*, and a rectangular grid with hole spacings *x* and *y* is to be used, then spacings

$$x<b$$
$$y<w$$

ensure the detection *of* the smallest minable deposit.

Actual drilling-grid design practice varies widely. However, the process *of* selecting grid parameters is well depicted in Figure 2.11. Information drillholes in (a) were located 1500-2000 ft (450-600 m) apart, based on the best geologic data available at the start of drilling. As a result of favorable findings, outline drilling (b) was carried out on 700-1200 ft (210-370 m) centers to define the deposit further. Finally, drilling for sampling purposes (c) was earned out on a 500-ft (150-m) grid.

Typical grid spacing used for different types of mineral deposits are as follows:

Porphyry copper	150 × 250 ft	(45 × 75 m)
	200 × 400	(60 × 120)
Taconite iron	200 × 300	(60 × 90)
	100 × 200	(30 × 60)
Bauxite (Al_2O_3)	50 × 100	(15 × 30)
Molybdenite (MoS_2)	200 × 200	(60 × 60)
Bituminous coal	max. 3000	(900)
	min. 500	(150)

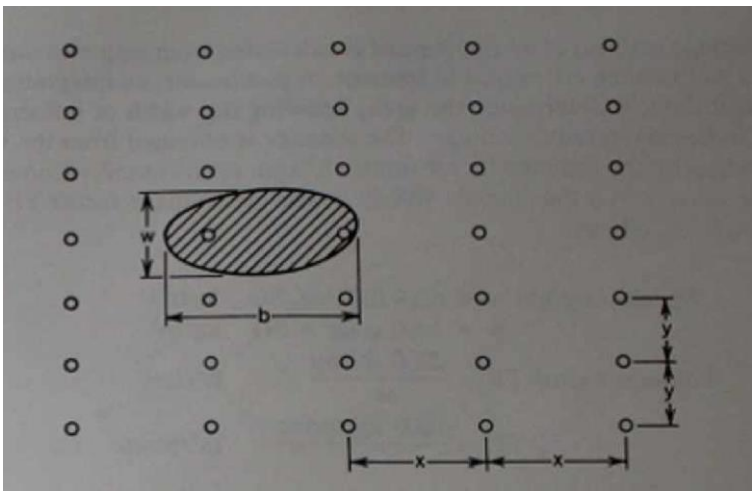


Figure 2.13. Principle of drillhole layout (plan view) to ensure detection of flat-bedded deposit Rectangular grid spacing *x* x *y*. Deposit horizontal dimension *b* x *w*. (After Payne. 1973. By permission from Society of Mining Engineers. Inc. Littleton, CO.)

Reserve Estimates

We have already defined (Section 2.5) two terms—*reserve* and *resource*— that now assume special significance. The quantitative and qualitative classification of ore (mineral, coal) reserves into three or more geoeconomic categories is one of the objectives of stage 2, exploration, and especially of phase 4, deposit evaluation (Fig. 2.1).

Mineral reserve estimation, or *ore estimation*, involves the calculation of *extent* (units; tons, tonnes, yd^3, etc.) and average *grade* or tenor (units: percent, oz/ton, g/tonne, $/ton, S/million Btu, etc.) of reserves in a deposit. Ore estimation is a process that begins during exploration, with continuing refinement during the life of the mine. Data derived from ore estimation arc the basic input for the feasibility study, the outcome of which determines if a mine is to be developed. Two general methods are in use for ore estimation: (I) the *classical,* employing two-dimensional maps and hand calculations; and (2) the *geostatistical.* a more sophisticated approach requiring a digital computer to prepare statistically derived estimates. Because the classical method is entirely satisfactory for small, uncomplicated ore bodies, and because an audit of the geostatistical results by another procedure is always useful anyway, the classical method is presented here (Readdy et al., 1982). It is followed by a brief explanation of geostatistics.

The raw data entering a reserve estimate are, of course, derived from sampling the deposit. The average grade of an ore deposit or of a specific block within a deposit is based on assays (analyses) of samples representative of the deposit or block. The average is calculated from individual samples using the width, area, or volume of influence exerted by each sample.

The tonnage (extent) of an ore deposit is calculated from map measurements of area and volume converted to tonnage. A planimeter, an integrating measurement device, determines the area; knowing the width of influence, the area can be converted to volume. The tonnage is obtained from the volume by dividing

by the tonnage factor (units; ft/ton, m^3/tone.) Knowing the specific gravity SG, the specific weight w and ore tonnage factor TF can be calculated as follows:

$$\text{Specific weight } w = 62.4 \text{ lb/ft}^3 \times SG \quad \text{lb/ft}^3 \qquad (2.1)$$

$$w = 1000 \text{ g/kg} \times SG \quad \text{kg/m}^3 \qquad (2.1a)^1$$

$$\text{Tonnage factor } TF = \frac{2000 \text{ lb/ton}}{w} \quad \text{ft}^3\text{/ton} \qquad (2.2)$$

$$TF = \frac{1000 \text{ kg/tonne}}{w} \quad \text{m}^3\text{/tonne} \qquad (2.2a)$$

(For a table of specific gravities of common rocks and minerals, see Behre and Stuart, 1973, p. 4.18, or Readdy et al. 1982, p.24).

The results of a reserve estimate, however, are not derived from straight calculation—they must be tempered by judgment and modified by technical and economic realism. At least three factors must be estimated: (1) mining, and processing recoveries. (2) waste dilution during mining, and (3) cutoff grade. Their incorporation into the reserve estimate provides a much more honest and accurate result.

Different classical procedures arc used in calculating reserve estimates (Readdy et al.. 1982). They differ mainly in the ways in which they combine the sample data. While computer programs reduce the drudgery in repetitive jobs, hand held calculators arc adequate for occasional work. Three procedures arc discussed here (see Fig. 2.14).

In the *polygon method*, areas of influence (polygons) are assigned to drillholes, either on plan or cross-section views. The procedure for construction of polygons is illustrated in Figure 2.14a and b, based on the theory that the influence of a hole extends halfway to the next adjacent hole. Areas are then measured by planimeter and subsequent calculations tabulated in finding tonnage and grade.

The *triangle method,* a modification of the polygon method, differs from it by constructing triangles with the drillholes at the apices; see Figure 2.14c. The advantage is that data from three points (holes) enter into the calculation of the thickness and grade parameters for each triangular block. Calculations then proceed in the same manner as for the polygon method.

In the *section method,* blocks of ore are outlined on regularly or evenly spaced cross sections of the ore body; see Figure 2.14d. Areas of influence are determined in the usual way and all calculations tabulated.

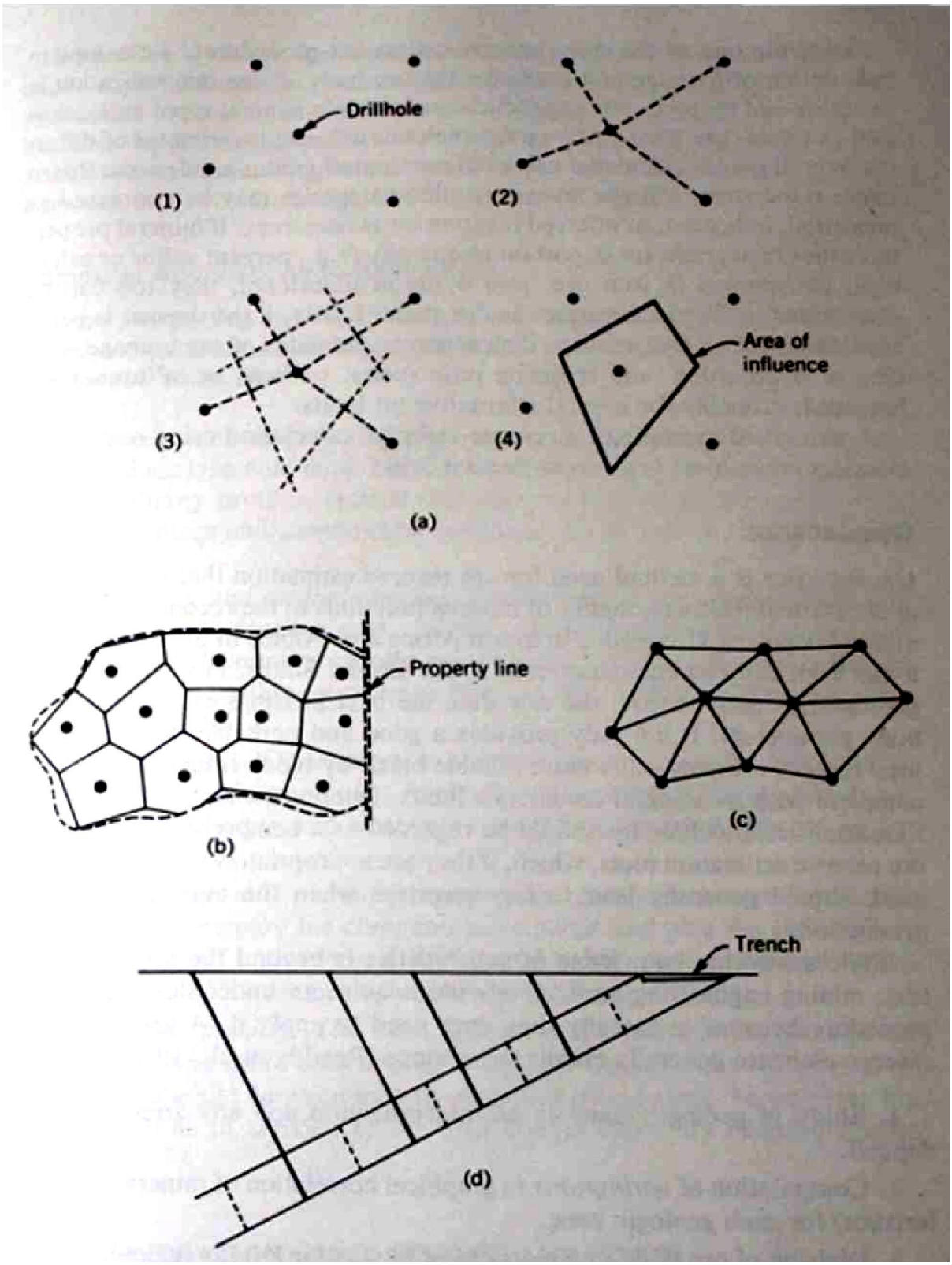


Figure 2.14. Traditional procedures for calculating reserve estimates by areas of influence. (a) Polygon method: (1) drillhole plan. (2) connecting lines between drillholes, (3) perpendicular bisectors of connecting lines. (4) final polygon, (b) Polygon method: plan view of completed diagram, (c) Triangle method: plan view of completed diagram, (d) Section method: cross-sectional view of completed diagram, with four drillholes and one trench sample. (After Readdy et al.. 1982. By permission from Society of Mining Engineers. Inc.. Littleton. CO.)

Selecting one of the three reserve – estimation procedures, we complete calculation of tonnage and grade for the ore body. If the mineralization is complex and by – products, coproducts, or multiple mineral types exist, several estimates are prepared, one for each constituent. If estimates of different overall grades calculated using different cutoff grades are desired, then a curve rather than a single answer results; categories may be expressed as measured, indicated, or inferred reserves or as resources. If mineral properties other than grade are important to quantify (e.g. percent sulfur or ash in coal, phosphorus in iron ore, iron oxide in limestone), they too can be determined in the same manner as the grade. Lastly, if the deposit is being considered for surface mining, then separate estimates of ore tonnage, volume of overburden, and stripping ratio (units: yd^3/ton, or m^3/tone) are prepared, probably for several alternative pit limits.

A numerical example of a reserve – estimate calculation using one of the classical procedures is given in Section 2.10.

Geostatistics

Geostatistics is a method used for ore reserve estimation that involves the *application of the* mathematics of random functions to the reconnaissance of mineral *deposits. Originating* in South Africa and France in the early 1960s, it has been adopted enthusiastically in the United States. Properly applied, geostatistics derives from the raw data the best possible estimates of ore body parameters. Ii not only provides a good and perhaps more accurate total reserve estimate but a more reliable block-by-block reserve inventory, complete with meaningful confidence limits. Quoting Readdy et al. (1982), " Geostatistical techniques should be regarded as a comprehensive suite of ore reserve estimation tools, which, if they are appropriately understood and used, should generally lead to few surprises when the mine comes into production."

While a working know ledge of geostatistics is beyond the scope of this text, mining engineering students should acquire an understanding of the procedure because eventually they may need to apply it. A geostatistical reserve estimate generally entails these steps (Readdy et al., 1982):

1. Study of geologic controls on mineralization and any zoning of the deposit.
2. Computation *of variograms* (a graphical correlation of mineral characteristics) for each geologic zone.
3. Division of ore body into matrices of blocks for ***kriging*** (calculation of block grade related to adjacent samples); blocks are then classed as measured, indicated, or inferred reserves.
4. Estimation of tonnage and grade of each block at given cutoff grade.
5. Printing of recoverable grade distribution plans by level or bench in the proposed mine.

While Readdy et al. feel there will always be doubts as to whether the added sophistication and cost arc justified, geostatistics does provide a comprehensive suite of tools—not only to find the total ore reserve parameters but also the distribution of

recoverable grade throughout the deposit. For additional treatment of geostatistics, see Journal and Hujjbregts (1978), Knudscn et al. (1978), Rendu (1981), and Kim et al. (1981).

Computer Applications to Exploration

Modem prospecting and exploration have become heavily dependent on the digital computer, and indications are that this dependency will increase. To name a few examples:

1. ***Geophysical surveys.*** Initially, the computer was used to record, store, and analyze field data. Later it was used to solve complex equations and plot survey profiles. Now it aids the geophysicist in interpreting field data and findings and supplies him confidence limits as well (Van Blaricom,1980).
2. ***Sampling and drillhole layout.*** Determining a sampling strategy and drillhole pattern, spacing, and grid are routinely performed by computer programs. Also sampling results are depicted and analyzed by computer graphics, aided by three-dimension plots of hole deviation, core recovery, etc. (Bruce, 1982).
3. ***Geostatistics.*** Ore-reserve estimation by geostatistics would not be practical without the computer. Again data arc analyzed, results calculated, and confidence limits and alternative strategies provided (Readdy et al., 1982).
4. ***Scheduling and planning.*** Large-scale prospecting and exploration programs now employ the computer to schedule and plan the entire effort, using the critical path method (CPM) and project evaluation and review technique (PERT) from operations research (Payne, 1973).
5. ***Decision making.*** The most difficult of all management tasks is rendered faster and more effective by computer. Optimization techniques are employed to assist decision making at critical points along the way (sec Fig. 2.1) as well as in arriving at the final crucial feasibility decision (Payne, 1973).

2.8 EXPLORATION: CASE STUDIES AND COSTS

Case Studies

The sequence of activities and methods employed during the four phases of prospecting and exploration of varied mineral deposits can best be grasped from hypothetical case studies. Table 2.7 contains particulars for 10 different properties (ventures), representing a wide spectrum of mineral commodities, geologic conditions, and locations. A significant aspect in six of the 10 ventures is that the progression of mineral search and definition involved all four phases of activity, which Bailly (1966, 1968) has described as full-sequence prospecting-exploration. Commoditywise. the deposits which require less elaborate search procedures are typically coal and many nonmetallics (in Table 2.7, limestone and gravel). Geologically, exploration is usually less involved for tabular sedimentary deposits of large extent (in Table 2.7, placer, coal, limestone, and gravel).

Case studies providing details of the prospecting and exploration procedures

used with particular mineral occurrences are given by Payne (1973) for vein deposits, sedimentary deposits, disseminated ores, placer deposits, coal, uranium, asbestos, potash, phosphate, and sulfur.

Costs

In estimating mining costs, it is helpful to break them down on a unit basis. The basis for calculation of unit costs is dollars per unit of time (hr, mo, yr) or unit of production (ton or tonne, yd^3 or m^3). In prospecting and exploration, we generally estimate costs for each of the methods employed, cumulating them to obtain overall costs. Because the methods arc usually applied over a specified area, it is easier first to express costs per unit area or profile length of survey and then convert them to the desired basis. The principal operating costs for prospecting-exploration are itemized below, normalized to prevailing (1986) price level in the (contiguous) United States.

Geology. Approximate costs for geologic prospecting and exploration are as follows (Metz and Campbell, 1982):

A ir

Aerial photography	S 70-140/mi2	($20-50/km2)
Photo interpretation	$ 90-150/mi2	($30-55/km2)
	S160-290/mi2	($50- 105/km2)

Ground

Geologic field work	$150-650/mi2	($60-240/km2)
Geologic inference	$180-450/day	

Geophysics. Geophysical survey costs, per line-mile (line-km) or per day, are given in Table 2.3 for all methods. Additional air and ground costs for several methods are as follows (Bruce, 1982; Metz and Campbell, 1982):

	AIR	**GROUND**
Gravitational	N / A	$450-1300/line-mi ($250-800/line-km)
Electrical- IP	N / A	$50-100/line-mi ($30-55/line-km)
M a g n e tic2	$20-65/line-mi ($ l5 - 40 /line- k m)	$150-200/line-mi ($100-130/line-km)

Electromagnctic2	$20-65/line-mi ($ l5 - 40 /line- km)	$100-200/line-mi ($65-130/linc-km)
Radiometric2 `	$20-65/line-mi ($15_40/line-km)	-------

Geochemistry. Costs of geochemical surveys consist of two main items, field work and laboratory (analytical) work. They range as follows (Metz and Campbell, 1982):

Geochemical prospecting	
Water	$50-100/mi^2 ($20-40/km^2)
Soil	*$* 1200-6000/mi^2 ($500-2500/km^2)
Analytical determinations	S4-10/sample

Drilling and Excavation. Drilling costs vary with hole depth, rock conditions, number of holes and footage drilled, terrain, location, and the skill of the driller. For estimating purposes, the following unit cost ranges may be helpful (Bruce, 1982):

Diamond drilling	$10—25/ft ($30-80/m)
Rotary drilling	$6-15/ft ($20-50/m)
Percussion drilling	$4— 10/ft (Si5-30/m)

By comparison, excavation of small underground exploration openings costs approximately as follows:

Shaft sinking	$300/ft ($ 100/m)
Adit or drift driving	$ 100/ft ($30/m)

Surface trenching costs range as follows: (Metz and Campbell, 1982):

Soil	$2-35/yd^3 ($3-45/m^3)
Rock	$7-60/yd^3 ($10-80/m^3)

Since speed in exploration is as crucial as cost, excavation is seldom employed today in place of drilling. If exploration drilling is contracted out, a common basis of charging is per foot (meter) of completed hole with a specified minimum core

(or cuttings) recovery, say *9 0 %* or 95%. To the actual drilling cost also should be added the cost of analytical determinations, $4-10 per sample.

Borehole Logging. Geophysical logging of drillholes, like drilling itself, is often contracted out. Costs may be estimated as follows (Payne, 1973):

Electromagnetic	$0.50-1.50/ft (SI.50-5.00/m)
Electrical	$0.50-2.00/ft (SI.50-6.50/m)
Radiometric	$0.20-0.50/ft ($0.70- 1.50/m)

Claim Staking. Staking a mineral location on public land, cither federal or state, costs S50-350 per claim (Bruce, 1982).

Overall Cost. Based on the individual operating costs given above, the overall cost of prospecting-exploration of a mineral deposit can be calculated. Costs can then be computed on the basis of unit production cost (S/ton, or S/tonne) or unit time ($/yr).

In Table 2.8, Bailly (1966) has estimated the total costs in three of the case studies given in Table 2.7 (based on 1966 prices), tabulated for the four phases. By dividing by area or time, also given, the costs per mi2 (km2) or per yr can be obtained (time periods arc cumulative). The tonnages of the ore bodies would have to be specified to find the unit production costs. Bear in mind, however, that these ventures were all successful; to find the actual costs, the costs of all unsuccessful searches would have to be added to these, raising the totals to 5 to 10 times these values.

This estimation procedure is the basis for the costs given in Table 1.3, ranging from *20* to 50$ per ton for prospecting and from 100 to Si per ton for exploration (or total costs of $100,000-5,000,000 for prospecting and $500,00-10,000,000 for exploration). W e can now understand better the significance of the figures stated earlier in this chapter, in which the total cost of finding a successful mine, including unsuccessful ventures, ranged from $30 million to $300 million.

TABLE 2.8 Prospecting–Exploration Costs and Time Expended for Three of the Mineral Properties (Ventures) of Table 2.7

	Prospecting		Exploration	
Venture	Phase 1 Regional Appraisal, One Region Selected	Phase 2 Detailed Reconn., One Target Area Selected	Phase 3 Detailed Surface, One Target Selected	Phase 4 Detailed Physical Sampling, Target an Ore Body
1. Porphyry Copper Deposit				
Area under consideration (mi^2)	1000–100,000	10–100	10–50	3–20
(km^2)	(2600–260,000)	(26–260)	(26–130)	(8–52)
Cumulative expenditures ($)	5000–100,000	10,000–75,000	50,000–150,000	500,000–4,000,00
Cumulative elapsed time (mo)	1–12	13–18	18–24	24–60
2. Blind Stratiform Lead–Zinc Deposit				
Area under consideration (mi^2)	1000–100,000	10–50	5–10	0.5–10
(km^2)	(2600–260,000)	(26–130)	(13–26)	(1.3–26)
Cumulative expenditures ($)	5000–500,000	50,000–1,000,000	150,000–1,500,000	300,000–3,000,000
Cumulative elapsed time (mo)	1–12	13–20	20–30	30–60
3. Massive Sulfide Deposit				
Area under consideration (mi^2)	1000–100,000	10–50	1–10	0.1–2
(km^2)	(2600–260,000)	(26–130)	(2.6–26)	(0.3–5)
Cumulative expenditures ($)	5000–500,000	20,000–600,000	100,000–1,000,000	300,000–3,000,000
Cumulative elapsed time (mo)	1–6	7–15	16–24	24–60

Source: Bailly, 1966; Payne, 1973. (By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Expenditures for all mineral prospecting and exploration in the United States over the decade 1974-1983 averaged 5319 million per year (Emerson and lvosevic, 1984). The most striking feature of the figures is their reflection of the boom nature of the mineral industries; a good example is gold, which in 1974 accounted for only 12% of the expenditures but in 1983 represented 55%. Precious metals very typically are in great demand during recessions, which correlates well with the gold boom of the 1980s.

2,9 FEASIBILITY STUDIES

Prospecting and exploration of a mineral deposit, stages I and 2 in the life of a mine, culminate in the preparation of a detailed study of the feasibility of proceeding with the development and exploitation of the mine, stages 3 and 4 (review Table 1.3). The report that results is primarily an economic one, but legal, technological, geologic, environmental, and sociopolitical aspects arc included as well. The key recommendation of a feasibility study is to proceed with, delay, or terminate further work on the project.

At this point, the company has already expended a substantial sum to find and define the deposit. There is a temptation for the investigating engineer in charge to take an optimistic view and recommend proceedings with the project in hopes of recovering the investment and realizing a profit for the company. However, realism is imperative at this point, and the engineer must retain an objective viewpoint and not recommend further expenditures unless the investment is a sound one, fully justified by the evidence.

A high degree of expertise and wisdom is required of the individual responsible for preparing the feasibility report. Doubts and uncertainties surely remain; the data may be impressive but have some gaps and fail to be totally convincing. The examining engineer must determine when to seek additional data and when to proceed without, perhaps qualifying or corroborating the evidence he or she has.

A carefully prepared report outline, compatible with management's investment objectives as well as mining interests, is essential. The ingredients of a feasibility report usually include the following; an expanded outline is given by Payne (1973) and a full treatment by Baxter and Parks (1957) and Jones and Pettijohn (1973);

1 *Preface; summary: definitions*

2 *General:* location, climate, topography, history, ownership, land status, transportation, etc.

3 *Environment concerns:* present conditions, standards, necessary protective measures, land reclamation, special studies, permitting

4 *Geologic factors:* deposit setting, origin, structure, mineralogy and petrography

5 *Mineral reserves;* exploration procedure, findings, calculation of tonnage and grade of reserves (e.g., 3.65 million tons, or 3,31 million tones, of measured reserves averaging 3,05 % Zn), extent and grade of by products in ore
6 *Mining plan;* development and exploitation
7 *Processing;* on site facilities needed
8 *Surface (and underground) plant;* location, construction plan
9 *Auxiliary and support facilities;* power, water supply, road access, waste disposal, housing, etc.
10 *Staffing;* work force (labor and supervisory)
11 *Marketing;* economic survey of supply and demand, price, long – term contracts, substitutes.
12 *Costs;* estimation of direct, indirect, and overall costs of development and exploitation; costs of processing, transportation, smelting, etc.
13 *Economic valuation;* valuation of deposit, classified as to type reserve or resource; calculation of present worth.
14 *Profit projection;* determination of profit margin, based on range of cutoff grades, prices.

(In an aside to the student: You are not expected to possess the knowledge to prepare such a report now. When you have completed study of this book, you will understand more of the factors involved. The topic is introduced now because the timing is appropriate to our discussion of the four stages of mining.)

Somewhat later, methods to calculate cost, value, profit, and present worth will be presented. Some simple examples now will demonstrate the relationship of these terms.

EXAMPLE 2.1. Estimate the unit profit in mining and processing a 0.60 % copper ore deposit if the selling price of copper in the concentrate is 74e/lb ($1.63/kg) and overall unit costs are $6.80/ton ($7.50/tonne). Overall recovery is 92%.

SOLUTION: Calculate the value o f the ore:

Value - grade x recovery x price x 2000 lb/ton (2.3)
= (0 0060) (0.92) (0.74) (2000) = 8.17/ton ($9.00/tonne)
Profit = value - cost = 8.17 - 6.80 = $1.37/ton ($l.5l/tonne) (2.4)

EXAMPLE 2.2. Calculate the cutoff grade for the copper deposit of Example

SOLUTION:

$$\text{Cutoff grade} = \frac{\text{cost}}{\text{price} \times \text{recovery}} = \frac{6.80}{(0.74)\,(0.92)} \qquad (2.5)$$

$$= 9.99 \text{ lb/ton} \div 2000 = 0.50\%$$

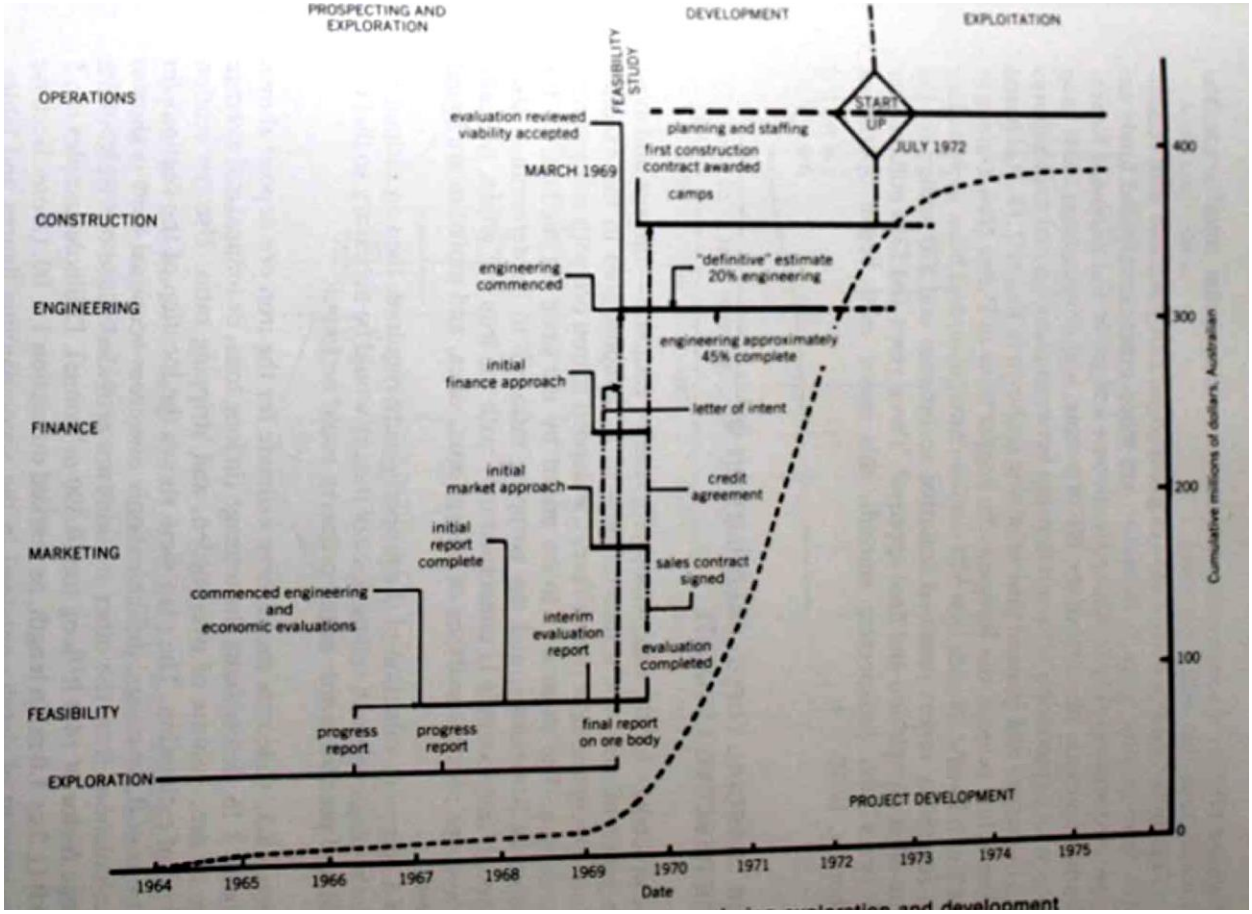


Figure 2.15. Relationship between planning steps during exploration and development and expenditures preparatory to mining a large open pit copper mine. Bougainville mine, Papua New Guinea. (After Hope, 1971. By permission from institution of Engineers, Barton, Australia)

Nilsson (1982d) discusses the determination of optimum cutoff grade, the lowest grade that will meet costs.

Feasibility studies do not always proceed smoothly nor end with conclusive findings after stage 2. Decisions are made progressively and funds are expended strategically to maximize income and profit and minimize losses. Studies continue throughout the life of a mine, and development must keep pace with exploitation.
The relationship between planning and expenditures and the stages and elapsed time in mining is shown in Figure 2.15. It is based on an *actual project*, the Bougainville copper mine in Papua New Guinea in the South Pacific. Notice that 20 $ million (Australian) had been spent before the feasibility report received tentative acceptance and S30 million by the time of its completion and final approval. Three years and $350 million later (8 years after prospecting started), this major mine began operations (Thomas. 1978).

2.10 SPECIAL TOPIC: CALCULATION OF ORE RESERVE ESTIMATE

The procedure for the calculation of a reserve estimate—tonnage and average grade of ore—for a mineral deposit is demonstrated in the example below. Occurring near the surface, the deposit is iron ore with soil and rock overburden. The deposit is to be mined by the open pit method, so the volume of overburden

and the stripping ratio are to be determined also. Although this example is concerned only with the iron ore grade, typically the average percent analyses of phosphorus, silica, and moisture are found as well.

If a complete valuation of the property were required, then an estimation of the mining costs and selling price of the ore would be necessary so that (he profit and present worth of the properly could be found.

EXAMPLE 2.3. Calculate the reserve estimate for the iron ore deposit shown in Figure 2.16, determining the tonnage (in long tons, or tonnes) and average grade of ore, volume of overburden, and stripping ratio. Use the section method of estimation. The plan view shows the location of the sections (in hundreds of ft, or m) and drillholes; only one cross-sectional view is shown, but calculations from the other sections are provided. Base ore weight on a tonnage factor of 14.0 ft^3/long ton (0.390 m^3/tonne). Drillhole samples are 5 or 10 ft (1.5 or 3.0 m) in length, as marked on section 1 + 00. (*Note*: Because of the amount of detail contained in the accompanying figures and tables, equivalent S.I. units are omitted. Answers, however, are given in both English and S.I. units.)

SOLUTION. Tabulate all calculations. Width of influence for each drillhole is one-half the distance to the adjacent drillholes; for end holes, extend influence one – half the distance to the drillhole on the opposite side unless specified otherwise. Determine the width of influence for sections in the same manner. Widths for both drillholes and sections are indicated on Figure 2.16. Convert planimeter readings in in.2 to areas in ft2 by using the map scale (note here that the section scales differ vertically and horizontally).

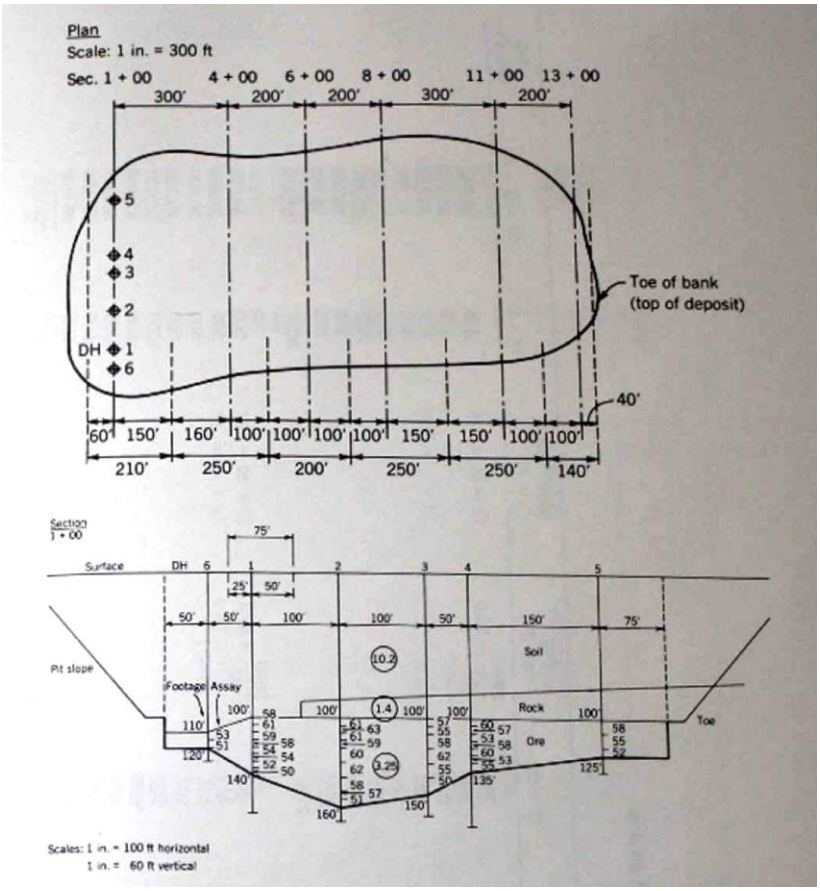


Figure 2.16. Plan view and cross section 1 + 00 of iron ore deposit in Example 2.3. Values to left of drillhole are depths in ft, those to right are ore analyses in percent Fe. Circled values are areas measured by planimeter in in^2

PROBLEMS.

2.1 From section information and drillhole analyses for the open pit iron ore mine shown in Figures 2.17 and 2.18, calculate the ore reserve estimate of tonnage and average grade. Also calculate stripping yardages (for soil, rock, and total) and the overall stripping ratio. Footages are given to the left of the drillhole and iron analyses (in percent) to the right. Circled areas are obtained by planimeter (units are in.^{2}). Use a tonnage factor of 14.0 ft^3/long ton (0.390 m^3/tonne). Set up calculations in neat tabular form, and show all steps. Express grade values to three significant figures, and record drillhole as well as section and mine averages.

2 2 From section information and drillhole analyses for the open pit copper mine shown in Figures 2.19 and 2.20, calculate the ore reserve estimate of tonnage and average grade. Also calculate stripping yardages (for soil rock, and total) and the overall stripping ratio Footages are given o the left of each drillhole and copper assays (in percent) to the right. Circled areas are obtained by planimeter (units are in.^{2}). Use a tonnage factor of 14.2 fl /ton (0.443 m'/tonne) Set up calculations in neat tabular form, and show all steps. Express grade values to three decimal places, and record drillhole as well as section and mine averages.

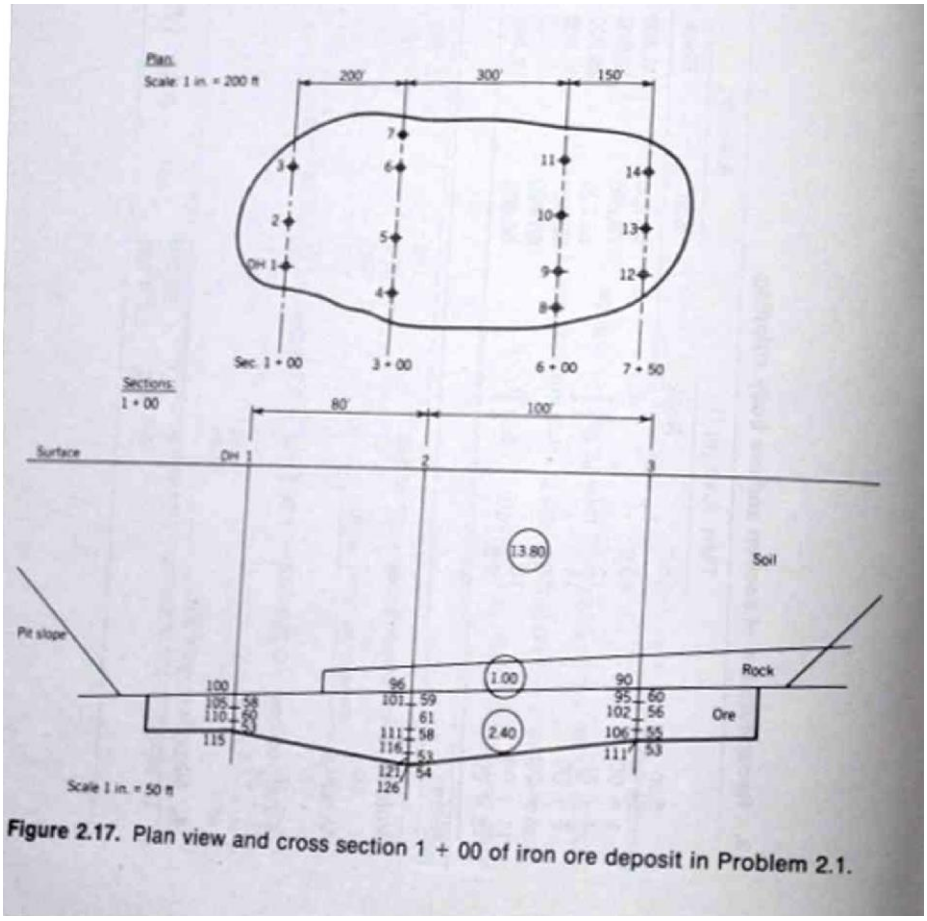


Figure 2.17. Plan view and cross section 1 + 00 of iron ore deposit in Problem 2.1.

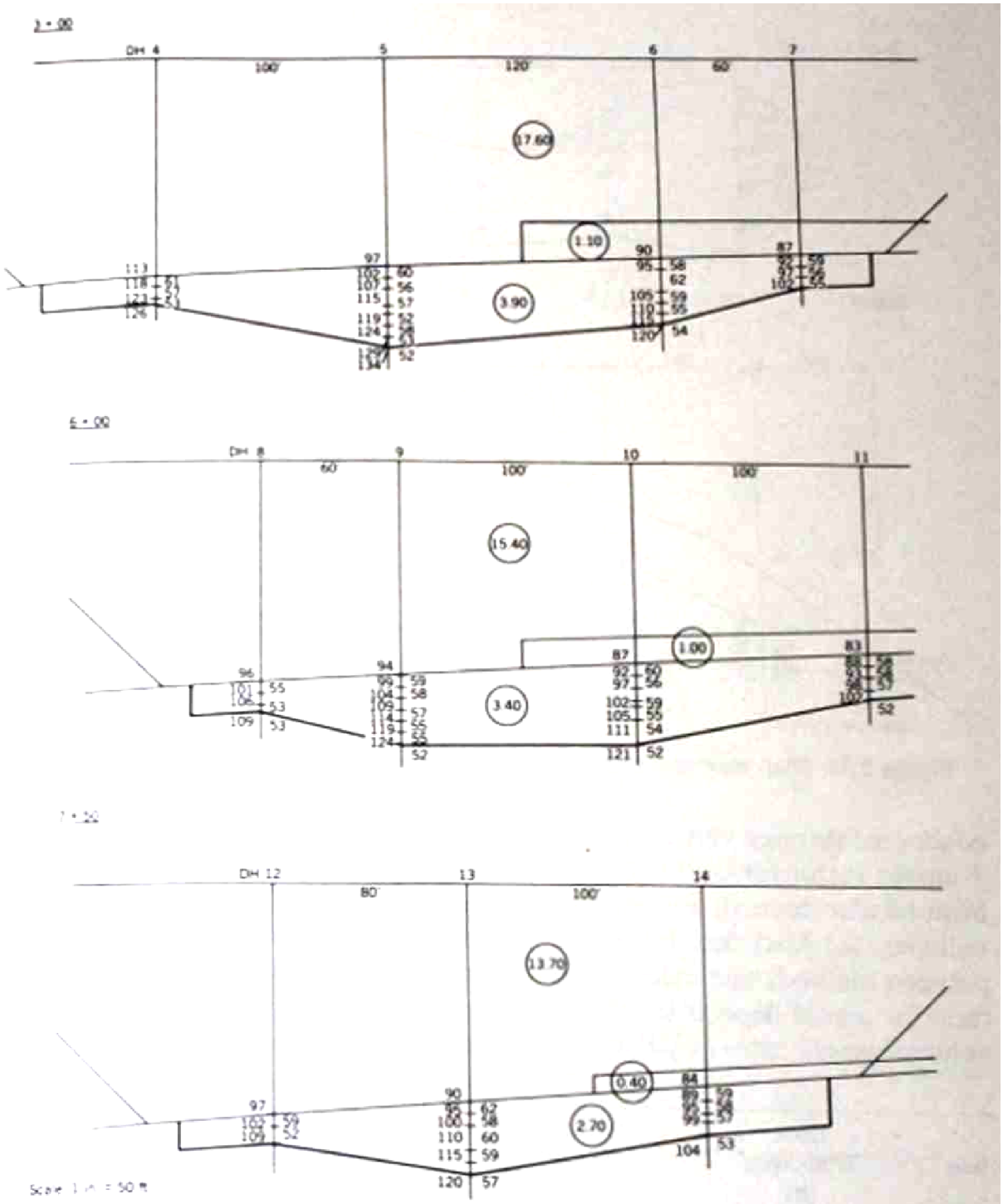


Figure 2.18. Cross sections 3 + 00, 6 + 00, and 7 + 50 of iron ore deposit in Problem 2.1.

2.3 From drillholes compiled in the accompanying table, calculate the (a) average seam thickness, (b) average depth of overburden, (c) average analysis (in percent sulfur), and (d) total tons (tonnes) of reserve for the coal scam shown in I Figure 2.21 in plan view and cross section. Exploration has been conducted through vertical drillholes, spaced equidistant in a triangular grid.

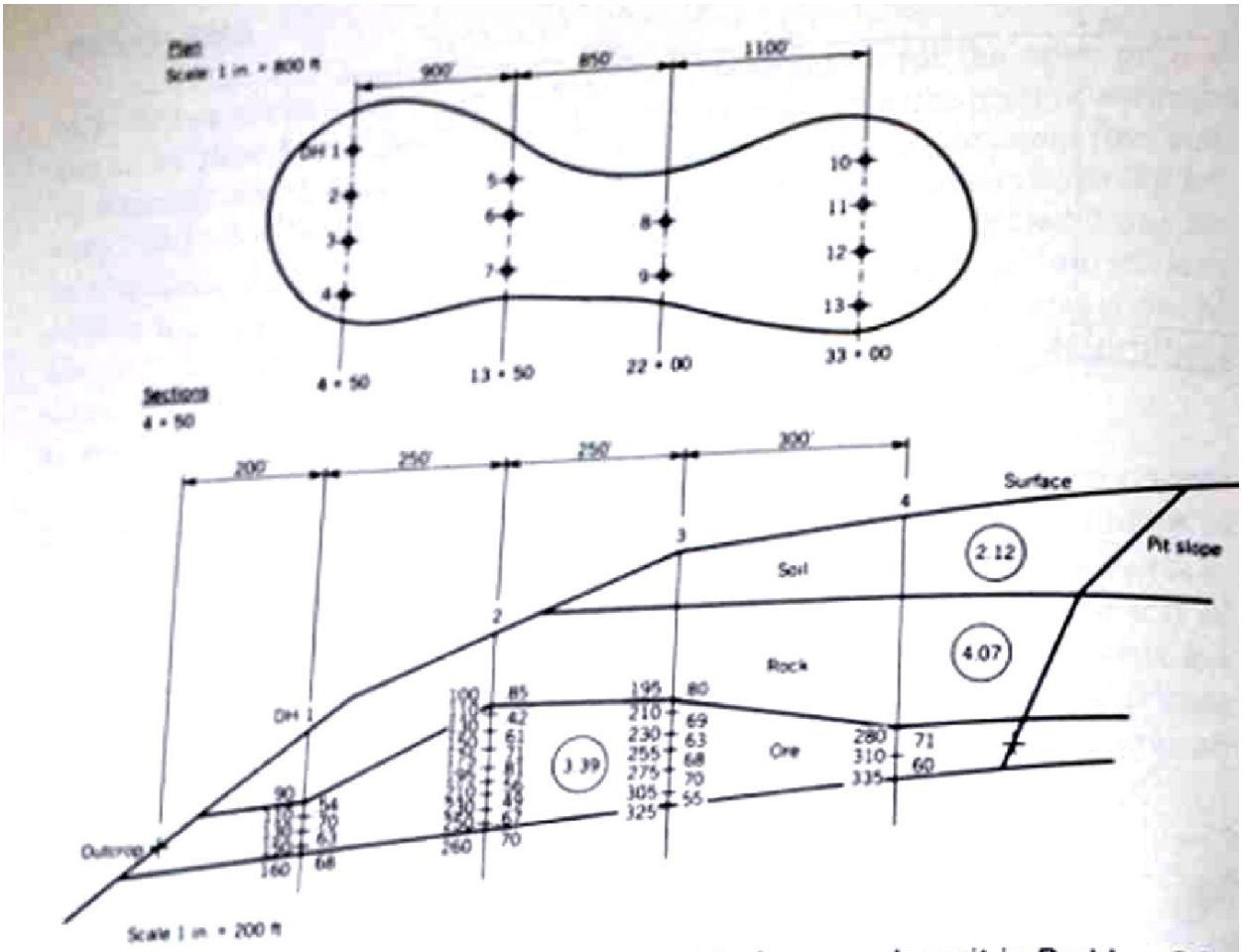


Figure 2.19. Plan view and cross section 4 + 50 of copper deposit in Problem 2.2.

Tonnage factor for coal is 25 ft^3/ton (0.78 m^3/tonne) *Suggestion:* In a uniform tabular deposit, statistical averages can be based on areas rather than volumes (e) Also sketch the areas of influence on the plan view, using the poison method, and calculate the stripping ratio (based on thickness, the ratio for a coal deposit in ft of overburden per ft of coal approximates the volume/weight ratio in yd^3/ton, or m^3/tonne).

Hole	Coal Thickness (ft)	Overburden Thickness (ft)	Analysis (% S)	Area of influence × 10^6 ft^2
1	3.1	133	0.8	1.56
2	2.8	175	1.6	2.10
3	3.2	283	1.4	1.34
4	3.0	182	1.7	2.32
5	3.1	327	0.9	1.70
6	2.9	156	1.2	2.21
7	3.2	298	1.5	1.73
8	3.2	417	1.4	1.20
9	2.8	149	1.0	2.38
10	3.1	275	1.3	1.98
11	3.0	444	1.2	2.61

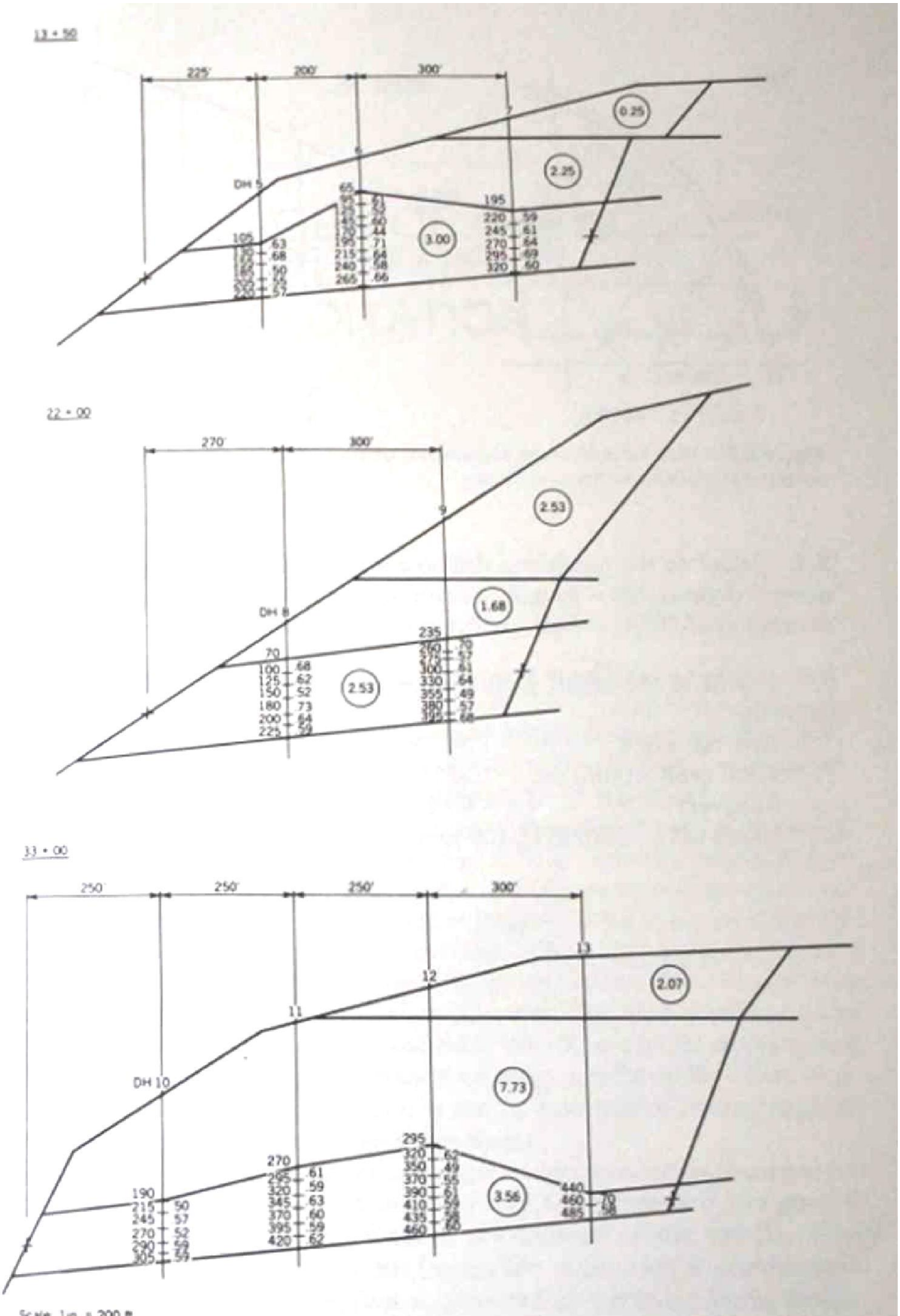


Figure 2.20. Cross sections 13 + 50, 22 + 00, and 33 + 00 of copper deposit in Problem 2.2.

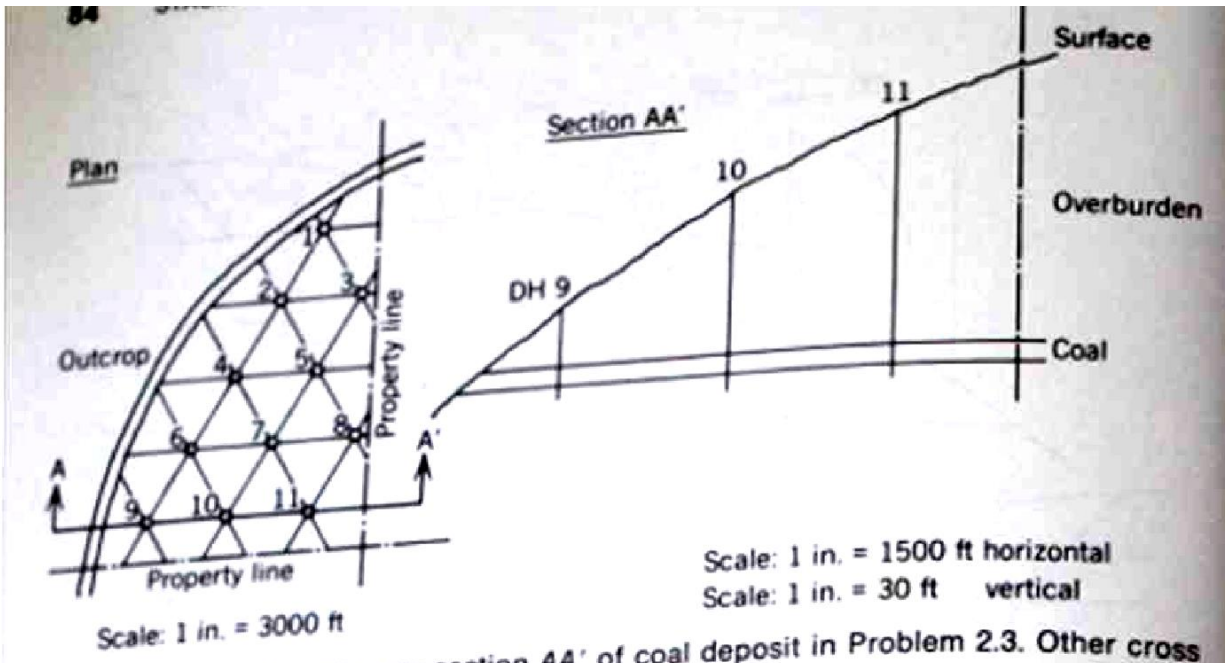


Figure 2.21. Plan view of cross section *AA'* of coal deposit in Problem 2.3. Other cross sections are similar to the one shown.

2.4 Calculate the maximum drillhole spacing that can be used to locate a mineral deposit that is roughly square in shape and has a minimum economic dimension of 200 ft (60 m). Select a suitable drillhole pattern.

2.5 Calculate the cutoff grade (in percent) for an iron deposit given the following.

Iron ore price	$85/long ton ($82.67/tonne), 100% Fe
Total production cost	$25/long ton (24.61/tonne)
Recovery	95%

I long ton = 2240 lb (1.106 tonne)

3

STAGES OF MINING: DEVELOPMENT AND EXPLOITATION

3.1 MINING PROPER

With successful completion of prospecting and exploration—and with a favorable outcome of the feasibility study—the work of mining proper can commence. What was originally an anomalous mineral deposit proved to be an ore deposit (or an economic deposit of coal or stone). What was recently rated a mineral prospect is soon to emerge as an embryo mine.

In the final two stages in the life of a mine, the work of the actual mining of a deposit takes place. Development, stage 3, to a considerable extent precedes exploitation, stage 4. Like prospecting and exploration, however, we find no sharp demarcation between these stages—at least, not in a chronological sense—even though their functions differ, and development gets underway first. Thereafter, development is apt to proceed but one step ahead of exploitation, and it concludes only when the mine is exhausted or closed, A review of Figure 2.16 indicates also that certain development activities commence during the exploration stage and feasibility study (e.g., economic analysis, land acquisition, tentative selection of mining method, preliminary financing, excavation of openings).

The reasons why development and exploitation proceed in sequence but overlap are both fiscal and technological: (1) The expense is too great to develop an entire mine at once, without any financial return; and (2) a mine evolves, and it is impractical if not impossible to develop it completely in advance, let alone maintain it. Just as exploration continues during mining, development occurs throughout exploitation, especially that closely associated with production openings. The proper coordination of all three activities become a key task of the mine management, and continual reassessment of the project's feasibility is required during its entire lifetime.

The principle of limited development in advance of exploitation is widely held in mining. There is an opposing argument, however, which advocates that as much development as affordable should be completed prior to the first production of ore, coal, or stone from a mine (Young, 1946). Efficient production scheduling and early exploitation of an ore body require that essentially all major surface plant facilities be in operation before any ore is produced. Many mistakes in mining enterprises and even mine failures can be traced to haste and greed in exploiting an ore body. As a compromise, then, the guiding rule of mine development is that a generous allocation of working capital be assigned for this purpose, and that with this allocation the maximum amount of development be completed prior to the

commencement of mining. In engineering terms, this is tantamount to saying that an optimal development – exploitation rate to maximize total profit must be determined for each new mine project.

Since development openings usually cost (in $/ton. or $/tonne) more to excavate than exploitation openings, *another rule is that development should be carried outto render accessible the maximum quantity of ore for a minimum volume of development openings* (Young. 1946). An exception to this rule occurs in room and pillar mining (commonly used for coal and several nonmetallics) where development openings (entries) closely resemble exploitation openings (rooms) and cost little more to excavate.

Specific practices in both development and exploitation arc presented in succeeding chapters on the various surface and underground mining methods. We now examine the principles.

3.2 DEVELOPMENT: GENERAL

As was indicated in Section 1.4 and Table 1.3, *development* is the work of opening a mineral deposit for exploitation. Involved in development are all the steps necessary to bring a mine into full, scheduled production. These include planning, design, соnя ruction, and other phases. Mine development fallows generally the feasibility study conducted at the close of stages I and expanded and updated as more and better information becomes available during successive phases of the project.

From the standpoint of physically opening a mine, the major purpose of development is to provide access to the ore deposit, permitting entry of the miners, equipment, supplies, power, water, and ventilating air, as well as egress for the mineral being mined and waste produced. Prior to the start of the exploitation phase of mining, development is limited insofar as possible to the construction of primary or main openings. In a surface mine, access to an ore body overlain by waste is gained by stripping the overburden (which is placed on disposal dumps and subsequently must be reclaimed). In an underground mine, small-sized openings are driven from the surface to intersect the ore body and eventually to connect with large exploitation openings. Because mainly waste material is involved, little income accrues during the excavation of development openings (any ore encountered is generally stockpiled until the mine goes into production).

Other purposes of development arc related to preparatory work, facilities, personnel, and services that support the mining and, usually, the mineral processing functions as well.

The major components of an operating mine—surface and underground—are well illustrated in Figure 3.1. Eventually, all the facilities shown (or some similar to these generalized ones) must be constructed during the development stage or early in the exploitation stage.

Factors In Mine Development

Following a successful exploration program and prior to developing a mineral property, consideration of a variety of influencing factors is necessary. Some of these may have been covered in the feasibility report, but additional information must be assembled on others (review Section 2.9). Thomas (1978) and others (Stoces, 1954; Banfield. 1973; Folinsbee and Clarke, 1981; Bullock. 1982a) discuss these factors, which may be organized in three categories.

Locational Factors. Ore bodies arc located ‘"where you find them”—and not always in the most advantageous setting. Few, for example, arc ideally situated from a business standpoint, cither for supplying or marketing. Because geography exerts a powerful influence in bringing a mine on stream but is beyond the control of the operators, we refer to its effects as locational:

1. Ease of transport of minerals to market and supplies to the mine
2. Availability of labor and support services (housing, educational and recreational facilities, health care, etc.)
3. Operational (and psychological) impacts of climate and weather

Recognizing that these factors are indigenous, a progressive mine management takes positive steps to offset disadvantages of location, for example, by subsidizing handsome fringe benefits and amenities for its work force and by "weatherproofing” its operations. Company towns are taboo in most industrial nations, but they may still be justified in unattractive regions or LDCs.

Natural and Geologic Factors. Mother Nature and geologic processes combine to govern key aspects of mine development, especially access

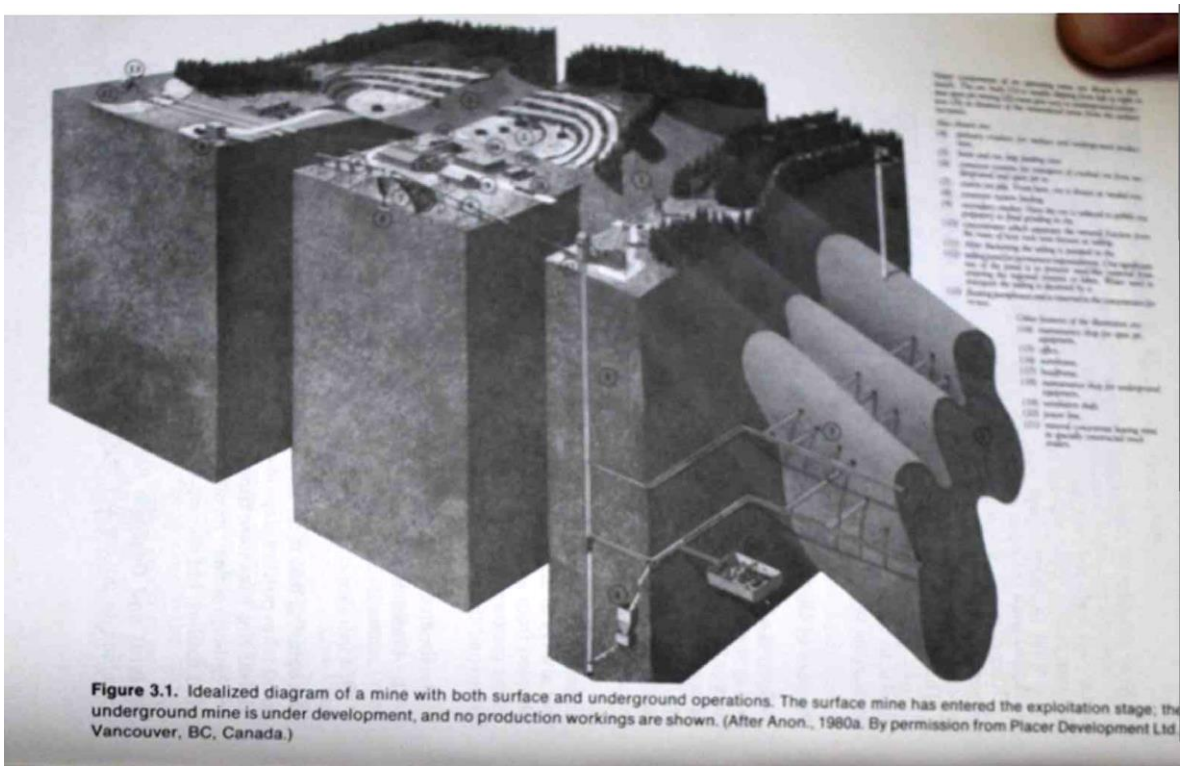


Figure 3.1. Idealized diagram of a mine with both surface and underground operations. The surface mine has entered the exploitation stage; the underground mine is under development, and no production workings are shown. (After Anon., 1980a. By permission from Placer Development Ltd., Vancouver, BC, Canada.)

openings and surface plant location Myriad factor» arc involved here, the more imptHiant ones bong the following:

1. Topography and terrain
2. Spatial relation (size. shape, altitude, etc.) of ore body, including depth
3. Geologic considerations (mineralogy, petrography, structure, ore body genesis. rock temperature gradient, presence of water, etc.)
4. Rock mechanics properties (strength, modulus of elasticity, hardness, abrasiveness, etc.)
5. Chemical and metallurgical properties (effect on storage, processing. smelting, etc.)

Factors in this category also exert a compelling influence on the selection of a mining method and operation of the mine.

Social-Economic-Political-Environmental Factors. Largely related to external forces, these factors can exercise disproportionate influence on mine development and operation. They are difficult to quantify and elusive to deal with. Some factors are:

1. Demographic and occupational skills of local populace (affect work force)
2. Means of financing and marketing (determines scale of operation, continuity of operation, etc.)
3. Political stability of host country (LDCs increasingly constitute high risks lo foreign investors)
4. Pollution legislation (air, water, wastes, etc.)
5. Other governmental aids and restrictions applicable lo the mining industry

These factors govern many critical aspects of both development and exploitation. Mining ventures judged to be attractive economically have been abandoned or bankrupted by the unforseen impacts of social, political, governmental, or environmental factors.

Sequence of Development

The steps generally carried out during mine development include the following. in rough order of accomplishment, for both surface and underground mining;

1. Adoption of feasibility report as a planning document, subject to modification as development progresses
2. Confirmation of mining method and general mining plan (see selection criteria, Section 3.7)
3. Arrangement of financing, based on confirmed cost estimates from the feasibility report.
4. Acquisition of land, including mineral rights and surface, as needed
5. Filing of environmental impact statement, obtaining of mining permit (including reclamation plan, if a surface mine), and posting of bonds subject to both federal and state statuses, as applicable.

6. Provision of surface access, transportation, communication, and power supply to the mine site.
7. Planning and constructing of surface plant, including all support and service facilities and administrative offices.
8. Erection of mineral processing plant, if required, and ore – handling and shipment facilities, and provision of stockpiling and waste disposal facilities
9. Selection of mining equipment for development and exploitation, with acquisition as needed.
10. Construction of main access opening to ore body and such secondary openings as required – including, in surface mining, advanced striping, and, in underground mining, shafts and certain other subsurface facilities.
11. Recruitment and training of labor force and provision of support services (housing, transportation, consumer stores, etc.), as necessary, with attention to other social – political – economic needs of employees

Several steps in development may proceed nearly simultaneously (e.g., steps 3, 4, and 5 in one effort, and 6, 7, 8, and 10 in another); others may be added to the list ***(e.g,.*** negotiation of shipping and sales agreements); and some may be omitted (e.g., land acquisition, if carried out during exploration). An overall planning model appears in Figure 3.2; it utilizes three levels of study (conceptual, engineering, and detailed design), as explained in Section 3.6. To ensure proper coordination and timely completion of all development tasks, careful scheduling is required, especially of ail construction, a subject dealt with in Section 3.4. Elaboration on several key steps follows.

3.3 DEVELOPMENT: LAND ACQUISITION AND ENVIRONMENTAL PROTECTION

Land Acquisition for Mining

Acquiring rights in the United States to lands and minerals is a development faced by every mining company, one chat is constrained by government regulations on mining and real estate transactions. The two (land and mineral rights) are separable here and often abroad. If they were not acquired during stage 1 or 2 (prospecting – exploration permits are sometimes obtained separately), then titles, permits, and/or leases must be arranged before actual development commences.

The classic yet current example given of the complexities of mineral ownership is the case of a third party, a natural gas company, discovering coalbed methane on a tract of land where the surface rights are owned by one party and leased to a second (a mining company) that has purchased the coal rights (Farnell, 1981). Questions: Who “owns” the coalbed methane? Is its production governed by the state regulatory body on natural gas? Or mining? Can coalbed methane be “severed” from the coal as a mineral right? Who pays royalties to whom?

Resolution of the legal issues is still pending in the courts, both at the state and federal levels.

Mining law in the United States pertaining to mineral ownership leasing, and surface rights is a branch of the law of real property Parr and Ely 1973). The applicable federal statutes are found in the Mining Law of 1872 (ownership) and subsequent legislation (leasing), as outlined in Section 1.8, While too lengthy and involved to be more than summarized here, provisions of these acts are contained and analyzed in publications of the Rocky Mountain Mineral Law Foundation (Anon., 1967a) and the U.S. Bureau of Mines (Ely, 1970).

While statutes, federal and state, apply to mineral rights on government lands, local ordinances and private contractural arrangements usually govern mineral transactions on private lands. Thus four distinctions in acquiring legal rights to minerals are recognized (Parr and Ely, 1973),

1. ***Ownership of public land.*** The federal government controls, wholly or partially, 38% of the land in the United States, known as the public domain. That portion of federal lands on which mineral rights may be claimed is administered mainly by the Bureau of Land Management (BLM) and termed *locatable lands:* excluded are national parks, wilderness areas, Indian tribal lands, and military reservations. Nearly all locatable lands lie in the western states. The federal law permitting acquisition is applicable mainly to metallic minerals; fuels and most nonmetallics are excluded. Following discovery and minimal development, a claim may be patented for a nominal fee. *Patenting* conveys title to the discoverer, who is then free to commence mining operations. In recent years, however, it has become increasingly difficult to acquire mineral rights in this manner because little promising, locatable land remains.

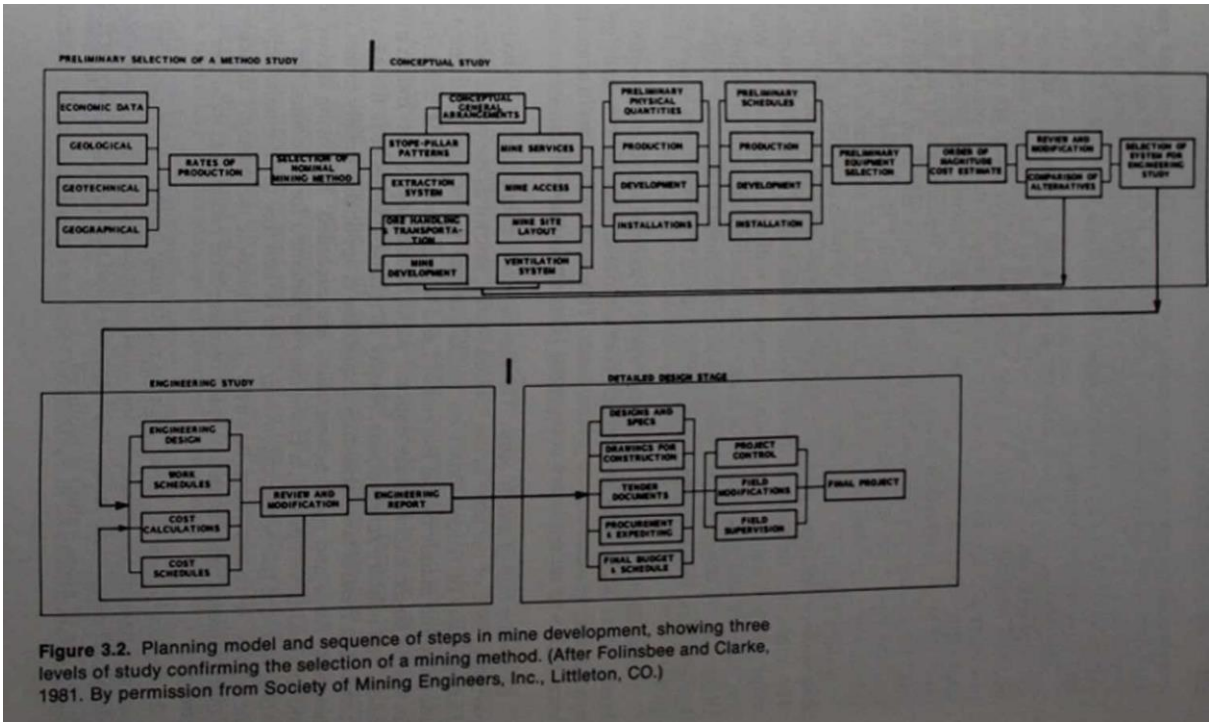


Figure 3.2. Planning model and sequence of steps in mine development, showing three levels of study confirming the selection of a mining method. (After Folinsbee and Clarke, 1981. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

2. ***Leasing of public land.*** Mining rights may be leased for coal, petroleum and natural gas, uranium, and most nonmetallic minerals occurring on federal and many state lands. Payment of a bond and royalties (varying from 5 % to 15 % of gross value) is required.

3. Ownership of private land. Formerly common, outright purchase of surface rights is less frequently practiced today. Reasons arc the escalating cost of real estate and the huge capital outlay required for a mining operation of even nominal size. Ownership of mineral rights is still customary in metal mining but is on the decrease in coal and nonmetal mining. Purchase costs are widely variable for metallic ore deposits, can range from Si.00 to $8.00/ ton ($ 1.10 to S9.00/tonne) or about 20% of the selling price for coal, and may be only a few cents per ton for sand and gravel (Bourne, 1983).

4. Leasing of private land. Transactions to acquire mineral rights by mining companies today increasingly involve leases (or lease with option to purchase). On private lands, royalties usually vary from 4-10% of gross value for coal and nonmetallics to 8-12% for metallics (Bourne, 1983). Royalties vary with deposit grade and size, market conditions, transportation cost, and investment climate. Both purchase and leasing agreements often involve deferred payments or options or are made contingent on future production, desirable features for the mining company.

Arrangements that permit multiple use of land—surface and subsurface— are becoming increasingly common in the United States (Banfield, 1973; Parr and Ely, 1973). Mining in the future may have to share or preserve surface rights for agricultural or recreational use. In some areas, different mineral deposits occur in the same stratigraphic column. For legal requirements of shared use on federal land, refer to the Multiple Surfacc Use Act.

Environmental Protection During Mining

Complying with the multitudinous federal and state statutes designed to prevent environmental harm during mining has become the major non-production-related task of mine development (Section 1.7) (Down and Stocks. 1977; Pachter, 1985). Nor does it begin nor end there; compliance with environmental quality legislation—principally dealing with air, water, land, and waste—starts during exploration and continues through the postexploitation stage of mining, with special emphasis on reclamation of the land surface in surface mining.

Requirements for environmental compliance are specified in the laws; discussions and summaries are plentiful in the literature, among the most helpful being ones by Brooks and Williams (1973) for surface mining, Parr (1982) for underground mining, and Dempsey (1975) on genera! compliance. The procedure of demonstrating intended compliance with environmental regulations in obtaining authorization to mine is termed *permitting.*

The greatest impact of all the environmental statutes in the United States today is contained in the briefest legislation, the National Environmental Policy Act (NEPA) of 1969. Every mine planner has occasion to refer to it, because NEPA requires the advance preparation of an environmental impact statement. EIS, for almost every mining project undertaken (Section 1.7 and 2.9). Its purpose is to ensure that the environmental effects of a

proposed activity are fully understood, so that adverse effects are contained within prescribed limits. From 18 to 36 months ordinarily is required to prepare and obtain approval of an EIS – and longer if the project proves controversial. Litigation may result during or even after the process.

NEPA compliance guidelines are set by the federal Council on Environmental Quality (CEQ) and enforced by CEQ through the Environmental Protection Agency (EPA) and various state agencies. The guidelines require the following (Parr,1982);

1. Description of project and environmental aspects.
2. Relation to land use plans of intended areas, usually obtainable from BLM
3. Probable impacts on environment, both favorable and unfavorable, primary and secondary
4. Proposed plan to contain adverse effects
5. Possible alternative plans

Because of the far-reaching consequences of environmental legislation to the mining industry, reflected in higher operating costs, it is highly advisable for the project management to conduct its own advance environmental study *(1)* (o determine if an EIS will be required and *(2)* to identify and assess problem areas, utilizing the tools of risk assessment and risk management (Davis, J9fM|. Voluntary compliance and full disclosure on environmental matters by a mining company pays dividends in its public relations and may actually save it money and time. Pollution control currently costs the mineral industry 17% of its total capital expenditures, over three times more than other industries spend (Dorr, 1984).

3.4 DEVELOPMENT: FINANCING AND IMPLEMENTATION

Financing of Mining Ventures

Mining is considered a high-risk industry for investment purposes. More correctly, the development of new mines is classed as a speculative venture. "Time and effort are lavished on the study of a mine's feasibility, since the risks diminish in inverse ratio to the extent and quality of analysis'* (Anon., 1980a). After technological feasibility is demonstrated, then economic feasibility must be established (Berry. 1984). Presumably, the feasibility report prepared at the conclusion of stage 2 (exploration), if favorable, builds the case for both, at least preliminarily.

In financing a mining venture, the developing company must both decide on and arrange a means of financing. Factors to consider are terms of the land acquisition agreement (lease, purchase, etc.); marketing arrangements (long-term contracts arc almost a necessity); time required to repay any loan incurred (preferably the same as the life of the mine); risk factor of the venture (set by lending agencies); the political climate of the host country; and the possible operation of the mine as a joint venture with another company. Ideally, funds become available when development expenditures begin and continue at a rate to match spending, until the mine generates sufficient revenues to cover

operating expenses. Financing of thc earlier prospecting-exploration stages may also have to be sought.

Alternative forms of financing popularly used to develop mines are as follows (Stewart and Lindley, 1971; Brooks and Hursh, 1982);

1. Loans from commercial banks and private investment sources. such as other corporations, trust funds, insurance companies, or foreign firms, unsecured or secured by collateral (sometimes arranged by lenders on a consortium basis, if amount needed is large)
2. Issues of securities, stocks and bonds, through investment houses or banks (a form of internally generated financing that avoids borrowing secured by collateral)
3. Rental of equipment by leasing firms (avoids debt financing, but limited in application)
4. Government loans—U.S., foreign, or international—for special-type projects (Synthetic Fuels Corporation loans, subsidies, etc. for oil shale development; World Bank or International Monetary Fund for projects in Third World countries, etc.).

Currently, U.S.-based mining companies rely about 50% on commercial loans, 25% on internal financing, 15% on leasing, and 10% on special government funding. A "combination" financial package is increasingly common.

Implementation of the Mine Development Plan

Once financing is secured, a critical milestone in the life of a mine has been passed. Mine development can commence in earnest and in fact does—and a great many things begin to happen all at once.

Speed is now of the essence (Anon., 1980a). Interest costs on any borrowed money begin to accrue immediately and will continue through the life of the mine. And while changes in the original feasibility report are to be expected, too many changes can be costly to implement in terms of both lime and money. Whether the design and construction arc done in-house or by outside contractor, every effort is made to get the mine into production at thc earliest possible moment. Termed fast-track development, the strategy minimizes front-end expenditures to realize an early cash flow from operations (Lightner et al.. 1983).

Referring to the typical sequence of events in mine development (Section 3.2), steps 6, 7, 8, and 10 comprise the *construction phase* and are undertaken in carefully coordinated fashion. Many factors can influence the construction schedule, including labor contracts, equipment deliveries, management skill, and weather delays. Site clearing and excavation for surface facilities, for example, can he carried out in bad weather, but foundations and main-building erection are best accomplished in summer and fall. Underground development, of course, can be done year round.

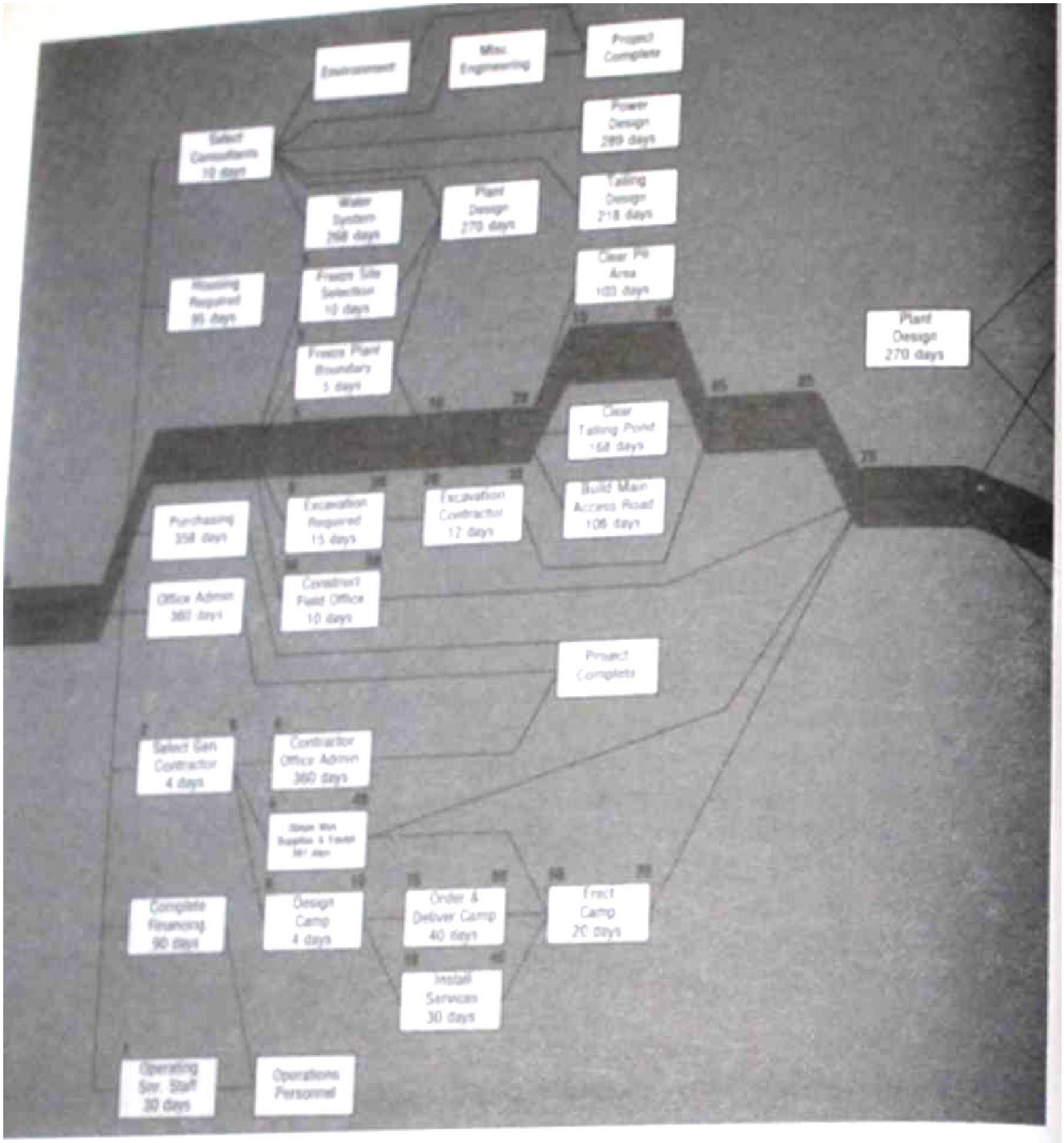


Figure 3.3. CPM (critical path method) network diagram for planning and scheduling mine construction tasks The shaded sequence of tasks is the critical path. (After Anon., 1980a By permission from Placer Development Ltd., Vancouver, BC. Canada)

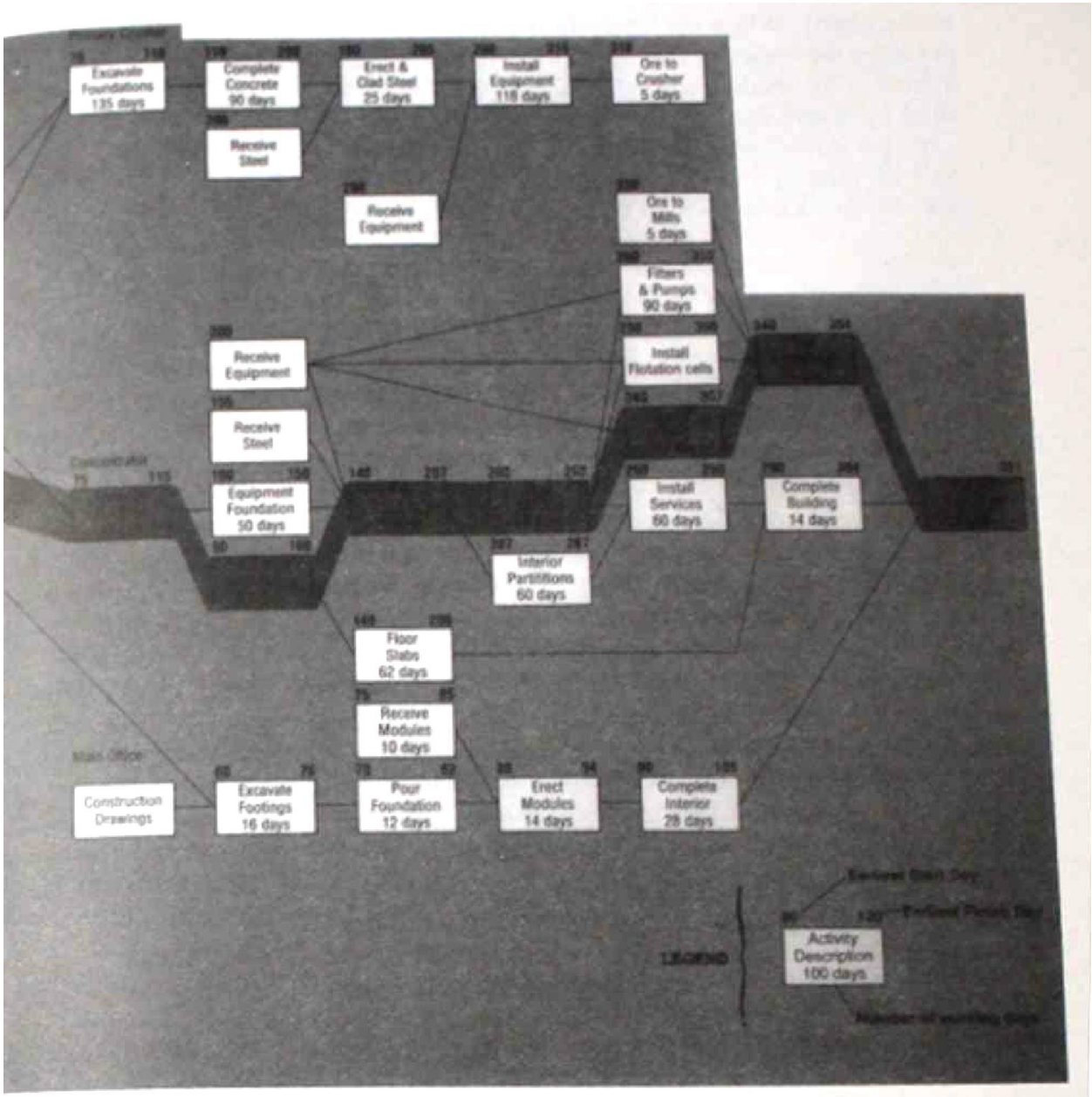


Figure 3.3. (*continued*)

Because of the complexity of mine construction, the scheduling and coordination of tasks is best done by applying operations research techniques, such as CPM or PERT, aided by CAD (computer-assisted design) (Anon., 1У80a; Gibson et al., 1982); see Section 2.7. CPM scheduling for a large mining project is illustrated in Figure 3.3. To prepare the chart, a comprehensive list of the hundreds of individual tasks is prepared. The time element for each task is then estimated and its sequential relationship to other tasks established (perhaps 1000 employee-hours would be required to prepare the entre chart). When the analysis is completed, a critical path emerges that portrays the series of events controlling the completion date of the project, indicated by shading in Figure 3.3. The completion time can only be shortened by rearranging or improving individual task times.

In the construction phase, one decision looms as critical and must precede all others, and that is selection of the plan site (Banfield, 1973; Bullock, 1982a). There groups of factors require consideration:

1. *Economic:* outside access, surface rights, labor and living conditions, power supply, water rights, means of transportation, government restrictions, building costs and contracts, royalties, and taxes
2. *Terrain:* space availability, topography, climate, weather exposure, drainage, and vulnerability to natural disasters
3. *Environmental:* air, water, soil, and waste pollution, reclamation, subsidence, noise, blasting damage, benefication, and public relations

It is instructive to compare this list with the general factors specified for mine development (Section 3.2).

All the physical facilities provided to operate a mine arc referred to as the mine plant. (If the mine is an underground one. the plant may be subgrouped into surface, shaft, and underground facilities.) The first principle in selecting the plant site and overall layout is to locate facilities in close proximity to the ore body and main mine openings without interfering with each other or with mine production. The second principle is to avoid having to relocate parts of the plant because of expansion or competition for space with the other facilities, encroachment by the mine itself, or subsidence of underground workings.

3.5DEVELOPMENT: TAXATION AND COSTS

Mine Investment Costs

In the case of the Bougainville copper project in Papua New Guinea (Section 2.9), begun in 1964, S350 million was required to finance mine prospecting, exploration, and development. No income was generated for 8| years, and probably another 10 years elapsed before the mine showed a profit. Another noted example is the White Pine mine (underground copper) in Michigan; it required II years after start-up to return a profit (Boyd. 1967); see Section 15.3. Neither project is an exception among world-class mines, recalling (Section 2.1) that today, prospecting-exploration costs alone to make a major discovery average a staggering $290 million. Total mine investment costs in the billion-dollar range have already been attained.

Taxation

Taxation of mineral property anti mining operations is unique, variable, and complex, both in the United States and elsewhere, and is a cost item to estimate during mine development. No other cost (the total amount of taxes charged is usually computed and deducted from gross revenues, along with operating costs, in determining net profit) is so difficult to project with accuracy. In part, this is traceable to the peculiar nature of mineral deposits, a *wasting* (or depleting) *asset,* which results in a mining company being permitted to claim a *depletion allowance* applied to its net income and based on the declining value of the unmined mineral. Depiction rates vary with the mineral, from 5% for most nonmetailic ores to 8% for coal to 15% or 22% for most metallics (Anon.,

1984b). Like other businesses, mining is also entitled to deduct depreciation on plant and equipment, but in addition can claim a deduction for mine development expenses (Halls, 1982),

In the United States, taxes on mining enterprises are levied by the federal government, most states, and some other political jurisdictions. Basically, they take these three forms: (1) income tax on earnings (revenues); (2) ad valorum tax on real estate or mineral reserves; and (3) severance tax on production (of minerals "severed" from the earth). Federal income taxes currently arc based on net income, with rates to 34%.

For specific provisions of U.S. mine taxation laws, consult the Internal Revenue Service and the appropriate state tax agency or any of several references (e.g., Stinson, 1982; Anon, 1983c; Schenck, 1984).

Cost Estimation

The engineering estimation of development or any other mining cost is a task that the beginning student is not, at this point, prepared to undertake. The procedure, complete with case studies and cost estimates, however, is illustrated in a useful compendium. *Mineral Industry Costs* (Hoskins, 1982), that covers all four stages in the life of a mine. Revised periodically, this publication has the advantage of currency, always a major concern in the arena of cost estimation. In succeeding chapters, we will develop our own procedure for estimating costs.

As an example, consider a ease study given by Hoskins (pp. 30-31) for a 2000 ton/day (1800 tonne/day), underground, lead-zinc mine on state-leased land. Required: Determine the total and unit costs of environmental control, including reclamation, over the life of the mine. This cost tabulation results:

Stages 1,2	Prospecting-exploration	3 yr	$ 30,000-90,000
	Feasibility study	2-3 yr	1.200,000-3,500,000
Stage 3	Development	2 y r	200,000-500,000
Stage 4	Exploitation	—	100,000-300.000/yr
	Reclamation-closure	5-7 yr	400,000-1,000.000

If the production life of the mine is projected as 20 years, then stages 4 environmental costs are $2 0-6.0 million, plus closure costs. Total environmental costs are thus estimated to be $ 3 .8 - 1 1.9 million over a period o f 32-35 years. Assuming total ore production over the life o f the mine o f 10 million tons 19 million tonnes), then the unit cost for all environmental work can he estimated as $0.38-1.19/ton <$0.42-1.32/tonne).

Early in its life, we also need to estimate roughly the capital investment cost to develop a mine. A common practice is to base the cost on a unit capacity factor,

in f/annual ton (tonne) of production. These factors vary widely with the type of mining, hut the following ranges are commonly employed (mine plant only):

Coal M IN U S

Surface	2S-IOO/ton/yr	($27-110/tonne/yr)
Underground	60-125	(65-140)
METAL M IN ES		
Surface	50-150	(55-165)
Underground	75-200	(85-220)

For example, if a deep coal mine is designed to produce 1.5 million ton/yr (1.4 million tonnes/yr), the capital investment cost will range from about $90 million to $190 million – a wide variation but helpful as a quick estimate.

Estimates of overall development costs are also needed, and values provided in Table 1.3 may be helpful in establishing the range of such costs. Over a period of 2-5 yr. development costs can vary from $10 million to $250 million or more, for a unit cost of $0.25-5/ton (SO.27-5.50/tonne).

3.6EXPLOITATION: GENERAL

Exploitation, stage 4 in the life of a mine, is not only the culmination of the three stages that occur prior to it, but it is the end process by which the others are economically justified. Without production of ore (or coal or stone) at a substantial, sustained rate, there can be no opportunity for a mining venture to succeed, and the mine is aborted or stillborn.

Exploitation is the work of recovering mineral from the earth in economic amounts and delivering it to shipping or processing facilities on the surface. While some exploration and development continue during stage 4, the chief objective of mine exploitation is geared toward mineral production. In the exploitation process, a number of extractive unit operations are employed, the primary ones constituting the ***production cycle*** and the secondary ones the ***auxiliary*** or support ***functions.*** We will examine these unit operations in detail in Chapter 4.

Production openings excavated in the mineral deposit tend to have names unique to the type of mine and locale (surface or underground) and to the commodity mined (coal, metal, or nonmetal)—in part because these kinds of mining have evolved with terminology distinctions the world over. On thc surface, for example, we excavate *pits* or *cuts* in all kinds of mineral deposits, but underground we call production openings *rooms* or *longwalls* in coal mines and *slopes* in noncoal mines.

A review of Section 1.4 reminds us of the present predominance of surface mining over underground insofar as tonnage output is concerned. What the future holds is not clear, although the trend since the 1940s has been toward surface mining (Pfleider, 1973a). That trend, however, has nearly leveled out sincc the early 1970s, and indications arc that

it may reverse. Certainly all the signs favor a gradual resumption of underground mining, al least in the United States (e.g., deeper deposits being discovered and mined, low-grade, near-surface deposits becoming more marginal). Nevertheless, that prophecy has proven erroneous before; and thanks to massive, more efficient surface mining machines, the trend back lo underground mining may be postponed. Specific arguments favoring one class or method of mining over another are best saved for later chapters.

The method chosen for exploitation distinguishes the final stage in the life of a mine. Selection of the mining method is the very crux of the exploitation process and probably the key engineering decision made also in mine development. The procedures presented for development earlier in this chapter now need to be reviewed and expanded in view of exploitation objectives and the procedure for the choice of mining method fully explained.

3.7EXPLOITATION: MINING METHODS

Method Selection

The cardinal rule of mine exploitation is to select a mining method that best matches the unique characteristics (natural, geologic, environmental, etc.) of the mineral deposit being mined, within the limits imposed by safety, technology, and economics, to yield the lowest cost and return the maximum profit. Let us now examine the factors which govern method selection (Morrison and Russel!, 1973; Boshkov and Wright, 1973).

Factors *in Selection*

1. *Spatial characteristics of deposit.* These factors arc probably the most important determinant, because they largely decide the choice of surface vs. underground mining and affect the production rate, the method of materials handling, and the layout of the mine in the ore body.

a. Size (dimensions, especially height or thickness)
b. Shape *(*tabular, lenticular, massive, irregular)
c. Altitude (inclination or dip)
d. Depth (mean and extreme values, stripping ratio)

2. *Geologic and hydrologic conditions.* The geologic characteristics of both the mineral deposit and adjacent country rock (host material) influence method selection, especially choices between selective and nonselective methods and extent of support required for ground control underground. Hydrology affects drainage and pumping requirements, both surface and underground. Mineralogy governs mineral processing requirements.

a. Mineralogy and petrography (sulfides vs. oxides)
b Chemical com position *(*primary, by-product minerals)
c. Deposit structure (folds, faults, discontinuities, intrusions)
d. Planes of weakness (joints, fractures, cleavage in mineral, cleats in coal)
e. Uniformity, alteration, weathering (zones, boundaries)
f. Groundwater and hydrology (occurrence, flow rate, water table)

3. *Geotechnical (soil and rock mechanics) properties.* Again, both ore and waste are involved. The mechanical properties of the materials comprising the deposit and country rock (and soil, if overburden) are the key factors in selecting the equipment in a surface mine and choosing among the classes of methods (unsupported, supported, and caving) if underground.

a. Elastic properties (strength, modulus o f elasticity, Poisson's ratio, etc.)
b. Plastic or viscoelastic behavior (flow, creep)
c. State of stress (original, modified by mining)
d. Consolidation, compaction, and competence (ability of opening to stand unsupported)
e. Other physical properties (specific gravity, voids, porosity, permeability, moisture content)

4. *Economic considerations.* Ultimately, economics determines the success of a mining venture. These factors govern the choice o f method because they affect output, investment, cash flow, payback period and profit.

a. Reserves (tonnages and grades)
b. Production rate (output per unit time)
c. Mine life (operating period for development and exploitation)
d. Productivity (output per unit of labor and time-e.g., tons or tonnes / employee – shift)
e. Comparative mining costs of suitable methods

5. *Technological factors.* The best match between natural conditions and mining method is sought. While a particular method may not be ruled out in mining, it may have adverse effects on subsequent operations (e.g., processing, smelting).

a. Mine recovery (portion of deposit actually extracted)
b. Dilution (amount of waste produced with ore)
c. Flexibility of method with changing conditions
d. Selectivity of method lo distinguish ore and waste
e. Concentration or dispersion of workings
f. Capital, labor, and mechanization intensities

6. *Environmental concerns.* Not only the physical environment but the social-political-economic climate is involved.

a. Ground control to maintain integrity of openings
b. Subsidence, or caving effects on the surface
c. Atmospheric control (ventilation, quality control, heat and humidity control)
d. Work force (recruitment, training, health and safety, living, community conditions)

Guidelines and Procedure. The basic objective in selecting a method to mine a particular mineral deposit is to design an exploitation system that is the most suitable under the actual circumstances (Hamrin, 1982). Experience plays a major role in decision making, which of necessity involves a large measure of judgment. Reaching an optimal solution, however, is considerably facilitated by the use of quantitative and engineering evaluation, including operations research techniques, aided by computerized information and data processing.

We are now in a better position to understand the planning model utilized for development and method selection in Section 3.2 (Folinsbee and Clarke, 1981). Engineering evaluation is carried on at three levels, in the manner depicted in Figure 3.2. First, in the *conceptual study*, the physical characteristics and output quantities of a number of mining methods, layouts, and systems are assessed. Next, in the *engineering study*, the preceding concepts arc quantified and compared, resulting in firm designs and costs. Finally, in the *detailed design study*, drawings and specifications for construction for the preferred method are prepared. The result is a final engineering герои on which investment decisions, equipment purchases, and a construction schedule are based.

In some cases, the conditions attached to a proposed mine project are quite distinctive, and the choice of a mining method may seem obvious. In other cases, several candidate methods appear equally suitable. Engineering evaluation is essential in either situation, however, and shortcuts are not to be tolerated. Although one or more standard methods may be applicable, an optimal solution may require modifying or hybridizing an existing method. New and better mining methods have evolved in this way. Few mines can afford to follow the textbook slavishly or copy exactly the solutions devised by other mines. However, once a decision as to methods is made, and development and exploitation get underway, it is extremely difficult to alter the plans or change to another method. Selection factors and procedures comprise much of the content of the chapters that follow.

Classification of Mining Methods

A variety of historical schemes exists to classify and help select mining methods (Pecle, 1941; Young 1946; Lewis and Clark, 1964), of which the oldest is probably still the best. The basis for method classification in these instances was some subjective combination of the spatial, geologic, and geotechnical factors we have just discussed. Recent schemes have introduced more quantitative or systematic approaches hut use the same basic approach as Pecle (Morrison and Russell. 1973; Boshkov and Wright. 1973; Thomas, 1978, Nicholas. 1981, Himrin. 1982) We will have occasion to refer to several of these again later (Sections 8,4, 13.4. and 15.4), after we have studied the methods themselves in some detail, want to compare various methods, and are prepared to undertake method selection on our own. Broadly, a failing of all existing schemes is that they are incomplete, coverining mainly underground, noncoal methods.

For our purposes now, we require and will devise a generic classification of mining methods that (I) if general (i.e.. applies to both locales of mining, surface and underground, and all commodities, coal and noncoal), but is not excessively detailed; (2) includes all current major methods and promising novel ones, under development but largely unproven; and (3) recognizes the major class distinctions and relative costs. The scheme was utilized in Chapter I when exploitation methods were first discussed. (Review of Section 1.4 will suffice for now to distinguish the methods, described more fully later.)

The categories utilized in our classification are *acceptance* (traditional or novel), *locale* (surface or underground), *class* and *subclass,* and *method,* with applications as to *commodities* and *relative cost.* Table 3.1 displays the mining method classification that we are adopting.

Features of the various methods are yet to be examined. These include a depiction of the method, sequence of development, cycle of operations, deposit conditions, advantages and disadvantages, production rate (large scale vs. small-scale), relative cost, and examples. A survey procedure is suggested in Section 3.9, employing mainly relative terms to compare conditions and features of the different methods. These characteristics are then collated and compared in summary tabulations of surface methods (Chapter 8), underground methods (Chapter 13), and all mining methods (Chapter 15).

The majority of our attention will be devoted to a handful of traditional methods, the most important and commonly used in this country. They number seven, and are marked by an asterisk in the table. Notice that two are surface methods, and that three are associated with coal mining. Cumulatively, they account today for probably 90 % of all U.S. solid mineral production.

3.8Exploitation: Organization and Costs

Mine *Organization and Administration*

By the time the exploitation stage of a mine begins, the organization of its management and work force should be complete. Personnel may be added or dropped from the mine complement as growth or reduction in output occurs, but the basic organization is in place and remains unchanged. Actually, recruitment of the staff and labor force was carried out at the close of the development stage, synchronized with the opening of the mine.

The administrative structure employed by most mining companies the world over is a *staff- and-line organization (*Boyd, 1973). Mining and its subsequent processing operations are technologically and economically complex, requiring the staff services of experts. They are also diverse, so that *line* supervision is required in each production unit.

The functions of management largely determine the line organization and staff assignments, as they do in all business enterprises. In a mine, production is supported by the functions *o f* planning (central coordination), engineering, finance and data processing, personnel and industrial relations, safety and environmental control, legal affairs, public relations, and research and development. Approximately three employees in support functions are required for every two in production (Anon.. 1980a). The organization chart for a large mining operation, vertically integrated to include processing and smelting as well, is shown in Figure 3.4. It is of the staff-and-line type, with the separate functions identified and classified.

Details o f mine organization, administration, and operation are best left for an advanced text. For current, complete works on the subject, refer to Britton (1981) and Sloan (1984).

The special socio engineering obligations of mine management are well stated by Boyd (1973):

Mining companies are groups of people working together using materials and capital to produce goods useful to society. In this total effort, hundreds of thousands of people use billions of dollars. Management means assessing the needs of society and providing the benefits of the mining industry at the lowest feasible human and dollar cost. Management techniques arc tools used to help provide what society warns. These tools have costs and benefits. They are not intuitive tools but must be carefully learned, and the correct tools must be used for each situation.

Mining Costs

Building on the discussion of cost estimation introduced during mine development (Section 3.5), we can extend the process to include exploitation costs as well.In addition to the data sources *(Hoskins, 1982) referenced then. Halls (1982) provides* cost estimation procedures and *problems* lo explain mine operating costs.

For the present, and elementary understanding of cost terminology will suffice. (Budget nomenelature is coveret Section 15.3). The sum of all direct costs associated with bringing a mine into production through the four stages of prospecting, exploration, development, and exploitation is called the direct mining cost. If calculated on a gross basis, it is a total cost; if on a unit (S/ton, or S/stone) basis, it is a unit cost. In addition, indirect mining cost is an overbead that usually includes an allowance of 5-10%for administration, engineering, and other nonitemized services. To find the overall ming cost, sum the two, either on total or unit basis:

Overall mining cost = direct cost + indirect cost

If to this figure is added all other costs *(processing,* smelling, transportation, etc.), then the *overall production cost* results.

From Table 1.3, *we* may approximate *exploitation costs,* which include both direct and indirect charges. The unit cost ranges from $2 to $ 100/ton ($2.20to $110tonne). *The* total cost varies from $5 million to $50 million/yr, over a customary mine *life o f* 10-30 yr. *If* we then wish to determine the overall mining cost, we must sum *the* individual costs, unit or total, for all four stages *of* mining (assume a mine life of 20 yr for stage 4):

Stage	Unit cost	Total cost
1.Prospecting	$0.02-0.50/ton	$ 0.1-5 million
2. Exploration	0.10-1.00	0.5-10
3. Development	0.25-5.00	10-250
4. Exploitation	2.00-100.00	100-1000
	$2.37-$106.50/ ton	$110.6-1265 million $I 10-1265 million
Overall mining cost say.	$2.40- 107/ton ($2.70-119/tonne	

These final figures include all direct and indirect costs in mining. It is important to recall, however, that the information taken from Tabic 1.3 is intended only to demonstrate the range of costs in mining and not to provide detailed cost estimates.

.

Because the determination of costs is imperative for the mining engineer to master , cost estimation examples will occur throughout this text. Because actual costs are difficult to obtain (they are considered proprietary by industry) and fluctuate unprediatably over time with price inflation, changes in labor rates, and technological progress, it is of little value to cite and memorize the figures themselves.(To update old costs, one can employ cost-of-living or consumer price indeces compiledby the U.S. government, but the results are approximations at best. Undated costs used in this text may be taken as current at the time of the writing.)

To avoid use of absolute costs(in $/ton, or $/tonne) in our forthcoming discussions of mining methods-which rapidly become obsolete and useless- it will be more helpful if we employ *relative costs* (in *%)* for comparisons. Notice that Table 3.1 adopts that convention. Arbitrarily, the mining cost of the most expensive methods (quarrying and square set stoping) is established at *100%;* the costs of ail other methods are expressed relative to that standard. Thus the cheapest method is dredging *(<5%).*

Even relative costs are limited in value and must be used with caution. Primarily, they arc intended to facilitate economic comparisons between mining

methods. No dollar equivalence can be attached to them, at least not ***with*** any confidence. Relative costs represent average conditions that can be substantially in error for actual mines if converted to absolute costs. For the determination of absolute costs, see Sections 13.7 and 15.3.

3.9 *SPECIAL TOPIC: MINING METHODS* **NOTEBOOK**

Probably the surest way to master the intricacies of the various mining methods is to (I) sketch each method, (*2)* describe how it is developed and operated, and (3) summarize its distinctive conditions and features. Figure 3.5, ora similar form, serves as a convenient reminder of this discipline.

In the course associated with this text, a form is filled out by the **student** for each method studied. The information required is obtained from **the** chapter (or assignments) dealing with that method, it is the **student's** job to distill it, condense it, and summarize it neatly on a form prepared from Figure 3.5.

At the conclusion of the course, the forms for all assigned methods **are** assembled in a *Mining Methods Notebook.* Thus a lasting, **well-engineered** summary record is produced for future reference.

4

UNIT OPERATIONS OF MINING

4.1 FU N D A M EN TA L O PER A T IO N S AND C Y C LE S

Throughout the latter three stages of mining, commencing with drilling and excavation during exploration and continuing through development and exploitation, certain fundamental operations are performed to free and transport the material being mined. As introduced in Section 1.5, these basic steps are referred to as the *unit operations o f mining.* If they contribute directly to mineral extraction, we call them *production operations* ; *auxiliary operations* support the main mining activity but are usually not directly part of it unless essential to worker safety or operating efficiency. Our interest is primarily in those production operations employed for development and exploitation.

The material extracted during mining varies widely, from unconsolidated soil or broken rock to the toughest and most compact rock in place (e.g., gabbro, quartzite, jasper, taconite). Both waste and ore (or coal or stone) typically are involved. Further, they must be transported and disposed of, ultimately, to mineral-processing, shipment, or waste disposal facilities.

Two basic functions, then, are required in mineral extraction: breakage and handling. Since only well-consolidated materials have to be freed from the earth, the first function is more specifically termed *rock breakage.* In most mines, breakage is accomplished by drilling (rock penetration) and blasting (rock fragmentation). Materials handling is usually performed in two steps, loading (excavation) and haulage. If considerable vertical lift is involved, then hoisting may be required as well. Each of these four fundamental operations— drilling, blasting, loading, and haulage— is discussed as to principle in this chapter and application in subsequent chapters (see especially illustrations in Sections 5.3 and 9.4).

Unit operations are characterized mainly by the equipment that performs them. Mining today is almost totally mechanized. Thus distinctions between them unit operation in surface and underground mining are mainly a matter of scale (but note again the modern scale of underground work in Fig. 1.1 The equipment used in both is remarkably similar, in principle and function, as we shall see.

The sequence of unit operation utilized to accomplish mine development or exploitation is called the cycle of operation. In most mining, as explained above, there are four basic operations in the production cycle:

Basic production cycle = drill + blast + load + haul

The production cycle is used or modified to suit conditions. For example, in "hard-rock" mining (the ores of most metals and nonmetals fall in this category), the basic cycle is used in nearly all surface and underground extraction. An exception is dispensing with drilling and blasting in stripping waste or mining ore that is loose or unconsolidated (soil, weathered rock, placer material, etc.). The basic cycle (with the same exception) is also applicable to surface coal mining, although the coal itself may be broken and excavated mechanically and blasting avoided. A departure customarily occurs in the underground mining of coal and the softer nonmetallics (e.g.. salt, potash, trona), there cutting of the coal or ore precedes blasting or continuous mining replaces the separate operations of drilling, blasting, and loading An extension of mechanical excavation to soft and medium-hard rock is also occurring with shaft-sinking, raise-boring, and tunnel-boring machines that eliminate drilling and blasting. Finally, the basic cycle is modified in quarrying, where the stone generally is not blasted but channeled, cut, or sawed.

The term cycle implies that mining operations are cyclic in nature, which (he majority are. It is interesting to note (above) that in two of (he more radical departures from the basic cycle (continuous mining and tunnel boring), (he resulting cycle of operations is (1) streamlined and (2) more continuous. Both represent improvements that the mining industry sorely needs if it is to progress from essentially an intermittent technology to a continuous one. The ultimate in mining, however -a truly continuous extraction system that breaks and moves material without interruption — is still some years away.

4.2 DRILLING AND ROCK PENETRATION

Rock Breakage

The freeing or detaching of large masses of rock from its parent deposit is termed rock breakage Man made a giant leap forward in prehistoric times when he devised a method to break rock by fire building and water quenching, an ingenious way of utilizing thermal stresses to overcome the rock's cohesive strength (Section 1.3). When he eventually discovered explosives and ventured to use them to blast rock, he loosed an awesome energy source and in so doing made the greatest technological advance in mining of all times. Without knowledge of rock mechanics, he had succeeded in amassing and focusing energy in amounts sufficient to break rock.

Interestingly, early man first displayed the ability to work rock in ways unrelated to mining. With the addition of the first sharpened, stone-tipped weapon to his arsenal, he demonstrated the skill to break and shape rock in a controlled way to suit his purposes. He applied energy, focused and concentrated, to overcome the strength of rock by chipping, abrading, and so forth. From experience, these early craftsmen learned—again, without benefit of an understanding of rock mechanics—how to apply stress in tension or shear so as to

capitalize on the weaknesses of the brittle pieces of rock they carved into ax blades, spear points, and, eventually, arrowheads.

Mining today exists with the same basic objectives that it did in ancient times and employs the same basic elements of the production cycle, breakage and handling. It was with the introduction of blasting into the cycle in the seventeenth century, however, that a flood of improvements in production operations began, culminating in revolutionary new machines and processes of the nineteenth and twentieth centuries. Mechanization had come to mining. and with it a realization that utilizing abundant amounts of energy allowed more rock to be broken and transported per unit of time and labor.

Generally, two different kinds of rock breakage operations are performed in mining. In *rock penetration* (drilling, culling, boring, etc.), a directed hole or kerf is formed, usually mechanically but sometimes hydraulically or thermally, (1) for the placement or relief of explosives in blasting or for other purposes requiring small holes, (2) to produce a finished mine opening or tunnel, or (3) to extract a mineral product of desired size and shape (dimension stone). In contrast, *rock fragmentation* aims to loosen and fragment large masses of material, conventionally by chemical energy in blasting but additionally by mechanical, hydraulic, and novel applications of energy.

Although operating on different scales, penetration and fragmentation function by applying energy through a variety of similar basic mechanisms to break rock. What is also remarkable in concentrating and focusing that energy to overcome the strength of the rock is that the rate of application is as critical in producing failure as the energy form and amount. Thus we note that all successful rock breakage processes arc time-dependent and that, in general, the more rapid the application, the more effective. In rock mechanics terms, we employ *dynamic loading* (in preference to *static loading*. except in caving methods of mining) to accomplish penetration or fragmentation—although it must be conceded that drilling is orders of magnitude slower than blasting as a dynamic-breakage process.

Mathematically, this is equivalent to saying that the amount of rock damage accomplished, usually measured as the weight W or volume of rock broken, is proportional to the energy E consumed in the process:

$$W \neq E$$

This basic relation is useful in blasting and other fragmentation processes. In drilling and other penetration processes, we are more concerned with the rate of advance R, the penetration per unit time, than the amount of rock broken; and hence the basic relation becomes

$$R \propto d E/ dt$$

where dE/dt is the time rate of energy application. Size dE/dt is also the power P consumed in the process, we may write Eq. 4.2 in the form

$$R \propto P$$

which states that – in drilling, cutting, boring, or any similar process – the penetration rate is directly proportional to the power.

In modeling rock breakage processes, we must also deal with the behavior of the rock. At an empirical level, we say that the physical strength of the rock (i.e.. compressive, shear, tensile) resists failure induced by the process loads. Since rock strength decreases in the order stated, it makes eminent sense to design a penetration or fragmentation process to apply its loads in the reverse order (i.e., tensile, shear, compressive). Unfortunately, that is more difficult to do than say. and most processes succeed by applying some energy in tension or shear but relying on excess amounts of energy in compression. In an era of increasingly expensive energy, however, process and equipment designers arc paying more attention to Eqs. 4.1 and 4.3 in a search for energy conservation and power efficiency.

Principles of Rock Penetration

Rock penetration methods can be classified on several bases. These include *size of hole*, *method of* mounting, and type of power. The scheme that we will use is a generic one. based on the form of rock attack or mode of energy *application leading* to penetration. It is general, applicable to all kinds of mining, and inclusive of all forms of penetration. Thus machines used for cutting and boring as well as drilling are included. This classification bears some resemblance to one for rock fragmentation methods, such as blasting, ripping, and other wholesale breakage techniques, since the principles are identical, and rock breakage is the common objective (see Table 4.3). Notice that the use of *drilling* as a process name, however, is reserved for mechanical attack systems.

All the known methods or concepts of rock penetration arc listed in Table 4.1 in general order of importance, with those having widest application appearing first. Where a commercial or operational machine exists that employs a particular form of attack and method, it is so identified. Discussion of the leading categories is in order.

TABLE 4.1 Classification of Rock Penetration Methods, Based on Form of Attack[a]

Form of Energy Application	Method	Machine
Practical		
Mechanical (Drilling)	Percussion	
	Drop tool	Churn or cable-tool drill
	Hammer	Rock drill, channeler
	Rotary, drag-bit	
	Blade	Auger or rotary drill, boring
	Stone-set	Diamond drill
	Sawing	Wire-rope, chain, or rotary saw
	Rotary, roller-bit	Rolling-cutter drill, boring
	Rotary-percussion	
	Hammer	Rock drill (independent rotation)
	Rotary	Rolling-cutter drill (superimposed percussion)
Novel		
Thermal	Flame	Jet piercer, jet channeler
	Plasma	Plasma torch
	Hot fluid	Rocket
	Fusion	Subterrene
	Freezing	(Conceptual)
Fluid	Jet	Hyraulic jet, monitor, cannon
	Erosion	Pellet-impact or abrasion drill
	Bursting	Implosion drill
	Cavitation	Cavitating drill
Experimental		
Sonic	Vibration	High-frequency transducer
Chemical	Explosion	Shaped charge, capsule, projectile
	Reaction	Rock "softener," dissolution
Electrical	Electric arc or current	Electrofrac drill
	Electron beam	Electron gun
	Electromagnetic induction	Spark drill
Light	Laser	Electromagnetic radiation beam
Nuclear	Fission	(Conceptual)
	Fusion	(Conceptual)

[a] Energy applications listed in approximate order of present practicality.
Source: Hartman, in press.

Mechanical Attack. The application of mechanical energy to rock can be performed basically in only one of two ways: by percussive or rotary action. Combining the two results in hybrid methods, termed roller – bit rotary and rotary – percussion drilling. The mechanical category, of course, encompasses the vast majority (probably 98 %) of rock penetration applications today. In surface mining, roller – bit rotaries and large percussion drills are the machines in widest current use, with rotaries heavily favored; while underground, percussion is employed for hard rock and boring machines utilize drag – bit or, more often, roller – bit action.

Thermal Attack. Although penetration principles other than drilling are known, only two novel forms have been utilized commercially in mining, thermal and hydraulic penetration. The only thermal method having practical application today but limited to surface mining (for environmental reasons) is flame attack with jet piercer or channeler. It penetrates rock by spalling, an action applicable to hard rock of high free – silica content. Because of its ready capability of varying the shape of openings, oxygen or air jet burners are used not only to produce blastholes but to chamber them as well and to cut dimension stone. Jet piercing of blastholes, however, has decreased in popularity in recent years as mechanical drills have improved in versatility and penetrability.

Fluid Attack. While disintegration of rock by fluid injection is an attractive concept the end result is more likely fragmentation than penetration. To produce a directed hole with pressurized fluid from an external source, jet action or erosion appears to be the most feasible means of attack, but commercial application to date is limited. Hydraulic monitors have been used for over a century to mine placer deposits and to strip frozen overburden; and more recently, high-pressure hydraulic jets have been applied successfully to the mining of coal, gilsonite. and other consolidated materials of relatively low strength. Both pulsating and steady jets have been utilized. In some penetration-fragmentation devices, hydraulic- and mechanical-attack mechanisms assist and complement one another. For large holes, the hydraulic jet alone may be competitive with drilling.

Other Methods and Future Applications. While some attempts to employ other forms of energy (sonic, chemical, electrical, etc.) have been made, the remaining methods in Table 4.1 must be classified in the experimental or conceptual categories at present.

Maurer (1980) among others is optimistic about the future of novel and untried penetration devices, citing successful laboratory and field tests where they have outperformed conventional drilling methods. It is probable, however, that their value for rock penetration in the near future will be limited to (I) supplementing mechanical energy (drilling) systems for special circumstances and (2) creating very large or deep holes, such us tunnels or \,l wells. Their application for general drilling purposes in mining seems less attractive.

Drilling

With five exceptions, drilling is employed in mining for the placement of explosives; In exploration, it is the primary method of sampling; in develop* merit, it may be used to provide drainage, to stabilize banks by the placement ъt anchors, and to test foundations; and in exploitation, it is used for the placement of roof or rock bolts (in coal mines, more drilling is done for bolting than blasting). If used in conjunction with blasting, its major application it called *production drilling.* We took now at drill performance and the principles of drill selection.

Operating Components of System. There are four main functional components of a drilling system (and of most other penetration systems). They are related to the utilization of energy by the drilling system in attacking rock in the following ways:

1. The *drill,* the energy source, is the prime mover, converting energy from its original form (fluid, electrical, pneumatic, or combustion engine drive) into mechanical energy to actuate the system.
2. The *rod* (or drill steel, stem, or pipe) transmits energy from the prime mover or source to the bit or applicator.
3. The *bit* is the applicator of energy in the system, attacking rock mechanically to achieve penetration.
4. The *circulation fluid* cleans the hole, controls dust, cools the bit, and at times stabilizes the hole.

The first three are physical components of the drilling system, controlling the penetration process, while the fourth is supportive of penetration through removal of cuttings.

In commercial drilling machines, attention has focused to some extent on reduction of energy losses in transmission. This has led to the introduction of down-hole (in-the-hole) drills, both of the large percussion variety and the roller-bit rotary (electrodrill and turbodrill) type, although the latter has found application mainly in oil well drilling. They replace mechanical energy transmission with fluid or electrical transmission, which usually results in more energy reaching Ihe bit and faster drilling.

Mechanics of Penetration. As indicated previously, there are only two basic ways to attack rock mechanically—percussion and rotation—and the four classes of commercial drilling methods to be discussed utilize these principles or combinations of them. It is the bit-rock interaction that governs the efficiency of energy transfer and the nature of the breakage process.

Causing rock to break during drilling is a matter of applying sufficient force with a tool to exceed the strength of the rock. This resistance to penetration of rock is termed its *drilling strength,* an empirical property; it is not equivalent to any of the well – known strength parameters. Further, the stress field created by the tool must be so directed as to produce penetration in the form of a hole of the desired shape and size. These stresses are quasi static in nature, because forces are applied relatively slowly in the drilling process.

The different ways in rich percussion, rotary, and combination (rollerbit, rotary – percussion) drills attack rock are compared in Figure 4.1. The resulting cutting action of the bit, however, is remarkably similar: In each case, alternating phases of crushing and chipping occur. What differs is the relative importance of each phase in advancing the bit; for example, crushing predominates in percussion drilling, chipping in a rotary drag bit, and a hybrid action in combination drills.

Factors In Drill Performance. A number of factors influence rock penetration or cuttings removal during drilling, which in turn largely determine drill performance. There are four groups of these factors:

1. *Operating variables.* These affect the four components of the drilling system *(drill, rod. bit.* and fluid). They are largely controllable and include two categories of factors: (a) *drill power,* blow energy and frequency, rotary speed, thrust, and rod design: and (b) fluid properties and flow rate.

2. *Drillhole factors.* These include hole size, length, and inclination; they are dictated by *outside* requirements and thus independent variables. Hole diameters in surface mining are generally 6-18 in. (150-450 mm); underground. they range from 11-7 in. (40-175 mm).

3. *Rock factors.* They are environmentally derived and also independent; they consist *of the* properties *of the* rock, geologic conditions, and the state of stress acting on the drillhole. Often referred to as *drill ability factors,* they *determine the drilling* strength of the rock and limit drill performance.

4. *Service factors.* These consist of labor and supervision, power supply, job site, weather, and so *forth.* They. too. are independent, largely external variables, having some effect on *drill* performance.

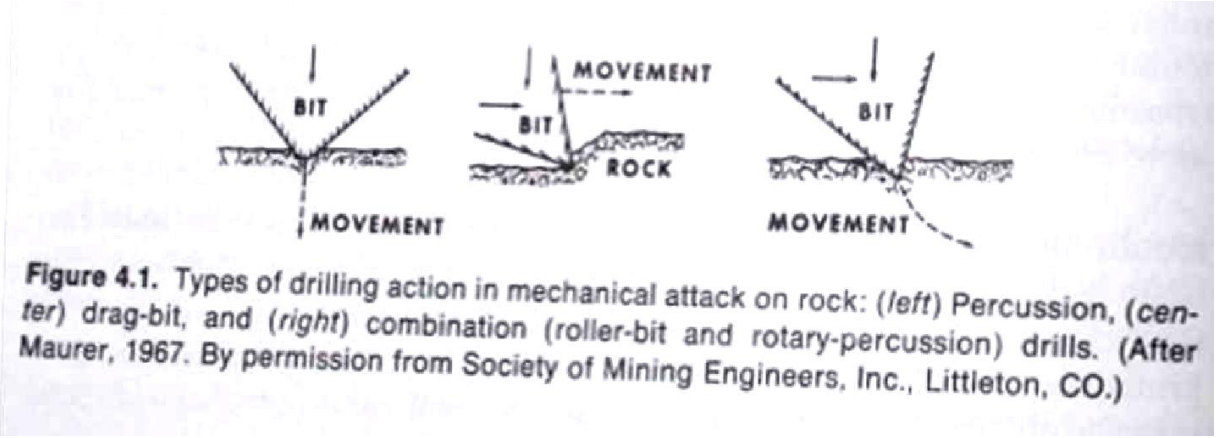


Figure 4.1. Types of drilling action in mechanical attack on rock: (*left*) Percussion, (*center*) drag-bit, and (*right*) combination (roller-bit and rotary-percussion) drills. (After Maurer, 1967. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Performance Parameters. In selecting the optimal drilling system or evaluating drill performance, four parameters are measured or estimated most frequently:

1. Process energy and power consumption
2. Penetration rate
3. Hit wear (life»
4. Cost (ownership + operating = overall)

The effects of key operating variables on penetration rate for the three common drilling systems are shown in Figures 4.2 to 4.4. Figure 4.5 relates a dri I lability factor, the coefficient of rock strength (drilling strength), to penetration rate in percussion drilling.

Energy and power affect operating cost, but they are more important as determinants of penetration rate (Eqs. 4.2 and 4.3). Both penetration rate and bit

wear arc major criteria of performance, with wear more critical in deep holes and hard rock. Cost is the ultimate measure of performance; a drill can have excellent performance, but if it is not cost effective, then an alternative system should be sought. (It is well to understand, however, that a goal of mining is the minimization of all rock breakage costs and that drilling docs not stand alone. Overall breakage costs include costs for blasting and also for crushing and grinding if mineral processing occurs.)

Drill Selection. The selection of a production drill follows a well-defined procedure; it is a true engineering design problem, requiring value judgments. The steps in selection are as follows (Capp, 1962);

1. Determine and specify the conditions under which the machine will be used, such as the job-related factors (labor, site, weather, etc.), with safety the ultimate consideration.

2. State the objectives for the rock breakage phases of the production cycle of operations in terms of tonnage, fragmentation, throw, vibrations, and so forth (in surface mining, consider loading and haulage restrictions, pit slope stability, crusher capacity, production quota, pit geometry, etc.).

3. Based on blasting requirements, design the drillhole pattern for surface mining or drill round if underground (hole size and depth, inclination, bur-den, spacing, etc.).

4. Determine the drillability factors, and, for the kind of rock anticipated, identify the drilling method candidates that appear feasible (manufacturers can perform rock drillability tests and recommend drills and bits).

5. Specify the operating variables for each system under consideration, including drill, rod, bit, and circulation fluid factors.

6. Estimate performance parameters, including machine availability and costs, and compare. Consider the power source and select specifications.

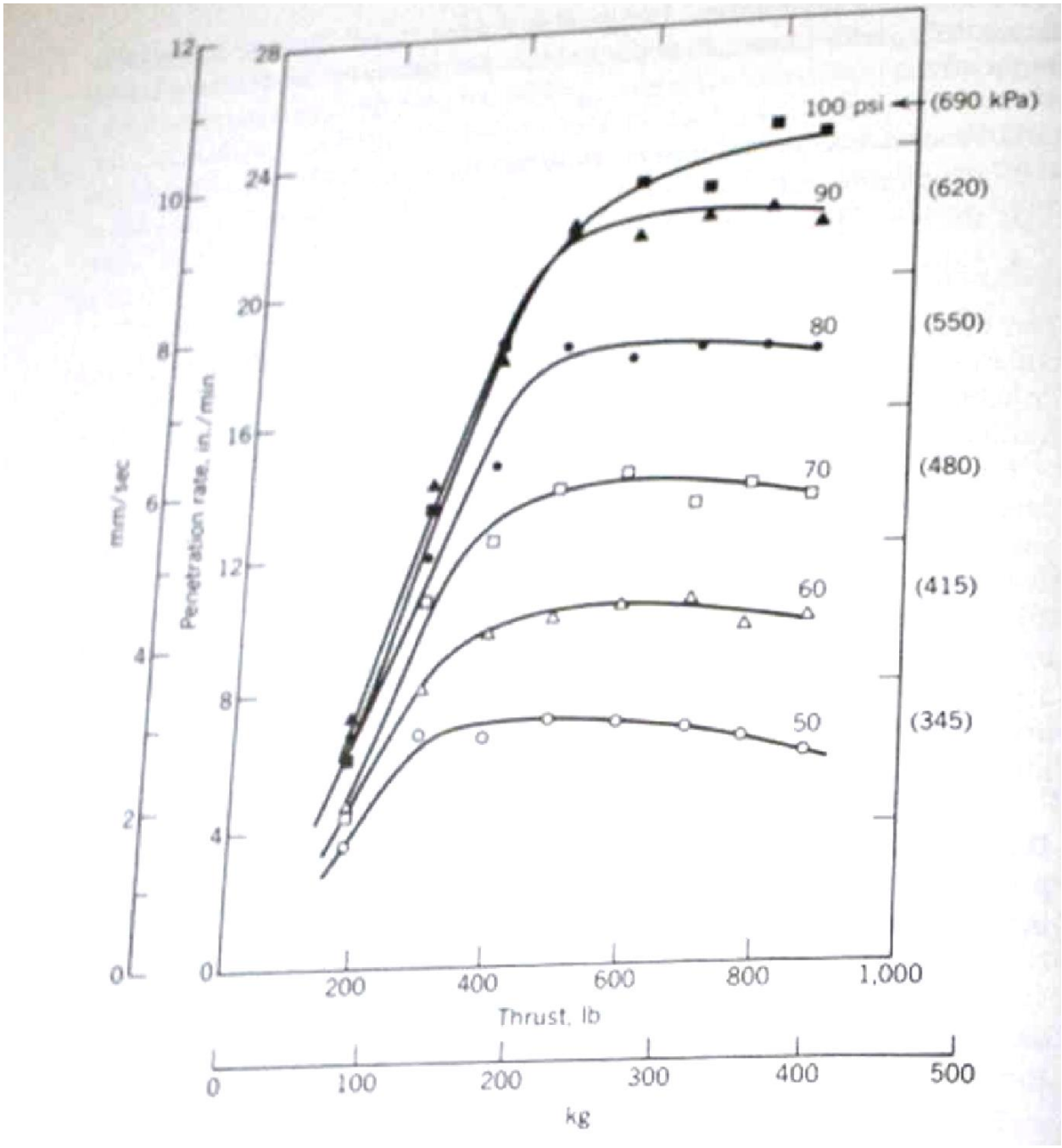


Figure 4.J. Percussion drilling Effect of thrust on penetration rate for a range of operatic air pressures Rock type, facomte (After Paone et al.. 1969.)

Major cost items arc hits, drill depreciation, labor, maintenance, power, and fluids. Hit life and costs are critical but difficult to project.

7. Select the drilling system that, in best satisfying all requirements, has the lowest overall cost, commensurate with safe operation.

Items 4 and 6 are the most difficult steps to accomplish in the entire design procedure, primarily because of the present unreliability of drillability determination and drill performance prediction.

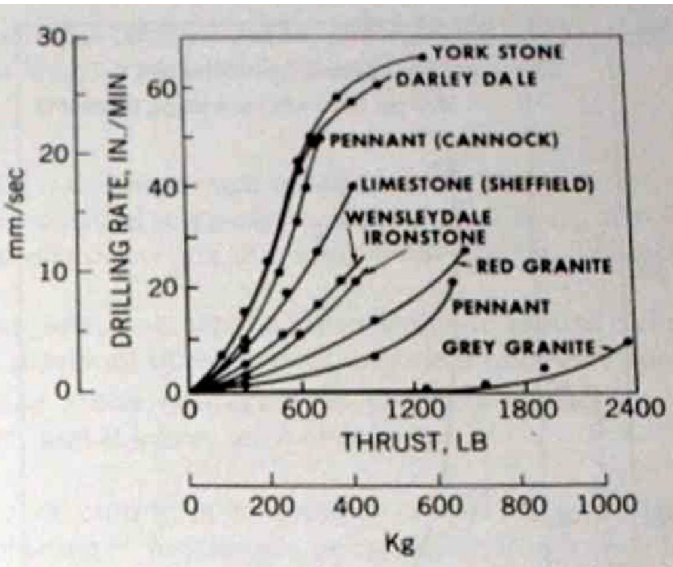


Figure 4.3. Drag-bit rotary drilling. Effect of thrust on penetration rate for various rock types. Bit diameter: $1\frac{11}{16}$ in. (43 mm). Rotary speed: 180 rpm. (After Alpan, 1961; Maurer, 1967. By permission from Institution of Mining Engineers, Doncaster, U.K.)

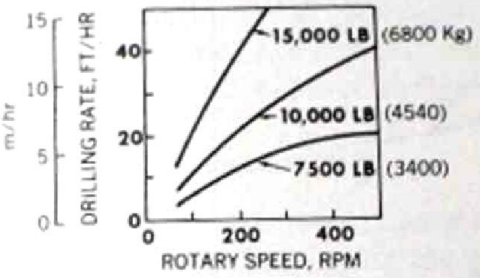


Figure 4.4. Roller-bit rotary drilling. Effect of rotary speed on penetration rate for a range of thrusts. Rock type: Beekmantown dolomite. (After Rowley et al., 1961. Copyright 1960, Society of Petroleum Engineers of AIME, Dallas.)

TABLE 4.2 Application of Drilling and Penetration Methods to Different Types of Rock

	Rock Type/Drillability			
Drilling Method	1 Soft (Shale, Weathered Limestone, Coal)	2 Medium-Hard (Limestone, Weathered Sandstone)	3 Hard (Granite, Chert)	4 Very Hard (Taconite, Quartzite)
Hydraulic jet	×	×		
Rotary, drag-bit	×	×		
Rotary, roller-bit	×	×	×	
Rotary percussion		×	×	
Percussion		×	×	×
Thermal jet piercing			×	×

Source: Hartman, in press.

4.3. BLASTING AND ROCK FRAGMENTATION.

Principles of Rock Fragmentation.

As we said earlier, rock fragmentation is the breakage function carried out on a large scale to fragment masses of rock. In both the mining and construction industries, blasting is the predominant fragmentation method employed, but other techniques are coming to the fore. Based on distinctions in the way they apply energy to break rock, a classification of rock fragmentation methods is given in Table 4.3.While all have received some application in mining (the last named, electrical, on an experimental basis only and limited, like impact, to secondary breakage of boulders), only chemical energy (blasting) has widespread use for all consolidated materials, both surface and underground. Excluded are excavating machines which, even though they may fragment soil and rock, primarily perform a loading function (Table 4.6); and continuous excavating machines, which mainly produce an opening and are classified as penetration methods (Table 4.1).

Further discussion here is limited to blasting. In blasting, we shall find that many of the principles o f rock penetration are valid, especially the one relating the amount of rock damage to the amount of energy applied: $W \propto E$ (Eq. 4.1). Both the properties o f explosives and the properties of rocks are involved (Hemphill. 1981). For our present purposes, however, we shall focus on the theory of explosives and explosions, reserving rock response and blasting applications for subsequent discussions under exploitation methods (Sections 6.6. 11.6).

TABLE 4.3 Classification of Rock Fragmentation Methods Based on Form of Energy Application*

Form of Energy Application	Method	Agent or Machine
Chemical	Blasting	High explosive, blasting agent, liquid oxygen (LOX), black powder
Mechanical	Pneumatic	Compressed air or carbon dioxide cylinder
	Ripping	Ripper blade, dozer blade
	Impact	Hydraulic impact hammer, drop ball
Fluid	Mining (soil)	Hydraulicking (monitor)
	Mining (rock)	Hydraulic jet
Electrical	Electric arc or current	Electrofrac machines

* Energy applications listed in approximate order of present practicality.

Theory of Explosives

Nature of Explosions. An explosive is an agent, compound, or mixture that undergoes very rapid decomposition when initiated by heat, impact, friction, or shock (Dick et al., 1983). The decomposition is a high-velocity, exothermic reaction, accompanied by the liberation of vast amounts of energy and very hot gases at tremendously high pressure. The process is termed detonation and the agent a high explosive if the speed of the reaction is supersonic. It is called deflagration and the agent a low explosive if the speed of the reaction is subsonic. Detonation may properly be termed an explosion. accompanied by the formation of a shock wave. Deflagration is very rapid hurtling but not an explosion.

It is both (1) the impa.cl of the shock wave and (2) the expanding effect of the high-pressure gas formed during detonation that cause the rock to fragment, in varying proportions, depending upon explosive and rock properties. Four of the key explosive properties are (1) energy density, (2) bulk density, (3) rate of energy release, and (4) pressure-time history of the gas release. Important rock properties are (1) density and porosity, (2) strength, (3) energy absorption properties and modulus of elasticity, and (4) rock structure, including jointing, bedding, fractures, alteration, and so forth (Clark, 1968; Hemphill, 1981).

Detonation Zone in Explosives. During an explosion, the chemical reaction of an explosive produces a detonation zone that propagates through the charge and into the surrounding rock (Dick et at., 1983). Figure 4.6 demonstrates the passage of the zone and attendant shock front at the explosive's midpoint (borehole charges typically are cylindrical in shape, unless the hole has been sprung or chambered). The primary reaction occurs in the zone bounded by the shock wave on the leading edge and the Chapman-Jouguet (C-J) plane al the roar boundary. (This zone is very narrow in high explosives, which have correspondingly smaller critical diameters than blasting agents, i.e.. the smallest-diameter cartridge which can be detonated.) The pressure profiles in Figure 4,6 represent the explosive forces applied to the rock being blasted. A comparison is given for two types of explosive. The shock (stress) wave moving out from the explosive creates the initial or detonation pressure Pj. It is this pressure which gives the explosive its shattering action in breaking rock. (A low explosive generates no shock wave and hence no P_d.) The detonation is followed by a sustained or explosion pressure P_r, also called the borehole pressure. This effect is due to the gas pressure action, equally or more important than the shattering action in breaking rock.

Chemical Reactions of Explosives. Most ingredients of explosives (discussed in the next section) are composed of the elements oxygen, nitrogen, hydrogen, and carbon, plus certain metallic elements (aluminum, magnesium, sodium, calcium, etc.). The composition of explosives is varied and balanced chemically to produce the desired effects in blasting.

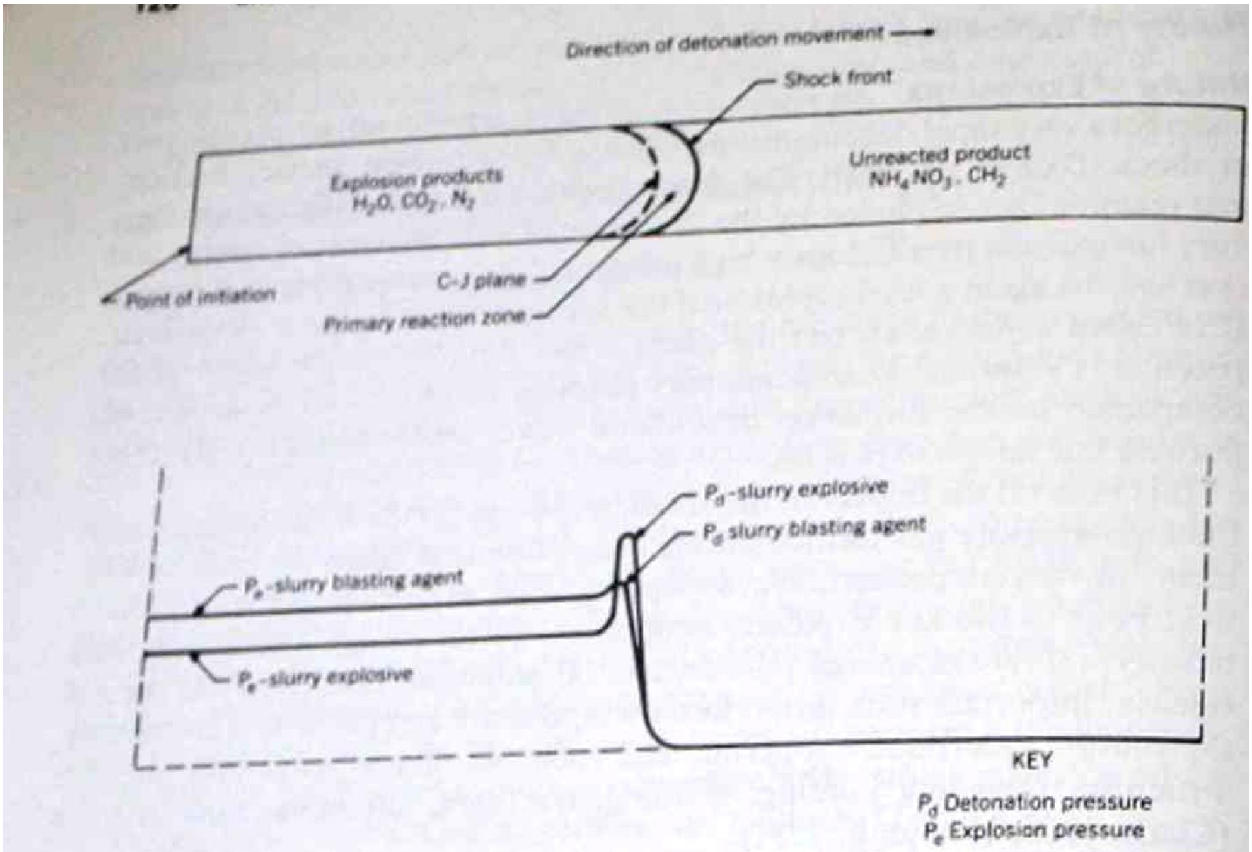


Figure 4.6. *(Top)* Propagation of detonation zone through an explosive charge. (*Bottom*) Resulting pressure profiles in rock produced by slurry blasting agent and slurry explosive. (After Dick et al., 1983.)

For example, because an explosion is essentially a combustion process, the chief criterion of efficient energy release is the oxygen balance; that is, an oxygen-balanced explosive is one that has an optimal energy release. *Zero oxygen balance* is the *point* at which an explosive mixture has sufficient oxygen to completely oxidize all the contained fuels, but no excess oxygen to react with the contained nitrogen (Clark, 1968).

At zero oxygen balance, the theoretical products of detonation, all gaseous and harmless, are water, carbon dioxide, and nitrogen. In reality, small amounts of other gases are generated, too: oxides of nitrogen, carbon monoxide, methane, and a few others. Any departure *from* zero oxygen balance *permits* greater amounts of these extraneous gases— all toxic— to form. Efficiencywise, they are mostly endothermic (“ heat robbers”), which means they consume energy rather than liberate it (Dick et al., 1983).

An equation may be written for a basic, oxygen-balanced reaction involving only oxygen, hydrogen, and carbon (Clark. 1968):

$$OB = O_o - 2CO_2 - H_20 = 0$$

If metals or other reactive elements are present, then in computing the oxygen balance, enough oxygen must also be allowed to form the appropriate oxides of these elements (e.g., A12O j, MgO, Na20, CaO).

The detonation of three different ammonium nitrate-fuel oil (ANFO) mixtures illustrates the principle of oxygen balance (Dick et al., 1983). In these idealized equations, fuel oil is represented approximately by the formula CH_2 , and the energy release is expressed in kcal/kg:

1. 94.5% AN–5.5% FO (oxygen-balanced)
$$3NH_4NO_3 + CH_2 \rightarrow 7H_2O + CO_2 + 3N_2 + 930 \text{ kcal/kg}$$
2. 92.0% AN–8.0% FO (fuel excess)
$$2NH_4NO_3 + CH_2 \rightarrow 5H_2O + CO + 2N_2 + 810 \text{ kcal/kg}$$
3. 96.6% AN–3.4% FO (fuel shortage)
$$5NH_4NO_3 + CH_2 \rightarrow 11H_2O + CO_2 + 4N_2 + 2NO + 600 \text{ kcal/kg}$$

In each ease, the energy release is obtained by calculating the difference of the heats of formation of the ingredients and the products. The consequences of departing from zero oxygen balance are obvious; different products or product amounts are formed, but the major penalty is in reduced energy release. The field mixture normally employed consists of 94% AN and 6% FO , which ensures a slight oxygen deficiency-fuel excess as a safety measure (N O is more toxic than CO) and sacrifices less energy.

The reactions of a few other common explosives are also important. Although simple A N FO mixtures provide the maximum energy release per unit cost of explosive, products with higher densities and energies and improved water resistance are sometimes required. Examples are blasting agents, dry or slurry, with fuel-sensitizer additives, such as trinitrotoluene (TNT) or aluminum:

1. 78.7% AN–21.3% TNT
$$21NH_4NO_3 + 2C_6H_2CH_3(NO_2)_3 \rightarrow 47H_2O + 14CO_2 + 24N_2 + 1010 \text{ kcal/kg}$$
2. 81.6% AN–18.4% Al
$$3NH_4NO_3 + 2Al \rightarrow 6H_2O + Al_2O_3 + 3N_2 + 1620 \text{ kcal/kg}$$

Both of these mixtures liberate more energy per unit weight than ANFO and possess higher densities.

Physical-chemical data for common explosive ingredients and explosives, useful for calculations involving chemical reactions, oxygen balance, and energy release, are given in Table 4.4.

Properties of Explosives

Classification and Types. In the broadest sense, explosives can be classified as follows (Anon., 1977):

CLASS	ACTION	EXAMPLE
Low-explosive	Deflagrate (burn)	Black powder
High-explosive	Detonate (explode)	Nitroglycerin (NG),dynamite
Blasting agent	Detonate (explode)	Ammonium nilrate-fuel oil (ANFO), slurry (or emulsion or gel)

The term *high explosive* requires practical definition, because under varying circumstances, ANFO may or may not act as a high explosive. If Ihe product is cap-sensitive (i.e., can be detonated by a no. 8 blasting cap), then it is technically classed a *high explosive* under U.S. Department of Transportation regulations. Therefore, depending on ingredients and particle size, ANFO (dry or slurry) may be either a blasting agent or high explosive (Dick et al.. 1983).

In recent years, the consumption of explosives in mining (3.7 billion lb, or 1.7 billion kg. annually, which accounts for 86% of U.S. commercial use) has shifted dramatically toward ammonium nitrate blasting agents, both dry and slurry, primarily because of their economy, safety, and versatility. Nitroglycerin-based (NG) explosives are diminishing in importance, and black powder has nearly vanished from the scene (it is essentially outlawed in underground coal mining). The distribution of explosives consumption currently is ANFO 85%. slurries 10%, NG dynamites 3%, and permissibles 1% (Martens, 1982). *(Permissibles* are specially formulated mixtures that are safe to use in flammable atmospheres in underground coal mines; they may ho NG-based but today are principally AN.)

Ingredients. The principal reacting ingredients in an explosive are *fuels* and *oxidizers.* Common fuels are fuel oil, carbon, aluminum, and TNT. Common oxidizers are AN, sodium nitrate, and calcium carbonate. Other ingredients may include *sensitizers* (NG, TNT, nitrostarch, aluminum, etc.), *energizers* (metallic powders), and *miscellaneous agents* (water, thickeners, gelatinizers, emulsifiers, stabilizers, flame retardants, etc.).

A summary of the principal ingredients used in high explosives, together with some of their associated properties, appears in Figure 4.7. Detailed flow sheets of ingredients and products which represent the processes involved in the manufacture or field mixing of blasting agents, both dry and slurry, are shown in Figure 4,8. Depending on their formulation, these agents may or may not be classed as high explosives.

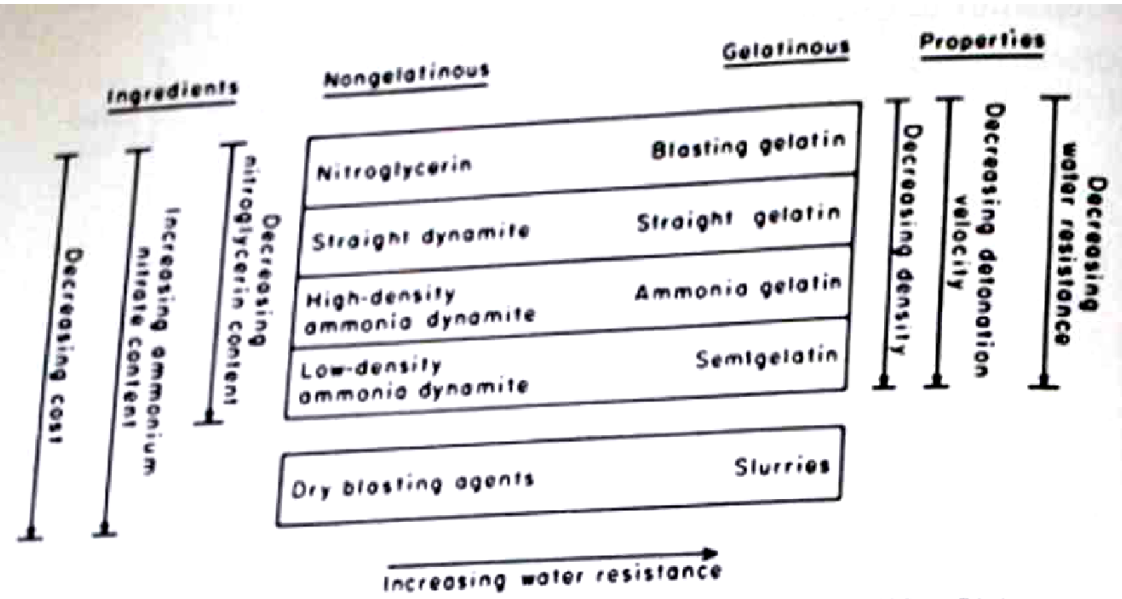


Figure 4.7. Relation of ingredients and properties of high explosives. (After Dick et al., 1983)

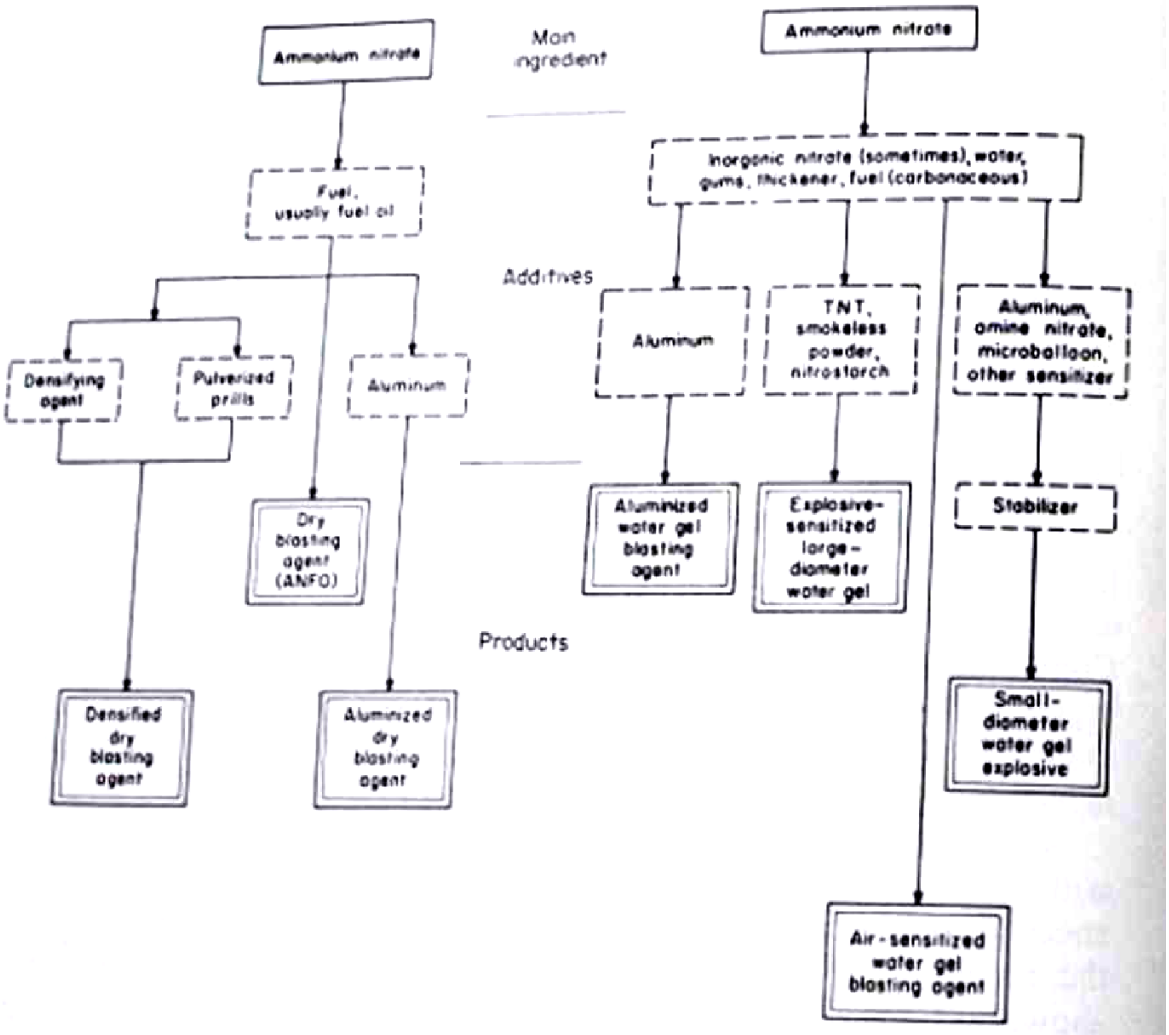


Figure 4.8. Formulations of blasting agents, showing ingredients and products. (*Left*) Dry blasting agents. (*Right*) Water-based slurries, emulsions, or gels. (After Dick et al., 1983.)

Blasting Properties of Explosives. Ammonium nitrate-based explosives, as mentioned, have completely revolutionized the field of rock blasting. An oxygen-balanced ANFO is the cheapest source of explosive energy available today. Its chief disadvantages, however, are its poor water resistance and modest energy release. Slurries, which are mixtures of AN, water, and a fuel sensitizer, cither explosive or nonexplosive, were developed to extend the range of properties of blasting agents (Fig. 4.8). Water gels and emulsions are similar to slurries.

Explosives and blasting agents are characterized by various properties that determine how they will function under field conditions in blasting. While not necessarily the fundamental properties that govern the behavior of explosives, they are practical measures of the ability of explosives to perform useful work. The most important follow (Anon,,1977; Dick et al.,1983), with a summary of some numerical values in Table 4.5.

1. *Strength.* Formerly based on weight or cartridge strength that reflected the NCJ content, the strength of explosives is now commonly expressed as the measured or calculated energy release or as an energy value relative to that of ANFO at 100%. High strength is needed to shatter hard rock, but excessive energy is wasted on soft, plastic, or fractured rock.
2. *Detonation velocity.* This is the velocity at which the detonation front moves through the explosive, and it varies from 5500 to 25,000 ft/sec (1580 to 7620 m/sec). High velocity is associated with shattering action, important in hard rock.
3. *Density.* Density is usually expressed as a specific gravity, relative to that of water at I. A practical measure is loading density, the weight of explosive per unit length of charge. Specific gravity of explosives varies from 0.5 to 1,7. A dense explosive releases more energy per unit volume or length of drillhole.

TABLE 4.5 Important Properties and Relative Costs of Explosives

Explosive	Specific Gravity or Density (g/cm^3)	Relative Weight Strength (ANFO = 100)	Relative Bulk Strength (ANFO = 100)	Relative Cost/Unit Volume (ANFO = 100)
ANFO	0.85	100	100	100
ANFO (dense)	1.10	100	130	130
15% Al/ANFO	0.85	135	135	183
15% Al/ANFO (dense)	1.10	135	175	237
Pelletized TNT	1.00	90	106	392
1% Al/NCN slurry	1.35	86	136	397
20% TNT slurry	1.48	87	151	421
40% Dynamite	1.44	82	139	551
25% TNT/15% Al slurry	1.60	140	264	722
95% Dynamite	1.40	138	193	824

Source: Modified after Anon., 1976a.

4. *Water resistance.* A practical consideration in wet ground, water resistance is the ability of an explosive to withstand exposure to water without losing sensitivity or efficiency. The scale is a relative one, ranging from very poor for ANFO to good for slurries and very good for NG gelatin dynamites.
5. *Fume class.* This is the measure of the amount of toxic gases (carbon monoxide, oxides of nitrogen, etc.) produced by detonation of an explosive. Because most commercial explosives today are oxygen – balanced, fume production is less a problem than formerly. Fume classes vary from low (0.36 ft^3/LB, or 0.023 m^3/kg, fume volume weight of explosive) to high (0,76 – 1,51 ft^3/LB or 0.047 – 0,094 m^3/kg), ANFO has excellent fume characteristics, tending not to cause the "powder headaches" associated with NG explosives.
6. *Detonation pressure.* Already defined, P_d is a function of the detonation velocity and the square of the density. It varies from 5 to 150 kb (500 – 15 000 kPa). An explosive with a high P_d is effective in hard, massive rock.
7. *Borehole (explosion) pressure.* I Also a calculated quantity, P_d varies from 10 to 60 kb (1000 - 6000 kPa). It is generally considered the most important explosive parameter in breaking and displacing rock in blasting. Some ANFO mixtures have higher P_c and P_d values, although most high explosives display the reverse behavior.
8. *Sensitiveness*. Defined as an explosive s susceptibility to initiation, sensitiveness reflects both safety in use and difficulty of detonation. Dynamites are *highly* sensitive by comparison with ANFO, although sensitiveness can be regulated by additives.

Initiation Systems. Because of the highly concentrated energy source that is required to detonate commercial high explosives and blasting agents, special initiators that arc sensitive, yet relatively safe and reliable, arc employed. An initiation system consists of three components (Dick et al., 1983):

1. Initial energy source
2. Distribution network to convey energy to individual blastholes
3. In-hole detonator that initiates explosive (the detonator and explosive are termed a *primer)*

An example of a simple (but nearly obsolete) initiation system in an underground metal mine is (1) a match igniting (2) a burning fuse connected to (3) blasting caps inserted in cartridges of explosives. A more elaborate system for surface or underground consists of (1) a detonating cord (explosive pentaerythritoltctranitrate or PETN) initiated by (2) a blasting cap. The most widely used systems are electrical, initiated by a power line or condenser discharge blasting machine. and with an electrical distribution network in place of fuse. Loading and charging are related subjects; the trend it to mechanized bulk systems. Particulars of loading and initiation arc provided b> Dick (1973) and Dick et al (I9H3).

Selection of Explosives

Both the properties of explosives and field conditions enter into the proper selection of explosives. Objectives arc to select an explosive and blasting system that will yield the lowest cost per unit of rock broken, while assuring that fragmentation and displacement of the rock are optimal for the conditions. As stated for drilling, comparison costs should be overall, consisting of drilling, blasting, and comminution. Selection criteria include the following (Anon., 1477; Dick el al., 1983):

1. *Explosive con.* Relative cosis of common mining explosives on a unit weight basis are (Gregory. 1979)

Bulk AN prills	100%
Bulk A NR)	150
Bagged ANFO	230
Cartridged ANFO	300
Bulk or bagged AN slurries	550
Cartridged dynamites	800
Cartridged gelatin dynamites	850

(For comparison, costs in Table 4.5 are given on a unit volume basis.) In estimating blasting costs, the expense of all other components in the system — transportation, storage, charging, initiation, and so forth — must be considered as well. Absolute costs for bulk ANFO currently average 4-5 e/lb (9- 11e/kg).

2. ***Charge diameter.*** Charge diameter is limited by hole diameter, and these factors arc among many in blast design that are interrelated. Critical charge diameter for bulk ANFO is less than 2 in. (50 mm), but efficiency of energy release suffers unless the holes arc pneumatically loaded.
3. ***Rock blastability.*** Special requirements for blasting because of rock conditions must be considered and evaluated (e.g.. very tough, dense, brittle, plastic, soft, or variable rock may necessitate a unique blasting system).
4. ***Water conditions***. Wet ground requires the use of a water-resistant explosive or water-repellent container.
5. ***Fume release***. If fumes are substantial, adequate ventilation must be provided in mines. Explosives that are particularly offensive in fume release are banned for use underground.
6. ***Other conditions.*** These include ambient temperature (low or high), propagating ground (short circuits cause detonation between holes), storage requirements (less stringent for blasting agents), sensitiveness (affects safety and dependability), and explosive atmosphere (permissible explosives required in coal mines and others classified as gassy).

4.4 LOADING AND EXCAVATION

Materials Handling

All unit operations involved in excavating or moving bulk minerals during mining an: termed materials handling. In cyclic operations, the two principal operations are loading and haulage, with hoisting an optional third when essentially vertical transport is accomplished. In continuous operations, where machines combine the breakage and handling functions, cutting, drilling and blasting are eliminated, and extraction and loading are performed in a single function (excavation). In combined loading-haulage machines, materials handling is performed in a single operation.

Materials handling in modern mechanized mining is equipment-centered. Unit operations are characterized by and sometimes identified with the equipment which performs them. Thus field terminology refers to a mining shove/, stripping dragline, or coal loader.

The scale of materials - handling equipment in surface mining grows ever larger. Upper size limits have risen to 300 tons (270 tonnes) for trucks, 220 yd^3 (170 m^3) for draglines, 180 yd^3 (140 m') for shovels, and 11,000 yd^3 /hr (8400 m^3/hr) for bucket wheel excavators. The latter three, boom-type machines are employed for stripping overburden in coal mining and arc believed to be the largest man-made, dry-land, mobile structures in existence. Reasons for the growing, gigantic scale of surface mining equipment are found in its high productivity and low unit operating cost, in part, this is due to its worldwide commercial availability from multiple, competing sources, standardized in product lines and graduated sizes.

As we have done throughout this chapter, our attention here is on principles, reserving applications for later discussion.

Principles of Loading - Excavatlon[2]

The extraction and elevating of minerals, either broken or in place, is termed loading or excavating. If the material is soil or very weak rock, it can probably be "dug" in place. If it is rock or unusually well-consolidated soil, it likely will have to be blasted or mechanically broken prior to excavation. In noncoal mining, 20% of the material has to be blasted (Martens, 1982).

A classification of mine loading-excavating equipment is presented in Table 4.6. Bases for the classification are the locale of mining (surface or underground) and continuity of operation (cyclic or continuous). Categories and common examples of individual machines are given for each. For the sake of completeness, continuous mining and boring machines appear, even though they were mentioned with penetration devices (Table 4.1).

The variety of equipment may overwhelm the student. Certain machines dominate the scene, however, and they are easily learned. For surface mining, shovels, loaders, draglines, and scrapers are in most common use. For underground mining, loaders, load-haul-dump units (LHDs), and continuous miners are prevalent. In underground mines with large openings and sufficient headroom,

surface equipment—dozers, shovels, trucks, loaders, and drills, modified to meet safety and operating conditions—has increasing application.

Each category and individual machine has operating characteristics that distinguish it and help to earmark it for selection. Several have a unique feature: They perform joint functions of loading and haulage (in addition to boom-type machines that cast the material being excavated). Examples of loading equipment that perform substantial haulage are dozers, rubber-tired scrapers, rope-drawn scrapers, and LHDs.

TABLE 4.6 Classification of Loading–Excavating Methods and Equipment

Operation	Category or Method	Machine (Application)
Surface		
Cyclic	Shovel	Power shovel, front-end loader, hydraulic excavator, backhoe (mining ore, stripping overburden)
	Dragline	Crawler, walking (stripping overburden)
	Dozer	Rubber tired, crawler (blade)
	Scraper	Rubber tired, crawler
	Blasting	Explosives stripping (overburden)
Continuous	Mechanical excavator	Bucket wheel (BWE) (overburden), cutting-head (soil, coal)
	Highwall mining	Auger, highwall miner (coal)
	Hydraulicking	Monitor or giant (placer)
	Dredging	Bucket ladder, hydraulic (placer)
Underground		
Cyclic	Loader	Overhead, gathering arm, shovel, front-end
	Shaft mucker	Clamshell, orange peel, cactus grab
	Self-loading transport	Load–haul–dump (LHD)
	Slusher	Rope-drawn scraper (metal ore)
Continuous	Continuous miner	Milling type, drum, ripper, borer, auger, plow, shearer (coal or nonmetallic)
	Boring machine	Tunnel-boring machine (TBM), roadheader, raise borer, shaft borer (soft rock)

Most loaders-excavators are required to operate in three working zones, with a variety of operating constraints(martin et al, 1982).These zones-digging, maneuvering-transport, and dumping-and some oif their constraints are illustrated for surface mining in Figure 4.9. The situation pictured occurs in stripping overburden or loading ore with a boom-type excavator (power shovel, dragline, or bucket wheel). Other conditions prevail where scrapers, dozers, front-end loaders, or dregred find unique application.

The major features o large shovel, dragline, and wheel excavators are summarized as advantages and disadvantages in Table 4.7. Competitive and commonly used to strip overburden in surface coal mining, these three machines warrant carefull comparison, although the trend in recent years has been strongly to draglines because of their greater range, both digging and casting(Anon.,1976a).

Selection of Equipment

Four groups of factors largely determine the selection of excavating equipment (the discussion applies specifically to surfacc equipment, but the factors are applicable to underground as well) (Pfleider. 1973a; Martin et al.. *1982)*

1. *Performance factors.* These relate directly to machine productivity and include cycle speed, available force (power), digging range, buckel capacity. travel speed, and reliability.

DIGGING
__FACE
WORKING
FLOOR
DISPOSAL
IP lL C •
H O P P E R TRUCK)
M C E \
MEIGMl
SLOPE
OEGREE
CONSOL IDAt
A BR A SlV EN ESt
V O L U M E ^lonnee*^
R E Q U IR E D
D IS C H A R G E
H E IG H T
v x M A XI MU M \
/ S IZ E
y
S L O P E .F L A T N E S S . W A T ER
S E V E R IT Y O P
D IS C H A R G E
TRANSPORT O IST A N C E .
M A N EU V ER IN G A R EA
R EQ U IR ED

Figure 4.9. Working zones and constraints in loading or excavating in a surface mine.

Conditions: shovel, dragline, or wheel excavator. Zones: digging, maneuvering-transport, and dumping. (After Martin et al., 1982. By permission from Martin Consultants Inc., Golden. CO.)

T A B L E 4 .7 Comparison of Feature of Shovel. Dragline,and Bucket Wheel Excavators

Machine	Advantages	Disadvantages
Shovel	1.Lower capital coat per yd1 (m) of bucket capacity, although when boom length or machine weight is considered the capital coats are roughly equivalent 2.Digs poor blasts and tougher materials better 3.Can handle partings well	1 More coal damage can result in tower coal recovery 2 Susceptible to spoil slides and pit flooding 3 Cannot easily handle spoil having poor stability 4 Cannot dig deep box cuts easily 5 Reduced cover depth capability compared with a dragline of comparable cost 6 Difficult to move
Dragline	1. Flexible operation, easy to move 2. Large digging depth capability 3. Can handle and stack overburden having poor stability 4. Completely safe from spoil pile slides or pit flooding during normal operation 5.High percentage of coal recovery; less coal damage 6Will dig a deeper box cut 7.Low maintenance cost 8.Can handle partings well 9.Is not affected by an uneven or rolling coal seam top surface 10.Can move in any direction	1.Requires bench preparation 2.Does not dig poor blasts well 3 Higher capital cost per yd* (m*) of bucket capacity, although when boom length or machine weight is considered, capital costs are roughly equivalent
Bucket whell	Continuous operation; no swinging necessary Long discharge range Can be operated on a	1. Will not dig hard materials 2. Some surface preparation

	highwall bench or on the coal seam 4. Can easily handle spoil with poor stacking characteristic and poor stability 5. Can extend range of shoveor dragline when operated in tandem 6. Can facilitate land reclamation as it dumps surface material back on top of the spoil pile	required 3. Lower availability 4. Large maintenance crew required 5. High capital cost compared with output 6. Can be susceptible to spoil slides and flooding 7. Can cause coal damage with resulting lower coal recovery 8. Poor mobility

2. *Design factors.* The design factors provide insight into the quality and effectiveness of detail design, including the sophistication of human-machine interface for operators and maintenance personnel, the level o technology employed, and the kinds of control and power available.
3. *Support factors.* Sometimes overlooked in machine evaluation, sup-port factors are reflected in servicing and maintenance. Ease of servicing, special skills involved, parts availability, and manufacturer's support are important considerations.
4. Cost factors. Probably *the most* quantitative (and ultimate) factor, costs are determined *by standard estimating* procedures *for* large mining and construction equipment. If reasonable assumptions as to life, interest rates, inflation, *fuel, and maintenance are* made, then results should be meaningful. *The customary* basis is to use unit costs, estimating *overall costs* as the sum *of ownership and operating costs,* all computed on a $/hr basis and converted to $/ton ($/tonne) or S/yd^3 ($/m^3).

For *estimation purposes* involving excavation, data such as those in Table 4.8 may be used *if actual* specifications are lacking. Numerical selection and cost estimation procedures will be demonstrated later (Sections 7.6, 8.7). *Operating characteristics o f large* surface mining excavators (power *shovel, dragline,* bucket wheel, front-end loader, and hydraulic excavator) are *available in* several references (Pfleider. I973a: Anon., 1976a: Martin et *al.. 1982).* Ratings for underground equipment are more difficult to obtain and *the choices are m ore* restricted *(Hamrin* . 1982). For the latest information on surface or underground equipment, consult the manufacturers or *appropriate trade* associations.

TABLE 4.8 Estimating *Parameters tor* Surface Excavators*

	Capacity $yd^3(m^3)$	Est. Weight, lb/yd^3 (kg/m^3)	Est. Power hp/yd^3 (kW/m^3)	Est. Life. hr	Est. Price. S/ yd^3 (S/m^3)
Rubber-tired scraper	*25-52* (19-40)	2.900 (1.700)	14 (14)	12.000	9.500 (12.400)
Front-end loader	3.5-30 (2.7-23)	12.000 (7.100)	52 (51)	12.000	36.000 (47.100)
Hydraulic excavator	4-30 (3-23)	30.000 (17.800)	72 (70)	30.000	85.000 (111.000)
Electric power *shovel*	6-75 (4.6-57)	54,000 (32.000)	41 (40)	75.000	110.000 (144.000)
Walking dragline	9-180 (69-138)	114.000 (68.000)	102 (99)	100.000	210,000 (275.000)
Bucket wheel excavator	0.1-5.2 (0.1-4)	—	—	30.000	

4.5 HAULAGE AND HOISTING

Principles of Haulage and Hoisting

Bulk materials in mining are transported by *haulage* (primarily horizontal movement) and *hoisting* (primarily vertical). Machines to accomplish these functions are identified in Table 4.9, classified on the same basis as excavating equipment. Information is also provided as to (I) normal range of haul distance and (2) gradeability, average and maximum, important performance parameters that distinguish machine application.

The primary function of all equipment in Table 4.9 is haulage, although some (front-end loader, dozer, rubber-tired scraper, rope-drawn slusher,

TABLE 4.9 Classification of Haulage and Hoisting Methods and Equipment

Operation	Method	Haul Distance	Gradeability (degrees)	
			Avg.	Max.
Surface Cyclic	Rail (train)	Unlimited	2	3
	Truck, trailer	0.2-5 mi (0.3-8 km)	8	12
	Scraper (rubber-tired)	500-5000 ft (150-1500 m)	12	15
	Front-end loader	<1000 ft (300 m)	8	12
	Dozer	<500 ft	15	20

		(150 m)		
	Skip	<8000 ft vert. (2400 m)	Unlimited	
	Aerial tramway	0.5-5 mi (0.8-8 km)	5	20
Continuous	Belt conveyor	0.2-10 mi (0.3-16 km)	17	20
	High-angle conveyor	<1 mi (1.6 km)	40	60
	Hydraulic conveyor (pipeline)	Unlimited	Unlimited	
Underground				
Cyclic	Rail (train)	Unlimited	2	3
	Truck, shuttle car	500-5000 ft (150-1500 m)	8	12
	Slusher (scraper)	100-300 ft (30-90 m)	25	30
	LHD	300-2000 ft (90-600 m)	8	12
	Skip, cage	<8000 ft vert.	Unlimited	
Continuous	Conveyor (belt, chain and flight, monorail)	0.2-5 mi (0.3-8 km)	17	20
	Hydraulic conveyor	Unlimited	Unlimited	
	Pneumatic conveyor	Unlimited	Unlimited	

LHD are self – loading. The only true hoisting machines are the skip and cage, but as indicated under gradeability all haulage devices can elevate the material1 they transport to миле extent (hydraulic and pneumatic conveyors are completely versatile and virtually unlimited as to distance and grade). Elevating *material* is generally crucial in any mining operation, because most mines, surface or underground, expand vertically to depth.

Again the variety of equipment seems overwhelming, but again certain machines predominate. In surface mines with multiple benches, as in open nits trucks team with shovels *in* most applications, with lesser use of belt conveyors and rubber-fired scrapers. In underground mines, rail, trucks (shuttle cars). LHDs. and belt conveyors are all popular in various segments of the overall haulage system.

Just as excavators perform in prescribed working zones, so do haulage units. For the most widely used haulage machine, the truck, as well as other machines, there are four zones: loading, traveling loaded, dumping, and traveling empty (Fig 4.10). Important design and operating decisions are required in specifying conditions for all these zones, but loading and dumping arc especially critical (Martin et al.. 1982). Alternative arrangements are sketched for each; the use of double-truck back-up or spotting is preferred for quick loading if a drive-through is not possible, and a drive-by grizzly (coarse screen made o f rails) affords the

fastest dumping. Different working zones characterize other haulage machines, and they all require careful analysis in selecting equipment and methods.

Comparable to Table 4.7 for excavators. Table 4.10 summarizes some of the main features of haulage equipment. Most of these machines are used both on the surface and underground.

Machine	Advantages	Disadvantages
Dozer	1. Flexible 2. Good gradeability 3. Negotiates rough terrain	3. Limited to short haul 4. Discontinuous 5. Low output, slow
Truck, trailer	1. Flexible and maneuverable 2. Handles coarse, blocky rock 3. Moderate gradeability	1. Requires good haul roads 2. Slowed by bad weather 3. High operating cost
Scraper (rubber – tired)	1. Flexible and maneuverable 2. Good gradeability	4. May require push loading 5. Limited to soil, small fragments 6. High operating cost
Rail	1. High output, low cost 2. Unlimited haul distance 3. Handles coarse, blocky rock	1. Track maintenance costly 2. Poor gradeability 3. High investment cost
Belt conveyor	1. High output, continuous 2. Very good gradeability 3. Low operating cost	1. Inflexible 2. Limited to small or crushed rock 3. High investment cost

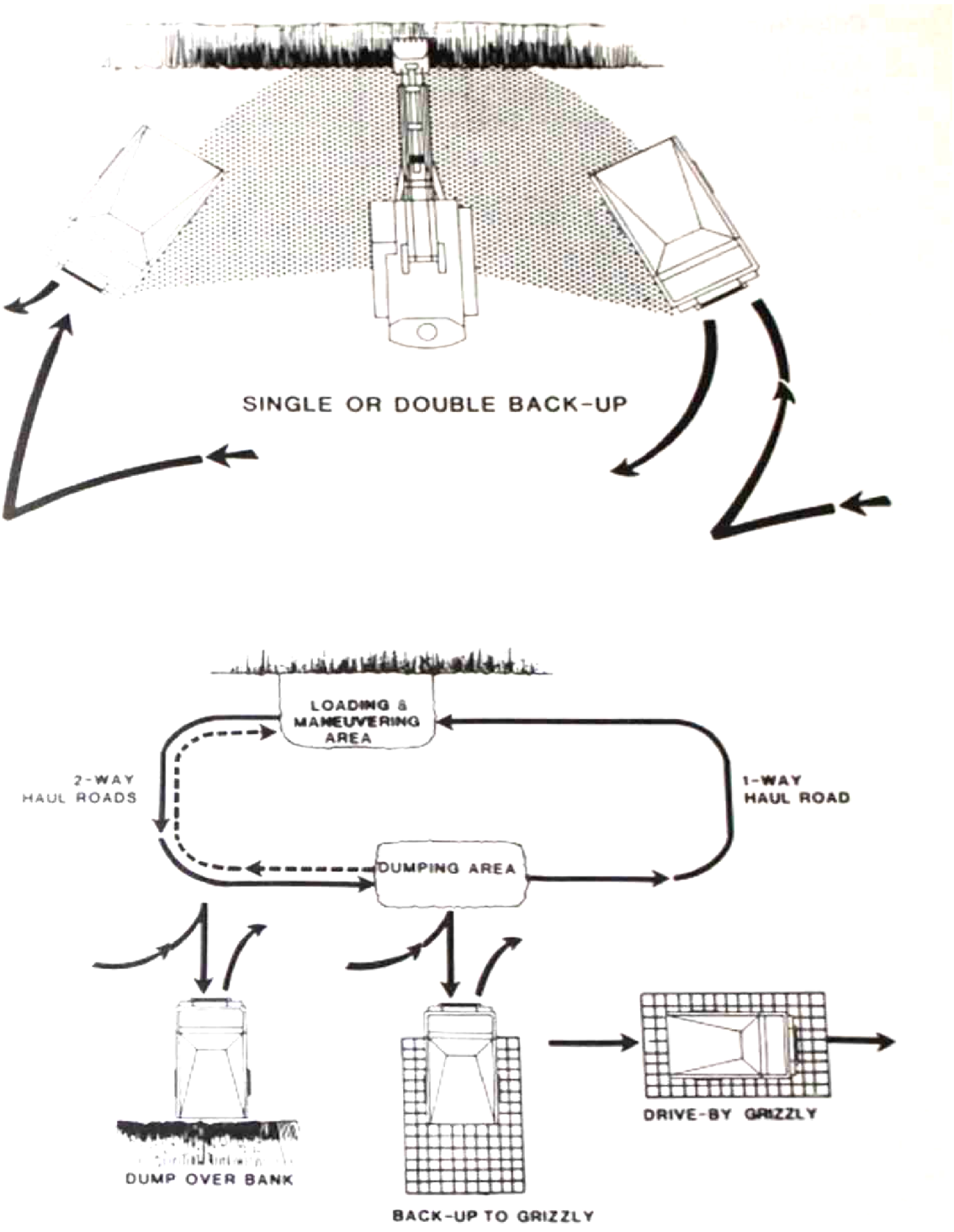


Figure 4.10. Working zones in haulage in a surface mine. Conditions: truck haulage, shovel loading, grizzly or bank dumping. Zones: loading, traveling loaded, dumping, and traveling empty. (After Martin et al.. 1982. By permission from Martin Consultants. Inc., Golden. CO.)

4.6 AUXILIAWV OPERATIONS

Auxiliary operations consist of all activities supportive of but not contributing directly to the winning of ore, coal, or stone. Most are scheduled prior to or following the production cycle so as to support but not interfere with production operations. A few may be conducted as an integral part of the cycle if they are essential to health and safety or the efficiency of operations.

Because auxiliary operations generate no income, there is a tendency in mining organizations to assign them a staff function and a low priority. It is the responsibility of the mainline production managers to ensure that all these tasks receive proper attention, are assigned appropriate priority, and are performed well. Recalling our discussion of mine administration (Section 3.8), approximately three support personnel (overall) are required for every two in production In terms of the total mining enterprise, probably 60% of the lime and effort, if not the expense, goes to auxiliary functions, too.

Classification of Auxiliary Operations

Auxiliary functions and operations carried on in mining are listed in Table 4.11. They are tabulated as (I) exploitation or development associated with production and (2) surface or underground operations. Those which may be incorporated into the production cycle arc indicated. The list is remarkable both for its length and diversity. Not every activity is performed in every mine, nor does every activity require full attention every shift. Every operation which is required in a mine, however, should be scheduled regularly.

Granted their importance, auxiliary operations warrant attention in our discussion. The chief of these activities—surface slope stability and underground ventilation and ground control — will be treated individually and in detail in later chapters. For accounts of these and other auxiliary operations, also see the pertinent sections of the *SME Mining Engineering Handbook* - for example, underground (Robinson. 1973), reclamation (Brooks and Wilhams. 1973), power (Ehrhom and Young. 1973), and maintenance (Ehrhom, 1973). Other references are underground (Lewis and Clark, 1964), surface (Hy slop, 1968; Westwater and Magnuson, 1968; Anon., 1976a), service vehicles (Sundeen and Wenbcrg, 1982), and underground coal (Stefanko and Hise. 1983).

4.6 CYCLES AND SYSTEMS

Having studied the principles of the unit operations comprising the production cycle and something of the procedure for selecting equipment to perform the various functions, we conclude with a discussion of mining systems and cycle balancing.

A flow sheet for a typical cycle of operations for surface mining, complete with both production and auxiliary operations, is depicted in Figure 4.11. Standard

equipment for the four main production operations (drill—blast—load – haul) is assumed, although specific types of machines are not identified (Martin et al., 1982). The sequence types of activities is apparent, as are the alternative approaches (e.g., ripping may replace drilling and blasting, and augering may be used in place of conventional coal recovery). Depending on circumstances, certain operations could be omitted (e.g. drilling and blasting, if the overburden is soil) or repeated (e.g. the entire cycle, if multiple coal seams are being mined).

How would the flow sheet differ for underground mining? Certainly the auxiliary operations would change (e.g., site preparation, clearing etc., would not be required, nor would the reclamation procedures; but additional *operations* such as roof control and ventilation, would have to be included). Details of the production cycle would also vary, assuming specialized underground equipment is required. But the scquent.al scheme and the basic production operations would be very similar. Preparing a flow sheet for the cycle of operations in a proposed mining venture is a necessary first step in system design.

Specifying the individual machines to perform all unit operations is followed by a careful balancing of the cycle. Ideally, the units of the system should be matched in capacity so that there is a uniform, uninterrupted flow of material from the working face to the surface disposal point (plant, loading pocket, or dump). This infers that all units have the same productive capacity, arc compatible size wise, will handle the material being mined, and will function within constraints of the workplace (Martin et al., 1982). Sometimes different equipment capacities can be brought into balance by varying the work schedule (e.g., drilling and blasting may be performed on one or two shifts, excavation may be scheduled for all three shifts).

In designing the production cycle, once individual machine capacities are established, the number of units (e.g., drills or trucks) can be determined from the required mine output. Notice that the overall availability in the production cycle is the product of the individual machine availabilities. For example, if the shovel is available *90%* of the time, the trucks are rated at *85%.* and the crusher at the dump point has a *95%* availability, then the system availability is 73%. Likewise, the output of the system is limited by the slowest and lowest-capacity machine in the cycle. Sometimes balance in a system can be achieved by over sizing all units, providing backup units, or incorporating surge piles—all of which incur added costs. Clearly, the output, cycle time, and number of units must be estimated with reasonable accuracy if a production cycle is to be properly balanced, and these calculations will be introduced later.

Several alternative mining systems typically are investigated. Thus in surface mining of coal, we may compare (1) front-end loaders and trucks, (2) rubber-tired scrapers, and (3) draglines for stripping soil overburden. In surface ore mining, (1) rotary drills, ANFO explosives, power shovels, and trucks may have to be compared with *(2)* percussion drills, TNT slurries, front-end loaders, and conveyors. In underground coal mining, the choice may he between *(1)* continuous miners and belt conveyors and (2) convention equipment (short wall cutter, auger drill, permissible explosives, gathering arm loader, and shuttle cars). Martin et al. (1982) provide some guidelines for selecting surface mining systems, and Bullock (1982a) does the same for underground systems.

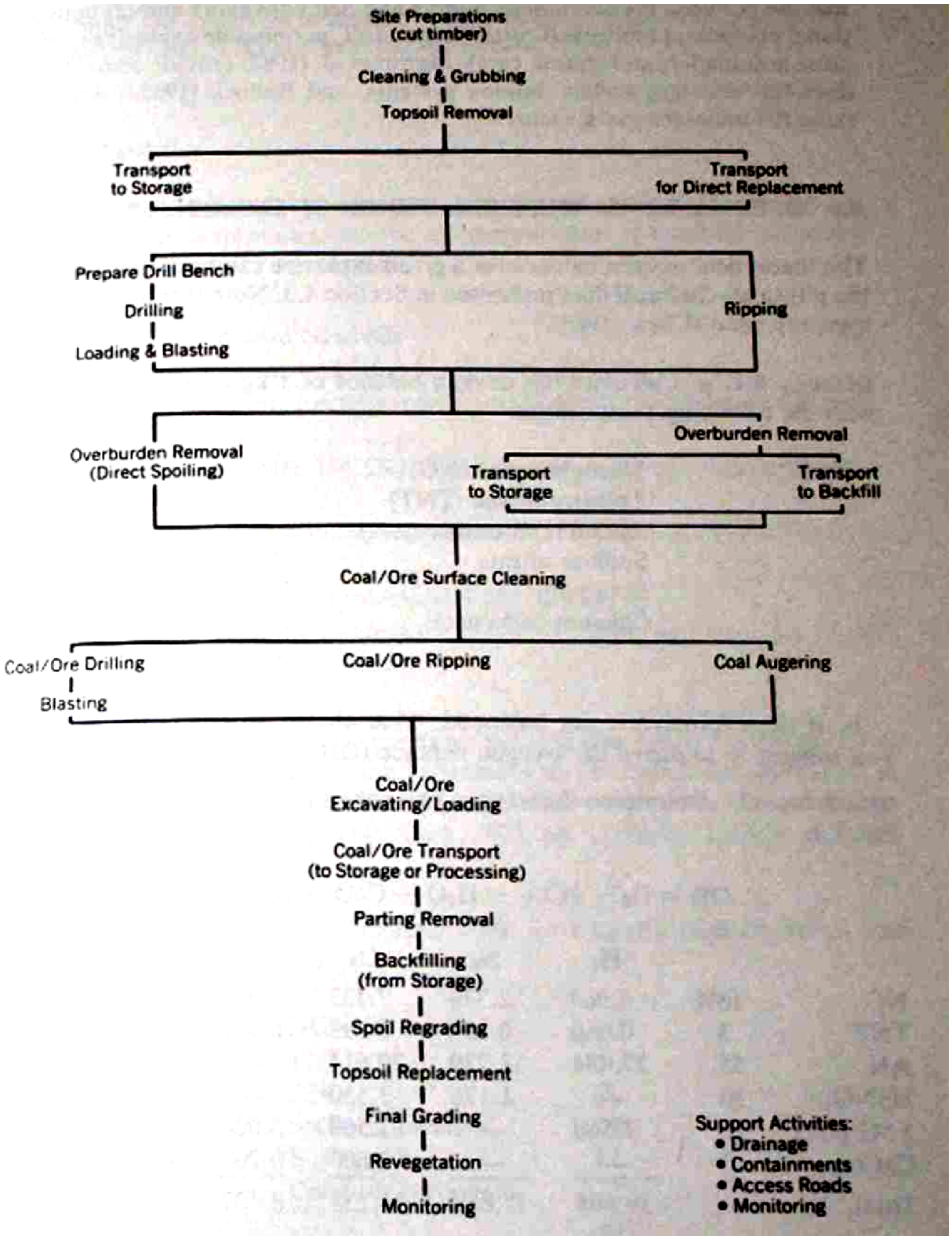


Figure 4.11. Typical cycle of operations for surface mining, incorporating both production and auxiliary operations. (After Martin et al.. 1982. By permission from Martin Consultants, Inc., Golden, CO.)

4.7 SPECIAL TOPIC: CHEMICAL DESIGN OF EXPLOSIVES

The theoretical oxygen balance for a given explosive can be calculated from the physical-chemical data presented in Section 4.3. Note that S.I. units are typically used (Clark. 1968).

EXAMPLE 4.1. a. Calculate the oxygen balance of I kg of a high explosive with the following composition:

Nitroglycerine (NG)	18%
Trinitrotoluene (TNT)	3
Ammonium nitrate (AN)	55
Sodium nitrate	10
S:Gpulp	12
Calcium carbonate	2
	100%

b. If the explosive is not balanced, what change in composition would you suggest to improve the oxygen balance (OB) solution, a. Determine the oxygen balance using data from Table 4.4 and Eq. 4.4:

$$OB = O_0 - 2CO_2 - H_2O - CaO - Na_2O = 0$$

		H_0	N_0	O_0	C_0	Ca	Na
NG	18%	3.964	2.378	7.133	2.378	—	—
TNT	3	0.660	0.396	0.793	0.925	—	—
AN	55	27.484	13.739	20.614	—	—	—
$NaNO_3$	10	—	1.176	3.530	—	—	1.176
S:G pulp	12	7.560	—	2.568	5.004	—	—
$CaCO_3$	2	—	—	0.600	0.200	0.200	—
Total		39.668	17.689	35.238	8.507	0.200	1.176

$$OB = 35.238 - 2(8.507) - \tfrac{1}{2}(39.668) - 0.200 - \tfrac{1}{2}(1.176)$$
$$= -2.398 \text{ g-atom/kg}$$

b. Since the explosive is slightly oxygen-deficient (fuel-rich), reduce the fuel content (NG, TNT) or increase the oxidant (AN).

PROBLEMS

4.1a. Write simplified, balanced equations for detonation of the following Maying agents, expressing also the exothermic heal of reaction in kcal/kg (obtain from reference or calculate):

1. Ammonium nitrate NH_4NO_3
2. Ammonium nitrate and lamp black NH_4NO_3+ C
3. Ammonium nitrate (96.6%) and fuel oil (3.4%)—fuel shortage NH_4NO_3+ CH_2

4. Ammonium nitrate (94.5%) and fuel oil (5.5%)—oxygen-balanced NH_4NO_3 + CH_2
5. Ammonium nitrate (92%) and fuel oil (8%)—fuel excess NH_4NO_3 + CH_2
6. Metalized ammonium nitrate NH_4NO_3 + Al
7. Metalized (9.9%) ammonium nitrate (87.6%) and fuel oil (2,5%) NH_4NO_3 + CH_2 + Al

b. Compare the heats of reaction, ranking the seven explosives from greatest to least. Account for the variation. Can the higher energy release of certain explosives be utilized in blasting? How? Are there limits? (References: Clark, 1968, pp. 341-346; Dick, 1973, pp. 11:78-80; Dick et al., 1983, pp. 3-4),

4.2a. Calculate the oxygen balance (OB) for 1 kg of a high explosive with the following composition:

NG	54%
TNT	9
AN	19
Sodium nitrate	4
S:G pulp	12
Calcium carbonate	2
	100%

b. Is the explosive in balance? What change in composition would you suggest to improve the OB?

4.3a. Calculate the OB for I kg o f a high explosive with the following composition: 8

N G	*9 %*
T N T	*2*
A N	63
Sodium nitrate	12
S : G pulp	12
Calcium carbonate	2
	100%

b. Is the explosive in balance? What change in composition would you *suggest to improve the OB?*

5

SURFACE MINE DEVELOPMENT

5.1NATURE AND SCOPE OF THE TASK

A review is in order (Sections 1.4, 3.2, 3.3, 3.4) as we examine more closely development applied to surface mining, preparatory to a discussion of individual surface exploitation methods.

Most surface mining methods by their very nature are large-scale (or mass-production) *methods.* The sheer magnitude of the volume or tonnage of material broken and handled in surface mining is staggering. Table 5.1 compares estimated U.S. production of ore and waste for different commodity classes in surface and underground mining and in summary for the United States and the world. Significant observations are that, in the United States. (1) six times as much ore, coal, and stone (85%) is excavated by surface mining as underground, but 20 limes as much total material (95%) is excavated: (2) coal and its associated waste accounts for 65% of the total material excavated: (3) essentially all (>99%) the waste handled in mining originates m surface operations; and (4) 73% of the material excavated in all mining is waste. World statistics are similar except that surface mining is somewhat less dominant.

Certain factors in mine development (Section 3.2) receive special attention in preparation for surface mining. Of the locational factors, climate is of more critical concern in surface operation than underground. Today, harsh climates at high altitudes or in northern latitudes rarely mitigate against surface mining, but they can be detrimental. Among natural and geologic factors, terrain, depth and spatial characteristics of the deposit, and presence of water arc most important in surface mining. Among environ – mental factors, certainly antipollution and reclamation requirements rank highest as concerns in surface mining.

Noting the steps *in* the sequence of mine development enumerated in Section 3.2, there are three that are unique to surface mining:

1. Initiation of a land reclamation plan as part of the environmental impact statement (EIS).
2. Provision of topsoil stockpiles and waste-disposal dumps.
3. Carrying out of advanced stripping of overburden to gain access to the deposit.

Normally, all the other steps identified must be carried out, too, but these three require special consideration in the scheduling of development tasks. As an example, Figure *5,*1 depicts the scheduling diagram for a hypothetical surface

metal mine, designed to produce 20,000 tons/day (18,100 tonnes / day). Stage 1, prospecting, consumes about 2.5 years, and stage 2, exploration and feasibility study, another 5.5 years. Stage 3, development, requires 3 more years, with an additional year of production to reach rated capacity. The total elapsed time is 12 years (an underground mine typically requires at least one to two years longer). Land reclamation, waste disposal, and advanced stripping arc scheduled in Figure 5.1 during stage 3. (Notice that it is customary usage to *mine* ore, coal, or stone but to *strip* overburden or waste.)

We consider these three tasks, together with plant layout, in more detail.

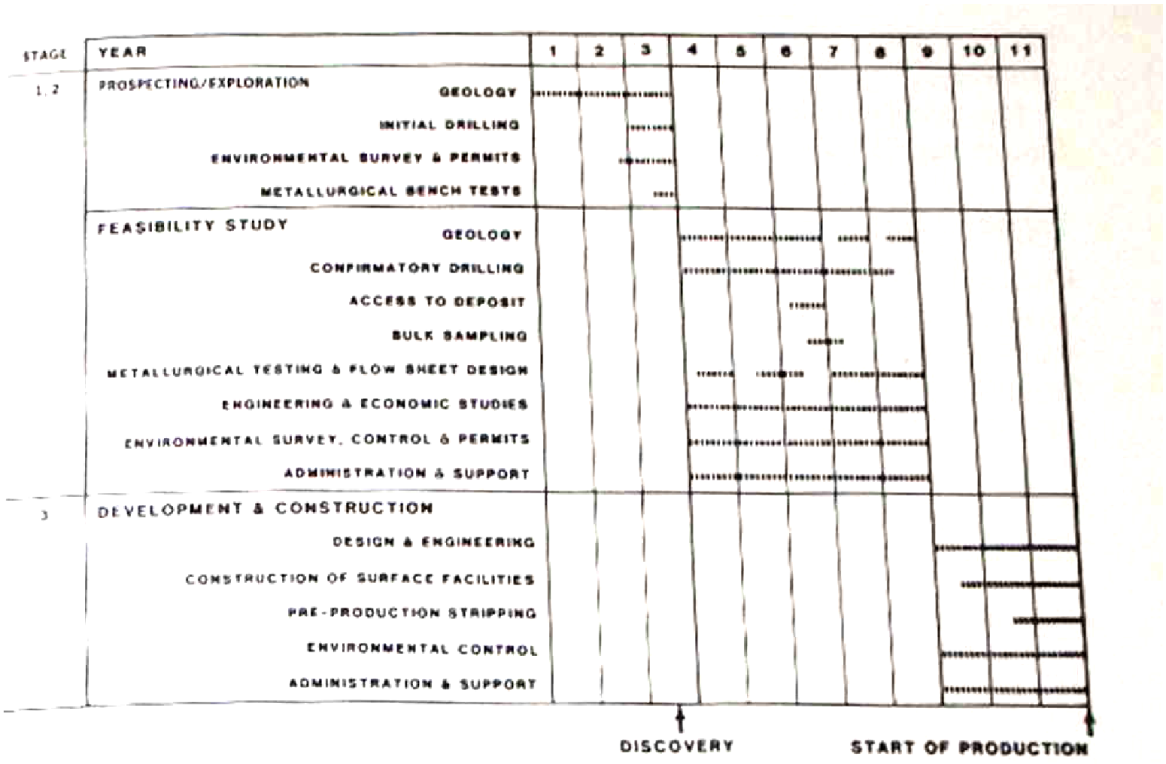


Figure 5.1. **Scheduling diagram** for surface metal mine. (After Petty. 1981. By permission **from American** institute o1 Professional Geologists. Colorado Section. Golden, CO.)

Land Reclamation

The federal Surface Mining Control and Reclamation Act of 1977 requires that land disturbed by mining be reclaimed and restored to its premining condition or belter. In essence, the law stipulates that if land cannot be successfully reclaimed, it cannot be mined (Anon., 1984c). During mine development, the first steps are taken to ensure that the EIS filed by the company is fully implemented. To obtain a mining permit, the company has to post a sizable bond (in the western United States, up to $10,000 per acre, or $24,700 per hectare) to cover the estimated reclamation cost. The bond is not released to the operator until adequate plant growth is restored, and a portion of the bond must be held for 5 or 10 years. Provisions of the law, which are especially applicable to surface coal mines, are considered stringent, Restoration of the land to its "approximate original contour" during and following mining is a particularly binding requirement that necessitates

careful planning, surveying, and mapping during mine development. Preserving surface drainage may require stream relocation or diversion. Maintaining wildlife is a special concern on land that is to be surface-mined; the two are not incompatible, but provisions to protect game and provide acceptable habitats for both birds and animals must be initiated early in the life of a mine. Finally, archaeological sites, known or uncovered, must be protected.

For provisions of a typical state code (Illinois), see Anon. (1985a); for foreign regulations, sec Atkinson (1983).

Topsoll Stockpiles and Waste Disposal

During .he development stage of surface mining, topsoil stockpiles for reclamation purposes and dumps for waste disposal are located. Separate areas are provided for topsoil, soil, rock, low-grade or potential ore, and tailings from mineral processing, permitting separate handling and recovery if necessary. As minted out in Section 3.4. site selection to ensure convenience but to avoid interference with production and auxiliary operations is a first objective, with advance planning to prevent conflicts in land use a second objective.

Advanced Stripping

The geometry of the mineral deposit and overburden together with the planned production rate of the mine largely determine the amount of advanced stripping that must be done (Pfleider. 1973a). The general rule in mine development (Section 3.1) is to limit preproduction activities to a minimum so as not to aggravate the cash flow problem. However, some advanced stripping is an absolute necessity, the amount being a discretionary decision by management. One rule of thumb, common in truck-shovel operations, is to maintain a 30-day supply of ore available for mining ahead of the shovels. In a typical bench-mining operation, I ft of ore face must be exposed for each 4 to 5 tons of required daily production (1 m per 12-15 tonnes). For example, if the required production is 20,000 tons/day (18,100 tonnes/day), then advanced stripping must continue until 4000-5000 ft (1200-1500 m) of bench face is exposed and can be maintained.

An additional consideration is climate. Severe cold weather may favor stripping during summer months when the ground is thawed and mining during Ihe winter. In the iron ranges of the Lake Superior district, however, transportation by boat can occur only when the Great Lakes are thawed, so stripping is emphasized in winter and mining in summer.Whether the stripping is to be done in-house or by contractor is a final decision. Large mining companies usually prefer to do their own stripping, but a smaller company may find it expeditious to contract it out. Contract stripping is often more expensive but faster and saves the mining company from purchasing specialized equipment.

Plant Layout

The factors that must be considered in selecting the mine plant size and layout were identified in Section 3.4. For a surface mine, the task is complicated by the special considerations just discussed; land reclamation, topsoil stockpiling and waste disposal, and overburden stripping. The plant layout for the Black Thunder mine in Wyoming, among

the largest surface coal mines in the world, is shown in Figure 5.2. The pit, waste dumps, and topsoil stockpiles are off the map to the lower right.

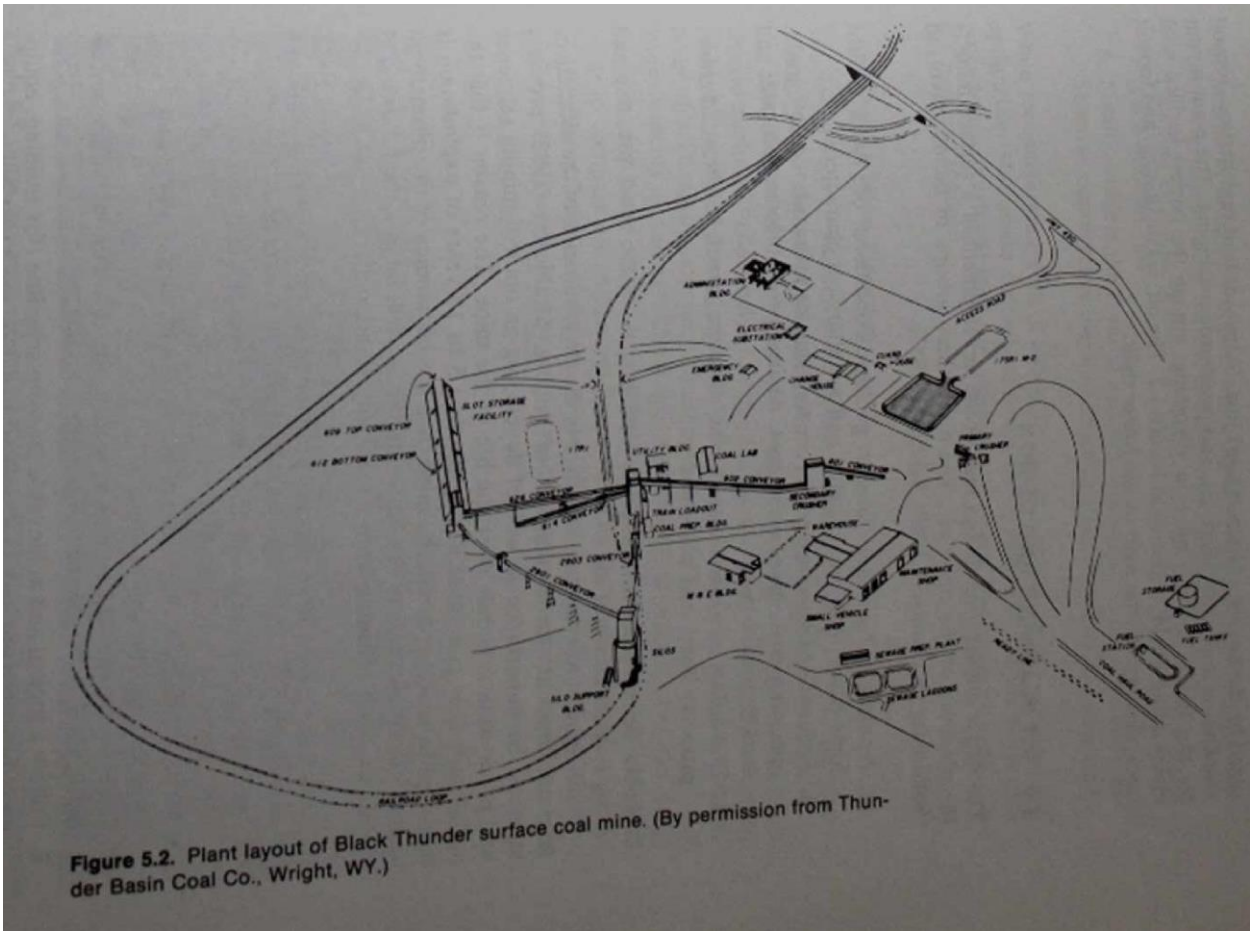


Figure 5.2. Plant layout of Black Thunder surface coal mine. (By permission from Thunder Basin Coal Co., Wright, WY.)

5.2 PIT PLANING AND DESIGN

The major engineering design task in the development of a surface mine is the planning of the open pit. There are three groups of factors involved (Soderberg and Rausch, 1968; Atkinson, 1983).

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

Probably determination of the last—pit limits is the most pressing and vexing consideration of all.

While much importance is attached to the determination of the ultimate or final pit limit at the termination of mining, Mathieson (1982) properly stresses the importance as well of developing an optimal mining sequence and production schedule over the life of

the mine. The reason is that the initial cash flow generated during the first 5 to 10 years of exploitation is more apt to make or break (he mine than (he economics of the ultimate pit limits forecast 20 years in the future. In this regard, he lists the following as objectives of pit planning from a feasibility standpoint:

1. Mine (he ore body so (hat the production cost per lb (kg) of metal is a minimum (i.e., mine (he "next best ore" in sequence).
2. Maintain operational viability (adequate bench width and ready haulage access for equipment).
3. Maintain sufficient exposure of ore to counter miscalculations or insufficient data from exploration.
4. Defer stripping requirements as long as possible without binding equipment, manpower, or the production schedule.
5. Follow a logical and achievable start-up schedule (for training, equipment procurement and deployment, logistics, etc.) that minimizes (he risk of delays in positive cash flow.
6. Maximize design pit slopes, while minimizing likelihood of bank failures (provide safety berms, employ rock mechanic», etc.).
7. Examine economic merits of reasonable production rate and cutoff grade alternatives.
8. Finally, subject the favored method, equipment, and schedule to exhaustive contingency planning before proceeding with development.

While recognizing the role of computer-based techniques in analyzing and optimizing, Mathicson recommends the use of manually based pit-design procedures, interfaced with the computer, to ensure operational viability in the resulting model.

Long-Range Mine Planning

The initial step in surface mine design is the compilation of a long-range mining plan or final pit design (Plleider, 1973a). In preparing the plan, plotting; the location of the various ore types and deposit boundaries based on refined exploration data helps to establish the ultimate pit limits (Fig. 5.3). In reality, long-range mining plans usually change over time to reflect the effects of a changing economy, increased knowledge of the ore body, and improvements in mining technology. As a result, as discussed above, long- range mining plans should be updated at regular intervals, using the computer to compare alternative pit limits, such as shown in Figure 5.3.

The determination of the *maximum allowable stripping ratio* SR_{max} of a surface mine is a break-even ratio, based solely on economics, which establishes the pit limits. It is defined as the ratio of overburden to ore at the ultimate boundary of the pit where the profit margin is zero. Mathematically, it is computed as follows (Pfleider, 1973a):

$$SR_{max} = \frac{\text{value of ore} - \text{production cost}}{\text{stripping cost}}$$

$$= \frac{\text{stripping allowance}}{\text{stripping cost}} \qquad \frac{\$/\text{ton}}{\$/\text{yd}^3} \text{ or } \text{yd}^3/\text{ton } (\text{m}^3/\text{tonne}) \qquad (5.1)$$

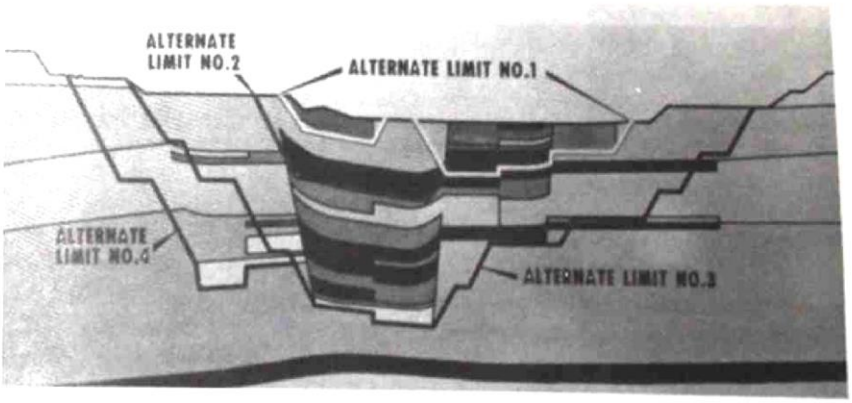


Figure 5.3 Cross section of iron ore pit showing different types of ore deposit boundaries and alternate pit limits (After Phelps, 1968. By permission from Society of Mining Engineers, Inc., Littleton. CO.)

The value of ore in S/ton (S/tonne) the recoverable value, and the production costs in S/ton (S/tonne) is the overall costs through refining or final processing, exclusive of stripping. The tripping cost in S/yd^3 (S/m^3) is the cost of breaking and handling a unit volume of standard overburden. Since value minus cost normally equals profit, and the profit is assigned /ero at I he pit limit, the numerator becomes the stripping allowance.

Because ore grades actually vary throughout the pit and ore prices fluctuate, the maximum allowable stripping ratio can change with lime. Thus it is helpful for a given property to prepare a table or graph showing the variation of SR_{max} with ore grade and price.

The maximum allowable stripping ratio also has physical significance (So - derberg and Rausch. 1968; Pfleider. 1973a). It enables us to locale the ultimate pit boundary or limit for prevailing economic conditions and for exist- mg physical and geometric conditions in the pit (e.g.. for a given pit slope and formation dip), losing computer graphics and Mathieson's interactive mode, various alternatives can be explored readily, even for complex geometries and in three dimensions (Fig. 5.4) Al the value of SR_{max} where the cost of surface mining first exceeds that of underground mining, the portion of the ore body lying beyond the ultimate pit limit would have to be mined by underground methods. A copper pit designed in this manner with varying ore grades and critical SR_{max} =3,0 yd^3/ton (2.5 m^3/tonne) is shown in Figure 5.5. Ore occurring in the ore body beyond this maximum stripping ratio will have to be left or mined underground, as indicated.

Short-Range Mine Planning

Once a long-range surface mining plan has been established, it is essential to develop a series of short-range mining plans (Pfleider, 1973a). These plans define the intermediate steps required to ascertain the final pit limits under physical, operating, and legal constraints. They also provide the pit boundary, ore grade, stripping ratio, and anticipated profit information necessary for future production forecasts and equipment needs.

In preparing a short-range plan, the engineer lays out on a set of horizontal sections a series of proposed mining cuts. The location and extent of the cuts reflects the engineer's judgment of the various operational factors involved. Very likely, more than one ore type occurs in the pit, and more than one pit may be mined. In this ease, an annual mining

program is developed which depicts the daily production rate scheduled for each ore type. A mining schedule for a hypothetical copper deposit producing four different ore types over a 12-year period is shown in Figure 5.6.

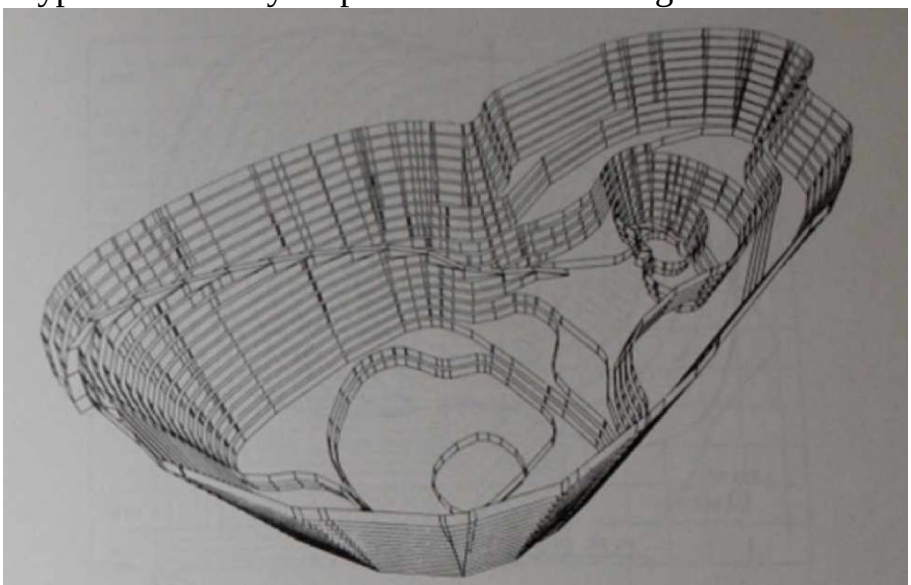


Figure 5.4. Computer simulation of pit boundaries of surface mine, in three dimensions. (By permission from Control Data Corp., Minneapolis.)

The mining strategy for either the short or long range must reflect corporate and management goals. For example, the time value of money suggests a sequence of mining the best ore grade first in order to maximize early profits. However, a strategy of mining a uniform grade may be supported by improved mineral-processing recovery and efficiency and income tax and depletion allowance considerations. Corporate plans must also reflect operating strategies for other of its mines.

Stripping sequence may be another operating variable to investigate; that is, how far in advance should certain ores or portions of the ore body be stripped? An important element of short-range planning is to program stripping ratios over the life of the property so they are not excessive during any period, especially at the outset of exploitation. An operating technique frequently utilized in the final years of mining is to steepen pit slopes, thus improving the stripping ratio, a practice made feasible by the foreshortened life span of the working benches (Fig. 5.7).

With the variety of operations research (O R) and computer simulation techniques at their disposal today, mine engineers are able to develop, evaluate, and optimize development strategies with ease. Monte Carlo simulation, linear programming, critical path analysis, game theory, dynamic programming, and queuing theory are some of the methods that permit the solving of even the most complex scheduling-optimization problems (Crawford, 1979; Crawford and Hustrulid, 1979; Gibson et al., 1982; Jeffreys and Hoare, 1985). Recalling cautions by Mathieson (1982), however, computer-OR techniques are tools that optimize a model only, not an actual mine.

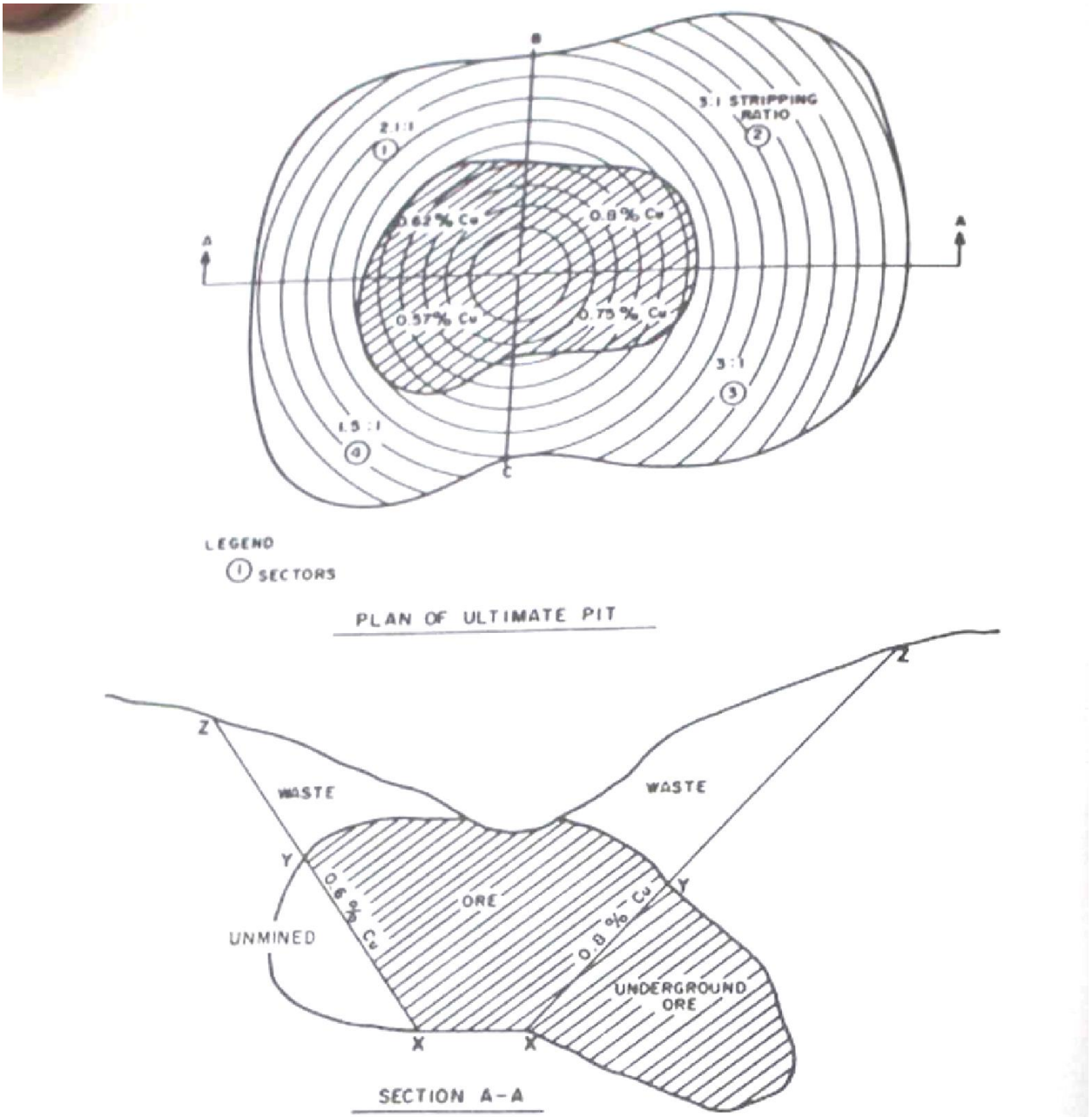


Figure 5.5. Location of pit limits for a copper ore body. Ore grades and maximum allowable stripping ratios are indicated in the four quadrants of the pit. (After Pfleider, 1973a. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

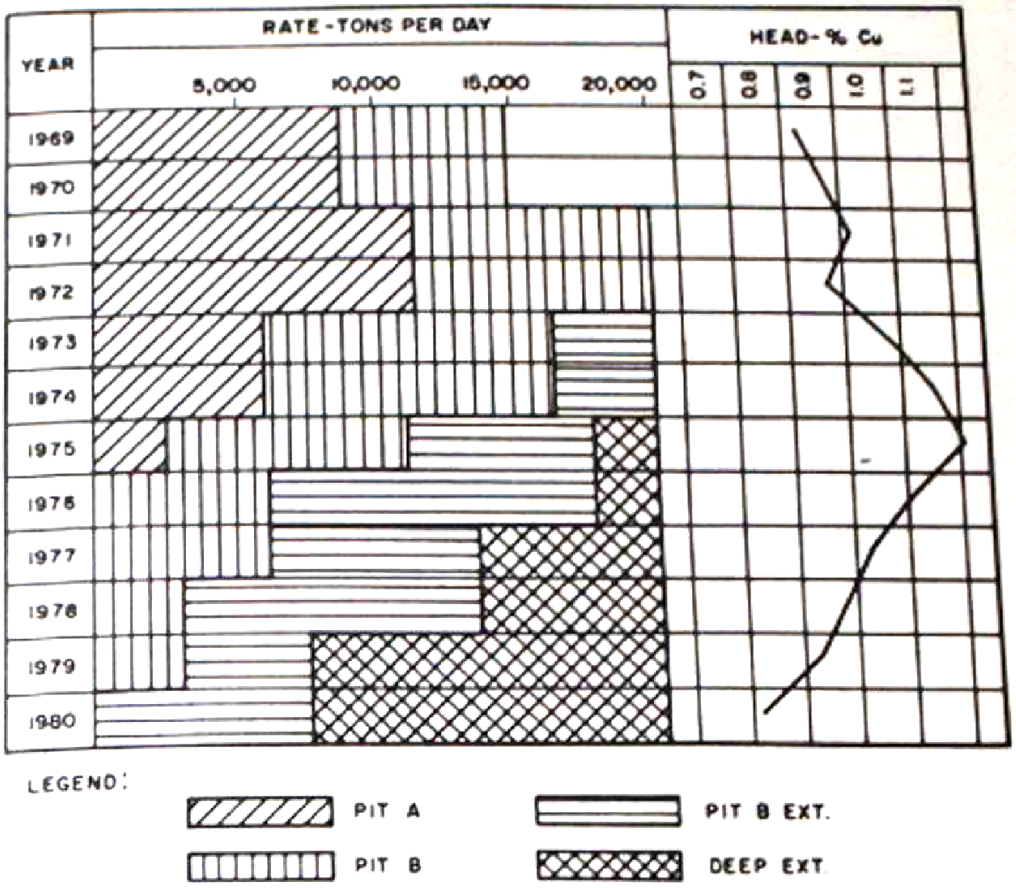


Figure 5.6. Average ore grade and mining schedule for copper pit depicting 12-year production sequence for different ore types. (After Soderberg and Rausch, 1968. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

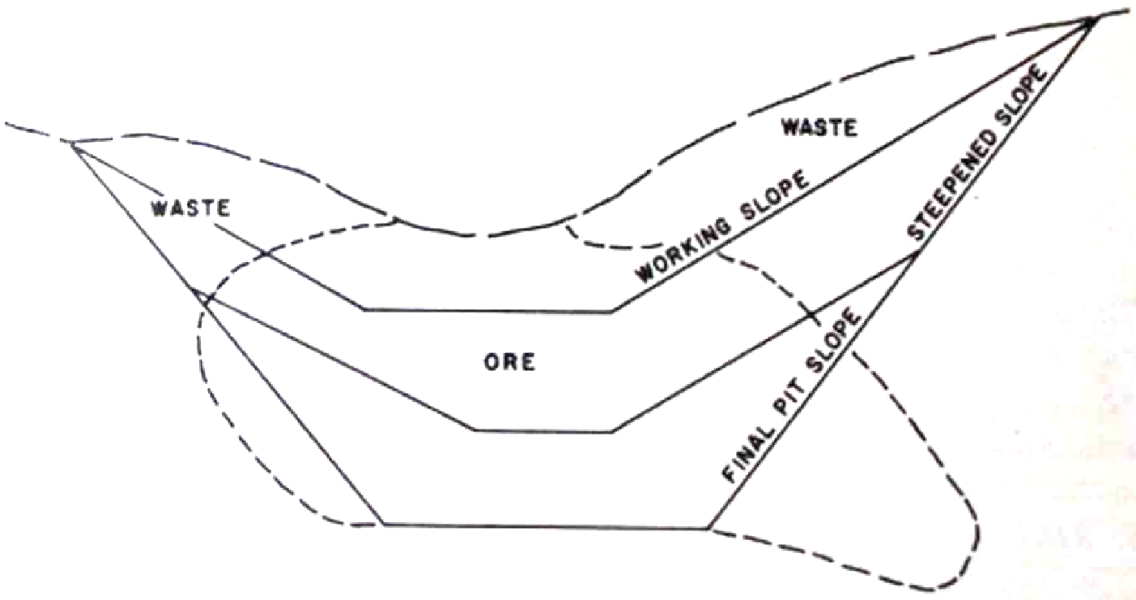


Figure 5.7. Steepening of pit slopes to final pit limits, with corresponding reduction in stripping ratio. (After Soderberg and Rausch, 1968. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

The general principles of equipment selection and system design were set forth in Section 4 7. Now we apply them to the operating circumstances of surface mining, ft is no. surprising .hat the selection of equipment ,s nearly synonymous with specifying the stripping method, because materials handling lies at the heart of surface mining.

In selecting a particular stripping method and its corresponding equipment .he ultimate aim is the removal of material (waste, overburden) at the least possible cost (Pfleider. 1973a). Many factors are involved, of which the size of the ore body, distribution of values within the deposit, and consolidation and compaction of the overburden are key and limit the range of method and equipment choices. Other factors to be evaluated, however, are the presence of geologic structures (faults, folds, shear zones, water-bearing zones, etc.) and alteration products (which may render roads impassable or mineral processing difficult): production rate and life; horizontal haul and vertical hoist distances; and possible future uses for specialized stripping equipment (subsequently during mining or reclamation or at other mines, etc.).

Pit geometry has a major effect on the type and size of equipment which can operate effectively. Each major mechanical extraction method of surface mining has its unique geometry: for example, open pit metallic and nonmetalic mines are designed with many low benches, open cast and auger coal mines generally consist of one or more high wall faces, and dimension- stone quarries operate from a single high bench. Open pits tend to be deep and laterally large, open casts are shallow and follow contours or advance across the country in long narrow panels, and quarries are usually very steep and may be quite deep. These geometric distinctions definitely favor or limit the kind of stripping equipment to be selected.

As a general rule, the largest equipment feasible and safe for the conditions is also the best-suited and lowest-cost equipment. Large materials- handling machines especially possess productivity and cost advantages that seem to have no upper limit, but practical limits (e.g.. mobility) do prevail.

The choices of equipment and systems to mechanically strip ore, coal, or stone deposits arc many in theory but few in practice (Martin et al.. 1982; Atkinson, 1983). Some of the machines employed in surface mining, arranged by cycle of unit operations, arc shown in Figures 5.8a-e. There are three main rock-breakage systems and six materials-handling systems in common usage today, as follows, in order of popularity:

1. Rock breakage
    a) No breakage necessary (material: typical soil)
    b) Drilling and blasting (usual choices: roller-bit rotary and ANFO or slurry; material: typical rock)
    c) Ripping (material: stiff soil, weak rock—for scale of rippabilitv see Pfleider, 1973a, pp. 17-25)

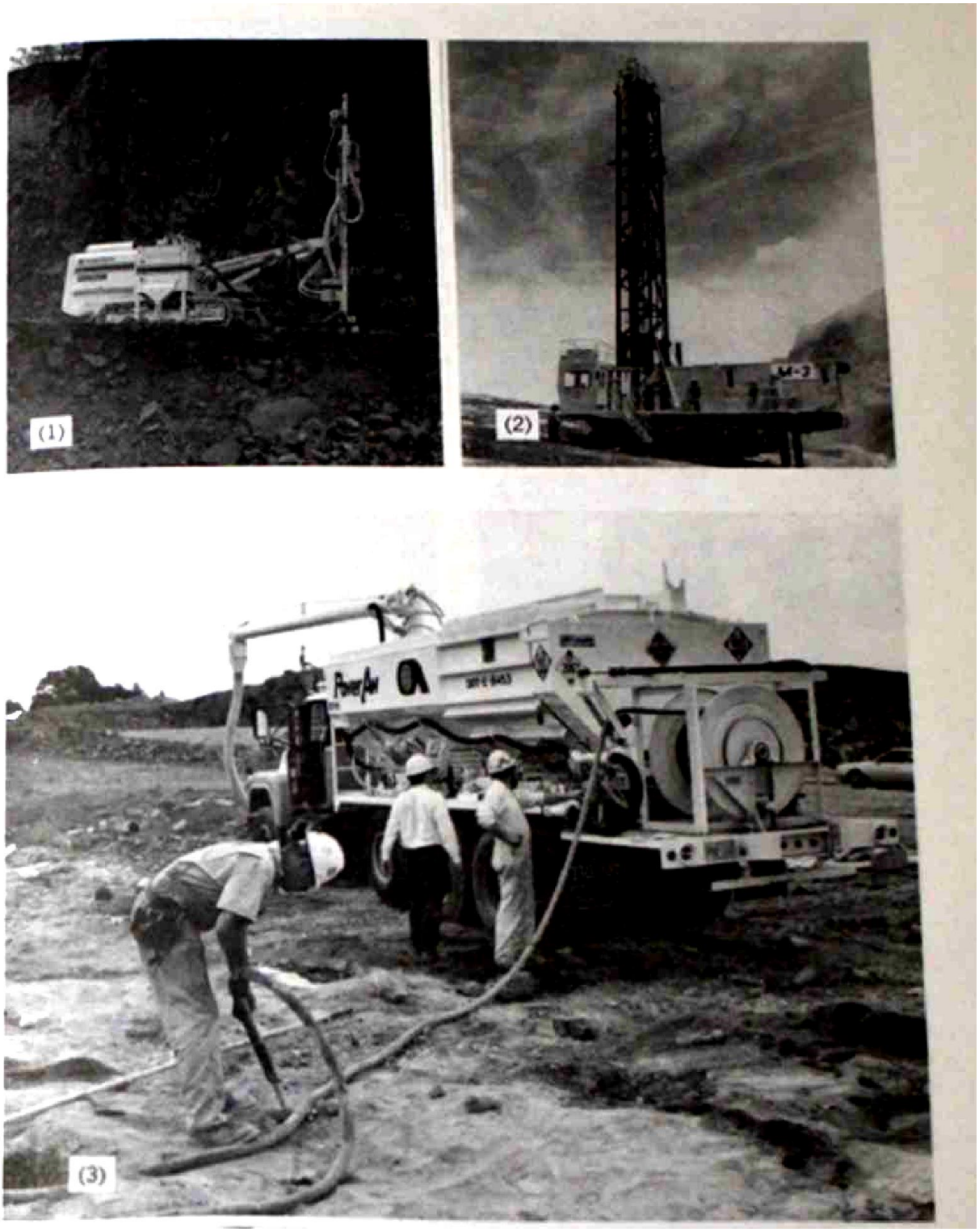


Figure 5.8a. Unit operations and equipment for surface mining: drilling and blasting (1) Small-hole, percussion, air trac drill. (By permission from Gardner-Denver Co.. Cooper Industries, Roanoke. VA.) (2) Large-hole, roller-bit, rotary drill. (By permission from Marion Div., Dresser Industries. Marion. OH.) (3) Bulk explosives truck (By permission from Atlas Powder Co., Dallas.)

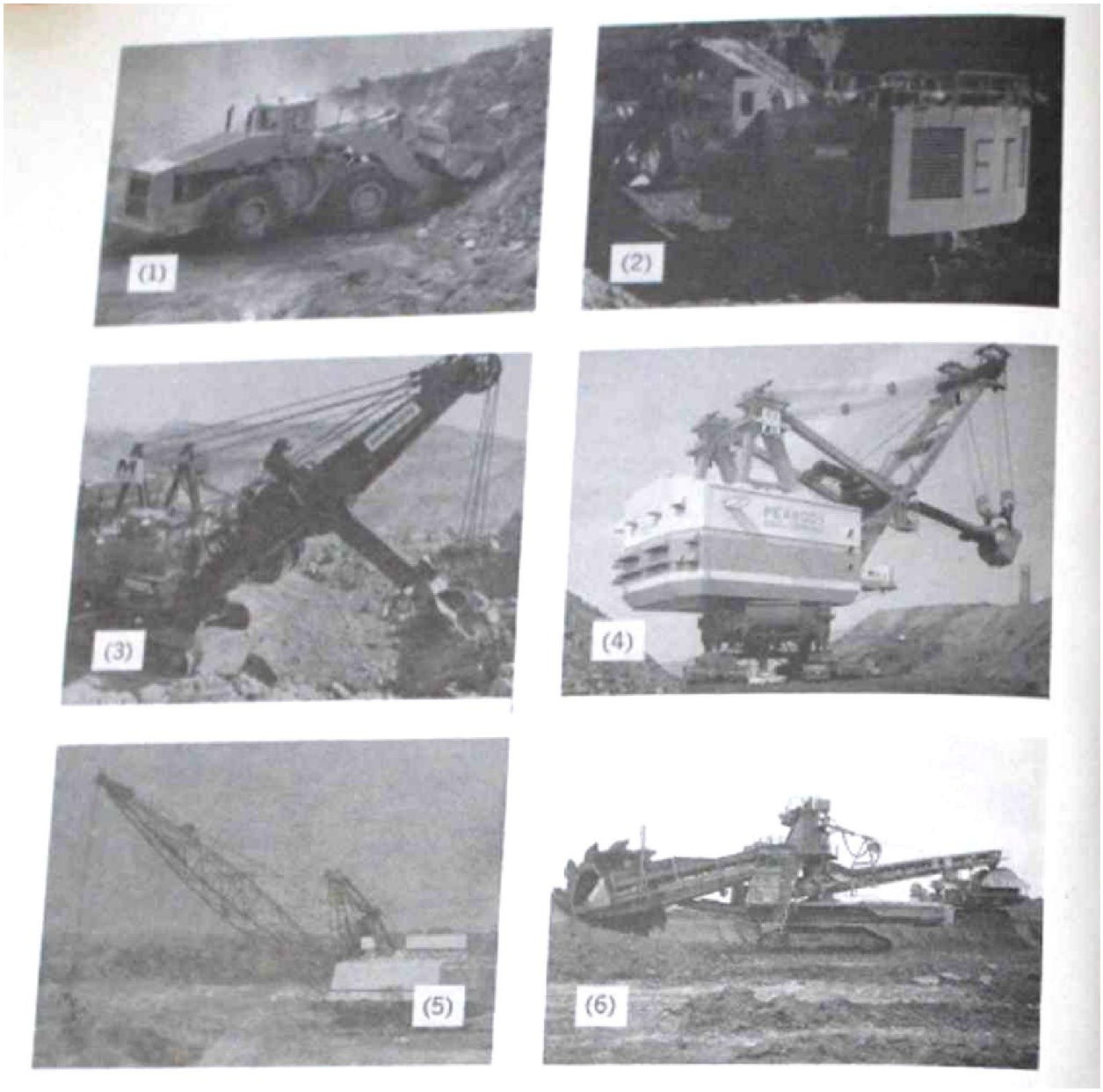


Figure 5.8b. Unit operations and equipment for surface mining: excavation. (1) Front-end loader. (By permission from Marathon LeTourneau, Longview, TX.) (2) Hydraulic excavator. (3) Mining shovel. (By permission from Marion Div., Dresser Industries, Marion, OH.) (4) Stripping shovel. (By permission from Bucyrus-Erie Co., South Milwaukee, Wl.) (5) Walking dragline. (By permission from Marion Div., Dresser Industries, Marion. OH.) (6) Bucket wheel excavator. (By permission from O&K Orenstein & Koppel. Edmonton, Can.)



Figure 5.8c. Unit operations and equipment for surface mining: excavation (*con tinued) (Top)* Dozer. (By permission from Maraton LeTourneau Co., Longview, TX.) (*Bottom)* Ripper. (By permission from Conway Technologies Corp., McAllen, TX.）

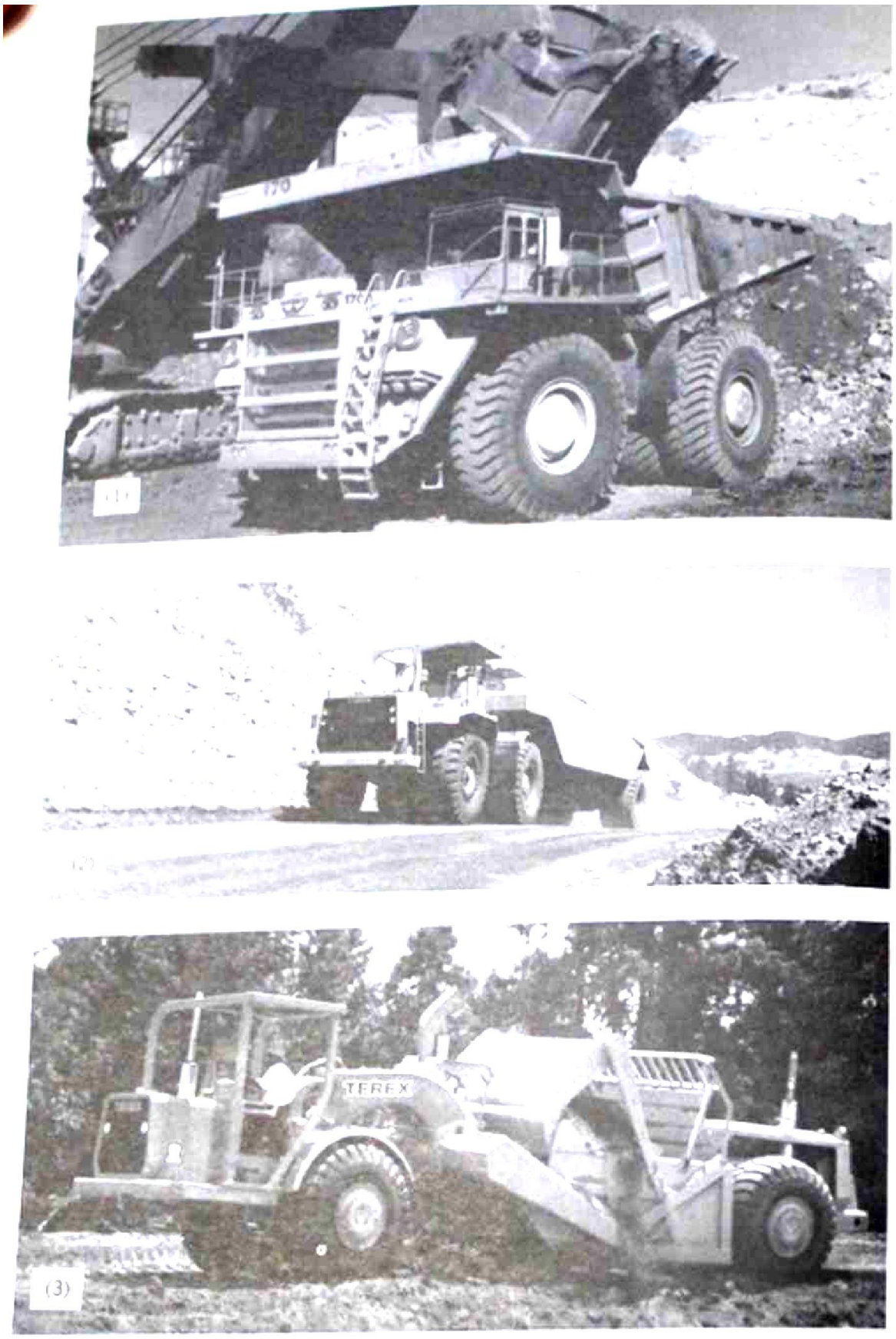


Figure S.8d. Unit operations and equipment for surface mining: rubber-tired haulage. (1) Diesel-electric rear-dump truck (By permission from Wabco Haulpak Div.. Dresser Industries, *Peoria, II.) (2)* Bottom-dump tractor-trailer. (3) Wheeled scraper. (By permission from Terex Corp.. Hudson. OH.)

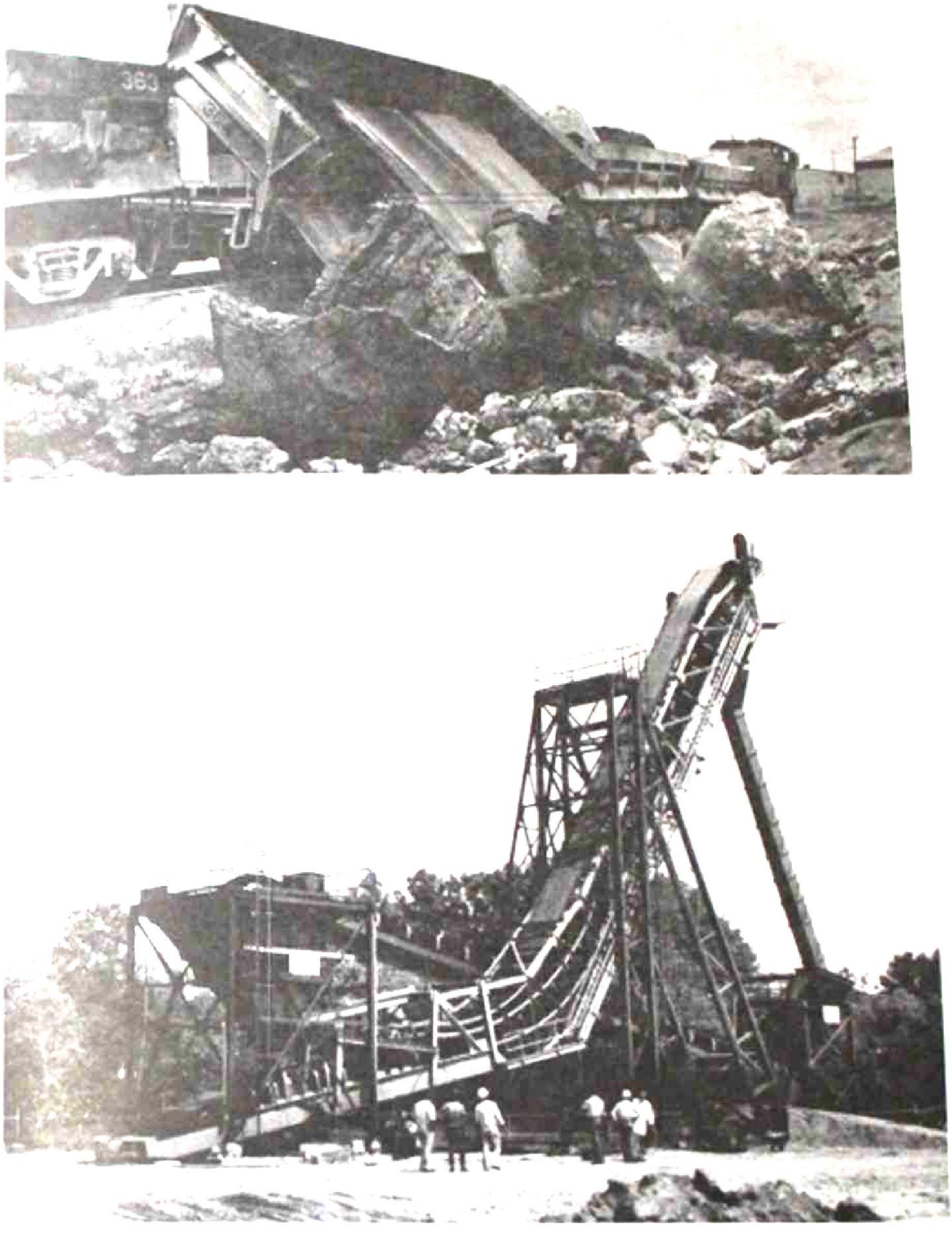


Figure 5.8e. Unit operations and equipment for surface mining: other haulage (Top) Side-dump rail car. (By permission from Difco, Inc.. Findlay. OH.) (*Bottom)* High-angle belt conveyor. (By permission from Continental Conveyor & Equipment Co.. Inc.. Winfield. AL.)

2. *Materials* handling
a. *Dragline (direct casting)*
b. Power shovel or front-end *loader* and trucks
c. *Dozer and front-end loader*
d Dozer and rubber-tired scraper
e. *Shovel or loader and* hopper, crusher, and conveyor
f. Bucket wheel excavator and conveyor.

Since the rock breakage alternatives are limited drastically by material characteristics, no further discussion is necessary. To assist in selecting the optimal materials - handling system for stripping or mining. Table 5.2 is presented (face preparation or rock breakage is specified, as are other important operating conditions, to aid in selection). Currently, draglines are favored for stripping in open cast mining, shovels and trucks in open pit mining, and dozers and scrapers in quarrying. Specialized haulage— hoisting equipment, such as hydraulic conveyors and skip hoists, are very limited in application (Section 4.5).

Prevailing relative costs for stripping systems are hard to obtain. For materials handling (haulage) equipment alone, the following arc helpful as guides (modified from Pfleider, 1973a, for an iron ore mine with a 5.5-mi, or 8,9-km. haul):

	Relative	Relative	Relative
	Ownership	Operating	Overall
Equipment	Cost	Cost	Cost
Truck	100%	100%	100%
Rail	85	86	85
Hydraulic conveyer	97	43	70
Belt conveyer	70	60	65

It may be significant that the wide, almost universal acceptance of truck haulage today is not always supported by economic data; and that in spite of its widely heralded flexibility and versatility, the truck may receive increasing competition in the future from the belt conveyor. Further, in steep pits (<60), high-angle conveyors are clearly advantageous (Dos Santos, 1984).

Helpful cost estimation procedures for surface mine development are provided for coal by Weimer (1968) and for noncoal by Phelps (1968).

5.4 STRIPPING RATIOS AND PITLIMIT S

Maximum vs. Overall Stripping Ratio

It is on the basis of calculating stripping ratios that we are able to locate pit limits and to express volumes of overburden to be moved per unit weight of ore, coal, or stone uncovered. W c must distinguish between two stripping ratios (units: yd '/ton. or m^3/tonne):

1. Maximum allowable stripping ratio SR_{max} =volume of overburden / weight of ore at economic pit limit = v/w
2. Overall stripping ratio SR_0 = volume of overburden / weight of ore for entire ore body or cross section = V/W

In only one instance docs SR_{max} not establish the pit limit or exceed SR_{max} in magnitude, and that occurs when (1) the surface is flat, and (2) the deposit is flat, tabular, and of constant thickness. In that singular case, SR_{max} lacks significance, and the pit limits are located at the property lines. Another distinction is that SR_0 is an actual numerical ratio of yd^3/ton (m^3/tone), whereas SR_{max} is expressed in units of equivalent yards (see the next section.)

Under long-range planning (Section 5.2), we discovered that the maximum *stripping* ratio, while a physical quantity, is determined solely by economics. The overall stripping ratio, on the other hand, has mainly physical significance. It is because of its economic basis that we can employ SR_{max} to locate the pit limits of a deposit in the general case, that is, an ore body of varying thickness, dip, or grade occurring beneath an inclined or horizontal surface.

Equivalent Yardage[2]

In developing a *relationship and* procedure to locate the pit limits in an ore body or *coal deposit,* we will employ a unit called the equivalent yard. *Equivalent yardage is the volume of* overburden which costs a unit amount to move *(expressed in $/yd*\ or S/m') and is accepted as a standard for the mine or district in which it occurs. It is a dimensionless unit. Examples of some standards and typical costs are

Lake Superior iron ranges (loaded and hauled)
Glacial till: S0.25 – 0.50/yd^3 (S0.33 – 0.65/m^3)

Eastern U.S. coal fields (cast)
Soil or decomposed rock: S0.10 – 0.30/yd^3 (S0.13 – 0.39/m^3)

Western U.S. porphyry copper district (blasted, loaded, and hauled)
Quartz monzonite porphyry: S0.50 – 1.00/yd^3 (S0.65 – 1.31/m^3)

The equivalent yardage rating *e* of a material is calculated with reference to the above standards, which are assigned a value of unity *(e* = 1). For example, at an eastern U.S. surface coal mine, if it costs $0.40/yd3 ($0.52/m3) to cast well-blasted rock, then the rating of the rock is *e* = 2, based on an average cost to cast *soil of e*=1 of $0.20/yd3 ($0.26/m3), the standard material in that district. A table of typical equivalent yardage ratings follows:

MATERIALE	Rating *e*
Dredged mud. water	0.5
Loose sand	0.7
Common soil (sand, loam, till)	1.0
Hard soil (clay, hardpan)	1.5
Shaley rock	1.5-2.5
Sandstone, limestone	2-3
Hard taconite	3-5

The concept of equivalent yardage is useful in dealing with a variety of stripping materials in calculating the maximum allowable stripping ratio and pit limits.

Relationships for Maximum Stripping Ratio and Pit Limit

The physical relationship of the maximum allowable stripping ratio at the pit limit enables us to develop a mathematical expression to locate the pit limit. Recalling that SR_{max} is an economic ratio but has physical significance, we $_{ean}$ use geometric relations in the pit to express the relationship.

Referring to Figure 5.9a, a simplified case of an inclined mineral deposit intersecting a horizontal surface is presented in cross-sectional view and the Key parameters indicated. The deposit thickness is t, its dip is a, its tonnage factor is TF, and its inclined length to the pit limit is *m*. The pit slope is *β,* its inclined length is *l*, its vertical height is *h*, and the horizontal distance from the outcrop to the pit limit is *d*. Note that *d* is measured to the *crest* of the bank and *m* to the *toe*. The volume of overburden on the cross section is V, *and the weight of ore is W. It is* common practice to express the angles *a* and *β in degrees, but β* may also be given as a slope ratio (horizontal-to-vertical), *which* then *must he* converted to degrees for the calculations that follow.

We *can develop* a geometric relation for *S R mal if we* reason that the crosssectional *view represents* a unit dimension through the deposit (I ft, or I m),

and that at the *pit limit, the* uncovering of a unit width of ore (1 ft, or 1 m) *requires the movement of If width of* overburden (Fig. 5.9b). Thus at the pit *limit, moving* a *prism of overburden of volume v* uncovers a prism of ore of weight *w.*

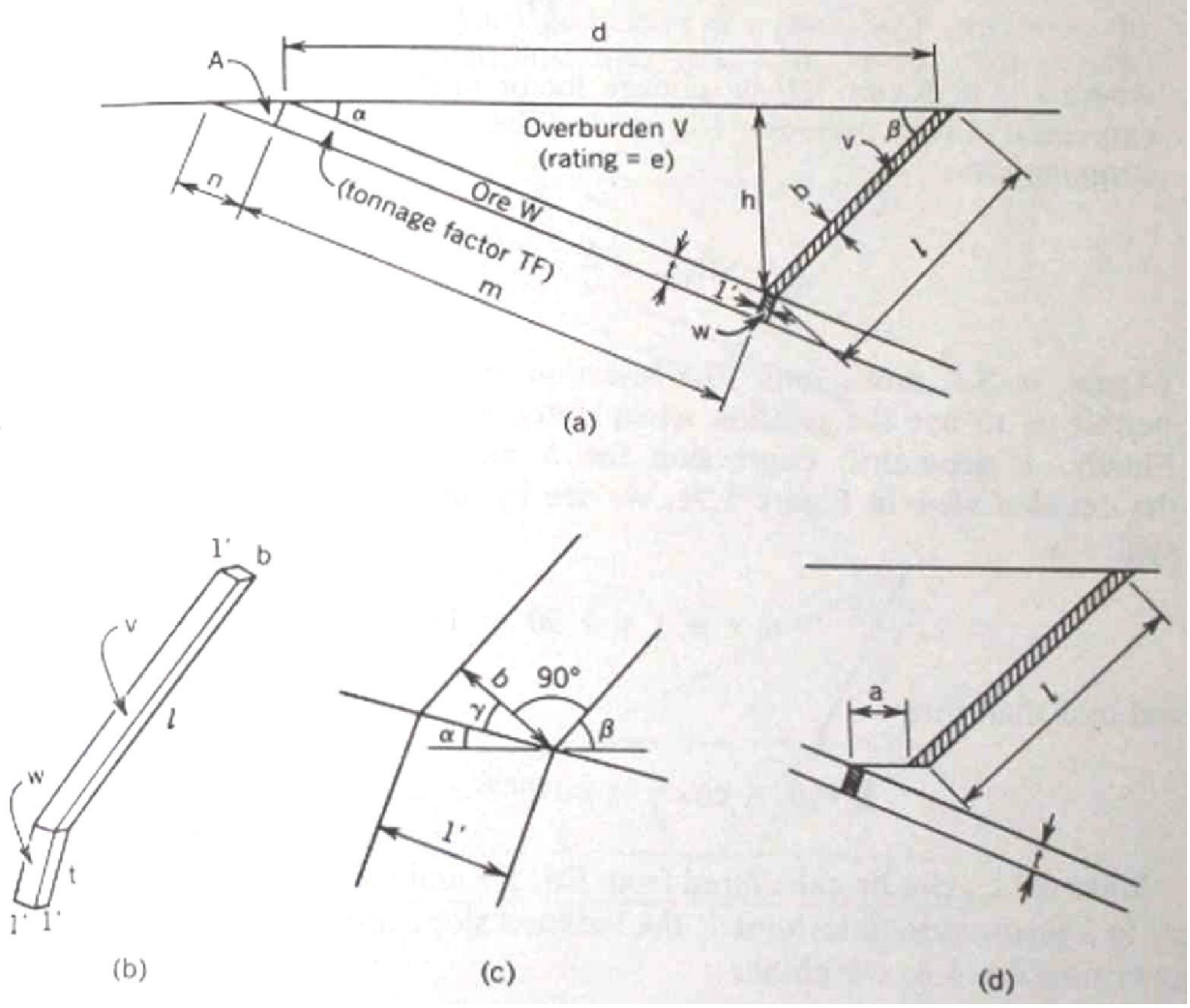


Figure 5.9. Geometric relation of pit parameters and SR^ at pit limit, (a) Cross section through ore body and overburden, (b) Detail at pit limit, (c) Detail at o re-overburden intersection (d) Effect of berm at pit limit. Unit dimension indicated by 1.

Mathematically, we can write, for the overburden,

$$v = \frac{1 \times b \times l}{27} \qquad (5.2)$$

where *b and l,* are in ft *(m)* and *v is* expressed in units of yd3/m^3, if the conversion factor 27 is omitted); and for the ore,

$$w = \frac{1 \times 1 \times t}{TF} \qquad (5.3)$$

where *t is in* ft (m). T F is tonnage factor in ft^3/ton (m3/tonne), and w is expressed in tons (tonnes). The *ratio of* the two is the maximum allowable stripping ratio

$$SR_{max} = \frac{v}{w} = \frac{ebl/27}{t/TF} \qquad (5.4)$$

(Again, in S.I. units, omit 27.) Insertion of the equivalent yardage *e* will permit us to use the relation when different overburden materials occur. Finally, a geometric expression *for b* must be obtained; referring to the detailed view in Figure 5.9c, we see by inspection that the sum of the angles is

$$a + \beta + y + 90° = 180°$$

and that therefore

$$b = 1 \times \cos y = \cos(90°) - a - \beta \qquad (5.5)$$

Since SR_{max} can be calculated from Eq. 5.1 and substituted in Eq. 5.4, we are in a position to determine /, the inclined slope at the economic pit limit.
Rewriting Eq. 5.4, we obtain

$$l = \frac{27t \times SR}{eb \times TF} \qquad (5.6)$$

By trigonometry, we can find the vertical height of pit slope *h* in ft (m):

$$h = l \sin\beta \qquad (5.7)$$

The horizontal distance from the outcrop *m* in ft (m), allowing for a *berm* of width *a* in ft (m) as a safety feature (Fig. 5.9d), is

$$d = a + \frac{h}{\tan \alpha} + \frac{h}{\tan \beta} \qquad (5.8)$$

Finally, the inclined length of ore from the outcrop *m* in ft (m) is

$$m = \frac{h}{\sin \alpha} \qquad (5.9)$$

There are several variations possible in deposit geometry and overburden composition that affect the determination of SR_{max} and the pit limits, and two o f them are shown in Figure 5.Ю. In both, two different overburden formations occur, with equivalent yardage ratings of *e t* and *e2t* respectively. In Figure 5 .10a, the surface is horizontal, and the deposit dips; in Figure 5.10b, the surface is inclined, and the deposit is flat. In writing a relation for maximum stripping , ratio, we modify Eq. 5.4 as follow:

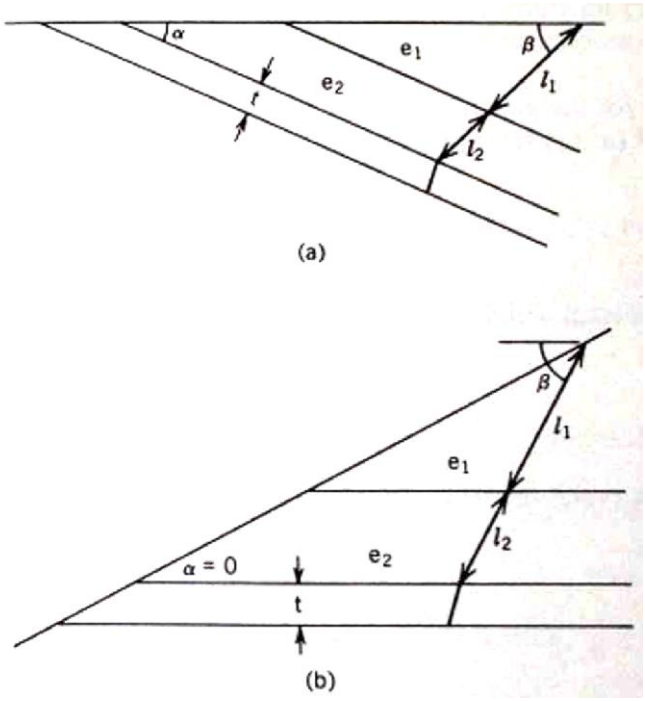


Figure 5.10. Variations in deposit geometry and overburden composition, (a) Typical dipping tabular noncoal deposit with different overburden formations e, and *e2.* (b) Typical flat tabular coal deposit with different overburden formations e_1 and e_2.

$$SR_{max} = \frac{v}{w} = \frac{(e_1 b_1 l_1 + e_2 b_2 l_2)/27}{t/TF} \qquad (5.10)$$

If the bank angle is constant in different overburdens, then $b_1=b_2$. After calculating a numerical value for l_2 by trigonometry, the equation can be solved for l_1, the only unknown parameter. Then the inclined pit slope $l = l_1 + l_2$ and values of h,d, and m can be found by Eqs. 5.7, 5.8, and 5.9, modified trigonometrically if the surface slopes. One caution: Care should be exercised, when calculating the value of b by Eq. 5.5, always to measure the angles n and (3 from the horizontal. Otherwise, an error in b will result when the surface slopes, as in Figure 5.10b.

If there are two variables in the deposit geometry (i.e.. both t and / vary), then Eq. 5.6 must be solved by trial and error. A graphical solution is quicker than an algebraic one, unless iterations can be carried out by computer.

Relationship for Overall Stripping Ratio

Relations can be developed to calculate the overall stripping ratio on a cross section or for an entire deposit. We will most often have need to determine SR_0, for a cross section where we have already calculated SR_{max}, and located the pit limit. Referring for the general case to Figure 5.9, the total volume of overburden V in yd^3 (m^3) is found trigonometrically:

$$V = \frac{1 \times \frac{1}{2} \times h \times d}{27} \qquad (5.11)$$

Similarly, the total weight of ore *W* in tons (tonnes) is

$$W = \frac{1 \times m \times t}{\text{TF}} \qquad (5.12)$$

Therefore, *the resulting* expression for overall stripping ratio in yd3/ton (m^3/tonne) *is*

$$\text{SR}_o = \frac{V}{W} = \frac{\frac{1}{2}hd/27}{mt/\text{TF}} \qquad (5.13)$$

An error results in ignoring the area *A* of the small tip of ore at the outcrop, which may be sizable if the deposit is thick (Fig. 5.9a). To consider A, use *the deposit* thickness *t,* find the base of the triangle *n* by trigonometry, and «hen calculate *A* as the area of the triangle. The value of *A x l* becomes a volume, which is applied as a correction to W in Eqs. 5.12 and 5.13. Also. |.q 5 .11 requires modification if the surface slopes (Fig. 5.10b) or a berm is employed d ig. 5.9d).

5.5 SPECIAL TOPIC: CALCULATION OF STRIPPING RATIOS AND PIT LIMITS

Using the equations developed in Sections 5.2 and 5.4, we can now determine mathematically the maximum and overall stripping ratios and the pit limit for an ore, coal, or stone deposit that is to be developed and exploited by surface mining.

Example 5.1. The following data are given for a mineral deposit occurring under conditions similar to those in Figure 5.9:

Value of ore = $4.80/ton (S5.29/tonne)
Costs (excluding stripping) = $3.30/ton ($3.64/tonne)
Stripping cost (for overburden of *e* = 1) = 50.20/yd^1 ($0.26/$m^J$)
Berm *a* = 0 ft (0 m)
Dip of deposit *a = 20°*
Pit slope ft = 60°
Deposit thickness *t* = 50 ft (15.2 m)
Equivalent yardage *e* = 2
Tonnage factor TF = 15 ft^3/ton (0.47 m^3 /tonne)

(a) Calculate SR_{max} (b) locate the pit limit for the deposit in terms of *h;* and (c) calculate SR_0.
Solution. (a) Use Eqs. 5.1, 5.5, 5.6, and 5.7 to find SR_{max} and the pit limit:

Stripping allowance (for 0 profit) = value – cost = 4.80 - 3.30 = S1.50/ton ($1.67/tonne)

$$SR_{max} = \frac{S1{,}50/ton}{S0.20/yd^3} = 7.5\frac{yd^3}{ton}\ (6.3\ m^3/tonne)$$

(b)

$$b = \cos(90° - 20 - 60) = \cos 10° = 0.9848$$

$$l = \frac{27(50)(7.5)}{(2)(0.9848)(15)} = 343ft$$

$$h = (343)\ (\sin 60°) = (343)(0.8660) = 297\ \text{ft}\ (90.5\ \text{m})$$

(c) Use Eqs. 5.8, 5.9, and 5.13 to find SR_0:

$$d = 0 + \frac{297}{tan20°} + \frac{297}{tan60°} = 816 + 171 = 987ft$$

$$m = \frac{297}{sin20°} = \frac{297}{0.3420} = 868ft$$

$$n = \frac{50}{tan20°} = 3427ft^2$$

$$SR_0 = \frac{\frac{1}{2}(297)(987)/27}{[3427 + (868)(50)]/15} = 1.739,$$

say 1.7 yd^3/ton (1.5m^3/tonne)

If a berm had been used, there would be no change in SR_{max},* or h, but SR_0, would increase.

PROBLEMS

5.1 Calculate (he maximum allowable stripping ratio, locate the pit limit (specify *h*= vertical depth to the deposit and *d* = horizontal distance along the surface from the outcrop), and calculate the overall stripping ratio (actual yd^3,or m^3**,**of overburden to tons, or tonnes, of ore) in mining the ore body shown in Figure 5.11 by surface methods. Allow a 30-ft (9, l-m) berm and pit slopes of 1:1. Cost of excavating 1 equivalent yard of overburden is $0.35 (S0.46/$m^3$): the stripping allowance is $2.10/ton ($2.31/tonne) of ore uncovered; and the ore tonnage factor is 16 ft^3/ton (0.50 m^3/tonne), Sketch the pit limit on a diagram, drawn to scale.

5.2a. Calculate the maximum stripping ratio and locate and sketch the economic pit limit, in terms of inclined distance along the surface and vertical depth of overburden, for the open pit shown in Figure 5.12, given the following conditions:

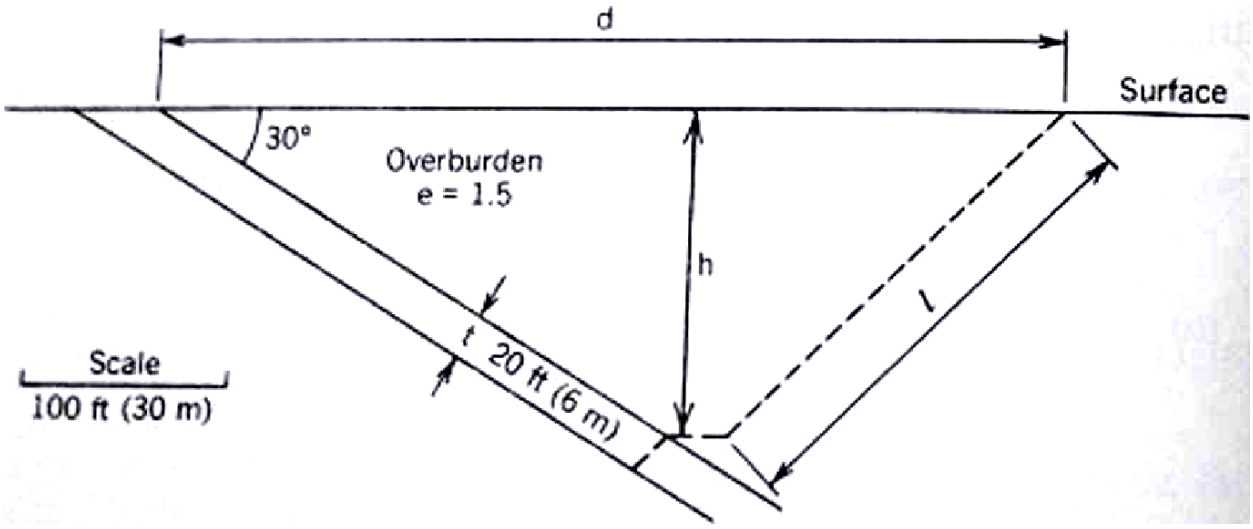


Figure 5.11. Cross section of surface mine in Problem 5.1 .

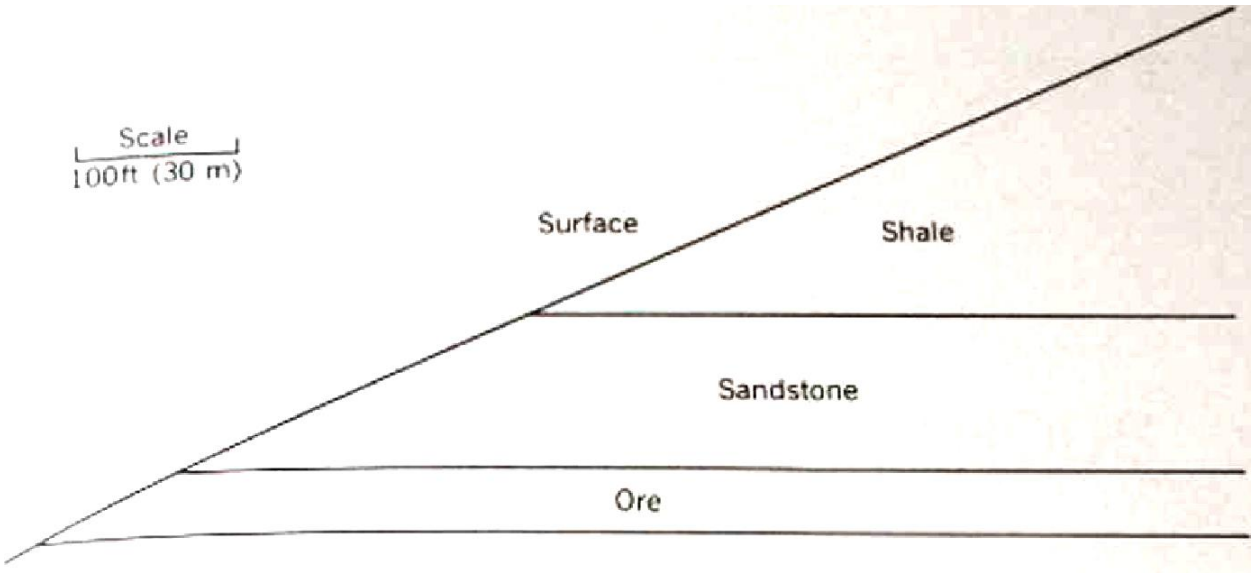


Figure 5.12. Cross section of surface coal mine in Problem 5.2.

Stripping allowance	S1.50/ton (S1.65/tonne) ore
Stripping cost (e = 1)	\$0.25/yd^3 (S0.33/m^3) overburden
Pit slope	0.466: 1
Berm, at ore	30 ft (9.1 m)
Surface inclination	25°
Dip of bed	0°
Ore thickness	40 ft (12.2 m)
Sandstone thickness	100 ft (30.5 m)
Tonnage factor	12 ft^3/ton (0.37 m3/tonne)
Equivalent yards, overburden	
Shale	1.5
Sandstone	2.5

b. Calculate the overall stripping ratio for the pit cross section.

5.3 Locate and sketch the pit limit (expressed as the vertical depth of overburden at the maximum stripping ratio) for the coal deposit shown in Figure 5.13, disregarding a berm. The following conditions arc given:

Value of coal	$24/ton ($26/tonne)
Production cost	$12/ton ($13/tonne)
Stripping cost	S0.6Q/yd^J (SO.78/m^5)
Soil thickness	80 fl (24.4 m)
Pit slope	2 / 3 : 1
Dip of seam	33.7°
Coal thickness	40 ft (12.2 m)

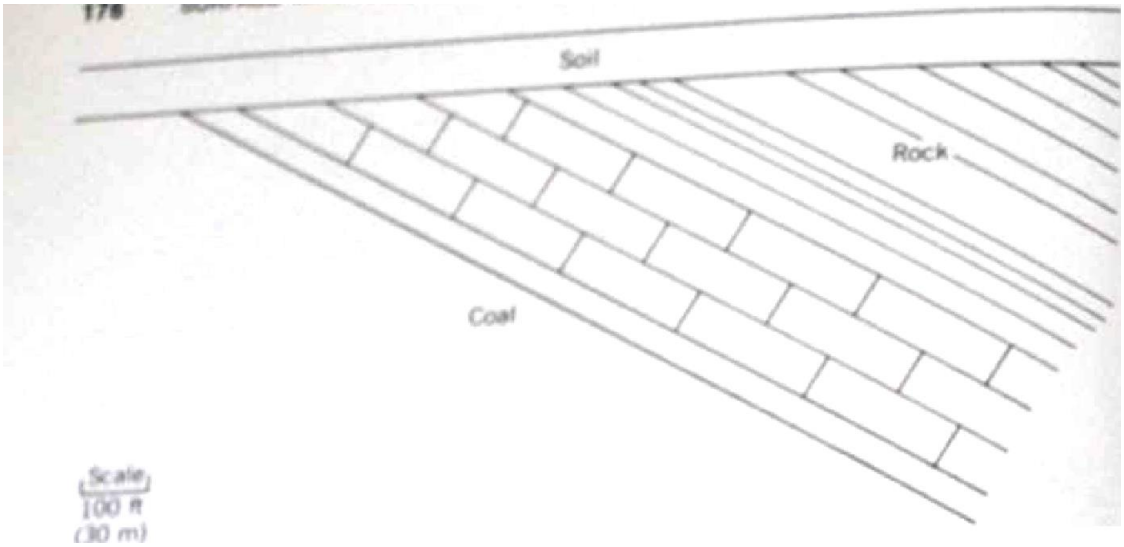


Figure 5.13. Cross section of surface coal mine in Problem 5.3.

Equipment yards, overburden

Soil	0.5
Rock	2.5
Tonnage factor	24.5 ft^3/ton (0.765 m^3/tonne)

Also calculate the maximum and overall ratios.

5.4 Locate the pit limit (*t,l,h,* and *d*) by trial and error for the ore body shown in Figure 5.14 under the following conditions:

Maximum stripping ratio	6 yd^3/ton (5.1 m^3 /tonne)
Pit slope	35°
Tonnage factor	13.5 ft^3/ton (0.421 m^3/tonne)

Give answers to the nearest 5 ft (1.5 m). and sketch the pit limit to scale.

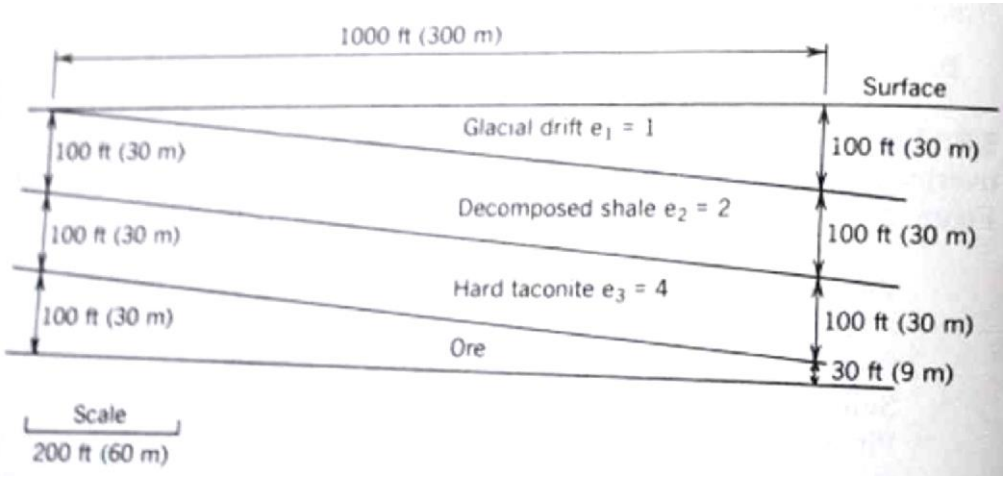


Figure 5.14. Cross section of surface mine in Problem 5.4.

6

SURFACE MINING: MECHANICAL EXTRACTION METHODS

6.1 CLASSIFICATION OF METHODS

Exploitation in which mining of ore, coal, or stone is carried on at the surface with essentially no exposure of miners underground is referred to as surface mining. While openings may sometimes be placed below the surface, and limited underground development work is occasionally required, this t\pe of mining is essentially surface-based.

We turn our attention first to the *mechanical extraction* class of surface methods, which employs mechanical processes in a dry environment to free minerals from the earth. Referring to Table 3.1, we observe that four methods comprise (his class:

1. Open pit mining
2. Quarrying (of dimension stone)
3. Open cast mining
4. Auger mining

These methods are responsible for over 90% of surface production and the bulk of the nation's total mineral output of coal, ore, and stone. On the basis of tonnage and application, two alone—open pit mining and open cast mining—rank as the most important surface methods and among the seven most important of all mining methods.

Broadly applicable, the open pit and open cast methods employ a conventional mining cycle of operations to extract mineral: Rock breakage is usually accomplished by drilling and blasting, followed by the materials-handling operations of excavation and haulage. Quarrying and augering are specialized and less frequently used methods where breakage is achieved by alternative means and explosives are essentially dispensed with. Mechanical methods are also distinguished from so-called aqueous methods, where extraction depends upon hydraulic action or solution attack.

This chapter is devoted to a discussion of the four surface mining methods that comprise the mechanical extraction class . (Note : The student should prepare a sheet on each method for his or her Mining Methods Notebook --- see Section 3.9).

6.2 OPEN PIT MINING

In *open pit mining,* any overburden is stripped and transported to a disposal area to uncover the mineral deposit. Both stripping and mining are conducted from one or a sequence of benches. A thick deposit, typical of metallic ores, requires many benches and resembles a (roughly) circular pyramid, inverted in the earth, each successive bench being cut to a smaller radius because of the slope imposed by safety considerations (Figs. 6.1, 5.4). A single bench may suffice if the deposit and overburden are relatively thin (50 – 150 ft, or 15 – 45 m), typical of some western U.S. coal and nonmetallic ores. (Fig. 6.2).

Provision of multiple benches for mining also ensures that enough length of face *is exposed to* allow sustained, uninterrupted production (see Section 5 I » After advanced stripping uncovers the deposit, stripping and mining are coordinated so that ore revenues will reimburse waste costs, while at the чаше time *long-range* objectives regarding *pit* limits arc met (Pfleider, 1973a).

Individual benches arc designed to accommodate the materials-handling equipment utilized. The height is limited by the reach of the excavator; a power shovel can trim a higher bank than a front end loader or hydraulic excavator. The width must be sufficient to contain most of the fly rock from a bench blast and provide enough maneuvering room for excavator and haulage units. The slope is the maximum dictated by rock or soil mechanics concerns (see Section 6-7). Common practice is as follows, assuming consolidated rock (Pfleider, 1973a):

Mineral	Bench Dimensions		
	Height, ft(m)	Width, ft(m)	Slope, deg
Copper	40-60 (12-18)	80-125 (24-38)	50-60
Iron	ЭО-45 (9-14)	60-100 (18-30)	60-70
Nonmetallics	40-100 (12-30)	60-150 (18-45)	50-60
Coat (western U.S.)	50-75 (15-23)	50-100 (15-30)	60-70



Figure 6.1. Open pit mine with multiple benches. (By permission from Voest-Alpine Inti.Corp.. San Antonio, TX.)

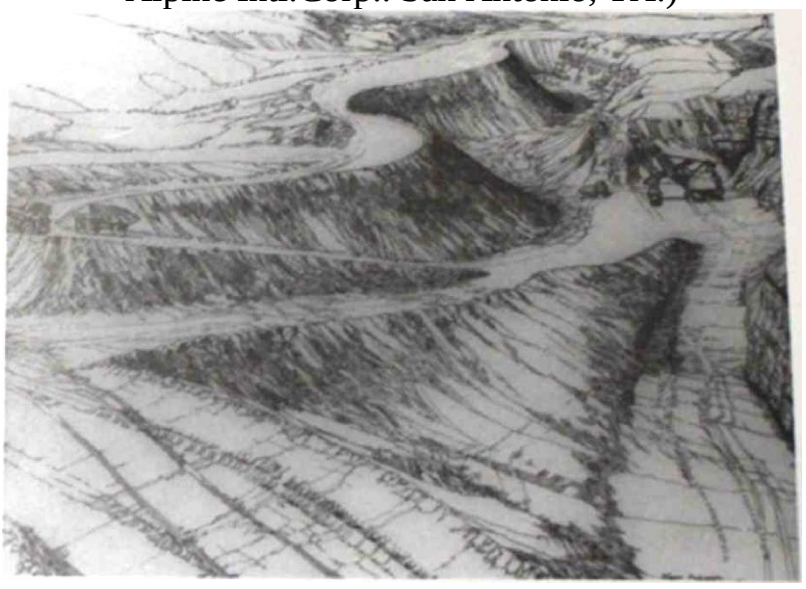


Figure 6.2. Open pit mine with single bench. (After Anon . 1979c Illustration by Frank Kjlzcak By permission from Skelly and Loy. Harrisburg, PA.)

Open pit mining is a large – scale method in terms of production rate, responsible for 60 % of all surface output. It permits the utilization of highly mechanized, mass production equipment that is capital intensive but labor conserving (Martin et al., 1982)

Sequence of Development

The elaborate procedure employed to lay out a surface mine has been detailed in Chapter 5. Those articles having special applicability to open pit mining are Section 5.1. (advanced stripping), 5.2 (stripping ratio), 5.3 (equipment selection), and 5.4 and 5.5 (location of pit limits). They should be reviewed now.

Open pit mining by its very nature normally involves the haulage of moderate to large amounts of waste and ore out of the pit for relatively long distances at steep grades. (For comparative costs of different haulage methods review Section 5.3.) These requirements affect the layout of the pit, the selection of equipment, and the required production rate. Because ore grades are typically low, equipment productivity must be high and stripping ratio held to modest levels (usually 1-5 yd^3 /ton, or 0.8-4 m^3/tonne). Thus depth limits are intermediate (generally <1000 ft, or <300 m) (Crawford and Hustrulid, 1979: Atkinson, 1983). Major steps in development are as follows. After the land is cleared and grubbed, the surface plant is located and constructed as described in Sections 3.2 and 3.4. Particularly important in the development of an open pit mine is the location of waste dumps and, if reclamation of the surface is required, topsoil stockpiles (Section 5.1). Ore dump, storage, processing, and transport facilities are located in proximity to the proposed final pit outline but with reasonable outside access (Section 5.1). Equipment is selected and acquired as needed (Sections 4.7. 5.3). Advanced stripping of overburden (hen commences, to permit the exploitation (ore production) stage to proceed on schedule (Section 5.1); stripping and mining then continue in a carefully meshed fashion, in keeping with short-range and long-range mining plans. Because of the high degree o f mechanization and the mobility required, layout and maintenance of haul roads is particularly important in open pits, providing access to loading benches, ore-handling facilities, waste dumps, and service areas.

Establishment of the first bench and each succeeding bench in waste or ore is a critical operation. The initial entry is referred to as the box cut (or drop cut), a wedge-shaped prism of rock which must be removed to establish a new bench face. Drillholes are placed in parallel rows in descending order of depth so that, when blasted, a ramp of negotiable grade is formed from the upper to the lower bench. Explosives consumption is higher than normal because only one free face exists; materials handling is impeded because of the grade and limited maneuverability (Anon., 1976a; Anon., 1979c).

Cycle of Operations

Production and auxiliary operations and stripping and mining cycles of operations arc examined separately.

Stripped Overburden. Stripping applies both to removing overburden external to 'he mineral deposit and removing waste within the confines of the open pit- The nature of the overburden or waste determines the cycle of operations: If unconsolidated (soil or broken rock), then breakage is not required: if consolidated (rock in place), then breakage is required (see Section 6.ft). Materials-handling equipment is then selected to satisfy the operating conditions, assuming that the waste must be conveyed some distance for disposal and cannot be cast in the pit or on an adjacent spoil bank (Atkinson, 1У83: Sassos, 1984a: Anon., 1985c). Systems of equipment for both stripping and mining were compared in Table 5.2.
The stripping cycle of operations and equipment commonly utilized consists of the following:

Drilling: auger (weak rock), roller bit (average rock), percussion (very hard rock)
Blasting: ANFO or slurry (***alternative:*** rip, if soil or weak rock); loading by machine (bulk) or hand (bag), firing by electricity or detonating cord
Excavation: power shovel, front-end loader, dozer, scraper (soil), dragline, bucket wheel (soil)
Haulage: truck, belt conveyor, dozer, scraper (soil)

Mining Ore, Coal, or Stone. In open pit stripping and mining, equipment and cycles of operations can often be very similar or even identical the determining factor is the difference or similarity of the ore and waste. If they are very much alike, then it is to the mine operator’s advantage to employ the same equipment and the same cycle. The reason is that equipment can be interchanged when there are breakdowns or unexpected production demands.
The mining cycle of operations and equipment usually consists of the following:

Drilling: roller bit (average rock), percussion (hard rock), jet piercing (hard siliceous rock) ***Blasting:*** ANFO or slurry ***(alternative:*** rip, if coal or weak rock); loading and firing similar to stripping
Excavation: power shovel, front-end loader, dragline, scraper (clay) ***Haulage:*** truck, belt conveyor, rail
Hoisting (very steep pits): **high-angle conveyor, skip hoist, hydraulic** conveyor

For a typical materials – handling combination, see the loader and truck in Figure 6.1.

Auxiliary Operations. Here operations are very similar whether stripping overburden or mining ore (Section 4.6). referring to Table 4.11, they usually consist of most of the following during the exploitation stages:

1. *Health and safety:* dust control (loading benches, haul roads, waste dumps), noise abatement, prevention of spontaneous combustion (if subbituminous coal or lignite)
2. *Environmental control:* air and water protection, sold waste management
3. *Ground control:* slope stability (soil or rock), erosion control (soil)
4. *Power supply and distribution: electrical* substation
5. *Water and flood control: pumping, drainage*
6. Waste *disposal:* storage, dumping
7. *Materiel supply :* storage, delivery of supplies
8. *Maintenance and repair: shop* facilities
9. *Lighting.: portable Hoods (for* night operation)
10. *Communications, dispatching: radio,* telephone (computer assisted) *(Sassos,* 1984b)
11. *Construction:* haul roads
12. *Personnel transport: trucks, buses*

Preceding, during, and following mining, reclamation activities are also performed as required.

Conditions

The natural, spatial, and geologic conditions which arc associated with or optimal for open *pit mining* arc the following (Pfleider, 1968, 1973a; Anon., 1979c; Crawford *and Hustrulid,* 1979):

1. *Ore strength:* any
2. *Rock strength:* any
3. *Deposit shape:* any. but preferably lecticular or tabular
4. *Deposit dip:* any. preferably horizontal or low dip
5. *Deposit size:* large, thick
6. *Ore grade:* can be very low
7. *Ore uniformity:* uniform, or variable by horizons (or sort or blend)
8. *Depth:* shallow to intermediate (technological limit of equipment economic *limit* of strip ratio)

Features

Features peculiar to open pit mining are best summarized as advantages and disadvantages (Pfleider, 1968, 1973a; Crawford and Hustrulid, 1979; Martin et al.. 1982).

Advantages

1. High productivity, characteristic of bulk methods which are highly mechanized and labor conserving (U .S. average for copper and iron mining 100-400 tons, or 90-360 tonnes, per employee - shift, including both ore and waste 2. Lowest cost (along with open cast mining) of broadly used methods, reflecting high productivity (relative cost 10%)
3. High production rate (essentially unlimited, although small surface mines also possible)
4. Early production, development can be programmed to permit early start-up
5. Low labor requirement; can be unskilled except key operators (e.g., drill, shovel)
6. Relatively flexible; can vary output if demand changes
7. Suitable for large equipment; permits high productivity
8. Fairly low rock-breakage cost; superior to underground mining where bench faces less easily maintained
9. Simple development and access; minimal openings required, although advanced stripping may be considerable
10. Little if any bank support required; proper design and maintenance of benches can provide stability
11. Good recovery (approaches 100%, except at pit limits); moderate to low dilution
12. Good health and safety; no underground hazards

Disadvantages

1. Limited by depth «1000 ft, or <300 m); technological limit imposed by equipment; deposit beyond pit limits must be mined underground or left in place
2. Limited by stripping ratio (range 1-5 yd^3/ton, or 0.8-4m^3/tonne); economics imposes limits

3. High capital investment associated with large equipment
4. Surface damaged, may require reclamation, an expense added to production cost
5. Requires large deposit to realize lowest cost, unless very-high-grade
6. Weather detrimental, can impede operations but rarely prohibits
7.Slope stability must be maintained; proper design and maintenance of benches plus good drainage essential

8.Must provide waste disposal; provision of dump area and proper siting necessary

Applications

Open pit mining is widely used for aluminum (outside U.S. – see White et al.,1983), bauxite (AR), coal (WY), copper (AZ, outside U.S.-see Sassos, 1983a, 1983b; Chenier et al., 1985), gold (NV-see Jackson, 1983;Argall,1986), iron (MN, outside U.S. – see White et al.,1983; Alport and Caines, 1984; Dayton and Sasson, 1985), uranium (NM) , and nearly all nonmetallics (Hartman,1969; Atkinson, 1983; Herod, 1984).

Case Study. Bingham Canyon copper mine, Kennecott Corporation, Salt Lake City, UT.

History: opened 1906, first surface copper mine, largest open pit excavation in world, undergoing major renovation in 1980s

Ore: chalcopyrite disseminated in porphyry

Grade:0.58%

Production: 100,000tons/day (90,000 tonnes/day) ore, 350,000 tons/day (320,000 tonnes/day) waste

Overall stripping ratio: 3.4 yd^3/ton (2.9 m^3/tone)

Commodity: (principal) 170,000tonnes/yr (155,000 tonnes/yr) Cu, 14% of U. S. production; (by-products) Mo, Au, Ag, and other metals

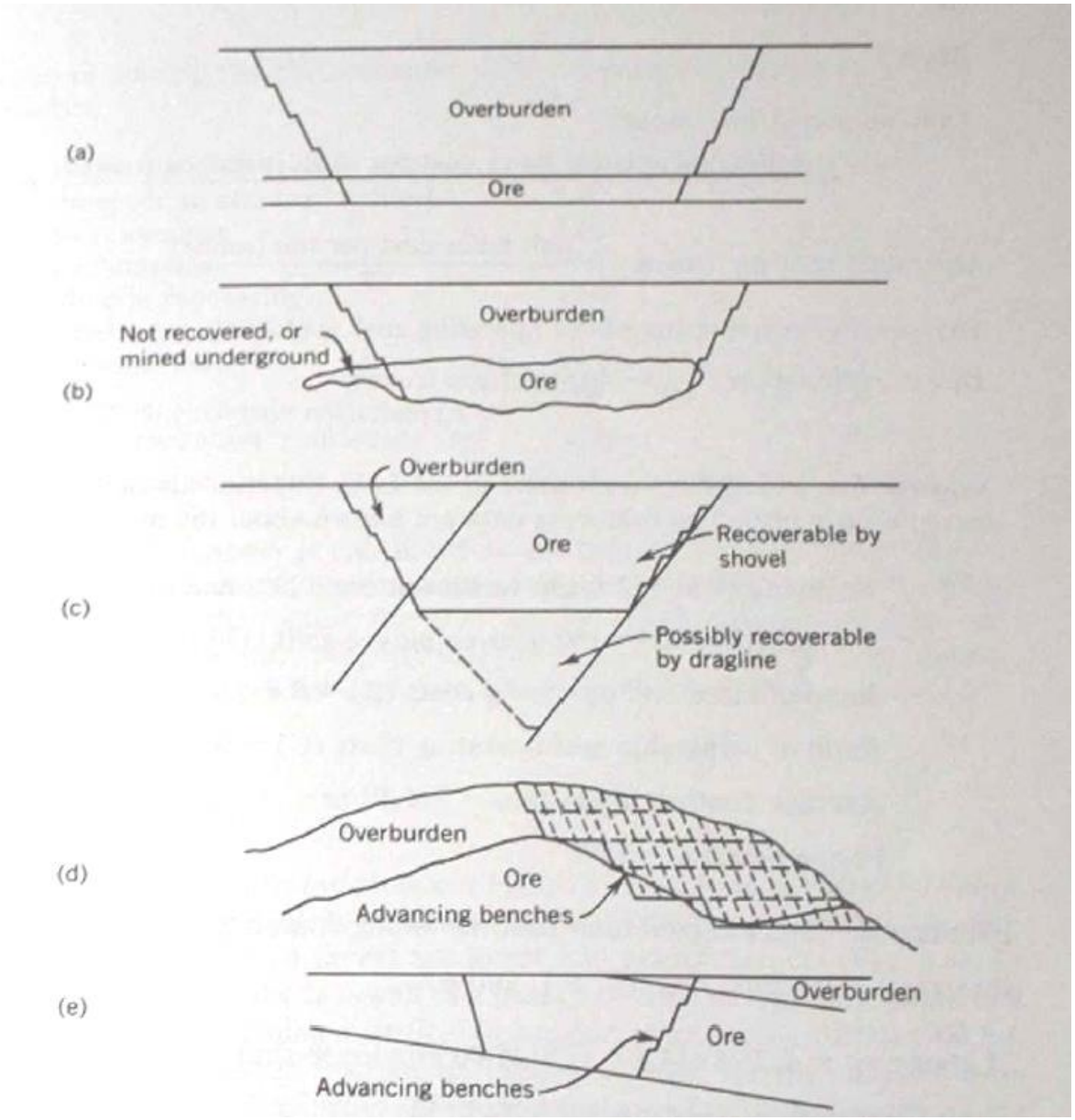
Variations

Depending on natural, geologic, and technological factors, several variations of open pit mining are possible *(Fig.* 6.3). The differences are mainly in pit design and equipment; the basic sequence of development and cycle of operations is very similar.

Cost Estimation

In taking up our first mining method, it is useful to learn an approximate *procedure for* cost estimation that is applicable to many exploitation methods. An approximation suffices for now, since it is time-consuming and *difficult to* make a detailed cost estimate, and actual costs arc almost impossible to obtain. Later, in Section 8.7, we will introduce a more exact method.

Our *approach relies* on certain empirical relationships that prevail in high



mechanized surface mining operations, which may be modified to

Figure 6.3. Variations of open pit mining, (a) Flat-lying seam or bed, flat terrain. *Example:* iron, taconite. (b) Massive deposit, flat terrain. *Example:* iron, (c) Pitching seam or bed, flat terrain. *Example:* anthracite, (d) Massive deposit, high relief. *Example:* copper, (e) Thickbedded deposits, little overburden. *Exam ple:* nonmetallics, western U.S. coal.

apply to mechanized underground mining also (Pfleider, 1973a). The cost terminology is that employed in Section 3.8.

Let A = average productivity in tons (tonnes)/employee-shift

B = estimated ratio of labor cost to operating cost

C = estimated ratio of ownership cost to operating cost

S R_0 = overall stripping ratio in tons/ton (tonnes/tonne); convert from Yd^3/ton (m^3/tonne).

unit *labor cost* per ton (tonne)

Then

Unit labor cost/ton (tonne)
=estimated average labor cost per shift, including fringes/A (6.1)

Operating cost/ton (tonne) =unit labor cost per ton (tonne)/B (6.2)

Exploitation cost/ton (tonne) = operating cost x (1 + C) (6.3)

Direct exploitation + stripping cost /ton (tonne)=exploitation cost*(1+SR_0) (6.4)

Example 6.1. A large *open pit mine in the* Lake Superior district produces aconite *(iron* ore). *The following data* are known about the mine:
Stripping ratio = 1.0 ton waste/ton ore (0.9 tonne/tonne)
Productivity (A) = /50 tons/employee-shift (136 tonne)
Ratio *of labor and* operating *costs (B)* = 0.5
Ratio of ownership and operating costs *(C) = 0.2*
Average *contract wage* rate = $14.50/hr
Fringe benefits = 25%

Estimate the direct exploitation cost, including stripping.
Solution . Use Eqs. 6.1, 6.2. 6.3, and 6.4:
Labor cost = 1.25(8)(I4.50) = $145.00/employee-shift
Unit labor cost = labor *cost/A = 145/150*
= $0.97/ton ($ 1.07/tonne)
Operating cost = unit labor cost/0 = 0.97/0.5=
= *$1.94/ton ($2*. 14/tonne)
Exploitation cost = operating cost x *(1 + C)* = 1.94(1 +0.2)
= $2.33/ton ($2.57/tonne)
Direct *exploitation* + stripping cost = exploitation cost x (1 + SR_0)
= 2.33(1 + 1.0)
= *$4.66/ton ($5.14/tonne)*
In determining the overall production cosl, to the direct exploitation cost (including stripping) must be added the costs of (1) indirect mining expense; (2) prospecting, exploration, and development prior to mining; and (3) reclamation, mineral processing, transportation, smelting, and refining subsequent to mining.In this example these costs might approximate and cumulate:

1. Indirect mining cost @ 10%	$ 0.47/ton ore	($ 0.52/tonne)
2. Prospecting and exploration	1.00	(1.10)
Development	2.00	(2.20)
3. Reclamation	0.50	(0.55)
Mineral processing	4.00	(4.41)

Smelting and refining	5.00	(5.51)
Transportation	3.00	(3.31)
Total, other costs	$ 15.97	($17.60)
Direct exploitation+stripping cost	4.66	(5.14)
Overall production cost	$ 20.63/ton ore	($22.74/tonne)

Notice that the answer is calculated on the basis of $/ton ($/tonne) of ore,*not* refined product. All intermediate costs must be calculated on the same basis. Also ote that before the profit can be calculated, (1) additional expenses for royalty, tate property and severances taxes, and federal income tax and (2) allowances for epletion and depreciation must be estimated.

6.3 QUARRYING

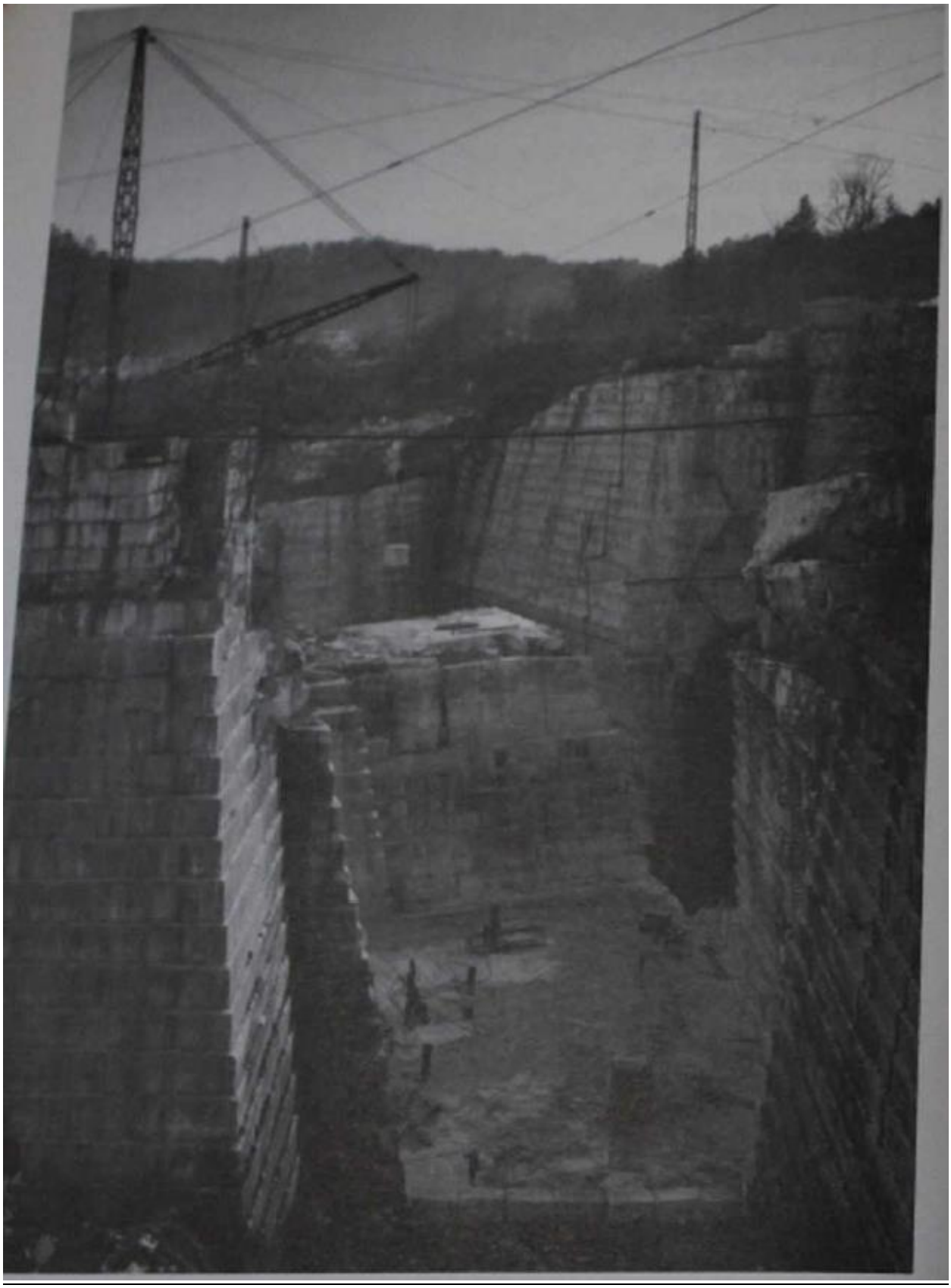
Dimension-stone *quarrying* produces from a deposit prismatic blocks of mineral which are both roughly sized and shaped. Quarries resemble open pits, but hebenches (called faces) are lower and nearly vertical (Fig. 6.4). In overall appearance, the highwall of a quarry is often of imposing height and steepness, some attaining a vertical dimension approaching 1000 ft (300 m).

Quarrying is now an uncommon mining method, for the simple reason that relatively little dimension stone is produced in the United States (or the rest of the w orld) today. Although the term *quarry* is sometimes applied to any surface mine producing a nonmetallic mineral, it is preferable to restrict *quarrying* to denoting a dimension-stone operation and the unique methods associated with it. Thus crushed limestone is produced in an open pit mine, while dimension limestone is produced in a quarry. W e will employ the latter terminology.

The products of all dimension-stone quarries are nonmetallics (Barton, 1968). Common dimension stones are granite, marble, limestone, sandstone, flagstone, and slate— in approximate decreasing order of difficulty to cut. Because of the difficulty and cost associated with cutting stone, quarrying is the most expensive surface mining method and. with square set stoping, the most costly mining method. It is also a highly selective, *small-scale* method, with low productivity.

Total dimension-stone production was 1.340.000 tons in 1983, valued at $153 million, less than 1% of all surface output (Anon., 1984b). Thus its

Figure 6.4. Dimension-stone quarry.(By permission from Georgia Marble Co



average value was $114/ton ($ 126/tonne), far in excess of that of crushed stone ($3.80/ton, or $4.16/tonne), and an indication of relative mining costs b> the two methods (quarrying and open pit).

Sequence of Development

The mineral properties which make a deposit of dimension stone commercial are largely physical rather than chemical. They include its color and appearance. ,competence, uniformity, strength, and freedom from cracks, flaws, discontinuities, and so forth (Singleton, 1980). Prospecting and exploration are therefore carried out in quite a different fashion than the traditional search for ore. Development also is unique.

After the surface is cleared, the processing plant and necessary support facilities are constructed adjacent to the quarry site (Bowles, 1958). Rather than concentrating mineral values, the plant performs finishing operations of sawing and polishing on the blocks of stone. Since blocks are cut roughly to size in the quarry, all mining and processing operations are geared to the handling of individual blocks. Waste is substantial, so that disposal areas must be provided nearby. Any overburden (usually minor) is removed by stripping (conventional or hydraulicking), and the deposit outcrop is cleaned and weathered rock removed by blasting if necessary.

Opening the first face is accomplished by forming a cut across the quarry with (the quarry is laid out appropriately with regard to stone properties, bedding , oints, etc.). The cut is established by cutting or channeling a *key block*, approximately 4 ft (1.2 m) wide, 12 ft (3.6 m) deep, and up to 30 ft (9.1 m) long (Fig. 6.5). Once the block is removed, a *key slot* can be cut across the quarry, which establishes the first cut. Now there are two free faces to break to. and blocks of desired size are readily cut and removed.

When cuts have been taken across the entire width and length of the quarry, then a new working level must be established by removing another key block (Morrison and Russell, 1973; Power, 1975).

Cycle of Operations

Stripping Overburden . Since the overburden associated with a dimension stone deposit is generally minimal, simple stripping methods suffice. If the quarry operators lack specialized equipment, they frequently contract out the stripping. Stripping methods parallel those in open pit mining, the cycle of operations consisting of the following:

Drilling : auger (weak rock), roller bit (average rock), percussion (very hard rock)
Blasting : A N F O (*alternative:* rip, if weak)
E x c a v a tio n : dragline, scraper, or monitor (soil); front-end loader (rock)
H a u la g e : truck, scraper, or cast (by dragline)

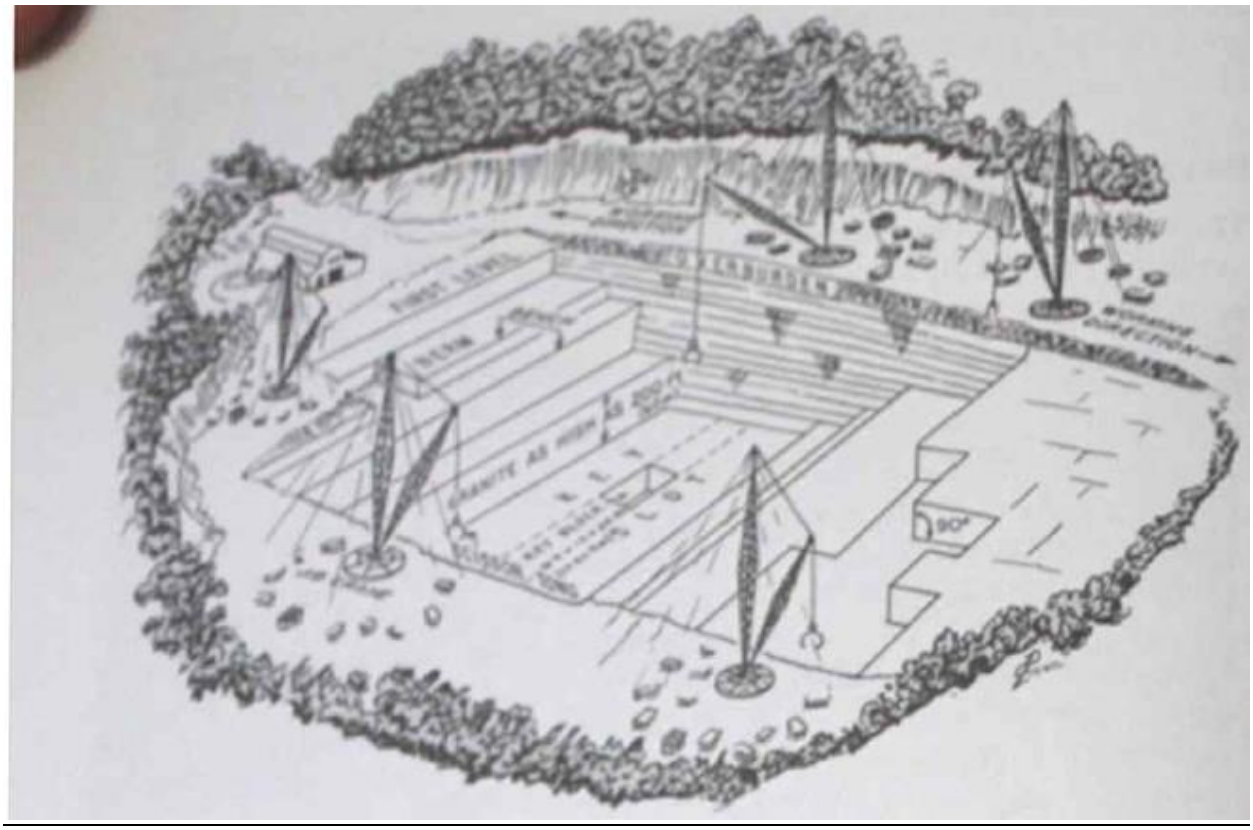


Figure 6.5. Diagramm of operations in dimension-stone quarry. (By permission from West Virgia Geological and Economic Survey, Morgantown)

Quarrying Stone. Because of the highly *specialized* nature of dimension stone quarrying, a customized cycle of unit operations is employed (Spielvogel, 1978). Rock breakage is almost always accomplished without blasting, which tends to shatter and spoil the blocks of stone. Cutting or channeling is *utilized* to *free* the block on three sides, assuming the top and front surfaces are free faces (Fig. 6.6). The sixth (bottom) side of the block is often freed by wedging, with *plug and feathers*, especially *if* the bedding or jointing is horizontal (Fig. 6.7). *If* wedging does not suffice, then the bottom of the block may be freed by cutting. *Hard* stone (e.g.. granite) is cut by a *channeler (*Fig. 6.8), while softer rock is cut by a *saw* (e.g., slate) (Browning et al.. 1965). After being freed, the blocks *of* stone are hoisted from the quarry.

Milling or finishing operations take place in the processing plant (Kostner,1976). Because of the customized nature of quarrying, it is difficult to mechanize the operations beyond the present extent. The cycle of operations consists of the following:

Cutting: rotary, chain, or wire-rope saw (soft stone); percussion, flamejct *or* watcr-jct channclcr (hard stone)

Wedging: drill and broach: wedge, plug, and feathers; light blasting

Excavation!hoisting: crane, derrick, hoist

Haulage: rail, truck, conveyor (chain or belt)

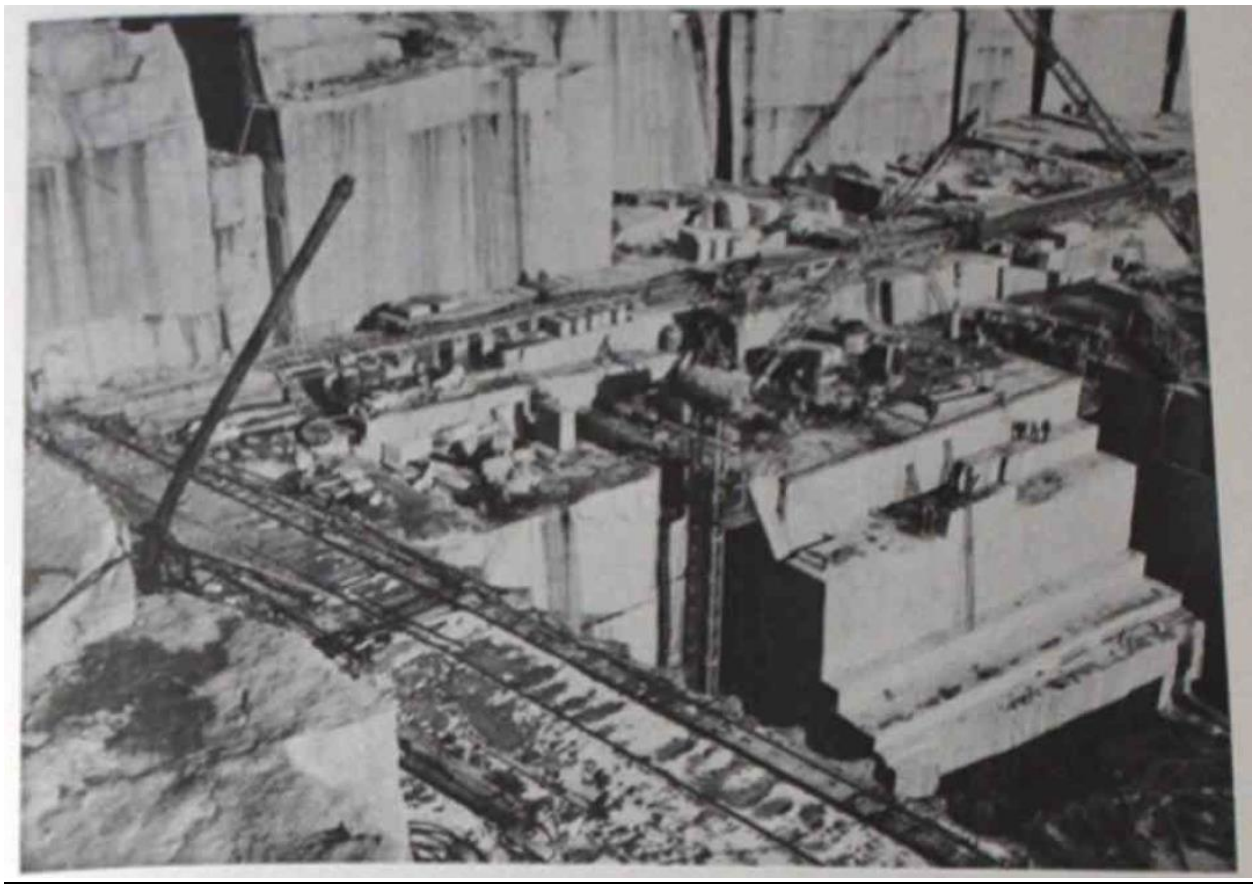


Figure 6.6. Quarrying granite with channeler and wire saw (After Brow ning et al., 1965.
B y permission from Society of M ining Engineers. Inc.. Littleton. CO.)

Auxiliary O p eratio n s. Auxiliary operations in quarrying tend to be simpler but otherwise parallel to those in all surfacc mining (sec “ Open Pit Mining," Seetion 6.2. and Table 4.11). The major ones arc power supply, equipment maintenance, pit drainage, waste disposal, materiel supply, and reclamation.

Conditions

The following is based on Bowles (1958), Barton (1%8), Morrison and Russell (1973) and Singleton (1980):

1. *Ore strength:* any; sound structurally
2. *Rock strength:* any
3. *D eposit shape:* thick-bcddcd or massive; large areal extent
4. *D eposit d ip :* any, if thick
5. *D eposit size:* large, thick
6. *O re g rad e:* high-quality (physical); assay not critical
7. *O re u n ifo rm ity:* uniform
8. *D epth:* shallow to intermediate; even with steep quarry walls, limits created by small area, bank instability, and hoisting constraints



6 *7 Quarrying limestone* with drillholes and *plug and* feathers. (After Pow er. 1975.
fyperm'ssion from Society of *Mining Engineers. Inc.. Littleton. CO.)*
Features
The following is based on Morrison and *Russell* (1973) and Power (1975):
Advantages
1. Low capital investment; *mechanization limited* and not sophisticated
2. Suited to small deposit, but limits scale of operation
3. Easily accessible; hoisting workers, stone, and *supplies may* pose difficulties

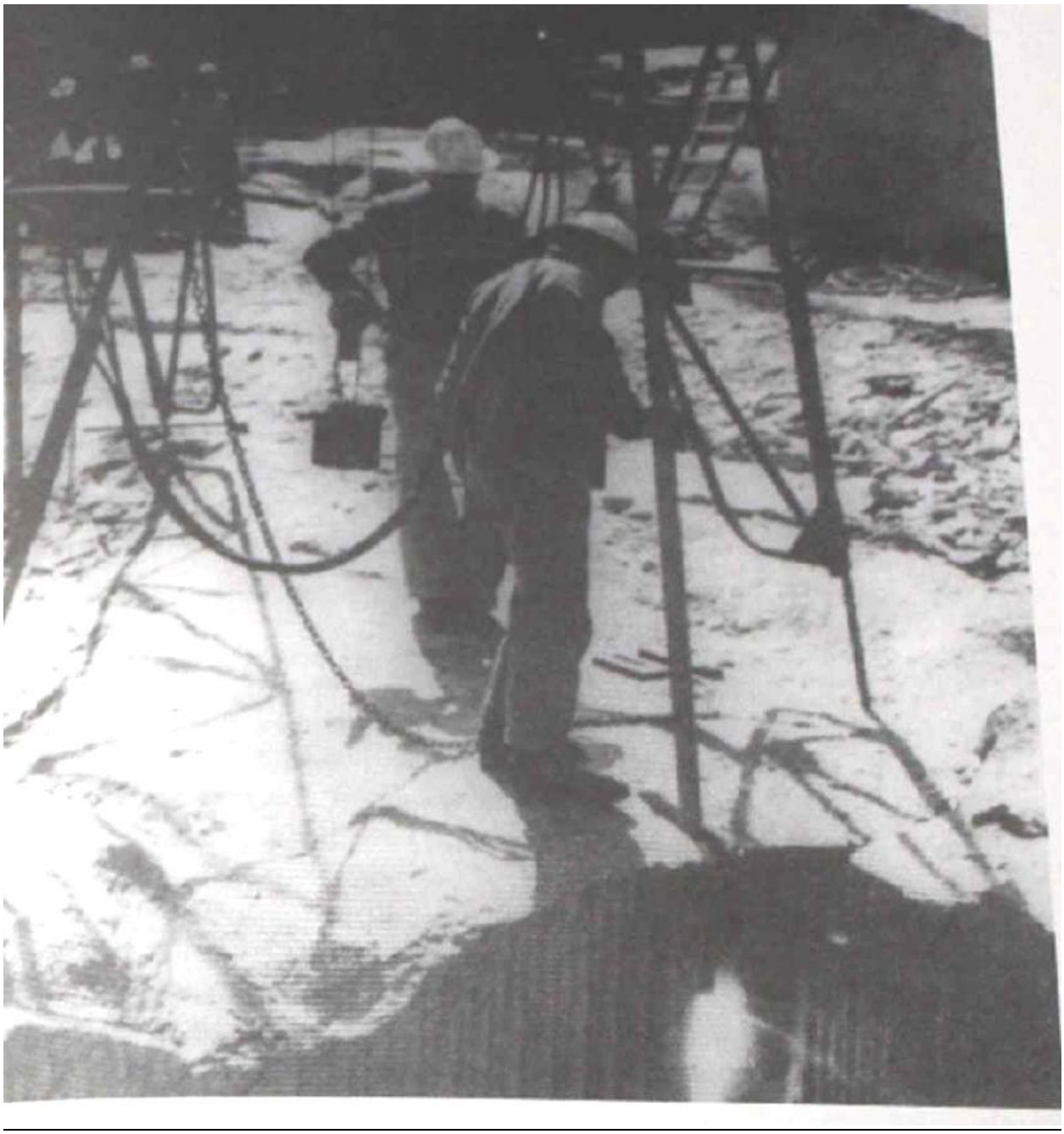


Figure 6.8. Cutting granite with flame-jet channeler. (After Calaman and Rolseth. 1968. By
permission from Society of Mining Engineers. Inc.. Littleton. CO.)

4. Very stable walls and benches, stone selected for its good physical condition; generally no bank support required, working face vertical but low
5. High selectivity: can discard low-quality stone to waste
6. Good health and safety, advantages of surfacc mining

D is a d v a n ta g e s

1. Limited by depth (usually <300 ft, or <90 m; maximum <1000 ft, or <300
2.Very low productivity; highly labor-intensive, unmechanize
3.Highest mining cost resulting from low productivity (relative cost 100%)

4.Low production rate due to nature of method
5.Skilled-labor-intensive; procurement and training of workers difficult
6.Inflexible, nearly impossible to change quaryying plan a depth
7.Unsuited for extensive mechanization due to nature of method
8.Complicated and costly breakage because because of inability to blast
9.Excessive waste (60-90%), much of which must be removed from the quarry
Applications
Dimension-stone quaryying is limited to the production of material used for structural purposes and monuments (Anon., 1984b). Examples are Barre granite (MA), Bedford limestone (IN), and slate (PA)
Case *Study* Tate marble quarry. Georg.a Marble Company. Tate, G A.
History: deposit discovered in late 1800s
Stone: crystalline marble
Grade nearly pure CaCO, (physical impurites detrimental)
Production: 100.000 tons/yr (90.000 tonnes/yr); output measured in ft»
(in')
Overall stripping ratio: negligible
Commodity: dimension stone tor building monuments, etc.
Variations. If a quarry proceeds below the surfacc. it becomes an underground mine (although colloquially it may be called an " underground quarry"). The mining method used is slope and pillar, with bench faces. Rarely can a dimension-stone quarry afford to go underground; already high operating costs become exorbitant.

6.4 OPEN CAST MINING

Open cast mining is a surface exploitation method, used mainly for coal, which resembles open pit mining but differs in one unique respect: The overburden is not transported to waste dumps for disposal but cast or hauled directly into adjacent mined-out panels. Materials handling thus consists of excavation and haulage (= *casting),* generally combined in one unit operation an performed by a single machine. Sometimes overburden is loaded into conveyances and then deposited in mined-out areas; at times, the stripping is done in stages, some by casting and some by haulage (e.g., topsoil is stockpiled for reclamation). It is casting in the pit. however, which makes this method distinctive, achieving for it the highest productivity and (with open pit) the lowest cost of any of the broadly used mining methods.

Like open pit mining, open cast mining is classed as a *large-scale* method and ranks with it as the most popular surface methods. Its importance is evident from Tables 1.4 and 5.1: Open cast mining produces 55% of the coal and 30*c/c* of the mineral which is surface-mined in the United States.

It is not just the replacement of haulage by casting that makes open cast mining attractive. The depositing of *spoil* (overburden) in mincd-out areas means that very little advanced stripping is necessary, that mining activity is concentrated in a relatively small area, and that reclamation can immediately

follow mining. Thus the key to productivity in open cast mining is in the output of the stripping excavator: By utilizing the largest land machines in the world, the number of active faces that has to be provided in the mine is sharply limited. (A corresponding disadvantage is that all production is realized from a single machinc— and major breakdowns and delays can have dire consequences.)

Another advantage of the open cast method is that the cut is open a relatively short time, permitting a much steeper slope to the bank (termed a *h ighw all).* The spoil banks also have a short life span and can be maintained at the natural angle of repose of the material. Typical dimensions in an open cast mine arc height of highwall 100-200 ft (30-60 m), width of panel cut 75-150 ft (23-45 m), slope of highwall 60-75°, and slope of spoil pile 35-50° (Anon., 1976a). Diagrammatic sketches of open cast mining appear in Figure 6.9. showing *(to p)* the main components of the method and (*bottom*) a typical pit layout.

Unlike open pit mining, the same equipment cannot usually be utilized in open cast mining for both stripping overburden and mining coal. Casting overburden requires specialized, boom-type excavators, while mining is carried on with conventional loading and haulage equipment. Further, differences in the overburden (soil or blasted rock) and the mineral mined (usually coal, blasted or not blasted) require different handling equipment.

The emphasis on stripping overburden in open cast mining led for many years to use of a colloquial term for the method, *strip m ining.* A very unfavorable connotation became attached to the name at the time of passage of federal environmental legislation in 1969, and much of the mining industry since has conspicuously avoided its use. *Open cast m ining* is a far more descriptive term and is to be preferred as a generic method name.
The unique, concentrated, and carefully orchestrated operations in stripping and mining a mineral deposit by the open cast method are shown in artist's renderings in Figures 6.10 and 6.11. Both the development sequence and cycle of operations can be followed. Notice the four different stripping techniques. In Figure 6.10, rubber-tired scrapers pushed by a dozer remove topsoil (and replace it during reclamation), while a dragline casts blasted

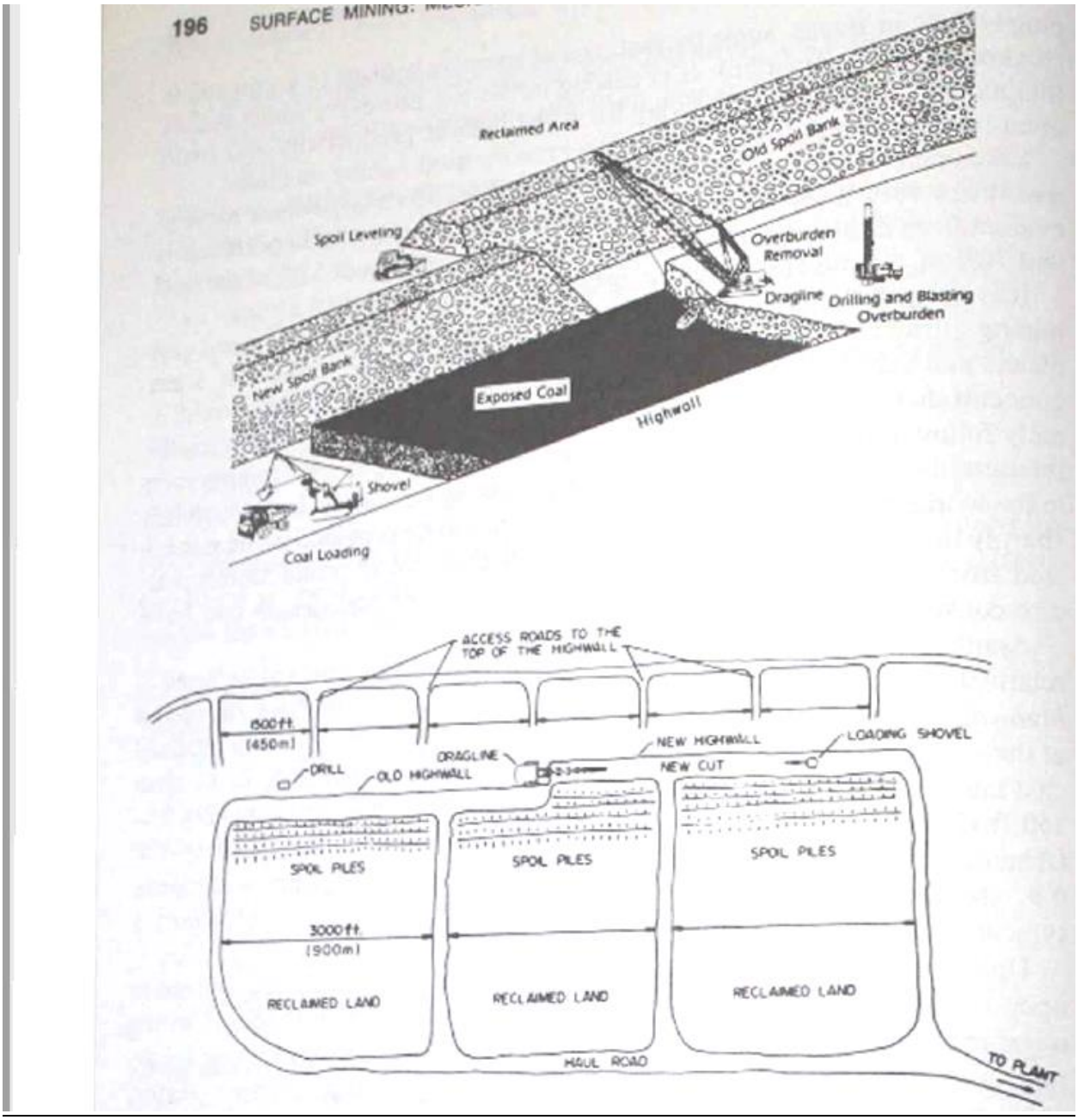


Figure 6.9. Open cast mine in single-seam area mining. *(Top)* Nomenclature and main components of stripping and mining. (After Anon., 1982a.) (*Bottom*) Plan view of pit, showing access roads to highwall and cut. Road interval on the highwall is limited to the maximum length of power cable for the dragline that can be handled readily. (After Anon., 1976a. By permission from Bucyrus-Erie Co.. South Milwaukee. Wl.)

rock (an overburden drill works at left center). In Figure 6.11, a bucket wheel excavator stripping soil works in tandem with a shovel casting blasted rock (a drill works at top left). Mining of coal in both cases takes place without drilling and blasting, loading and haulage being carried out by frontend loader and trucks (Fig. 6.10) and by power shovel and tractor-trailcrs (Fig. 6.11).



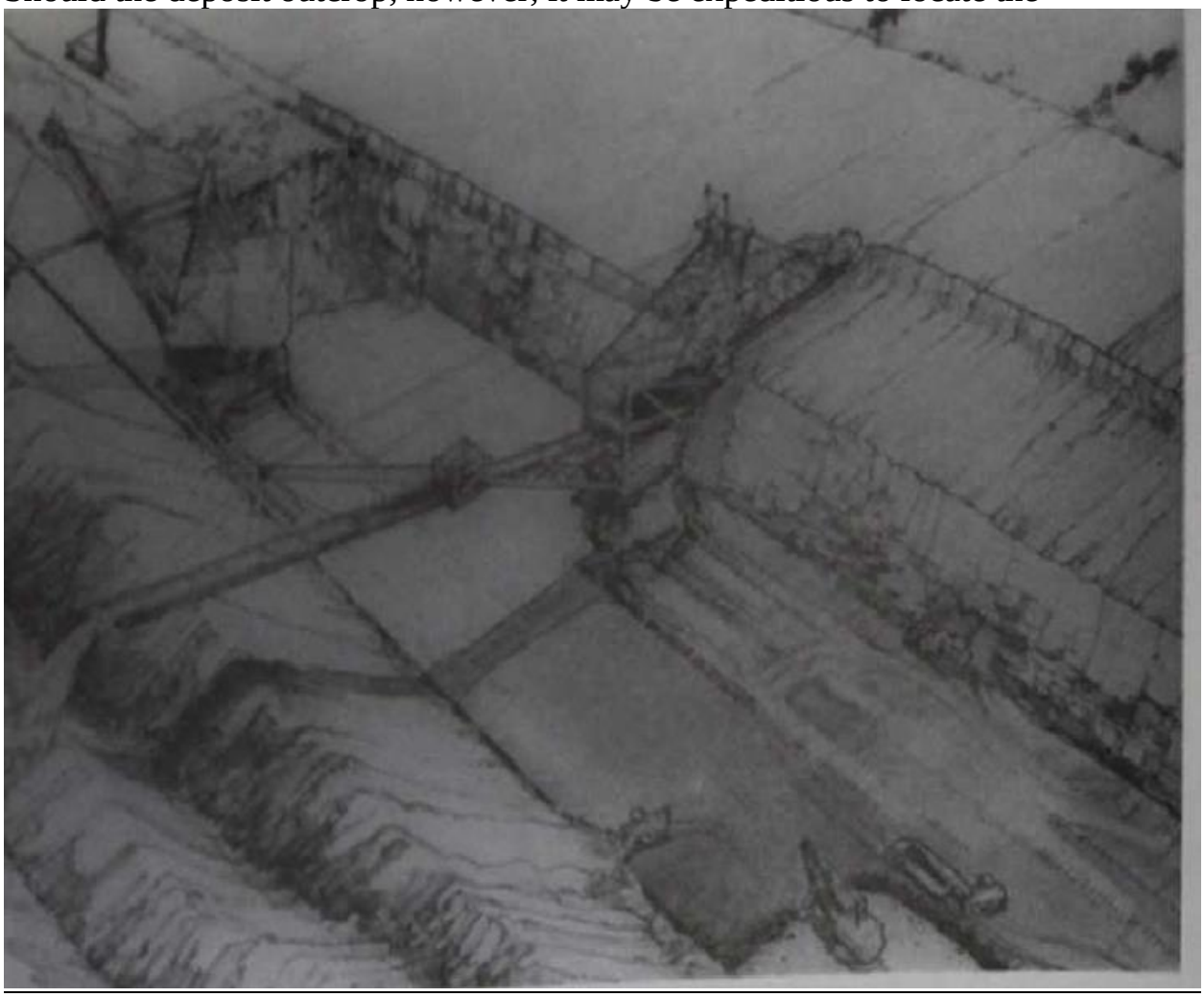
Figure 6.10. Stripping and mining operations in the open cast method. Stripping by rubber-tired scrapers and dragline. Mining by front-end loader and trucks. (After Anon.,
1979c. Illustration by Frank Kulczak. B y permission from Skelly and Loy, Harrisburg. PA.)

Sequence of Development

The factors and sequence of development outlined in general for mine development in Section 3.2 are, of course, applicable to open cast mining. The most important specific steps for this method are (2) confirmation of mining method and plan, (5) writing an environmental assessment and E IS (if required),
(7) locating the surface plant, and (9) selection of stripping and mining equipment. In connection with item 5, provisions of the federal Surface Mining Control and Reclamation Act of 1977 must be carefully adhered to, since open cast mining of coal is specifically covered in that act and reclamation procedures must be approved in advance (Section 5.1).

In developing a large open cast mine in a flat area or one of low relief, if other factors do not govern, the location of the surface plant is chosen at or near the center of the property. A central location ensures minimum haul distances and

ready access to all sectors of the mineral reserve. Even if the topography is rugged, a central location of the plant can usually be arranged.
Should the deposit outcrop, however, it may be expeditious to locate the



6.11. Stripping and mining operation in the open cast method. Stripping by bucketwheel excavator and shovel. Mining by power shovel and tractor-trailers.(After Anon.,1979c.Illustration by Frank Kulczak. By permission from Skillery and Loy, Harrisburg, PA.)

plant adjacent to the overcrop but on barren ground to avoid leaving mineral unmined or moving the plant. The location of surfacc transportation (highway.
rail, or water) and stream drainage may influence siting of the plant.

Both long-range and short-range planning of an open cast mine requires consideration of the factors previously discussed (Section 5.2). Because of the highly repetitive pattern of open cast mining, especially in flat tabular deposits, advanced planning need not be as elaborate as in open pit mining.
In mountainous terrain or with dipping deposits, however, prediction of stripping ratios and establishment of pit limits are every bit as critical. Computer modeling is a useful technique to utilize where ratios and limits must be monitored with care (Section 5.4).

By its very nature, open cast mining centers on the stripping function;

thus the selection of stripping equipment is the paramount decision to be made. Reviewing the system options of Table 5.2 and Section 5.3, for largescale operations one of the three boom-type excavators will undoubtedly be the choice: dragline, shovel, or bucket wheel— and probably in that order.

While other factors comc into play (e.g.. hard digging favors the shovel, deep digging or longer reach favors the dragline, high capacity favors the bucket wheel), the output of the excavator is the key performance parameter to consider (Pfleider. 1973a; Anon., 1976a; Atkinson. 1983). Table 6.1 compares the output of the three machines under similar operating conditions.

Cost studies arc also helpful in the selection process. A comparison of the two most popular machines, dragline and shovel, is shown in Figure 6.12. in which unit cost is plotted against bucket capacity. There is little distinction between the two when the buckets exceed 100 yd* (75 m*), the unit costs becoming asymptotic and nearly equal. (The curves also suggest that the limit in excavator size may now have been reached, since two excavators in the 100-yd\ or 75-mJ, range can produce as cheaply as one of 200-yd\ or l50-m\ capacity, and perhaps produce more because delays are less penalizing.) Another cost comparison is made in Figure 6.13, this for four common

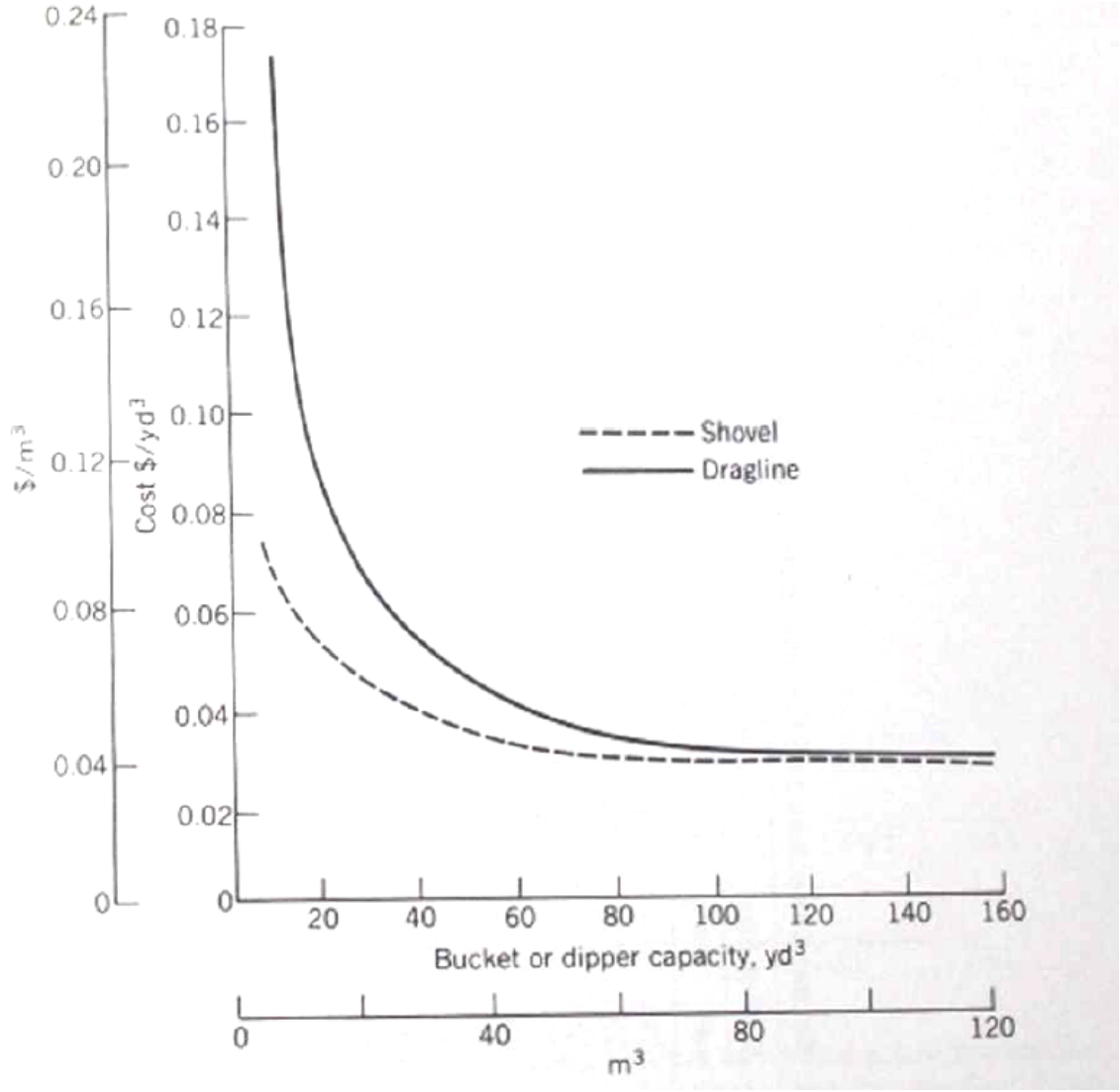


Figure 6.12. Comparison of unit operating costs (per yd^3, or m^3, of material stripped) for various bucket capacities. (After Pfleider, 1973 a. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Table 6.1 Bucket Capacities and Stripping Outputs of Boom-Type Excavators Under Similar Operating Conditions*

Excavator	Bucket Size		Output	
	yd^3	(m^3)	yd^3/hr	(m^3/hr)
Shovel	20	(15)	1000	(760)
	50	(38)	1900	(1500)
	65	(50)	2460	(1880)
	75	(57)	2850	(2180)
	80	(61)	3040	(2320)
	105	(80)	4000	(3060)
	125	(96)	4750	(3630)
	140	(107)	5320	(4070)
	180	(138)	6800	(5200)
Dragline	25	(19)	870	(670)
	30	(23)	1050	(800)
	45	(34)	1550	(1190)
	60	(46)	2060	(1570)
	75	(57)	2600	(1990)
	90	(69)	3080	(2350)
	105	(80)	3600	(2750)
	120	(92)	4200	(3210)
	130	(99)	4500	(3440)
	220	(168)	7500	(5730)
Bucket wheel	0.13	(0.10)	430	(330)
	0.20	(0.15)	1100	(840)
	0.33	(0.25)	1500	(1150)
	1.11	(0.85)	4160	(3180)
	1.77	(1.35)	5500	(4200)
	1.96	(1.50)	5600	(4280)
	5.23	(4.00)	11,000	(8490)

* Stripping soil overburden and casting on spoil pile.

Source: Modified after Pfleider, 1973a.

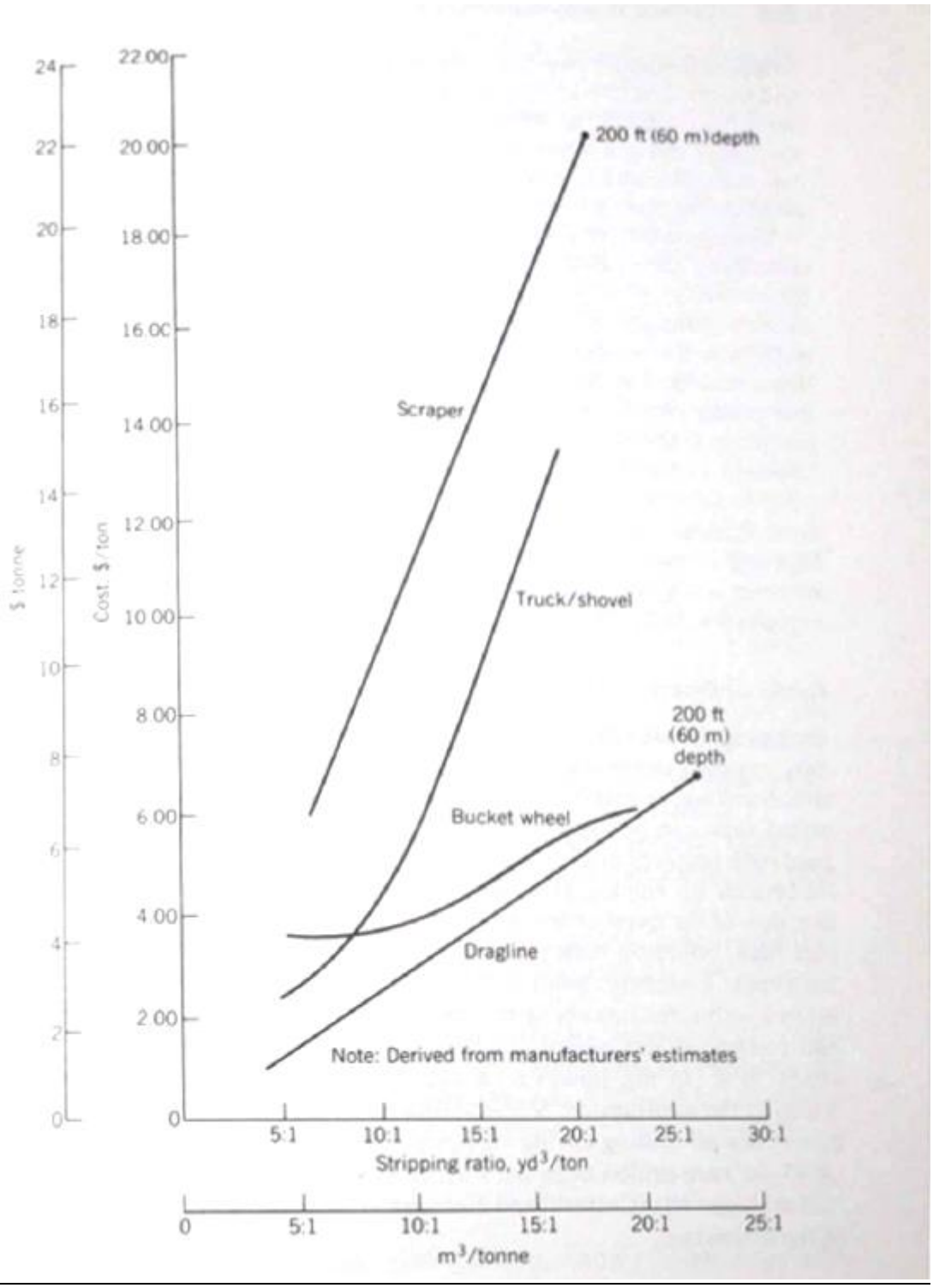


Figure 6.13. Com parison of unit operating costs (per ton of mineral produced) for four excavating system s for varying stripping rations. Conditions: hypothetical Gulf Coast lignite mine, annual production 5 million tons/yr (4.5 million tonnes'yr). (After Kanle and Mosely. 1983 B y permission from Society of Mining Engineers, Inc.. Littleton. CO.)

stripping methods: two involving baulacc (rubber-tired scraper and shovel and truck), and two casting (dragline and bucket whell).Again unit operating costs (in $/ton, or $/tonne) for stripping are plotted, this time against stripping ratio in a hypothical lignite mine. The dragline is most economical, with the bucket whell a

close second. A detailed total and unit cost estimate for open cast mining is provided by Pfleider (1973a).

The major activities in development of an open cast mine have now been described. The sequence of activities is as follows. After clearing the land, the surface plant is located and constructed (Sections 3.2 and 3.4). Because of the importance of reclamation in the open cast method, special attention is given to the location and maintenance of topsoil stockpiles (section 5.1).

Since reclamation must keep pace with mining, environmental and restoration procedures are worked out with care. Coal dumping, storage, processing(if any), and transport facilities are located centrally (Section 5.1). After selection of equipment, initial pit development takes place (Section 4.7, 5.3). As in open pit mining , the first cut, or drop cut,is the most difficult; some haulage of overburden is inevitable, and progress is slow. Since a highwall is maintained for the entire depth of cut in an open cast mine, However, only one drop cut is usually required. Simultaneous development, exploitation in the normal fashion can now proceed.

Cycle of Operations

Stripping Overburden. The cycle of operations during stripping in open cast ining is determined entirely by the nature of the overburden, in materials handling, as just discussed, and also in rock breakage. Soil and decomposed rock can be excavated without prior breakage, well-consolidated to hard rock requires drilling and blasting, and very stiff soil or weak rock maybe broken by ripping prior to excavation. The drilling method is also a function of the overburden material: Auger-bit rotary is used in soil or very soft rock, roller-bit rotary for intermediate rock, and percussion for very hard rock. Formerly, holes drilled with auger bits were placed horizontally in the overburden just above the coal seam; this practice was quicker, easier, and consumed less explosives than vertical holes. When the highwall ex- 50 ft (15 m), however, horizontal drilling ceases to be effective in ng the overburden. Vertical drilling with roller bits is now utilized for cally all drilling during stripping, except that inclined or angled holes 15-30°) are drilled from the surface in very hard rock or deep (>80 ft. or I m> high walls (Cottcnll and Nelmark. 1984). The stripping cycle consists the following:

Drilling ; auger (weak rock), roller bit (average rock), percussion (very hard rock) 7

Blasting: ANFO. AN slurry. AL/ANFO *(alternative:* rip. if soil or decomposed rock) loading by machine (bulk) or hand (bag); firing by electricity or detonating cord

*Excavation:*dragline, shovel, bucket wheel (soil), dozer, scraper (soil), explosives casting

Haulage (if multistage stripping or recasting necessary): truck, scraper (soil), conveyor

Mining C o al or Ore. When the mineral is coal or soft ore. mining by the open cast method is much simplified. In the majority of cases, the cycle of operations consists of an abbreviated breakage operation, if any. and materials

handling. If the mineral is hard to break or relatively thick, it may require drilling and blasting. If it is only moderately well consolidated, then ripping may suffice. If it is sufficiently weak, then it may be excavated by loading equipment without prior preparation, especially if equipment with digging capability, such as a power shovel or front-end loader, is used. The excavator is nearly always cyclic, although the cutting-head excavator, a small bucket wheel, has some application in loading coal. If coal, haulage equipment must be highly mobile and flexible and built to handle bulky material. Limited use has been made of hydraulic monitors employing highpressure
jets of water to slurry and transport phosphate ore in open cast mines. The mining cycle consists of the following:
Cleaning: rotary brush equipment sweeps top of coal scam clean (optional)
D rilling: small auger (weak rock. e.g.. coal), small percussion air trac (hard rock, e.g., uranium)
Blasting: AN FO *(alternative:* rip. if well consolidated but blasting unnecessary or undesirable); loading and firing similar to stripping
Excavation: front-end loader, power shovel, cutting-head excavator (continuous), hydraulic monitor (if ore weak and hydraulic transport used)
H aulage: truck, tractor-trailer, bell conveyor, hydraulic conveyor, rail (limited to largest mines, e.g.. German or Australian brown coal)
Auxiliary O peration s. The most important auxiliary operations are reclamation. slope stability, haul-road construction, equipment maintenance, pit drainage, communications, power distribution, and dust control and safety. Reclamation is probably the most critical of these.
Conditions
The following is based on Pfleider (1968. 1973a), Anon. (1976a), and Anon. (1979c):
1. *O re strength:* any
2. *Ro ck strength:* any
3. *Deposit shape:* tabular, bedded
• 4 OPEN CAST MININO 203
1: % £ 1 :Z £ £ fc ^ " ^ toPm^eraie , W c k - *ness*
6. Ore grade: can be very low
l: ^ iripping ratiw’techno,o8icai *limit* on equipment)
| M , * 0 MECHANICAL E ,Г Я Л С Т 0 , — OS
Features
The following is based on Pfleider (.968. -973a,. Anon. (.976a), and Kahlc and Moseley (1983):
1 Hi?hest productivity of any broadly used mining methods; low labor intensity and replacement of haulage with casting results m high outnut (U.S. average for coal 5(Ю-1000 tons, or 450-900 tonnes, per face

cmployec-shift. including both coal and waste; average for clean coal 35 tons, or 31 tonnes, per gross employee-shift)
2. Lowest cost (along with open pit mining) of broadly used methods (large equipment and ability to cast overburden minimize operating cost; relative cost 10%)
3. High production rate, near maximum output of all methods
4. Early production (modest development requirements permit rapid exploitation)
5. Low labor intensity, but skilled labor required for key operators (e.g., drill, dragline, bucket wheel, etc.)
6. Relatively flexible, can vary output by expanding operations, but less easily than in open pit
7. Suitable for large equipment, permits high productivity
8. Low rock-breakage cost; large bench faces efficient, provide multiple free faces for blasting
9. Simple development and access; minimal openings and advanced stripping required
10. Bank support rarely required; rapid advance allows face and cut to be abandoned promptly
11. Good recovery (approaches 100% cxccpt at property lines); very low dilution c
П . Normally eliminates hauling overburden (casting is essence of method)
13. Good health and safety; no underground hazards

Disadvantages

1. Economic limits of the method and technological limits of the equipment impose depth limits «300 ft, or <90 m); exceeding limits requires excessive rchandling of spoil
2. Economics impose limits on stripping ratios (maximum 30-45 ydVton, or 25-38 mVtonne; overall U .S. average for coal 11.3 ydVton, or 9.5 mVtonne), although much higher than for open pit mining
3. Surface damaged, extensive environmental protection and reclamation required by law; expense substantial, added to production cost
4. High capital investment (boom-type stripping excavators particularly expensive)
5. Requires large deposit to realize lowest cost; small deposit can be mined with small equipment
6. Weather detrimental, can impede operations, if not prevent use of the method
7. Requires careful sequencing of operations, especially during stripping
8. Slope stability must be maintained, usually not a serious problem Applications
In the United States open cast mining is used almost exclusively for the surface mining of bituminous coal (Anon., 1979c; Finch and Fidler, 1980; Chironis, 1984a; George et al., 1986). It also is used in producing anthracite (PA), bauxite (A R), bentonite (A Z), lignite (T X), phosphate (F L — see Hoppe, 1976). tar sands

(Canada— see Love. 1985), and uranium (W Y). *Case Stu d y.* Kellerman bituminous coal mine, Drummond Coal Company, Brookwood, A L.
G eneral: largest surface coal mine in Alabama, located in Warrior Basin of Appalachian coal field
C o al: high-volatile, metallurgical bituminous, five seams (Carter. Milldalc, Johnson. Brookwood. Guide), total thickness 90 in. (2.3 m)
Prod uctio n: 2.1 million tons/yr (1.9 million tonnes/yr) coal, 36 million ydVyr (28 million mVyr) overburden
Equipm ent: two 78-yd? (60-m3) draglines, wheeled scrapers, rotary overburden drills, front-end coal loaders, tractor-trailers for coal haulage
O verall stripping ra tio : 25 ydVton (21 mVtonne)
Pro d u ctivity: 28 tons/employee-shift (25 tonnes/employee-shift) of clean coal

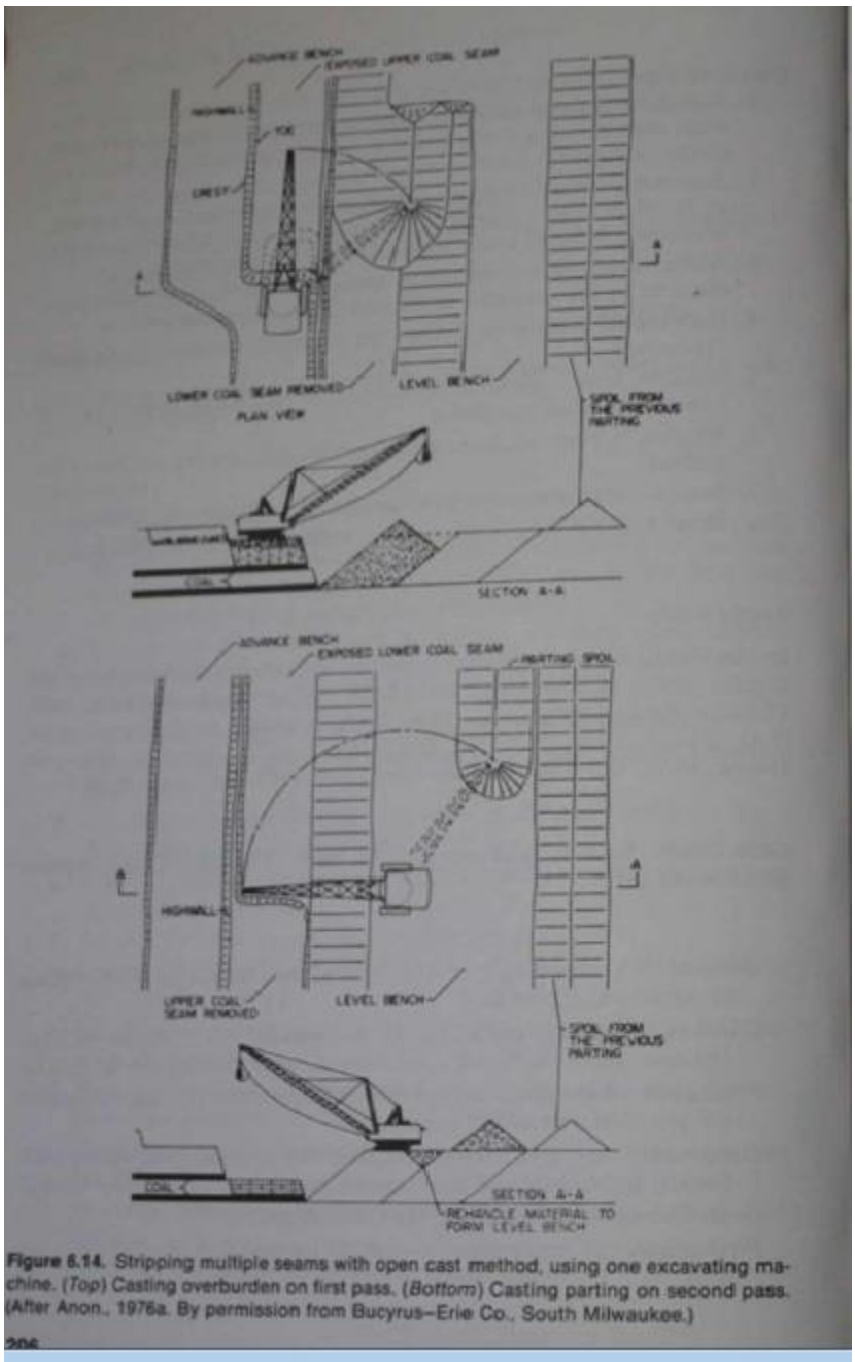


Figure 6.14. Stripping multiple seams with open cast method, using one excavating machine. (*Top*) Casting overburden on first pass. (*Bottom*) Casting parting on second pass. (After Anon., 1976a. By permission from Bucyrus–Erie Co., South Milwaukee.)

Variations

The two major variations of open cast mining in coal arc *area mining* and *contour mining* . Area mining is carried out in flat terrain with flat-lying seams; mining cuts arc made in straight, parallel panels, advancing across mthe property. It is common in the U.S. West and Midwest. Contour mining is conducted in hilly or mountainous terrain, with cuts placed on the contours to rim the mountains, progressing from the outcrop to a depth fixed by the stripping ratio. It is prevalent in the U.S. East.

Multiple-seam mining requires special sequencing and great care to avoid e x c e s s i v e rehandling (Olsson, 1983). One successful variation of open cast mining employs a single excavator, a dragline (alternatively, two machines may be used in tandem) (Anon., 1976a). The initial cut is performed as shown in Figure 6.14 *(top).* Here, the dragline strips to the upper coal scam in its first pass. The spoil toe is allowed to just touch the edge of the lower coal scam. Then the upper coal seam is mined, and the spoil-pilc top is leveled (rehandled) to allow the dragline to strip the parting from a location on the spoil bank, usually working in the opposite direction. The second pass is illustrated in Figure 6.14 *(bottom):* the dragline casts the parting on top of the previous spoils at a minimum swing angle. Elimination of practically all rehandling and employment of a single excavator are the main advantages of the method. The main disadvantage is the extreme dragline reach requirements.

Explosives stripping is being used to some extent in open cast coal mining to reduce the amount of conventional casting done by excavators (Chironis, 1980; Bauer et al., 1983). Under controlled circumstances, explosives energy can be less expensive for casting than electromechanical energy. With proper blasting practice, 40-60% of the overburden can be cast by explosives. with an overall savings in the stripping function.

6.5 AUGER MINING

Auger mining is a method for surface highwall or outcrop recovery of coal by boring or excavating openings into the scam beneath the overburden. Although the overburden is not removed, and the coal is extracted by an auger or a mining machine working underground, the crew operating the machine is located at the surface; hence we classify it a surface method. In most applications, augcring is a supplemental method used in conjunction with open cast mining; it accounts for < *2 %* of U.S. surface mineral production.

Augering is *intermediate* in production rate and productivity, falling between a large-scale and a small-scale mining method.

Auger mining originated in the late 1940s as a method for the recovery ot coal left in the highwall after conventional surface mining (Anon., 1979c). It was developed in the mountainous areas of the Appalachian coal fields for the mining of coal beyond the pit limits or at the property lines, coal that would otherwise be lost and left in the highwall (unless recoverable by underground methods).

Coal is generally extracted by an auger consisting of one, two, or three boring heads (Pfleider, 1973a). More elaborate mining machines may also be used

(see the section “Variations”, below). A coal auger works on the principle of a large-scale drag-bit rotary drill. No prior breakage of the coal is required, and the auger extracts and removes the coal from the hole, where it is elevated by conveyor or loaded by front-end loaders into trucks. A form of continuous miner, the coal auger is a relatively simple, inexpensive machine that is easily moved, operated by a small crew (three to four), and relatively productive.

Because true augers bore a circular hole, the cutting head and flight diameter must be chosen to be compatible with the seam thickness. Consequently, recovery is low (40-60%). To improve productivity and recovery, augers with multiple heads are used, especially in thin seams (Blakely, 1475). Designed to cut an overlapping pattern, they leave less coal in place in the webs or pillars between holes. Specifications of augers arc as follows:

Diameter, single head	1.5-8 ft (0.5-2.5 m)
double head	1.5-3 ft (0.5-0.9 m)
triple head	1.5-2 ft (0.5-0.6 m)
Length or depth	200-300 ft (60-90 m)
Production rate	100-2500 tons/shift (90-2200 tonnes/shift)

A small coal auger for thin-seam mining (<3 ft, or <0.9 m) is pictured in Figure 6.15. Operation of augering in conjunction with open cast contour mining is shown in Figure 6.16.

Sequence of Development

Developing and preparing a site for auger mining is a simple task if it is used in conjunction with and follows open cast or open pit mining. Practically no additional development is required for augering. After the highwall is trimmed of loose rock (for safety) and the pit floor cleaned of debris (for ease of operation), he auger can be set up and placed in operation. A cut width of 50-75 ft (15-25 m) is generally adequate. For access and transportation, a haul road must be constructed near the site, if not already available. If augering is being carried out on the outcrop, then dozing to clear a vertical mining face in the seam as well to provide a level bench and access for the auger is necessary.



Figure 6.15. Surface coal auger with single head in thin seam. (By permission from Richmond Manufacturing Co., Ashland, OH.)

Cycle of Operations

No stripping of overburden is required in augering. Uncovering of the coal seam in the highwall or at the outcrop has already occurred. During operation of the auger, the following abbreviated production cycle takes place:
Ex cavatio n : auger (1, 2, or 3 heads)
Haulage : (1) auger flights, (2) conveyor or loader, (3) truck
No auxiliary operations, other than advanced trimming of the highwall and cleaning the pit floor, are necessary.

Conditions

The following is based on Pflcider (1973a) and Anon. (1979c):
1. *Ore strength :* any
2. *Rock strenth :* any
3. *Deposit shape:* tabular, bedded
4. *Deposit dip :* low, nearly horizontal



Figure 6.16. Auger mining of coal used in conjunction with open cast contour mining. (Anon., 1979c. Illustration by Frank Kulczak. By permission from Skelly and Loy, Harrisburg, PA.)

5. *Deposit size:* small, confined, thin
6. *Ore grade:* can be very low
7. *Ore uniformity:* highly uniform
8. *Depth:* shallow, limited to highwall height where auger is placed

Features

The following is based on Blakely (1975) and Anon. (1979c):

Advantages

1. High productivity, among the highest of any mcchanical extraction surface method (25-500 tons, or 22-450 tonnes, per employee-shift)
2. Low mining cost, probably the lowest of any mechanical extraction surface method (relative cost 5%)
3. Intermediate production rate (100-2500 tons, or 90-2200 tonnes, per shift)

4. Little ore development required, when used in conjunction with open cast mining
5. Lo w labor requirement (crew: three to four)
6. Lo w capital investment, may require purchase of auger only
7. Surface preserved, no separate reclamation cost incurred
8. Recovers coal that would otherwise be lost
9. Good health and safety; no underground hazards

Disadvantage s

1. Application is strictly limited to certain conditions associated with open cast mining; method rarely employed alone
2. Application is restricted to small remnants of coal at pit limits and boundaries (maximum length 300 ft, or 90 m; maximum height 8 ft, or 2.4 m)
3. L o w coal recovery (40-60%)
4. Production capability dependent on single extraction unit

Applications

All applications of auger mining are for highwall (or outcrop) production of bituminous coal in mountainous areas and in conjunction with open cast (or open pit) mining (Chironis, 1984b, 1985).

Case Study. Shannon bituminous coal mine. Black Diamond Coal Company, Abernant, A L.

General: small surface mine, located in Warrior Basin

C o al: high-volatile, metallurgical bituminous; two seams (Blue Creek, Mary Lee), combined thickness 10 ft (3 m) maximum, dip 13-15°

Method: highwall miner, opening size 5 x 7 ft (1.5 x 2.1 m) depth 220 ft (67 m)

Production: 400-500 tons/shift (360-450 tonnes/shift), crew 3

Variations

Two interesting improved versions of auger mining have been attempted, both types of continuous mining machines remotely controlled. The first was the pushbutton miner, coupled to a train of rubber-tire-mounted conveyors stored in a mobile tower. Complex and never entirely successful, the conccpt is Currently inactive. The sccond is the highwall miner (formerly the thin-seam miner) (Fig. 6.17), newer and simpler in design (Chironis, 1982a, 1984b). Driving a cross section 3-8 ft (0.9-2.4 m) in height and 8 ft (2.4 m) in width, the highwall miner is capable of advancing 320 ft (98 m) into the coal. Gamma-ray sensors guide the cutting head in the seam, and broken coal is



Figure 6.17. Highwall miner used in auger mining of coal. (By permission from Metec, Inc.,Lexington, KY

convered from the hole by a spiral conveyor. The highwall miner is designed for a recovery up to 80% and an availability of 80%, with a production rate of 100-300 tons (90-270 tonnes) per hour. For recent experimental concepts in auger mining, see Treuhaft (1984).

6.6 SPECIAL TOPIC A: SURFACE BLASTING

While general blasting theory and practice will be covered later under underground mining (Section 11.6) and the topic of explosives has already been discussed (Section 4.3), surface blasting is a rather specialized topic and deserves its own coverage. In treating the subject, we note common blasting practices and explore the design of blasting patterns in surfacc mining.

Blasting Practice

Since the advent of ammonium nitrate-based explosives and blasting agents and the demise of nitroglycerin-based high explosives in mining, blasting practice has standardized substantially, especially in surface mining. Further, the provision of several free faces with the existence of benches in virtually all methods of surface mining facilitates breakage and simplifies blasting.

The major elements of surfacc blasting practice consist of the following:

1. *Inclination of drillhole:* generally vertical; may incline to improve efficiency or drill horizontal in open cast highwall
2. *Subdrilling :* depth of hole below grade or intended floor (provided to, ensure complete breakage to floor; increases in back rows of holes)
3. *Row s o f holes:* usually multiple (two, three, or more)
4. *H ole p attern :* square, rectangular, or staggered (triangular)
5. *Exp losives:* A N FO (bulk or bagged); AN or metallized slurry or gel
6. *Hole loading:* by machine (bulk) or hand (bag); explosives confined by stemming (to reduce blowouts); may deck charge to distribute explosives in hole
7. *Hole charging :* double line of explosive detonating cord in drillhole connected to special primers to explode A N FO ; cord detonated by blasting caps, fired electrically; millisecond delays provide delays between rows and holes (improve fragmentation and reduce noise and vibration)
8. *Secondary blasting :* additional breakage required for boulders remaining after primary blast; accomplished by mudcapping, blockholing, dropballing, or impact hammer (inefficient and expensive breakage)

Several aspects of blasting practice are illustrated in Figure 6.18. Additional particulars are available in the literature (Ash, 1968; Pfleider, 1973a; Anon., 1977: Gregory, 1979; Hemphill, 1981).

The design of blasting patterns in surface mining is based on theoretical concepts, modified by experience and empirical considerations. Factors to plan or evaluate blasts are employed for both drilling and blasting; they are termed the *d rillin g fa c to r* and the *pow der fa c to r.* For various mining methods and commodities, they range as follows:

	DRILLING FACTOR	POWDER FACTOR
Open pit mining	0.02-0.04 ft/ton	0.1-1.0 lb/ton
metallic ores	(6.7-13.4 mm/tonne)	(0.05-0.5 kg/tonne)
Open pit mining		0.05-0.3 lb/ton
nonmetallic ores		(0.03-0.1 kg/tonne)
Open cast mining		0.2-0.5 lb/ton
Overburden		(0.1-0.3 kg/tonne)

While empirical, they arc useful for design and evaluation.
The main parameters of the blast which have to be determined are the following:

1.Bank *height and slope: based on non*blasting considerations
2. *Hole diameter:* range 6-*18 in. (150-460 mm)*
3. Hole angle: vertical, *inclined (<30° to* vertical, horizontal On frequent)
4. *Hole depth:* bank *height plus subdrilling,* in ft (mf
5. *Burden: perpendicular distance from slope face to center of* charge, *in ft (m)*
6. *Explosives loading density: weight of explosives per* length of

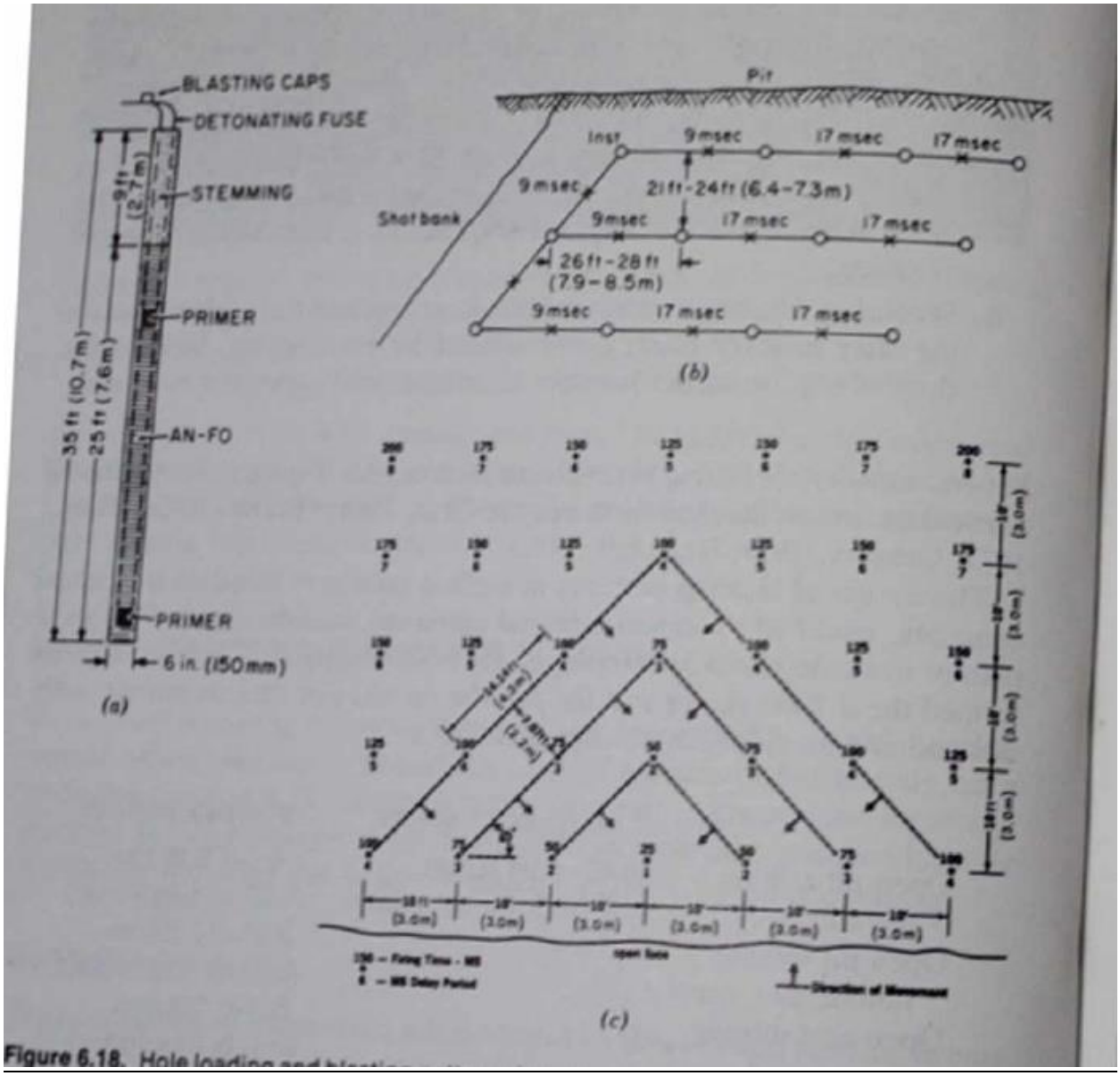


Figure 6.18. Hole loading and blasting patterns in surface bench mining (a) Cross section through drillhole showing loading and charging of explosives (b) Plan view of diagonal blasthole pattern; breaking to two vertical free faces with milliscecond delays. (After Stefanko and Bise, 1983 By permission from Society of Mining Engineers. Inc., Littleton, CO.) (c) Plan view of echelon biasthoie pattern; breaking to single vertical free face with millisecond delays. (After Anon..

1977, By permission from E. Co.. Wilmington, DE.) I. duPont de Nemours and Co., Wilmington, DE.)
7. *Height of charge in drillhole:* usually continuous but may stem; measured in ft Cm)
8. *Weight of charge per drillhole:* in lb (kg)
9. *Hole spacing:* distance between holes and rows, in ft (m)
10. *Number of holes required*
11. *Weight of rock (ore, coal) broken:* tons (tonnes)
12. *Drilling factor:* length of drillhole per unit weight of rock broken, in ft/ton (m/tonne)
13. *Powder factor:* weight of explosives per unit weight of rock broken, in lb/ton (kg/tonne)

Blasting theory will be introduced later, but empirical relationships suffice for now. They are demonstrated in this design problem for an open pit copper mine.

Example 6.2. Design the blasting pattern for the bench of a surface mine shown in cross section in Figure 6.19. given these data:

Copper ore specific weight w	2.0 tons/yd1 (2.4 tonnes/m')
Bank length *L*	200 ft (61 m)
Explosive	AN slurry
Loading density *r*	70 lb/ft (104 kg/m)
Powder factor PF	0.5 lb/ton (0.25 kg/tonne)
Height of charge in hole, κ	35 ft (11 m)
Hole diameter	12 in. (305 mm)

SOLUTION

Bank area $A = bh = (40)(50) = 2000$ ft1 (186 m^2)
Bank volume $V = LA/27=(200)(2000)/27=14{,}815$ yd^3 (11,325m^3)
Bank weight W=wV=(2.0)(14,815)=29,630 tons (26,880 tonnes)
Weight of chargc/drillhole, c =2.0 *rk* = (70X35)
= 2450 lb (1110 kg)

Weight of rock broken/drillhole, m =c/PF =2540/0,5= 4900 tons (4445 tonnes)

Number of holes needed, *n* = W/m=29,630/4900=6.05, say 6
Hole spacing *s=L/n=200/6=33ft (10m)*

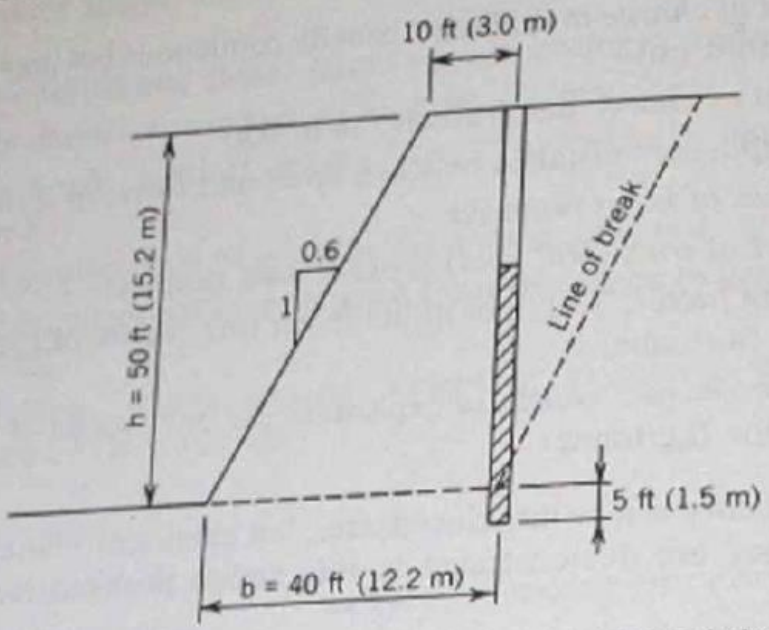


Figure 6.19. Design of surface blast, showing bank cross section and charged drillhole. See Example 6.2.

6.7 SPECIAL TOPIC B: SLOPE STABILITY

In mine development, (here arc many situations where soil mechanics can be applied to the design of structures and to mining activities. They include the following:

1. Structural stability (buildings, roads)
2. Foundations, retaining walls, dams
3. Slope stability (pit slopes as well as dumps, tailing ponds, etc.)
4. Seepage, drainage
5. Excavation
6. Compaction, fill

Although soils typically arc encountered only at the surface, some poorly consolidated and broken rock behaves much like soil. Also, rock in jointed or fractured masses may resemble soil and be susceptible to analysis by soil mechanics. Similarly, in underground mining, backfill, caved ground, broken ore in stopcs, and soft running ground behave more like soil than rock. Hence soil mechanics has widespread application in mining engineering, perhaps as much as rock mechanics. We shall examine only one common usage, largely in soils, that of slope design and maintenance in surface mining. For the analysis and design of rock slopes, see the work by Hoek and Bray (1981).

6.7 SPECIAL TOPIC B: SLOPE STABILITY

Principles of Soil Mechanics

Soil mechanics is the study of the properties of soils and their behavior in relation to the design, construction, and performance of engineering works. It is applicable, however, to man-made materials such as fills, wastes, tailings, dumps, stockpiles, bins as well as to soils, which broadens its utility in mine engineering. Soil mechanics is a companion field to rock mechanics, and the two share much in common in the analysis and design of geotcchnical structures.

Soil is a natural aggregate of mineral grains, mechanically adhering. For our purposes, all naturally occurring, uncemented or unconsolidated materials which arc naturally or artificially deposited— soils as well as loose or weathered rock— are considered soils. Typically, a soil is a complex, threephase system consisting of solid grains and voids, in turn made up of water and air (Fig. 6.20). If we visualize a unit volume of a soil, V, then the grain volume is *Vg* and the void volume is *Vv* . The voids consist of air *Va* and water *VH*.. On a specific weight basis, the weight of the unit volume of soil is W, the weight of the grains is *W* g, and the weight of water is Ww (the weight of air *Va* is usually neglected). Certain useful soil properties are derived from
these parameters:

Porosity $n = V J V$ x 100 (%) 6 .5)

Moisture content $m = W J W g$ as a decimal (6.6)

Specific weight w = *W IV* in lb/ft3 (kg/m3) (6.7)

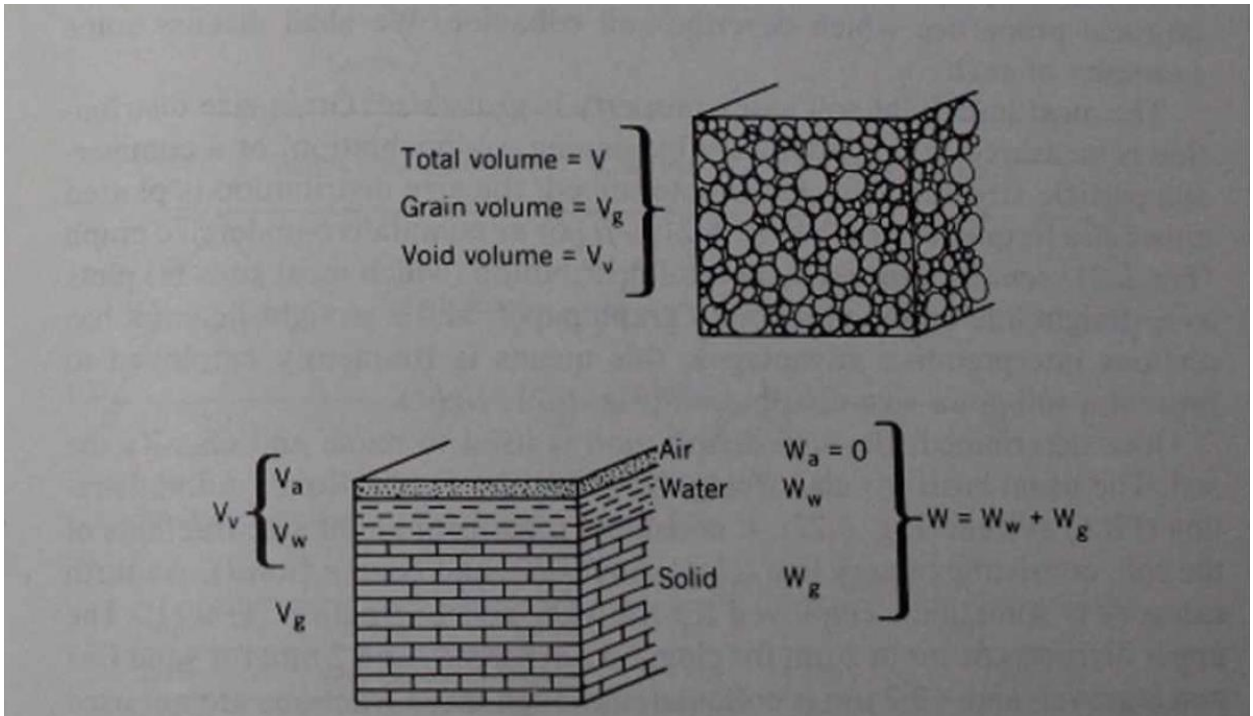


Figure 6.20. Soil represented as three-phase system, consisting of solid (grains), liquid (water), and gas (air). (*Left*) Natural composite sample. (*Right*) Sample stratified by phase.

Figure 6.20. Figure 6.20. So il represented as three-phase system, consisting of solid (grains), liquid (water), and gas (air). *(Left)* Natural composite sample. *(Right)* Sample stratified by phase.

Another common measure of w in surface mining is bank or solid measure, in units of tons/yd^3

We have already encountered (Section 2.7) use for tonnage factor TF, the specific volume of natural materials in ft^3/ton (m^3/tonne). The relationship between specific weight (in various units) and tonnage factor is

$$w=2000/TF \text{ lb/ft'} \qquad (2\text{-}2)$$

$$w = 1000/TF \text{ kg/m'} \qquad (2\text{-}2a»)$$

$$w= 27/TF \text{ tons/yd}^3 \qquad 6.8)$$

$$w=1/TF \text{ tonnes} \qquad (6.8a)$$

Some values of TF and w for common mineral commodities arc as follows:

	Tonnage factor	Specific weight
Iron ore	14 ft^3/long ton (0,4m^3/tonne)	1,9 long tons/yd^3 (2,5 tonnes/m^3)
Copper ore	13 ft^3/ton (0,4 m^3/tonne)	2,1 tons/yd^3 (2,5 tonnes/m^3)
Bituminous coal	25 ft^3/ton (0,8m^3/tonne)	1,1 tons/yd^3 (1,3 tonnes/m^3)

Soil properties are usually grouped in two categories. *Index properties* are identification properties used to classify soils. *Mechanical properties* are physical properties which describe soil behavior. We shall discuss some examples of each.

The most important soil index property is *grain size.* Grain size distribution is measured in the laboratory by sieving, sedimentation, or a commercial particle size analyzer. Once determined, the size distribution is plotted cither as a frequency graph (Fig. 6.21. *left)* or as cumulative-undersize graph (Fig. 6.21, *center).* Since a lognormal distribution (which most soils fit) plots as a straight line on log-probability graph paper, and a straight-line plot has obvious interpretative

advantages, this means is frequently employed to present a soil grain size distribution (Fig. 6.21, *right).*

Once determined, the size distribution is used to name and classify the soil. The usual basis for classifying the soil is the Public Roads Administration (PRA) system (Fig. 6.22). It considers three prominent size fractions of Ihe soil, consisting of very fine *(clay),* fine *(silt),* and coarse *(sand).* A fourth category is sometimes employed for the very coarse fraction *(g ravel).* The upper divisions occur at 2 /ш for clay. 60 Mm for silt, and 2 mm for sand (>2 mm is gravel, and <0.2 /ш is colloidal, although these fractions arc not used in the classification). Once the fractions are read from the size distribution graph, they are plotted on the PRA chart (a chart of three coordinates) and the soil name read.

100
Percent
frequency
size
distribution
0
Percent
cumulative 50r
undersize
<100
Percent
cumulative
undersize
(probability
scale)
Particle size Size Log size
(a)

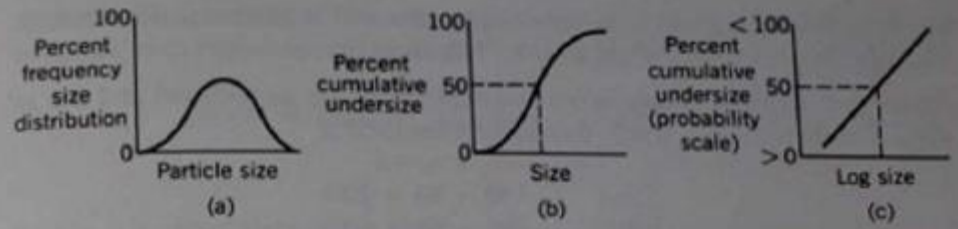
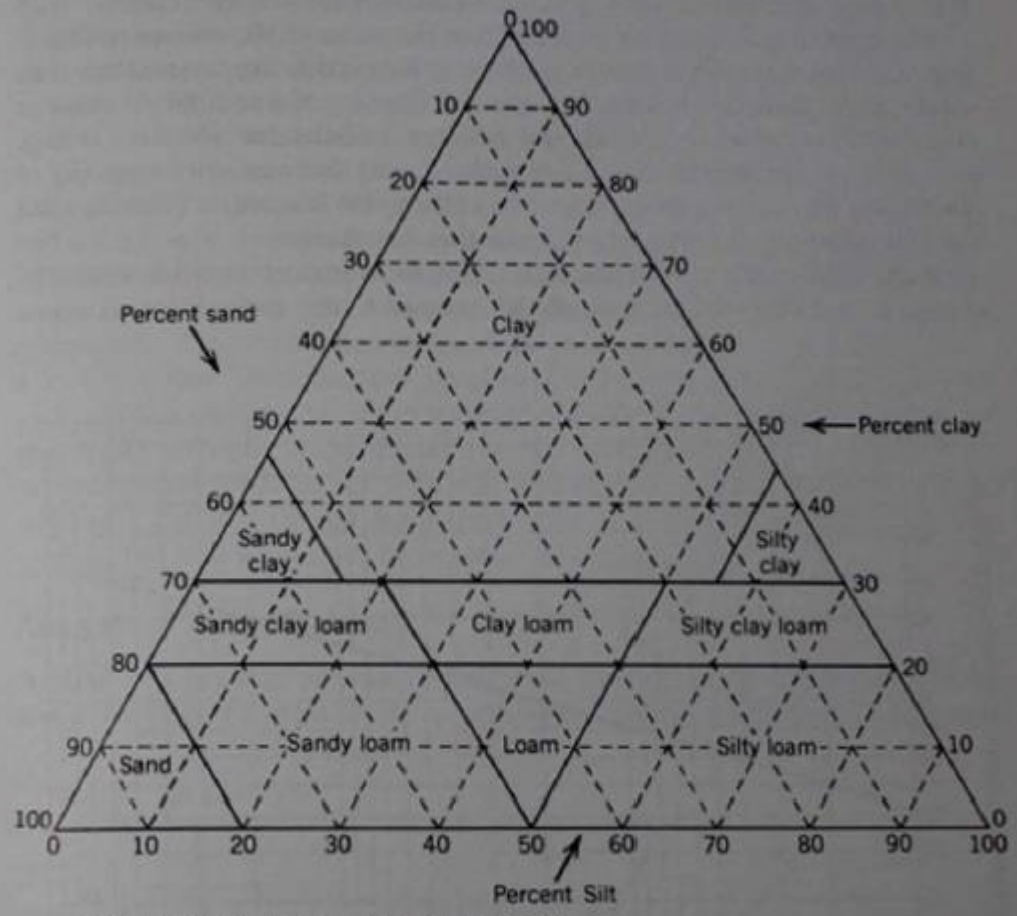


Figure 6.21. Grain size distribution of soil, plotted as (*left*) frequency graph, (*center*) cumulative-undersize graph, and (*right*) cumulative-undersize graph on log-probability paper.



Size-fraction classification, based on grain size:

Size	2000 / 2.0	60 / 0.06	2 / 0.002	0.2 / 0.0002	μm / mm
Name	Gravel	Sand	Silt	Clay	
Description	Very Coarse	Coarse	Fine	Very fine	Colloidal
Visibility	Macroscopic		Microscopic		Submicroscopic

(1 mm = 1000 μm)

Figure 6.22. Soil classification chart, based on grain size distribution. (Modified after U.S. Public Road Administration chart.)

Example 6.3. The grain size distribution of a soil is plotted in figure 6.23.

Classify the soil with *the aid of the PRA chart.*
Solution . Read the cumulative undersize at 60 m as 80% *and at 2 m 40%. The distribution in the three fractions then is*

Sand	100 – 80= 20 %
Silt	*80 - 40 = 40%*
Clay	*40* - 0 = 40%

Plotting the distribution on Figure *6.22* identifies the so I as a clay.

Much information can be gleaned from the name of the soil, once classified Certain behavioral and engineering properties arc associated with *sandy, silty, and clayey soils. It is the fine fraction o f* a soil, however— the clay *fraction— which is critical and* governs its behavior. (Notice, in Fig. 6.22. that clay predominates as a soil name, and that any soil consisting of >45% clay fraction *is* classified as clay.) Obert (1973a) gives a more detailed *soil* classification and *property* scheme that has merit.

Of the mechanical *properties o f* soil, the most important is *strength,* the allowable working *stress:* and of *the* strengths, the critical one is shear

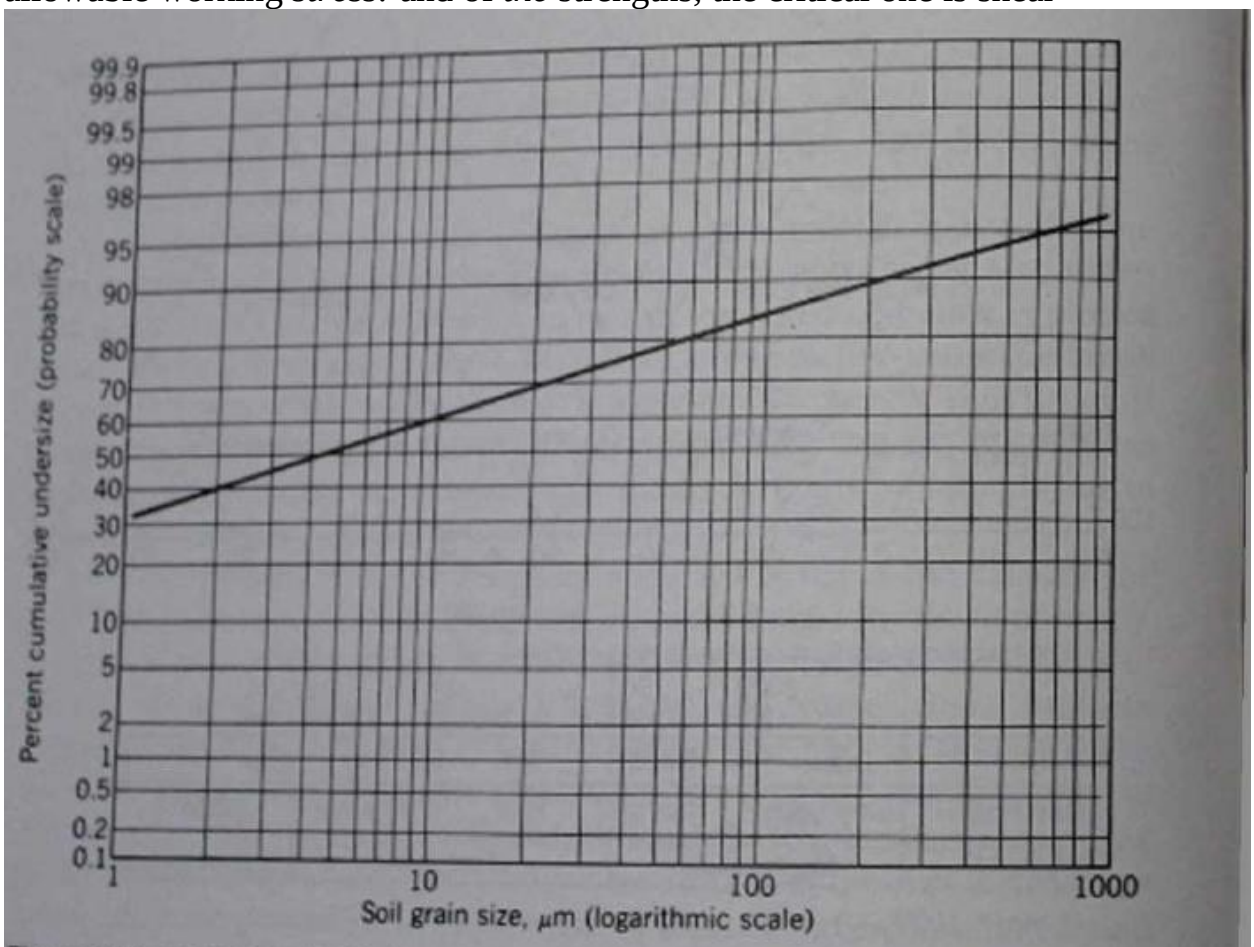


Figure 6.23. Grain size distribution of soil plotted on log-probability paper as percentage cumulative undersize. See Example 6.3.

strength (Obert, 1973b). The classical analysis of shear strength in a soil involves both cohesion and friction (Figure 6.24):

Shear strength = cohesion + friction

$s = c + p \tan \phi = c + wh \tan \phi$ (6 .9)

where *s* is shear strength in lb/ftl (κPa), *c* is cohesion in lb/ft2 (kPa). *P is n*ormal stress in lb/ft- (kPa), w is specific weight in lb/ft1 (kg/m'), *h* is depth in ft (m). and *Φ* is angle of internal friction or angle of repose in degrees.
Most sands have low cohesion and high friction, while clays have high cohesion and low friction. A wet soil usually possesses both a higher cohesion and larger angle of internal friction; if saturated, however, both may fall to /ero. A high shear strength in a soil allows a high allowable working stress, but if the strength is exceeded, the soil and any structure on it fails.

We can now offer some general observations about soil behavior. A *sandy soil* is usually stable and permeable and can be drained to prevent saturation and a decline of strength from occurring. Quicksand is a dangerous subtype of soil hibiting spontaneous liquefaction with shock; it has rounded grains and is unstable when wet. A *clayey soil* has an affinity for water, is cohesive and plastic, may well when wet, and can be treacherous when saturated. A *silty soil* (loam) demonstrates properties intermediate between sands and clays. W'hen designing a soil-supported structure— bank slope, building, or road— the strength properties of the dry (or moist) soil are utilized, and provisions are made to keep the soil well drained. Soil drainage in its own right is a subfield of soil mechanics, one of great importance in surface mining, but one we do not have space to explore here.

Analysis of Slope Stability

A *slide* is a gradual or sudden failure of the soil mass underlying a bank slope, causing the mass to move outward and downward. Since it is one of

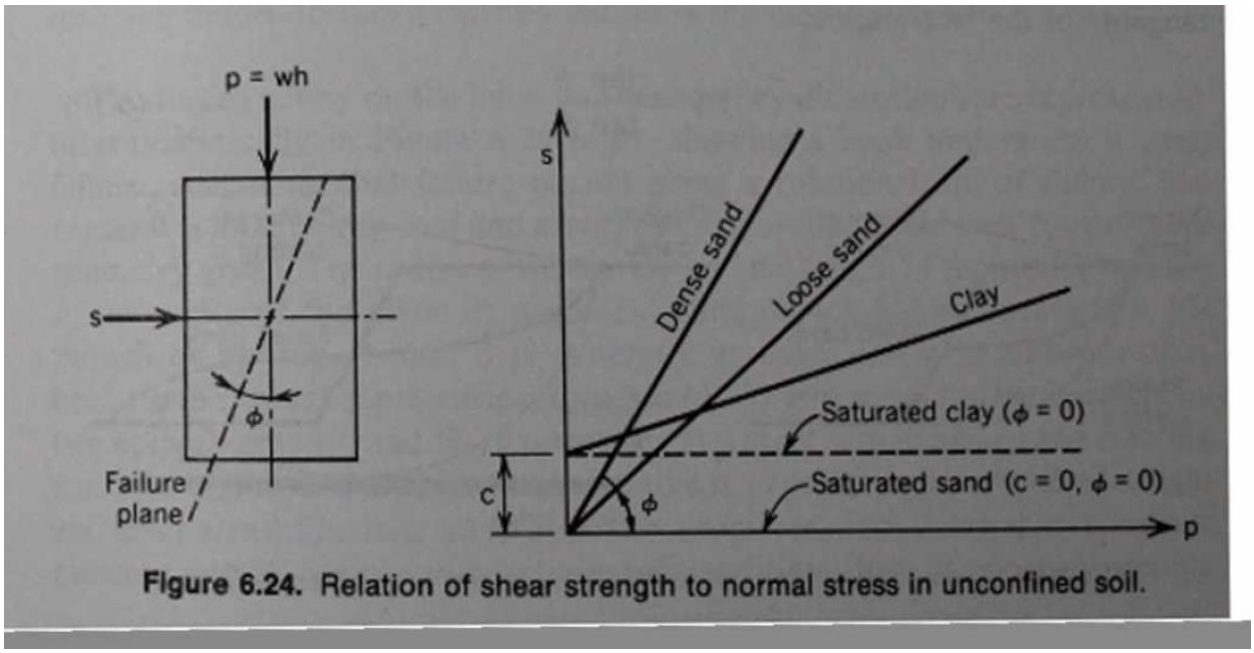


Figure 6.24. Relation of shear strength to normal stress in unconfined soil.

the most catastrophic natural failures that can occur in a surface mine, mining engineers must learn how to prevent slides and to design slopes properly so they do not occur. The latter comprises the subject of slope stability.

There are many causes of bank failure which may result in a slide: (1) undercutting the toe (bottom), (2) excavating too steep a slope, (3) rise in pore water pressure, (4) excessive shock or vibration, such as blasting or carthquake, (5) existence of discontinuities such as bedding, cleavage, faults, and fractures.Slides are prevented or controlled by (1) flattening the slope, (2) proper drainage, (3) placing a retaining wall or ballast at the toe, (4) driving pilling, (5) reinforcing the bank with rock bolts or bolts and wire mesh, and (6) closely monitoring the surface for tensile cracks near the bank crest (top) (Obert, 1973b; Thomas, 1978; Seegmiller, 1984)

Two types of failure may occur, reflecting the location of the sliding surface (Fig 6.25). A *slope failure* occurs intermediate between the crest and toe A *hose failure* occurs at a plane of weakness (such as causal situations 5 and 6. above), usually below the toe of the bank. In cither case, the result is a heaving and slumping of the bank, the failed material moving outward and downward in a massive flow.

Table 6.2 contains rccommcnded design practice for slopes in soil and rock, alues that are customary and normally safe in preventing slides.

These are empirical values, of course, which should be refined against operating experience at a given mine.

There are also analytic procedures to employ in the engineering design of 'it slopes. We will examine a method for cach pit of two soil types.

1. *Cohesionless soil.* In dry or drained sandy or silty soil, a stable bank can be designed based on the requirement that the bank angle be less than the angle of internal friction (or angle of repose) of the soil (Obert. 1973a).

Usually, the design is based on an empirical *facto r o f safety* F S relating the tangents of the two angles:

$$FS = \frac{\tan \phi}{\tan \beta} \qquad (6.10)$$

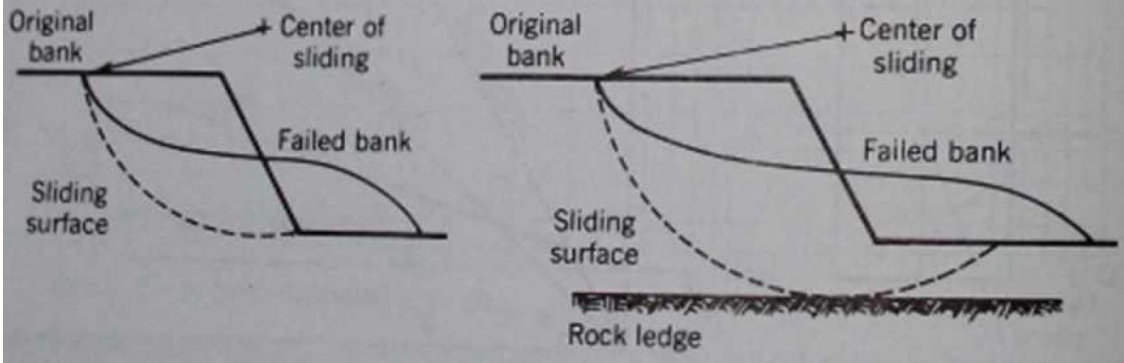


Figure 6.25. Types of bank failure. (*Left*) Slope failure. (*Right*) Base failure.

Table 6.2 Recom m ended Pit Slopes'

A. Soils	
Clean, loose sand	1½:1 (34°)
Sand and clay	1⅓:1 (37°)
Wet sand	2½:1 (22°)
Gravel	1½:1(37°)
Dry clay	1:1 (45°)
Wet clay	3½:1 (16°)
Loose rock	1:1 (45°)
Glacial till	3:1 (20°)
Mud	1:1 (45°)
B. Rock	
Solid country rock	1/3 or ½:1 (72-63°)
Solid iron ore, bench	1/2 or 2/3:1 (63-56°)
Solid iron ore, total bank	1:1 (45°)
Solid copper ore. Bench	2/3:1 (60°)
Solid copper ore. total bank	1½:1 (34°)

* Values expressed as slope ratio, horizontal to vertical, and slope angle (in degrees).

Numerically, values of FS in practice range from 1.0 to 1.5, with a commonly accepted design limit of at least 1.25 for nonpermanent slopes.

2. *Cohesive soil.* In a bank consisting of a cohesive soil such as clay or silty clay, the slope is also stable if *(1* < d>, but the soil's cohesion allows us to increase the bank angle to an even steeper value (Obert, 1973b). The method used for analysis is known in soil mechanics as the *slip circle technique.* It is based on the

assumption that the rotational arc of sliding during a bank failure is circular, permitting a simple analytic solution. The mathematical premise for analysis is the summing of the moments of all forces acting on the bank, including the soil weight and shear stress. If the moments are unbalanced in favor of a rotational movement, then the bank fails. If they are unbalanced to restrain movement, the bank is safe. The ratio of the moments resisting failure to those causing failure is the factor of safety.

The forces acting on the bank and certain key dimensions are represented diagrammatically in Figure 6.26 *(top),* showing a bank undergoing a base failure. Assuming that failure occurs along a rotational are of sliding, the center 0 is located (by trial and error) as the worst possible case for the bank geometry given. The radius of the are is *r* and the length of the sliding surface *L :* the depth of the plane of weakness along which failure occurs is *h* (although an average height *h* is generally assumed in the calculation of *s,* because the actual depth varies from 0 to). We represent the bank weight by two components W| and W2, each acting at a lever arm (/| and *I2)* and creating a moment (W_1, and W_2 , each cach acting at a ver arm l_1 and l_2) and (An alternative is to use a single combined weight and lever arm.) The two act in opposite directions, $W_1 l_1$ and W_2 l_2 an overturning moment nd *W 2I 2* a resisting moment. In addition, there is another

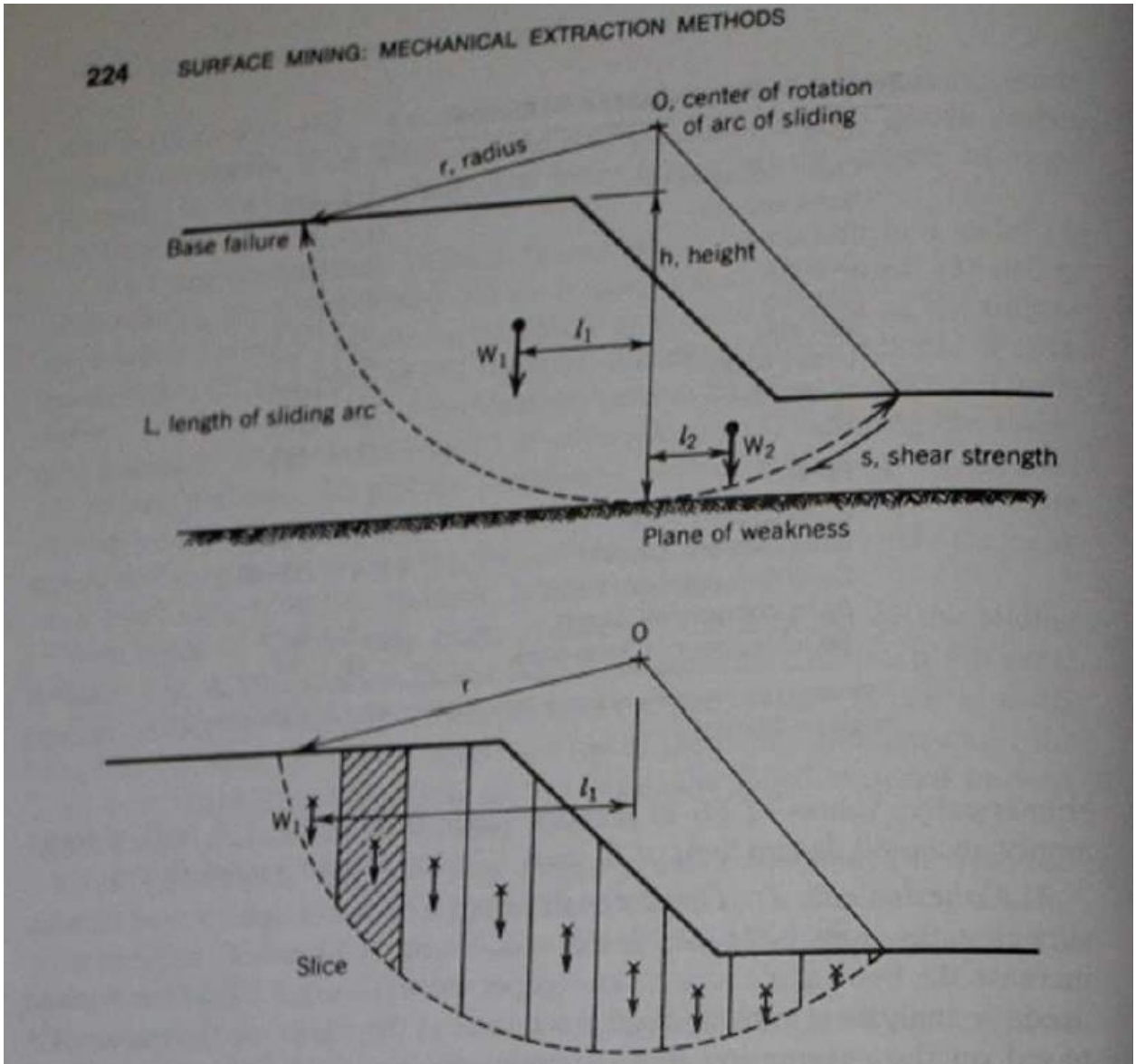


Figure 6.26. Bank failure analysis by slip circle technique. (*Top*) Analysis by opposing

moment, the shear strength j acting along the length of sliding *L* at lever arm *r*. Summing the three moments enables us to determine if the bank is stable:

$$W_1 l_1 < W_2 l_2 + sLr \qquad (6.11)$$

The factor of safety is

$$FS = W_2 l_2 + sLr / W_1 l_1 \qquad (6.12)$$

and we again usually assume stability if FS > 1.25.

Numerically, values ol *W* and IV2 are determined most easily by measuring the areas and converting them to weights per unit of bank width by multiplying by the soil unit weight. Lever arms *l* and *l2* can be approximated by geometric

construction or by cutting out cardboard replicas of cach bank component *Wt* and *W2* and locating their centroids by suspending them freely on a pm. Locating the worst-case rotational center 0 by trial and error

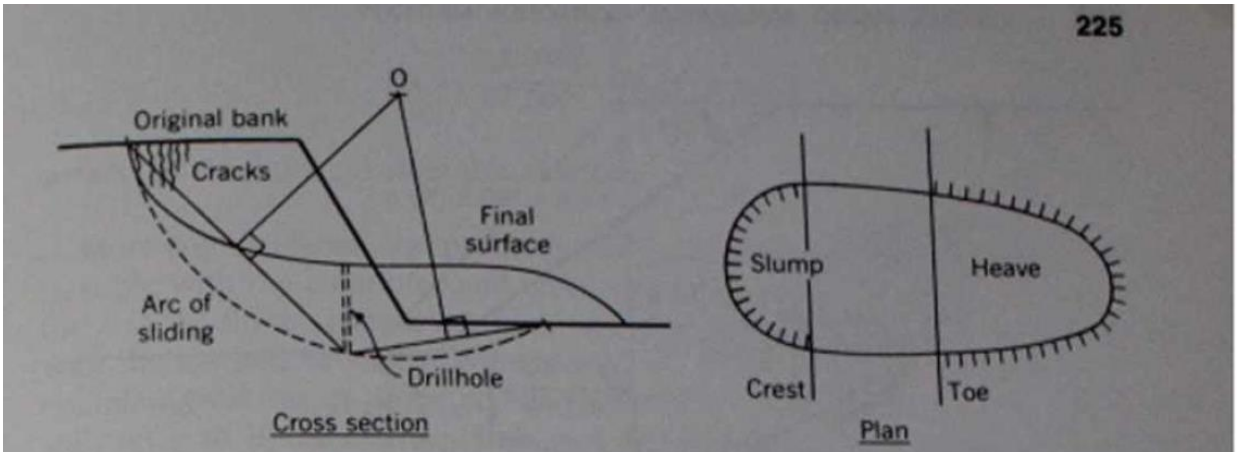


Figure 6.27. Field analysis of bank failure with drillhole to determine location of sliding surface and center of arc of sliding.

is very tedious by hand calculation; it is better pursued today by computer graphics, which can likewise be used to determine the weights and lever arm as well. For greater accuracy in analysis than obtained with two weights, the method of slices can be utilized (Fig. 6.26, *bottom*); using many segments rather than two provides a better mathematical approximation.

In the field, a failed bank can and should be analyzed experimentally. It can be accomplished by drilling an exploratory hole to locate the surface of sliding (Fig. 6.27). Again, based on the assumption that the rotational are is circular, simple geometric construction can be used to fix the center of sliding 0. Conceivably, future failures can be avoided or at least anticipated.

A sure sign of a pending slide is the appearance of tension cracks in the crest of the bank some distance from the slope (Fig. 6.27); preventive measures are sometimes possible even at this late stage. Stress instrumentation can also be installed to warn of threatening bank failure (Merrill and Rausch,1968).

The analytical procedure for analyzing the stability of a slope is demonstrated by a numerical example.

Example 6.4 Analyze the bank shown in cross section in Figure 6.28, with the worst possible case for the center of sliding as shown. The soil is a sand with properties as follow's:

w = 120 lb/ft' (1922 kg/m3)

c = 115 lb/ft2 (5.51 kPa)

$\phi = 30°$

Analyze for two cases; (1) saturated and (2) dry.

S o l u t i o n : (1) Assume soil is saturated. Then
c = 0, ϕ = 0, and s = 0
Therefore, *sLr=0*

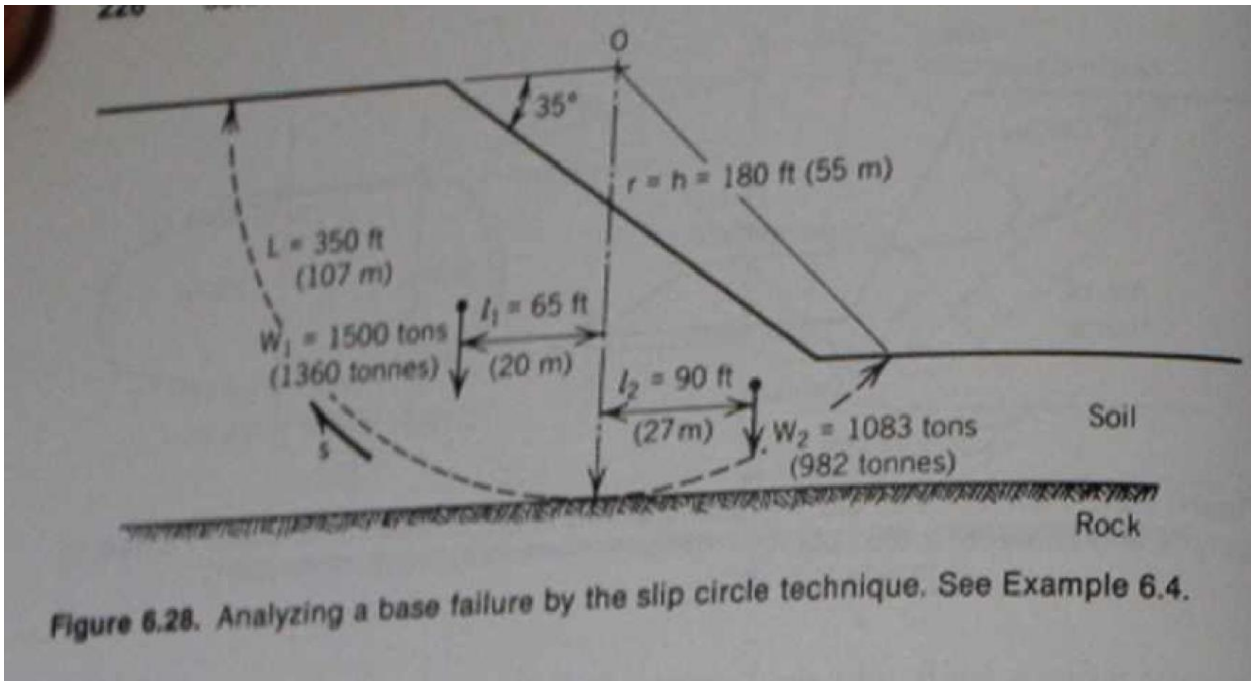


Figure 6.28. Analyzing a base failure by the slip circle technique. See Example 6.4.

Writing the moment equation (Eq. 6.11) per unit width of bank, using units of ft-tons (m-tonnes):

$W_1, l_1 \leq W_2 l_2 + slr$

(1500) (65) ≤(1083) (90) + 0

= 97.500 ft-tons (26.960 m-tonnes)

From Eq. 6.12.

F S = 97.500/97.500 = 1 < 1,25

which is unsafe.
(2) Assume soil is dry (or only moist). Calculate shear strength by Eq. 6.9:
h = h/2 = 180/2 = 90 ft (27 m)
s =*c* +wh tan ϕ = 115 + (I20)(90) tan 30°
= 6350 lb/ft^2 (304 kPa) + 2000
= 3.175 tons/ft2 (31.0 tonnes/m^2)

sLr = p. 175)(350)(180) = 200.020 ft-tons (55,310 m-tonnes)

Summing moments by Eq. 6.1 1 .

97.500 s 97.400 + 200.020 £ 297.500 ft-tons

(82,260 m-tonnes)

and finding the factor of safety by Eq. 6.12,

FS=297.500/97,500=3.05> 1.25

which is safe, by a considerable margin.

More sophisticated, and probably more accurate, methods of analysis are possible with the computer and numerical techniques. The most elegant of these is the finite clement method, and it is employed routinely now along with the method of slices (Wang and Sun. 1970; Obert, 1973b). For more complete treatises on slope stability and failure analysis, in rock as well as soil, refer to Black (1964), Hoek and Bray (1981), Smith (1982), Brawner ,(1982), and Huang (1983).

P R O B L E M S

6.1 Particle size analyses are obtained in index-property tests of five soil samples, and the resulting cumulative size distributions arc plotted on the log-probability graph in Figure 6.29. Using the PRA classification chart, identify and name each soil. Read the graph to the nearest *5%*.

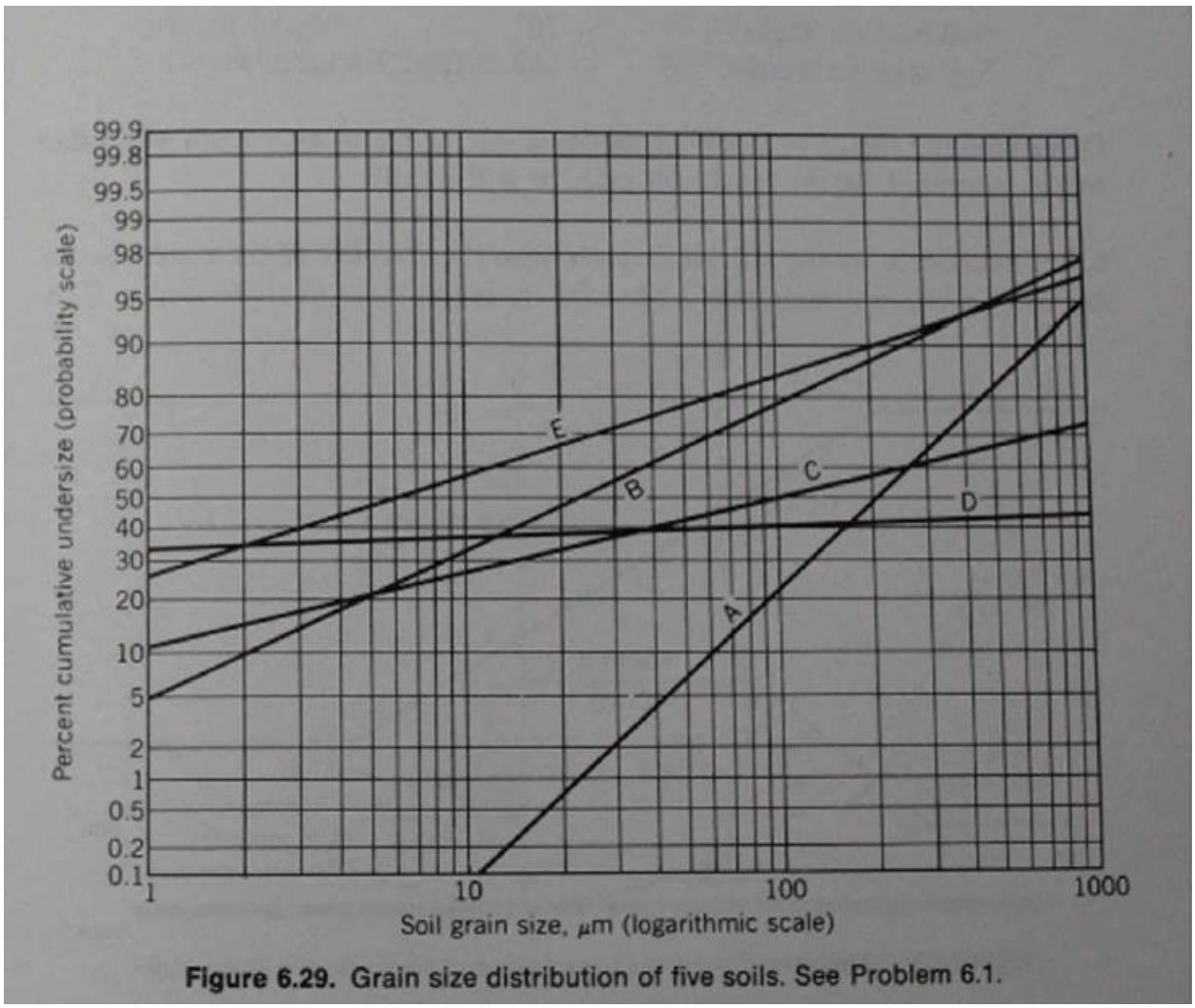


Figure 6.29. Grain size distribution of five soils. See Problem 6.1.

6.2 The following information has been determined in the stability analysis of a slope, using a compined combined weight and lever arm

Shear strength	1200 lb/ft^2 (57.5 kPa)
Length of are of sliding	290 ft (88.4 m)
Total weighth of bank mass failing	826 tons (746 tonnes)
Lever arm	26.3 ft (8.0m)
Radius of are of sliding	120 fut ft (36.6 m)

Is the bank safe, or will it fail? If safe what is the factor or safety?

6.3 An open pit bank in soil has been analyzed under dry conditions and the following data obtained, using a combined moment:

Factor of safety	1.3
Overturning moment	6154 ft-tons (1702 m- tonnes)
Bank height	52 ft (15.8)
Radius of are of sliding	60 ft (18.3 m)
Length of are	124 ft (37.8 m)

Bank angle	44^0
Soil cohesion	80 lb/ft2 (3.8 κ Pa)
Soil friction angle	20°
Soil specific weight	162 lb/ft' (2595 kg/m^3)

Determine the factor of safety of the bank during the rainy season when the soil is saturated. Is the bank still safe, or will it fail ?

6.4 Investigate by the slip circk technique the stability of the bank shown in Figure

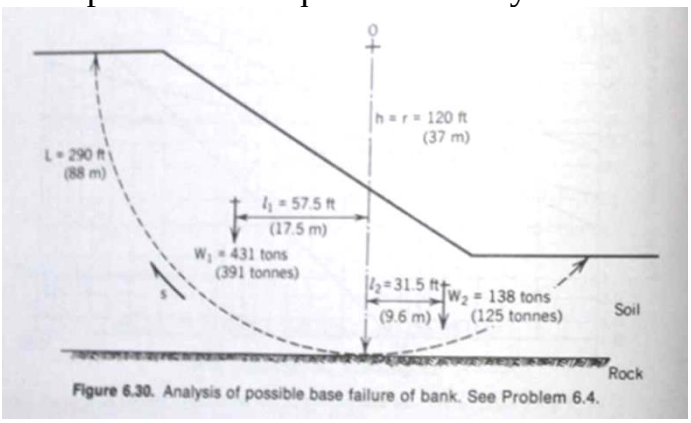


Figure 6.30. Analysis of possible base failure of bank. See Problem 6.4.

6.30. assuming that a base failure is the most likely to occur. The condition selected is considered the worst case. Analyte (a) dry and (b) saturated. Soil property and bank dimensions are as follows:

Soil specific weight	105 lb/ft* (1682 kg/m')
Soil cohesion	M 2 lb/ft1 (5.36 kPa)
Soil internal friction angle	15*
Pit slope	32*
Bank height K0 ft	(24 m)
Depth to rock ledge	120 ft (36 m)

Calculate the factor of safety. Is the slope stable?

6.5 Investigate by the slip circle technique the stability of the bank shown in Figure 6.31. assuming that a slope failure is the most likely to occur. The condition selected is considered the worst case. Analyze (a) dry and (b) saturated. Soil property and bank dimensions are as follows:

Soil specific weight	169 lb/ft' (2707 kg/m')
Soil cohesion	215 lb/ftJ (10.29 kPa)
Soil internal friction angle	25°
Pit slope	40°
Bank height	60 ft (18 m)
Depth to rock ledge	80 ft (24 m)

Calculate the factor o f safety. Is the slope stable?

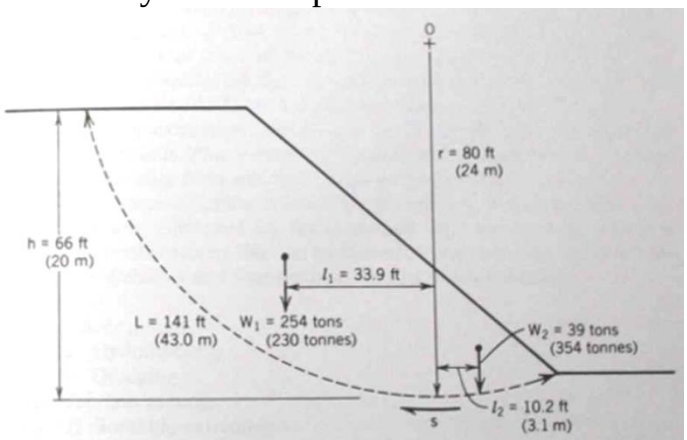


Figure 6.31. Analysis of possible slope failure of bank. See Problem 6.5.

6 6a Modify the design of the blasting pattern for *the* surface mine j

Example 6.2 by increasing the powder factor to *0.6. W hat is the* effect o ”
hole spacing and number of holes required? n
b. Design a blasting pattern to conserve explosives (lower *the* pow |
factor) in Example 6.2 by modifying the hole burden, depth, and/or spac' **

7

SURFACE MINING: AQUEOUS EXTRACTION METHODS

7.1 CLASSIFICATION OF METHODS

In addition to mechanical extraction, there is a second class of surface mining methods. ***Aqueous extraction*** includes those methods which arc reliant on water or a liquid solvent to recover minerals from the earth, either by hydraulic action or solution attack. Much less commonly employed than the mechanical methods (<10% of surface mineral production), the aqueous category consists of some of the oldest and newest mining methods. Restricted in their application, they nevertheless are attractive because of their very low relative cost: Table 3.1 indicates relative costs of 5% or less.

The aqueous extraction class is made up of two subclasses, each consisting of two methods. ***Placer mining*** is one subclass, intended for the recovery of heavy minerals from mainly alluvial or placer deposits, using water to excavate, transport, and/or concentrate the mineral. ***Solution mining*** is another subclass, employed for the extraction from the earth of soluble or fusable minerals or those that can be slurried, using water or a liquid solvent. The two subclasses and four methods consist of the following:

A Placer mining
1. Hydraulicking
2. Dredging

B. Solution mining
1. Borehole extraction
2. Leaching

In addition to being limited in their application, the four aqueous methods employ unique sequences of development and cycles of operations. This chapter discusses the major features of the four methods but, in view of their restricted use in modern mining, treats them more briefly than the mechanical extraction methods just considered. For more details on placer mining, consult references by Griffith (1960) and Macdonald (1983).

7.2. Placer mining: Hydraulic king.

Geologically, a placer deposit is mechanical concentration of heavy mineral, which may be an ore deposit if commercial in value. Common occurrences of minerals in placers are gold, diamond, tin (cassiterite), titanium (rutile), platinum, tungsten (scheelite), chromite, magnetite, and phosphate (found in placer – like deposit). Placer are classified by agent as alluvial (continental detrital), eolian (wind), marine, or glacial. By location, they are categorized as residual (elivual), bench (heelside), stream (fluvial), beach, buried, or desert. Commercially, the most important are, by agent, alluvial and marine and, by location, stream and beach.

The distinctive qualities of placer deposits that permit aqueous mining are the following (Daily, 1968a):

1. Material in place amenable to disintegration by the action of water under pressure (or mechanical plus hydraulic action)
2. Adequate water supply available at required head
3. Adequate space for waste disposal
4. Heavy concentrate is valuable mineral, amenable to simple mineral processing
5. Prevalent, low, natural gradient to permit hydraulic transport of mineral (if hydraulic king)
6. Can comply with environmental regulations regarding water clarity and waste disposal

The two principal methods of placer mining that lend themselves to high productivity are hydraulic king and dredging, ln *hydraulic king,* a high-pressure stream of water is directed against a placer bank to undercut and cave it (Fig. 7.1). As the bank disintegrates, the loosened placer material (mineral, sand, and gravel) is partially slurried in water and washed to a *sluice,* either a natural trough in the ground or a metal or wooden box, where it is transported by gravity to a *riffle box* or more elaborate concentrating device.

Water is directed at the bank by a hydraulic *monitor* or *giant,* a swivel mounted nozzle, which is connected to a supply pipeline or header (Daily, 1968a). One design of monitor, called an Intelligent, includes a section of curved pipe shaped in such a way that all reactive forces are absorbed in the unit, thus eliminating anchors. The Intelligent can be automated to cycle through a vertical arc of 120° and near-full-circle horizontal rotation. In operation, two or more monitors are located close to the bank but at a safe distance and positioned so that they overlap while water-jetting a complete section of bank, cutting at an angle.

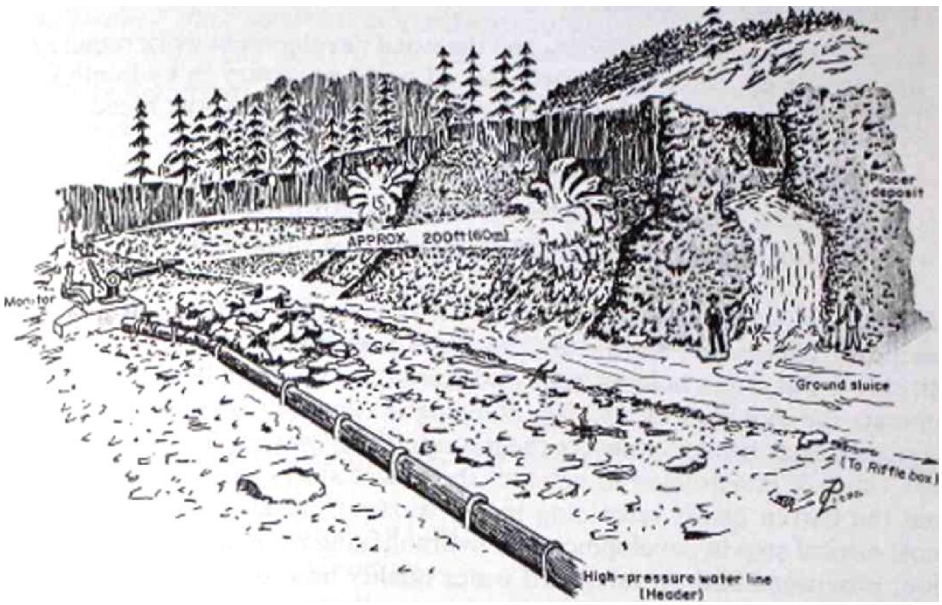


Figure 7.1., Placer mining by hydraulic king. (By permission from West Virginia Geological & Economic Survey, Morgantown.)

Normally, the bank height is 15-50 ft (5- 15 m), but it may attain 150-200 ft (45-60 m) (Morrison and Russell, 1973). Design specifications for monitors are the following:

Nozzle diameter		1.5-6 in. (40-150 mm)
Pressure head		100-450 ft (30-140 m) or 45-200 lb/in.^{2} (300-1400 kPa)
Volume flow rate		500-4000 gal/min (30-250 L/sec)
Water jet velocity: sand		30 ft/min (0.15 m/sec)
	gravel	300 ft/min (1.5 m/sec)
	boulders	600 ft/min (3.0 m/see)

Hydraulic king is considered to have an *intermediate* production rate. It reportedly originated in Russia in 1830 (Macdonald, 1983).

Sequence of Development

In addition to possible stripping and the usual development work required in surface mining, there are several special steps necessary in hydraulic king. They relate to some of the distinctive qualities of placers just listed:

1. Provision o f adequate water supply upstream
2. Location o f suitable waste disposal area downstream

3. Control o f water quality and reclamation o f surface

Stripping is usually minimal and performed by monitor or with any available stripping equipment. Preferably, water is supplied by gravity, fed in by stream or flume to a holding basin and stored at a pressure head sufficient to operate the monitors (ordinarily, pumping is ruled out because o f cost). A large-diameter header connects the basin to the monitors. Then a ground or box sluice is constructed to conduct the placer slurry a w a y for processing and the barren debris remaining to the waste disposal area. Probably the most critical step in development for hydraulic king is environmental protection; provisions both to safeguard water quality and to reclaim the surface after mining must he made. Because of past pollution abuses, California and several other western states have outlawed hydraulic king, and the method has fallen into general disuse in this country (<1 % o f surface mineral production.)

Cycle of Operations

Since both excavation and handling arc accomplished entirely hydraulically, the production cycle is an abbreviated, unified one. No prior breakage and no subsequent transport are required. In fact, water is integral to the processing as well as to waste disposal. The simplicity of the cycle and multipurpose use of water accounts for the attractive productivity and cost of the method. Auxiliary operations other than development are negligible, with reclamation incorporated into the cycle.

Conditions

The following is based on Daily (1968a), Morrison and Russell (1973), and Hoppe, (1976):

1. *Ore strength:* heavy mineral grains in unconsolidated soil or gravel matrix, few boulders.
2. *Rock strength:* unconsolidated.
3. *Deposit shape:* placer type, tabular, bank or beach.
4. *Deposit dip:* nearly flat (2-6% grade)
5. *Deposit size:* small to intermediate (thickness 15-200 ft, or 5-60 m)
6. *Ore grade:* can be very low
7. *Ore uniformity:* fairly uniform
8. *Depth:* very shallow, little overburden
9. *Other:* large quantities of water (500-4000 gal/min, or 30-250 Used at high heads (100-450 ft, or 30-140 m)

Features

The following is based on Daily 1968a and Morrison and Russell (1973):

Advantages

1. Fairly high productivity (100-300 yd\ or 75-230 m\ of gravel per employee-shift)
2. Low mining cost (relative cost: 5%)
3. Intermediate production rate
4. Low capital cost; simple equipment and cycle
5. Can automate operations

Disadvantages

1. Environmental damage severe unless elaborate protection exercised (regardless, several states prohibit)
2. Extensive water requirements
3. Limited to unconsolidated deposits that disintegrate under hydraulic attack
4. Inefficient cutting action, difficult to control

Applications and Variations

Hydraulic king is little used today for placer mining in the United States (except in Alaska) but continues in use elsewhere (mainly in Australia for gold and Southeast Asia for tin). Occasionally, it finds application for other purposes, especially stripping. Specialized uses of stripping have been removal of silt overburden in iron ore mining in Ontario (Li, 1976b), frozen or thawed muck overlying gold placers in Alaska (Daily, 1968b), and soft volcanic ashes and tuffs which form the overburden of the Bougainville copper deposit in Papua New Guinea (Thomas, 1978). In an unusual application, it is used in phosphate mining in Florida to slurry the ore matrix, a loose conglomerate (Hoppe, 1976).

Case Study. Clear Spring phosphate mine, International Minerals and Chemical Corp., Bartow. FL.

7.3 PLACER MINING: DREDGING

The dredge may have been the first continuous mining machine invented; a crude device of this type was in use in the Netherlands in 1565 (Macdonald, 1983). Dredging is the underwater excavation of a placer deposit, usually carried out from a floating vessel which may incorporate processing and wasic – disposal facilities. The body of water may be natural or man-made; depending on the sizes of the dredge and deposit, from 200 to 2000 gal/min (13 to 125 L/sec), of water may be required for both mining and processing waste disposal (Daily. 1968a : Macdonald. 1983). Once popular in the United States, there are few placer-mining dredges operating here today (except in one or two western states and Alaska), but 130 are still used in mining elsewhere in the world. Foster (1984) estimates a total of 1350 are currently active for mining and all other purposes.

Dredges are classified as follows (Turner, 1975):

A. Mechanical
1. Bucket line (endless chain of buckets revolving along ladder)
2. Bucket-wheel suction (buckets discharge in suction pipeline)
3. Dipper (shovel, grapple, or dragline mounted on barge)
B. Hydraulic
1. Suction (open intake suction line)
2. Cutter head (excavation by rotating cutter on suction line)

The bucket-line dredge is the classical, wet, continuous-excavating machine for poorly consolidated or loose materials with some boulders; hence its early application to placers. A recent, more versatile development, the bucket-wheel suction dredge, mounts buckets on a rotating wheel and discharges excavated material into a hydraulic conveyer. Dipper dredges have the disadvantages of all intermittent excavators but are able to dig tougher, more consolidated materials with boulders; their use in the United States today is practically limited to sand and gravel deposits. Hydraulic dredges were adapted to placer mining from channel excavating; the suction type is restricted to sand and gravel, while the cutter head type can manage consolidated materials. Both hydraulic dredges have limited application for placer mining because of low heavy-metal recovery and boulder restrictions although they offer more continuous excavation capability,

We direct our attention mainly to bucket-line and bucket-wheel dredges because of their suitability for placer mining. In operation, a dredge "carries" its pond with it as it excavates the placer bank ahead while depositing waste behind (fig. 7.2), Material dug by a rotating bucket line is elevated on board the dredge: the bucket ladder digs in a vertical arc up the placer bank then steps over with the aid of a spud and shore lines to a fresh face The sequence is pictured well by Stout (1980). The cut is about I ft (0.3 m) deep and 5 ft (1.5 m) wide. Dredges can excavate banks a maximum 50 ft (15 m) above water level to 160 ft (50 m) below (Morrison and Russell, 1973)

With a bucket-line dredge, mineral processing is carried out on board. A tail sluice disposes of fines while a stacker conveyor discharges coarser material. The output of a bucket-line dredge may be estimated from its design specifications.

Example 7.1. Find the low to high output of a bucket-line dredge (in yd^3/day, or m^3/day) rated as follows:

Bucket capacity	10 ft^1 (0.28 m')
Bucket line speed	22 bucket/min
Shifts	3/day, 22.5 hr total
Bucket factor	60-87%

SOLUTION

$$\text{Low output} = \frac{(10)(22)(60\ \text{min/hr})(22.5)(0.60)}{27\ \text{ft}^3/\text{yd}^3} = 6600\ \text{yd}^3/\text{day}\ (5050\ \text{m}^3/\text{day})$$

$$\text{High output} = \frac{(10)(22)(60)(22.5)(0.87)}{27} = 9600\ \text{yd}^3/\text{day}\ (7340\ \text{m}^3/\text{day})$$

As a rule of thumb, one thus can estimate the output of a bucket-line dredge as 500-1000 yd^3/day per ft3 (13,500-27.000 m^3/day per m3) of bucket capacity. A large dredge may excavate 9 million yd3 (7 million m3) of placer material per year.

Bucket-line dredging is not only capable of large production (it is a *large-scale* method of mining), but it is highly productive, perhaps the most productive of all methods. Capital costs for dredges in 1980 vary from SI 2,000 to $28,000 per yd^3/hr output ($9,000-S21,000 per m3/hr) (Macdonald, 1983).

Daily (196Kb) quotes operating costs as follows in 1962:

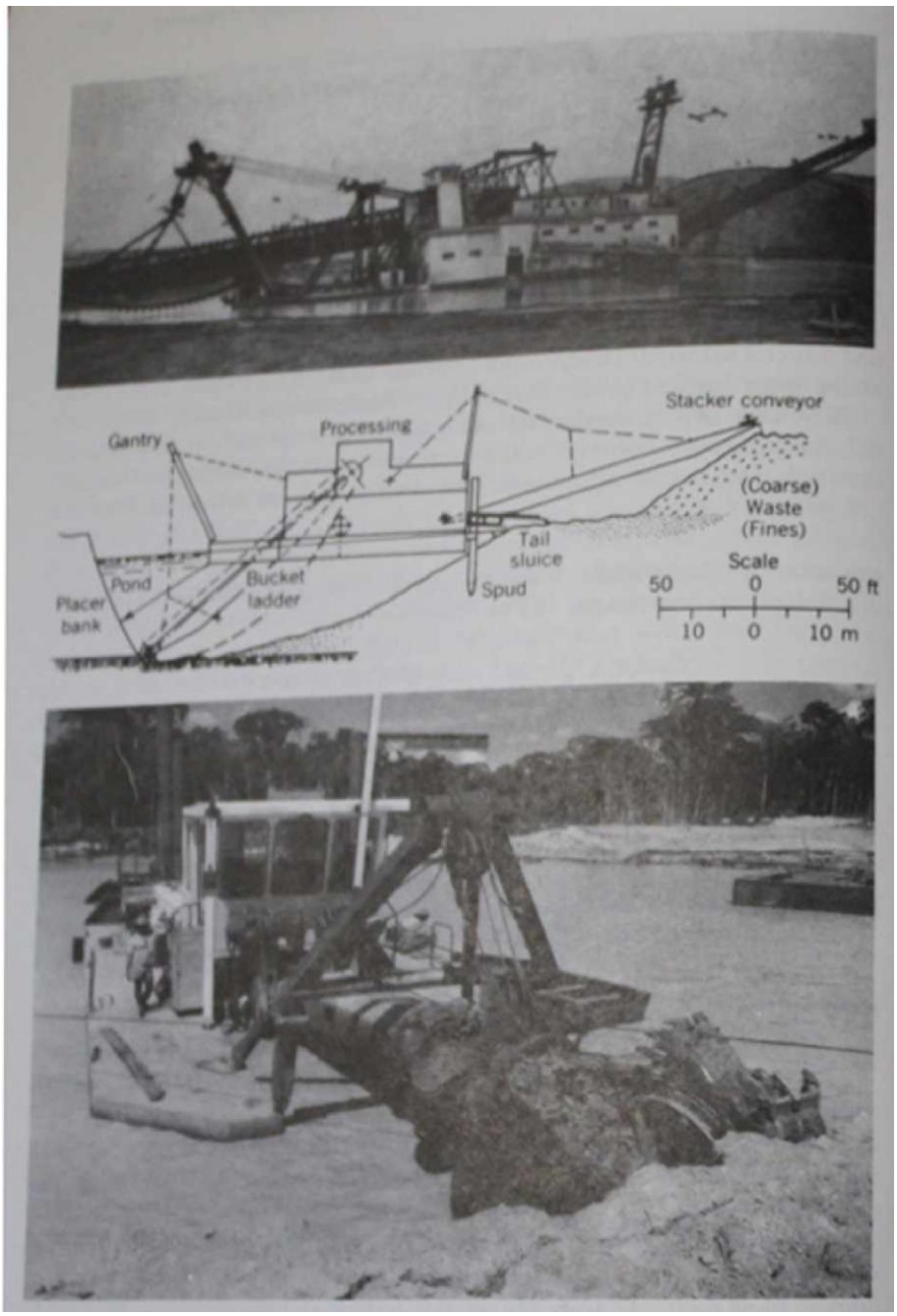


Figure 7.2. Placer mining by mechanical dredging (*Top*, *center*) Bucket line dredge (After Daily. 1966a. By permission from Society of Mining Engineers. Inc.. Littleton. CO.) (Bottom*)* Bucket wheel suction dredge (By permission from Ellicott Machine Corp.. Baltimore)

BUCKI-IT CAPACITY			OPLRATING COST
6 ft^3	(4.6 m^3)		18.0 e/yd^3 (23.5e/m^3)
7-8	(5.4-6.1)	8.3-13.8	(10.6-18.0)
12	(9.2)	7.3-18.5	(9.5-24.2)
13.5	(10.3)	5.5- 8.5	(7.2-11.1)

Even corrected for current price levels, these are remarkably low costs, probably the cheapest for all mining, particularly in view of the fact that processing costs are included as well.

The bucket-wheel suction dredge resembles and operates very much like its dry land successor, the bucket wheel excavator (Foster. 1984). It can be built more compactly and cheaply than the bucket-line dredge (but processing must be done on shore), and it holds great promise in reviving placer mining in the United States. Its output can be estimated in a manner similar to that outlined in Example 7.1. In excavating tailings, costs total about S0.75/ton (S0.83/tonne).

Hydraulic dredges (Fig. 7.3) enjoy special applications in placer mining, mainly for excavating heavy beach sands and diamond ocean placers or stripping overburden under unusual circumstances—for example, sediment and glacial till overlying the Steep Rock iron ore deposit in Ontario (Pfleider, 1973a; Turner, 1975).

Sequence of Development

There is little that differs in developing for placer mining, whether it is by hydraulic king or dredging. Again, provision of an adequate water supply is imperative (a pond is created by damming a stream or pumping water), as is suitable waste disposal and a sequenced reclamation plan.

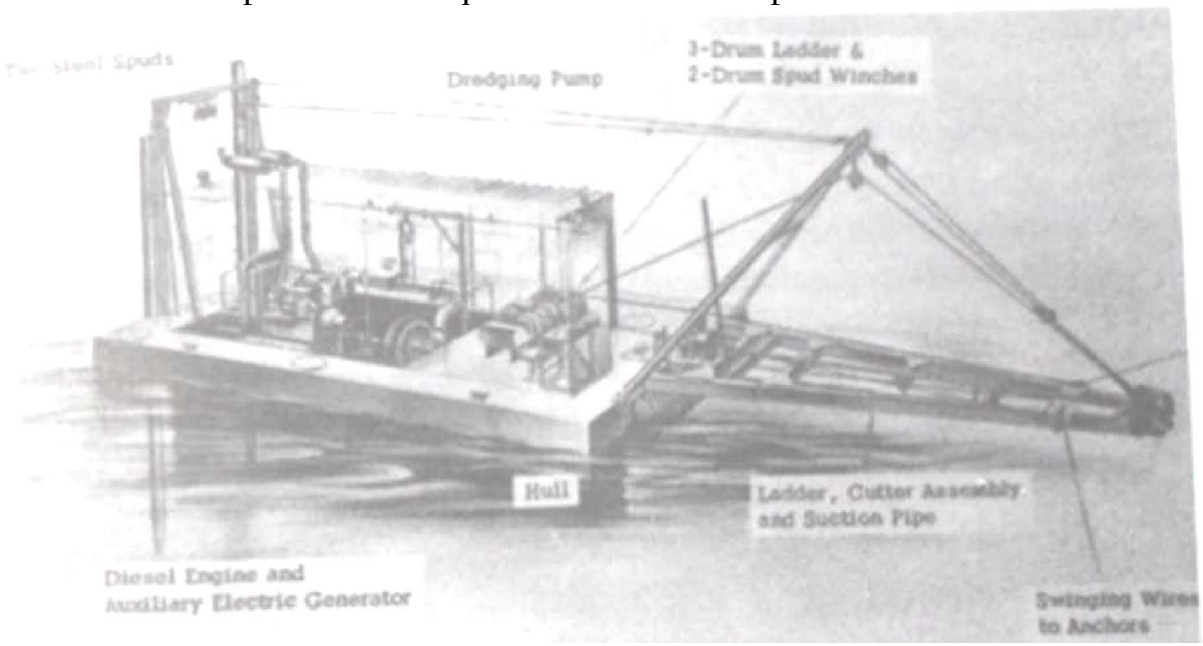


Figure 7.3. Placer mining by hydraulic dredging View of cutter head dredge. (After Daily, 1968a. By permission from Society of Mining Engineers, Inc, Littleton, CO.)

Overburden removal is minimal and done conventionally, unless overburden and placer are mined together without distinction (Pfleider, 1973a).

Cycle of Operations

Since the dredge is a type of continuous mining machines, no breaking is required, and materials' handling (excavation plus transport) is conducted without interruption. Water aids in excavation, although much of the digging is accomplished mechanically (except on suction dredge). Mineral processing is almost always wet gravity concentration, often carried out on board the dredge. Waste is dumped aft of the dredge, filling in the pond as it is excavated ahead of the dredge. Reclamation is conducted as an integral part of the operating cycle.

Conditions

The following is based on Daily (1968a. 1968b). Morrison and Russell (1973), and *Turner (1* 975):

1. *Ore strength:* mineral grains must be heavier than waste; placer-type *deposit* of unconsolidated soil or gravel matrix, with some boulders, depending on dredge type
2. Rock strength: unconsolidated
3. *Deposit shape:* placer type, tabular, bank or beach
4. *Deposit (Up:* preferably flat (maximum 2-6% grade)
5. *Deposit size:* intermediate to large (thickness 25-200 ft. or 8-60 m)
6. *Ore grade:* can be very low
7. *Ore uniformity:* fairly uniform
8. *Depth: very* shallow, little overburden
9. Other: moderate quantities of water (200-2000 gal/min, or 13-125 L/sec. at atmospheric pressure)

Features

The following is based on Morrison and Russell (1973). Pfleider (1973a), Macdonald (1983). and Foster (1984):

Advantages

1. Most productive of all methods (250-400 yd3, or 190-300 m3, of gravel per employee-shift)
2. Lowest mining cost (relative cost: <5%)
3. High production rate (maximum 9 million yd*, or 7 mi,,ion m, >
4. Low labor requirements (crew: 10-30 employees)
5. Good recovery (approaches 90%). but accompanied by high dilution
6. Continuous operation, with no breakage required

Disadvantages

1. Environmental damage severe unless elaborate protection exercised (prohibited in some states)

2. Moderate water requirements (600-800 gal/yd^3 or 1000 to 4000 L/m^3 of material mined)
3. Limited to unconsolidated deposits that disintegrate under hydraulic or combined attack
4. High capital investment with large dredges
5. Inflexible and unselective: limited to placer-type deposits

Applications and Variations

Little used today in the United States (<2% of surface mineral production), dredging finds application for placer mining in other parts of the world, mainly in South America, Southeast Asia, and the Far North (Daily 1968b; Pfleider. 1973a; Macdonald, 1983; Foster. 1984; Lewis, 1984). Nearly all placer minerals are produced by dredges. In this country, dredging is occasionally employed in mining for stripping overburden (e.g., phosphate deposits in the U.S. Southeast). Dry-land dredging is a variation of placer mining used with conventional surface excavating equipment in arid regions.

Case Study. Lee Creek phosphate mine, Texasgulf Company, Lee Creek *SC.*

Mining method: open cast surface mine, excavation by dragline

Stripping: cutter head hydraulic dredge removing unconsolidated overburden, in tandem with dragline

7.4 SOLUTION MINING: BOREHOLE EXTRACTION

As conventional ore production becomes more difficult and costly, the attractions of solution mining as a primary or secondary exploitation method increase. *Solution mining* is the subclass of aqueous surface mining methods in which minerals are recovered, usually in place, by dissolution, melting, leaching, or slurrying. (Even though certain development or exploitation on work may be carried out underground, most operations are conducted on the surface; hence we consider solution mining to be surface method.) First used 1922, and at one time limed lo the evaporites, sulfur and copper, solution mining now accounts for significant production of silver and uranium as well as over 15% of U.S. copper and 25% of the gold. It is also in the research and development stage for the exploitation of several nonmetallic, hydrocarbons, and metals (Bhappu, 1982).
While solution mining is discussed here in two categories of methods there is some overlap and basic similarity. We shall nevertheless recognize' the following as distinctive methods:

1. **Borehole mining**
2. Leaching

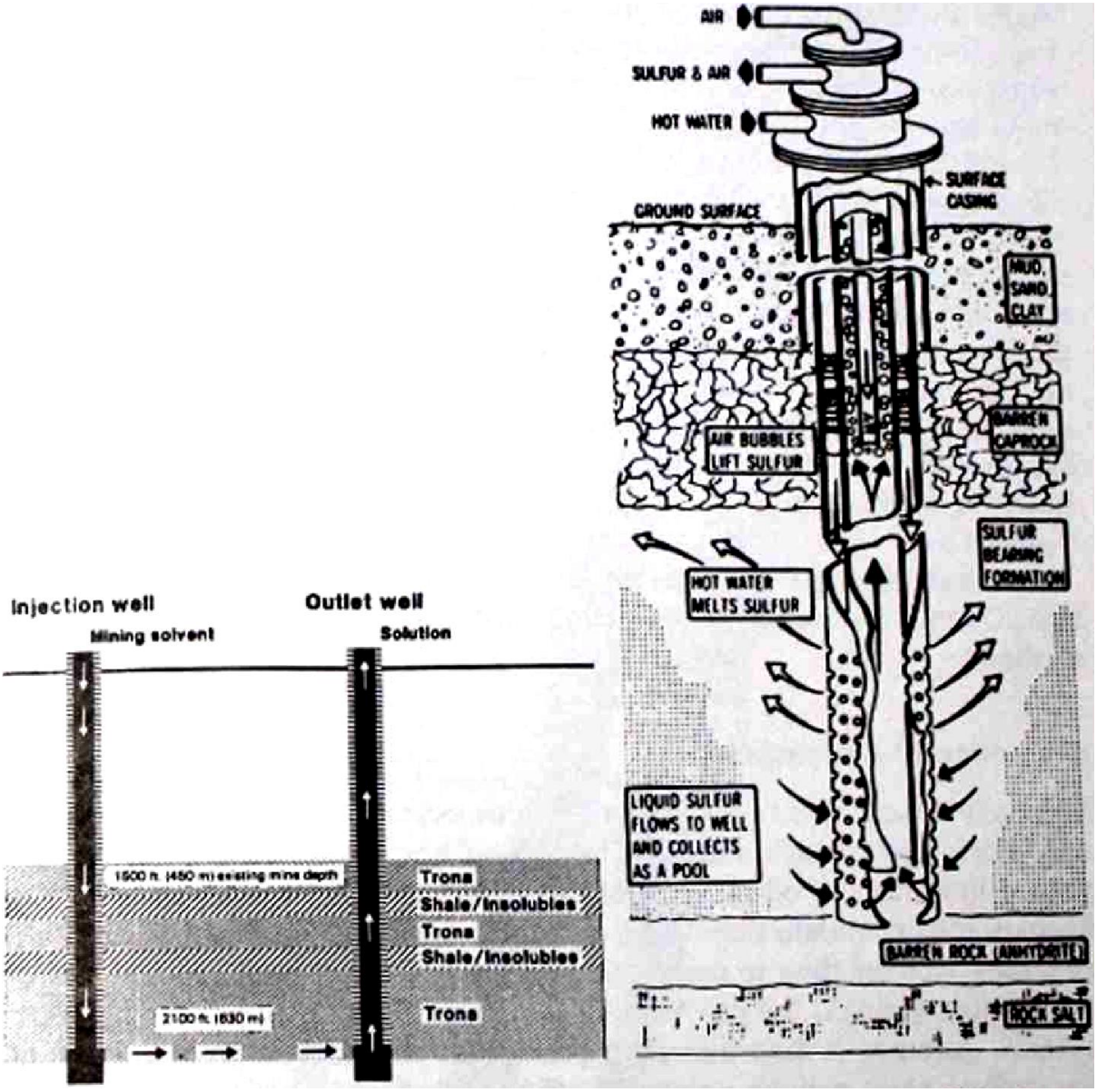
In borehole mining, water is injected by wells into a mineral formation where it dissolves, melts, or slurries the valuable mineral and is then returned to the surface through wellbores. Sometimes a reagent (solvent) is added to the water, and an aqueous solution is injected into the formation; strictly speaking, this constitutes chemical leaching, and the process should be so classified. It is acceptable also to term it borehole extraction, however, if wells are used for injection. Minerals recoverable by borehole mining include the following:

1. Evaporites: salt, potash, and trona—by dissolution
2. Sulfur—by melting *(Frasch process)*
3. Phosphate, kaolin, oil sand, coal, gilsonitc, and uranium— by slurrying (experimental)
4. Uranium and lignite—by chemical leaching

Much of the United States' salt and sulfur is produced by borehole mining, but it is responsible for *<2%* of surface mineral production. Success in borehole mining (and leaching as well) depends on several factors (Bhappu, 1982):

1. Elements of the leaching phase, such as accessibility for contact, physiochemical interaction, and transport of reactants and products to and from the surface (e.g., desirable features are impervious layers above and below the formation being mined as well as competent overburden that will not cave during mining)
2. Mineralogy and chemical composition
3. Solubility of minerals
4. Particle size (reduced size desirable for fast, thorough reaction)
5. Fragmentation or fracturing (usually not necessary in borehole mining but nearly always in in-situ leaching
6. Solution application and regeneration (recovery and reuse of reagents essential)
7. Recovery of metals or minerals from solution

In addition, environmental considerations may be critical (Hrabik, 1986). possible contamination of ground water is the major concern— and a somewhat unique one— with solution mining. In uranium mining, the risk of radioactivity escaping into an aquifer requires safeguards. Other potential hazards arc surface subsidence, normally associated with underground exploitation, and disposal of liquid and solid wastes at the surface.



Figurer 7.4. Solution mining by boreholes. *(Left)* Multiple wells for trona. (Alter Anon.. 1981c; Kostick, 1982. By permission from American Mining Congress, Washington.) *(Right)* Concentric well for sulfur, Frasch process. (After Marsden and Lucas, 1973. By permission from Society of Mining Engineers. Inc., Littleton, CO.)

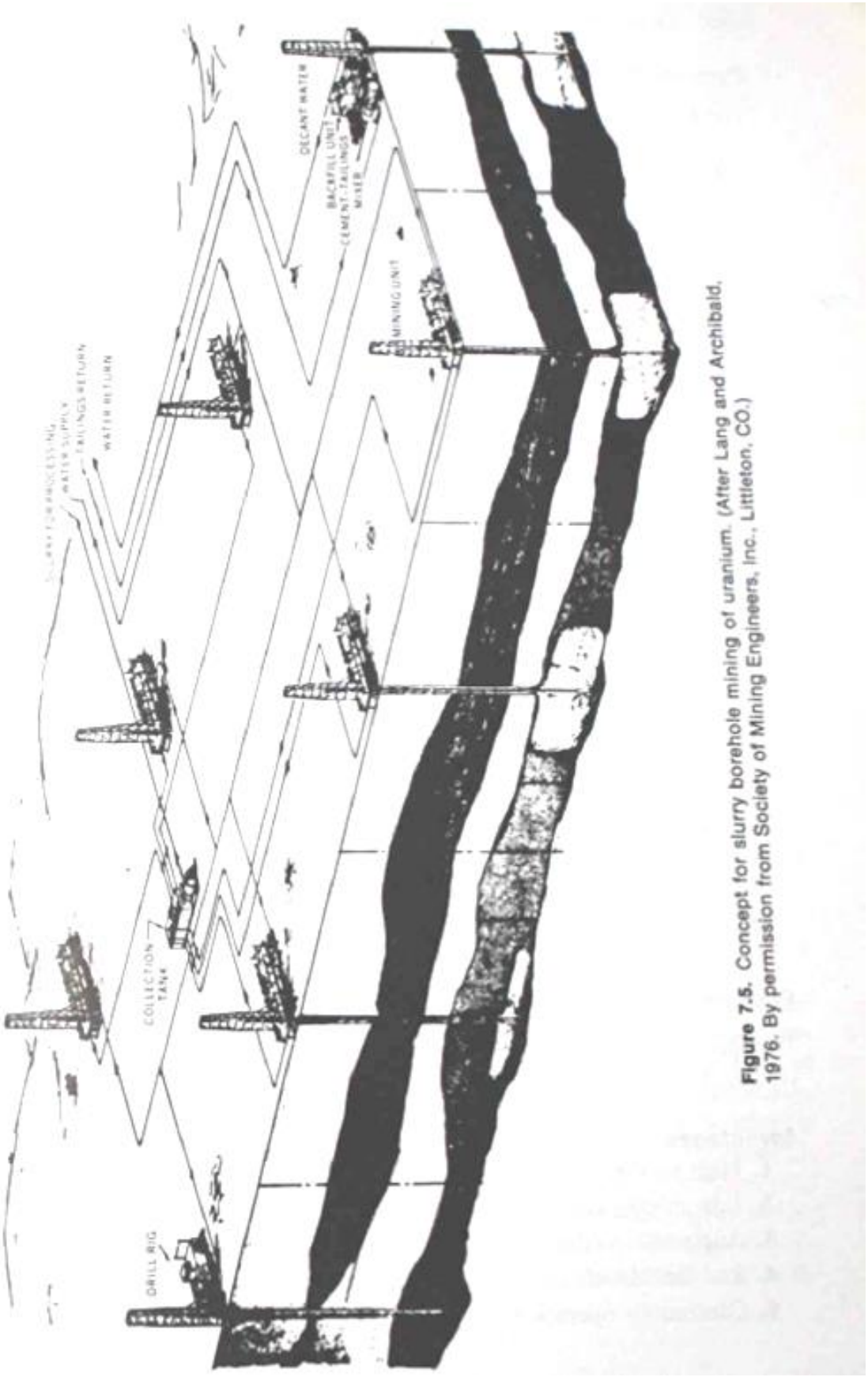


Figure 7.5. Concept for slurry borehole mining of uranium. (After Lang and Archibald, 1976. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Conditions

The following is based on Morrison and Russell (1973), White (1975), and Kostick (1982):

1. Ore strength: reasonably competent but porous and permeable; mineral must dissolve, melt, or slurry in water
2. Rock strength: overlain by competent, impervious rock
3. *Deposit shape:* any, but usually tabular, large extent
4. Deposit dip: any. but preferably flat or low'
5. *Deposit size:* moderate to large, thickness >50-100 ft (>15-30m)
6. *Ore grade:* intermediate (sulfur >20%)
7. *Ore uniformity:* variable to uniform
8. *Depth:* intermediate to great depth, <10.000 ft. or <3000 m (sulfur 200-2500 ft. or 60-750 m)
9. *Other:* moderate to large quantities of water required Features

The following is based on Marsden and Lucas (1973), Morrison and Russell (1973). Bhappu (1982). and Hrabik (1986):

Advantages

1. High productivity and moderate production rate
2. Low mining cost (relative cost: *5 %)* and labor requirements
3. Applicable to deep deposits
4. Reduced development time and cost
5. Continuous operation, with breakage usually not required

Disadvantages

15 Highly specialized method; limited to deposits that dissolve melt or slurry in water
16 Moderate water requirements (Frasch process high in energy)
17 Unselective; dilution may be high
18 Process control may be difficult, recovery low to medium
19 Possible environmental hazard, especially groundwater contamination

Applications and Variations

Most of these have already been discussed. While still limited in application, the use of borehole mining is expanding. If slurrying and leaching experiments are successful, the method may enjoy substantial growth in the future, tempered by environmental concerns (Mussey and Tyree, 1984).
Case Study (multiple wells). Diamond Crystal Sail Company, Akron, OH.
Case Study (concentric wells). Grand Ecaille sulfur mine, Freeport Sulfur Company, Grand Ecaille, LA.

7.5 SOLUTION MINING: LEACHING

Leaching is the chemical extraction of metals or minerals from the confines of an ore deposit as well as from material already excavated and mined (Schlitt. 1982). The process is basically chemical but may be bacteriological as well (certain bacteria act as catalysts to speed up the reaction in leaching sulfides). If the extraction is carried out on mineral in place, (hen it is termed *in-sit u leaching.* 11' it is performed on previously mined material in dumps, tailings, or slag piles, it is called *heap leaching.* A variation carried out in vats or tanks is termed *vat leaching.*

Heap leaching, then, is always a secondary mining method. In-situ leaching may also be secondary, but it is primary when it is conducted on a virgin deposit through boreholes or on an ore deposit which is blasted or caved especially for leaching. In either form, leaching is actually a combination process, because in addition to extraction, it accomplishes beneficiation in the first stage of mineral processing (Lastra and Chase, 1984). Consequently, production costs by leaching tend to be less than those by conventional mining and processing. One comparison (Bhappu, 1982) for copper mining estimates total production costs for an ore body by open pit mining of $5.50- $6.80/ton (S6.10-$7.50/tonne) and in-situ leaching of $3.60-$4.40/ton ($4.00- $4.85/tonne).

Applications of in – situ leaching so far are limited to copper and uranium, with gold and silver in addition being recoverable by heap leaching. Experimental study indicates that many metals, especially manganese, gold – silver, aluminum, and cobalt – nickel, are prime candidates for in – situ leaching. (Potter et al. 1982) In-situ leaching of lignite has also been investigated. (Sadler and Huang, 1981) Solvents include acid ferric sulfate for copper sulfide minerals, sulfuric acid for copper oxide, hydrogen peroxide for uranium, sodium cyanide for gold, and sodium hydroxide for lignite. Malouf lists chemical and microbial recreations for values mineral.

Factors important in leaching, such as solubility and permeability, are the same ones critical in boreholes mining. (Section 7.4) Additionally, successful in – situ leaching requires that these two criteria be satisfied. (Vandell 1982):

1. The ore deposit must located in a water – saturated zone.
2. The deposit must be confined both above and below by impervious layers.

Environmental damage from groundwater contamination is the most serious threat in in – situ leaching: criterion I above tends to raise the risk, while criterion 2 lowers it.

Heap and in – situ leaching are compared in Figure 7.6. In this instance, both are examples of secondary mining. Leach solutions are distributed (by sprays, ponds, injection holes, and ditches) at the top of the broken ore; after percolating by gravity downward through the heap or ore body, the solutions are collected at the lower extremity (in basins, ditches, or sumps) and pumped to the processing plant. In primary mining by in-situ leaching, a low permeability ore body is broken

up by drilling and blasting or caved by undercutting prior to circulating the leach solution. If the ore body is permeable it can be leached without breakage by borehole mining, similar to that depleted in Figure 7.4 (left). During mineral processing, the dissolved metal or mineral is recovered (by precipitation, called cementation), and the solvent is regenerated and revised. Leaching is highly productive and considered an inter mediate to large unit method of mining It constitutes <5*r of surface mineral production.

Sequence of Development. Development steps for heap leaching consist of the following (Marsden and Lucas. 1973):

1. Remove all vegetation in area to be utilized.
2. Compact soil base, and emplace and compact a 2-ft (0.6-m) clay layer as in impervious base for leaching.
3. Remove fines from ore to be leached.

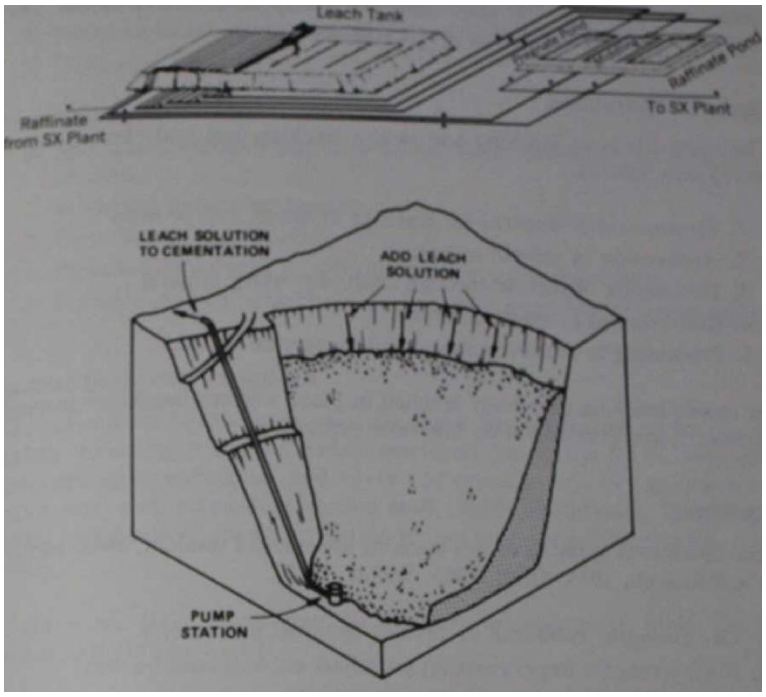


Figure 7.6. Solution mining by leaching. (Top) Heap teaching. (After Scheffel,1982) (Bottom) in-situ leaching. (Alter Crawford and Hustrulid, 1979. By permission from Society of Mining Engineers. Inc.. Littleton. CO.)

4. Spread material in heaps 400 ft (120 m) in width (maximum for natural aeration and oxidation of sulfides).
5. Remove top 2 ft (0.6 m) from dump surface (fines generated by haulage vehicles).

20 Rip dump surface in layers to a depth of 8-10 ft (2.4-3 0 m) to improve percolation of solution.

21 Leach in lifts of 50 ft (15 m), emplacing additional material after (caching of lower lift completed.

Development for in-situ leaching is similar to that for heap leaching if secondary mining, and to borehole extraction if primary through drillholes. If the ore body has

already been broken up by blasting or caving, providing collection sites for the pregnant solution after it has percolated through the deposit is the major task of development. Sometimes boreholes are adequate, but generally full – sized underground openings are required. They should be constructed during the development stage.

Cycle of Operations

The cycle for heap leaching and in – situ leaching (ore body broken up in place) is as follows:

1. Preparation o f material for leaching in dump, vat, or heap
2. Application o f solvent to heap
3. Percolation o f solvent through heap, dissolving mineral
4. Collection of pregnant solution
5. Processing to recover mineral and regenerate solvent

For in – situ leaching (ore body leached in place without breakage, through boreholes), see Section 7.4. on borehole extraction.

Conditions

These conditions apply to in-situ leaching specifically (Malouf, 1968; Morrison and *Russell, 1973; Stout,* J980: *Schlitt. 1982):*
1. Ore strength: rubblized or cavable *(porous*, permeable)
2. *Rock strength:* impervious *layers* adjacent; walls can be weak
3. *Deposit shape:* massive or *large vein or bed*
4. *Deposit dip: steep*
5. *Deposit size:* any, prefer *large*
6. *Ore grade:* can be very low
7. *Ore uniformity:* variable, minerals fine-grained
8. *Depth:* intermediate (-1000 ft, or -300 m)
9. *Other:* moderate to large quantities of water required ,

Features

The following is based on Marsden and Lucas (1973). Crawford and Hustrulid (1979), and Lastra and Chase (1984):

1. High productivity and production rate
2. Low mining cost (relative cost: *5%)* and labor requirements
3. Applicable to deep, low-grade deposits
4. Reduced development time and cost
5. Supplements primary mining
6. Perform s beneficiation of mineral as well as extraction

7. Good health and safety

Disadvantages

1. Specialized method: limited to soluble minerals that can be chemically leached.
2. Moderate water requirements
3. Unselective, high dilution
4. Process control may be difficult, recovery low to medium
5. Possible environmental hazard, especially groundwater contamination

Applications and Variations

Several of these have already been discussed. See the references for process details. Leaching is still not widely employed, but its use for the secondary recovery of copper, gold, and silver and primary recovery of uranium is substantial and increasing (Jackson ct a l,, 1980; Chamberlin. 1981- Areall 1985). Applications for other metals appear promising (Chamberlain et al., 1983).
Case Study (in-situ leaching). El Mesquite uranium mine, Mobil Oil Co Bruni. TX (Burger, 1981).
Case Study (heap leaching). Darwin silver recovery project. Anaconda Minerals Co., Lone Pine, CA (Milligan and Engelhardt, 1983).

7.6 SPECIAL TOPIC: SELECTION OF MATERIALS-HANDLING EQUIPMENT

The proper selection of equipment, carried out during the development stage, is one of the most critical decisions required in surface mining (Sections 4.4 and 4.5). Technological feasibility, economic suitability, and safety are the principal criteria that must be satisfied. Because it is central to all mine operation, the selection of equipment for materials handling and its two unit operations, excavation and haulage, is generally the first and foremost task undertaken. Equipment for other operations logically follows and complements materials handling. Since open pit mining always requires both loading and haulage in the production cycle for stripping as well as mining, we choose it as the typical surface method for which to demonstrate the equipment-selection procedure. The procedure is also applicable to coal loading and haulage operations in open cast mining, as well as to various underground methods in which materials handling of ore, coal, or stone occurs. With certain modifications, we will employ the basic selection procedure later for other equipment in the production cycles as well.

Cost estimation procedures are developed in the special topic in the following chapter (Section 8.7.)

Excavator

The machine we choose as a typical excavator to demonstrate the selection procedure is a power shovel. Although it competes with the front – end loader, dragline, hydraulic excavator, and other excavating machines, the shovel is still the workhorse of the surface mining industry. Because of that, performance data have been thoroughly measured and widely published, permitting reasonably accurate selection.

The selection procedure involves consideration of the following (Anon. 1976b).

1. *Idealized output,*. Power shovels are rated by their idealized output in yd^3/hr (m^3/h), which is a function of bucket size, type of material, working difficulty of digging, type of haulage unit and working conditions. The measured employed customarily is *bank (solid) measure,* the volume of material in place. When dealing with haulage units, however *loose (broken) measure* the volume of material after excavation, is employed. For estimation purposes, *the* following approximate measures of specific weight hold:

(Actual specific weights should be used, if the minerals arc known; e.g., with coal, use 1.1 tons/yd3, or 1.3 tonnes/m3, for bank measure; and 0.7 ton/yd3, or 0.8 tonne/m \ loose.)

Table 7.1 provides rated, idealized shovel outputs for machines in the 5- 25 yd3 (3.8-19.1 m3) range, typical of loading shovels in mining (stripping shovels and draglines used for casting are much larger; see ratings in Pflcider. 1973a). There are two categories of material: *earth* (soil) and *rock* (ore. coal, or stone). Operating conditions noted for Table 7.1 are assumed to be ideal.

2. *Operating factors.* The rated outputs obtained from Table 7.1 must be modified for “real-life" conditions. Corrections are made for three factors idealized in the table: working time, operating conditions, and rock fragmentation.

TABLE **7.1 Power Shovel Loading Output***

Bucket Capacity, yd^1 (m^5)	Earth, bank yd>/hf (rnVhr)	Rock, bank y<Я/цг (m*/hr)
5 (3.в) в (6.1) 9 (69) 10 (7.в> 15 (115) 25 (19.1)	420-605 (320-465) 600-825 (460-630) 680-930 (520-710) 750-1025 (575-785) 1140-1550 (870-1185) 1900-2500 (1455-1910)	375-500 (285-380) 490-675 (375-515) 555-770 (425-590) 615-845 (470-645) 925-1270 (705-970) 1540-2075 (1176-1585)

Assumed conditions:

1 *Specific weight, bank material:*

Earth 1.5 tons/yd1 (1.8 tonnes/yd3)

Rock 2 0 tons/yd» (2 4 tonnes/yd1)
2 *Range of digging* hard to easy (with rock, poorly to well blasted)
3 *Working time:* 60 min/hr (Job management factor - 100)%
4. *Swing* optimum (90‘)
5 *Depth o f cu t:* favorable (100%)
6 *Combined depth swing factor:* 100%
7 *Bucket factor:* 1.0
8 *Cycle:* loading into haulage units on grade
Source: Modified after Pfieider (1973a).

a. *Working time.* Actual working time may depart substantially from 100% of available time. Adjustments for job and management conditions are made as follows:

AVAILABILITY	ACTUAL TIME
Favorable	**55 min/hr, 7 hr/shift**
Average	**50**
Unfavorable	**40**

Unless otherwise specified, average availability is assumed, and 50 min/hr and 7 hr/shift are used.

b. *Operating conditions.* Operating conditions reflect a variety of working factors, such as angle of swing, depth of cut, *bucket factor* (percent of bucket capacity utilized), and loading conditions. Corrections to table outputs are made for combinations of these factors:

CONDITIONS	CORRECTION
Favorable	**80%**
Average	**70**
Unfavorable	**60**

c. *Rock fragmentation.* If the material being excavated is soil, we should express this factor as “easy, average, or hard digging.” In the case of rock, blasting is nearly always required, and the categories are related to the degree of fragmentation. The range of the effect of blasting can be read directly from the outputs in table 7.1:

FRAGMENTATION (DIGGING)	TABLE OUTPUT
Well blasted (easy)	Larger figure

Average (average)	Average value
Poor blasted (hard)	Smaller figure

Finally, in selecting the proper size of power shovel, the following rule of thumb gives an approximate indication:

$$\text{size } (yd^3) = \text{shift production (tons)}/1000 \qquad (7.1)$$

$$\text{sizc (m)} = \text{shift production (tonnes)}/1200 \qquad (7.1a)$$

The shovel size actually selected should equal or exceed the size calculated by EQ 7.1.
An example demonstrates the selec.ion procedure for a power shovel.

Example 7.2. Given are the following data for a shovel-truck open pit mine:

Required production, per bench	10.000 tons/shift (9070 tonnes/shift)
Operating period	1 shift/day
Material	Well-blasted rock
Operating conditions	Favorable

Find the size and number of power shovels required.

Solution. Assume 50 min/hr and 7 hr/shift, and choose operating factor = 80%. For fragmentation (well-blasted rock), use larger output in Table 7.1.

$$\text{Desired output} = \frac{10{,}000 \text{ ton/shift}}{7 \text{ hr/shift}} \times \frac{60 \text{ min}}{50 \text{ min}}$$

$$= 1714 \text{ tons/hr} \div 2.0 \text{ tons/yd}^3 = 857 \text{ yd}^3\text{/hr}$$

$$\text{Idealized output} = \frac{857}{0.8} = 1071 \text{ yd}^3\text{/hr}$$

From Table 7.1, *select one 15-yd-* (11-m^3) *power shovel* (rated output = 1270 yd'/hr. or 971 m^3 /hr).
Check by Eq. 7.1: 10,000/1000 = 10 < 15 yd^3 (8 < 11 m^3), which is satisfactory.
If a second bench were placed in operation, then another shovel would be required; the size of the units would have to be recalculated, based on the production specified per bench.
We choose the truck as a typical haulage unit in open pit and open cast mines because it is used far more extensively than conveyors, scrapers dozers and other conveyances. As with the power shovel, ample performance data are available to assist in the selection procedure. The following consider ations arc important (Anon., 1981a):

1. ***Size rating.*** Truck capacity ***(live load)*** is usually measured on a weight rather than volume basis to prevent overloading. Volume measure is struck capacity, based on loose (not bank) measure. Standard si2cs vary among manufacturers, but

the following ratings are representative. Trucks are divided somewhat arbitrarily into ***normal*** and ***giant*** sizes, reflecting extensive customizing in the upper range:

Normal sizes: ***22.*** 30, 35, 40. 55. 85, 100, 130 tons
(20. 27. 32, 36. 50, 77. 90, 117 tonnes)

Giant sizes: 150, 175, 200, 250, 300, 350 tons
(135, 158, 180, 225, 270, 315 tonnes)

Our interest is mainly in the normal sizes.
Unlike the procedure with excavators, there are no published tables of truck output because of wide variations in cycle time and working zone (review Section 4.5).
2. ***Operating factors.*** These factors are similar to those for shovels, but some are expressed differently.

a. ***Working time.*** The same available time estimates are made for **trucks** as for shovels. Again, 50 min/hr and 7 hr/shift are used unless other conditions are specified.

b. Optimal number of swings. This is a function of the target area of (he truck bed and the frequency of truck changes at the shovel. A proper balance is provided in these ranges:

Normal sizes 4-6 swings/truckload
Giant sizes 5-8

e. Bucket factor. The operating condition that reflects loading efficiency is bucket factor.

LOADING CONDITIONS	HUCKET FACTOR
Favorable	120%
Average	90
Unfavorable	60

d. Cycle time. For an actual mining situation, the individual time elements in the haulage cycle arc estimated and cumulated. Cycle time t is determined by

$$t = t_{de} + t_z + t_l + t_{l1} + t_d$$

where t_{de} *is* travel empty *time, t_s is spot* time at the shovel, t_l is load time, t_{l1} is travel loaded time, and t_d is dump time, all in minutes Time to spot, load and dump can he estimated or calculated readily, but travel times require *knowledge of* truck rimpull — speed characteristics and the haulage-road profile (Anon.. 1981a.) Since truck characteristics vary with individual units, we *will simplify* the selection procedure by using travel-time elements that have already been determined.

c. *Spare units. To* maintain a full haulage fleet in operation even when breakdowns occur, *spare* units are usually purchased. For every five to six production units at the mine, one spare is provided.

Truck selection procedure is demonstrated in the following example.

Example 7J. Given arc the following additional data for the shovel-truck mine of Example 7,2:

Bucket factor	*90%*
Cycle time *t*	15.0 min

Find the size and number of trucks required, based on the one 15-yd^3 (11-m^3) shovel previously selected.

SOLUTION.

Bucket capacity = (15 yd*XD.90K 1.3 ton/yd') = 17.55, say, 17.6 tons/swine expressed in loose measure.

Find truck sizes for 4 to 6 swings/load and compare with standard sizes:

4 swings x 17.6 tons/swing = 70.4 tons vs. (none available)
5 swings x 17.6 tons/swing = 88.0 tons vs. 85 ions actual
6 swings x 17.6 tons/swing = 105.6 tons vs. 100 tons actual

For a proper match of shovel and truck, a truck size within ±5% of the calculated load should be selected. (A truck size smaller than the load ensures that the truck is fully loaded, bur risk overloading.) Should no match occur with 4 to 6 swings, try 3 or 7. In this example, select 85 ton trucks, with 5 swings/truckload, as the closest match within ±5%.

Find

$$\text{Number of trips/shift} = \frac{(7\text{ hr})(50\text{ min/hr})}{15.0\text{ min}} = 23.3, \text{ say } 23/\text{truck}$$

Output/shift = (85 tons)(23 trips) = 1955 tons/truck (the smaller of the two figures, 88.0 or 85, is used in the calculation, assuming that truck overloading is not permitted.)

$$\text{Required number of trucks } n = \frac{10{,}000\text{ tons/shift}}{1955\text{ tons/truck}} = 5.12, \text{ say } 5$$

(Go to the next larger whole number if the decimal remainder is >0.2.) *Therefore, select Jive 85-ton* (*77-tonne*) *trucks and one spare unit.*

Synchronization of Loading and Haulage Units

One final check remains to be made before the choice of loading and haulage units is completed: the synchronization of the loading portion of the cycle must be investigated to be certain that (1) the shovel does not wait for the (rucks, and (2) the truck wait is not excessive (more than a few minutes). If the synchronization is unsatisfactory, then the selection of equipment must be modified or the cycle (travel, spot, or dump times) changed.
Loud time can be calculated for the equipment selected. Spot and dump times are usually estimated, based on operating conditions; values are given for both trucks and tractor-trailers (Anon., 1981a):

	SPOT TIME (MIN)		DUMP TIME, MIN	
CONDITIONS	TRUCKS	TRAILERS	TRUCKS	TRAILERS
Favorable	0.15	0.15	1.0	0.3
Average	0.30	0.50	1.3	0.6
Unfavorable	0.50	LOO	1.8	1.5

Once these time elements have been determined, and knowing the cycle time, we can check the synchronization by the following equation:

$$t \leq n(t_l + t_s) \qquad (7.3)$$

where n is number of trucks. For proper cycle balance or synchronization to exist, the cycle time for one truck, /, must be less than the time required to

PROBLEMS

7.1a. Determine the sustained production rate in a surface mine using a shovel and trucks for materials handling:

Shovel	One 9-yd^5 (7-m^J)
Trucks	Four 55-ton (50-tonne)
Material	Average blasted rock
Working	Average
Haulage cycle time	15.0 min

b. Which governs—truck or shovel output? Check cycle synchronization.

c. What improvements would you recommend in operating practice to increase mine production without changing equipment?

7.2a. Determine the sustained production rate in a surface mine, given the following conditions:

Shovel	One 5-yd' (4-т>)
Trucks	Six 30-ton (27-tonne)
Material	Well-blasted rock
Work time	55 min^hr, 7 hr/shift
Working conditions	Favorable
Haulage cycle time	14.0 min

22 Which govems-truck or shovel output? Cheek cycle synchronization.
23 What improvements would you recommend in operating practice to increase mine production without changing equipment?
24
7.3a. Determine the optimal number and size of excavating and haulage equipment (power shovel and trucks) for the following surface metal mining application.

Mining method	Open pit (single face)
Required production	16.500 tons/day (14,850 tonnes/day)
Working shifts	3/day, 7 hr/shift, 50 min/hr
Material	Well-blasted copper ore
Working conditions	Unfavorable
Bucket factor	80%
Truck cycle time	18.0 min

b. Check the shovel-lruck synchronization and suggest improvements in the operation if delays are excessive.
7.4a. Determine the optimal number and size of excavating and haulage equipment (power shovel and trucks) for the following surface coal mining application:

Mining method	Open cast (single face)
Required production	35,000 tons/day (31.500 tonnes/day)
Working shifts	2/day, 7.5 hr/shift, 55 min/hr
Material	Well-blasted rock overburden
Working conditions	Favorable
Bucket factor	85%
Truck cycle time	19.0 min

b. Check the shovel-truck synchronization, and suggest improvements in the operation if delays are excessive.

8

SURFAKE MINING: METHOD COMPARISON AND SUMMARY

8.1 METHOD RECAPITULATION

We have in the last three chapters examined in some detail the development and exploitation of mineral deposits by traditional surface mining methods. It is important now to compare and summarize principal features of these methods so we are better able to make wise decisions in the proper selection of a surface method to mining a given deposit. Finally, when we complete our study of underground methods as well, repeat the summarization process, and then compare costs, we should be able to select the optimal method to mine any deposit.

In Section 3.7. we encountered the cardinal rule of mine expiloitation concerned with method selection (“ within the limits imposed by safety, technology, and economics") and considered an exhaustive list of factors that determine the outcome. They were grouped in six categories:

1. Spatial characteristics of deposit
2. Geologic and hydrologic conditions
3. Geotechnical properties
4. Economic considerations
5. Technological factors
6. Environmental concerns

In our study of surface mining methods, we attempted to identify critical factors in the selection process and relate them to individual methods.

8.2 SEQUENCE Of DEVELOPM ENT

Repeating from Table 3.1 the portion of the classification of mining methods pertaining to surface mining, eight methods have been introduced ("Percent U.S. Use" refers to U.S. surface mineral production):

Class	Subclass	Method	Percent U.S.USE
Mechanical	-	Open pit mining	60%
		Quarrying	< 1
		Open cast mining	30
		Auger mining	< 2
Aqueous	Placer	Hydraulicking	< 1

	Dredging	< 2
Solution	Borehole mining	< 2
	Leaching	< 5
		- 100%

Employing this outline, which is also the format for the *Mining Methods Notebook* (Fig. 3.4). we will compare the cight methods as to development sequence, cycle of operations, conditions, and characteristics and finally summarize their importance in modern-day mining.

8.2 SEQUENCE OF DEVELOPMENT

Chapter 3 established a general sequence of mine development applicable to both surface and underground locales and to all mining methods, and in Section 3.2 presented a list of II steps in the normal sequence. Those aspects of development which are distinctive and critical to the various methods, however, are fewer in number, consisting for surface mining of step 5 (environment and reclamation), step 8 (waste disposal), and step 10 (overburden removal and access openings). For discussion, they may be grouped conveniently by the class of mining methods, mechanical extraction and aqueous extraction.

Environment and Reclamation

Although underground mining is characterized by certain environmental problems of its own such as surface subsidence and a life-threatening atmosphere. they are dwarfed by the environmental complexities of surface mining. Paramount of these are winning approval of an environmental impact statement and obtaining the necessary mining permits, stockpiling topsoil if required, solid waste management, prevention of air and water pollution during mining operations, and concurrent or subsequent restoration of the surface and revegetation if required. Environmental regulations, in the main, impact the mechanical extraction methods. In open cast mining of coal, land

reclamation assumes the proportion of a major auxiliary operation, integral to the *mining cycle, and constitutes a significant* cost item. Less stringent *laws make reclamation less critical in the* open pit mining and quarrying of *noncoal deposits,* but other environmental concerns *such as air*-and water- *quality* protection are equally demanding.

Aqueous surface methods vary in the extreme in their environmental onsequences. The placer mining methods, hydraulicking and dredging, require even more attention to environmental protection and reclamation of the surface than open cast mining (unless,in fact, their use is entirely legislated against at the state or local level). On the other hand, solution mining methods are more sparing of the environment-borehole mining especially and leaching somewhat-by disturbing the

surface less, but risk ground-water contamination. Water supply requirements with all aqueous extraction methods are high, however, and may constitute a separate environmental concern.

Waste Disposal

Provision of adequate waste disposal facilities is inherently a major task during development in most surface mining, particularly when mechanical extraction is used, and may constitute a solid-waste environmental problem as well. Overburden and other waste materials associated with the ore (or coal or stone) deposit being mined must be excavated, transported, and disposed of in a manner and location that interferes minimally with production operations. Mineral processing as well generates waste in the form of crushed rock or tailings (any water runoff complicates disposal and environmental protection). Open pit mining as a method has the most demanding waste disposal requirements, often necessitating separate facilities for lean ore, rock, and soil. One of the major attractions of open cast mining is the disposal of overburden in mined-out areas, eliminating haulage and dedicated disposal sites (although grading to original contour is required). Quarrying generally has modest waste disposal requirements (little overburden is moved), and augering has none. Placer mining may require substantial waste disposal areas, since both overburden stripping and waste rejection during mineral processing arc generally involved. In contrast, little waste disposal is associated with solution mining, except for liquid waste in borehole mining (often treated and recirculated or disposed of in deep wells) and provision of dump space for heap leaching (may have to be reclaimed) and water treatment facilities.

Overburden Removal and Access Openings

By its very nature, surface mining is conducted through excavations that are large in extent and open to the atmosphere. If surface access is not readily available, it must be provided through artificial openings, generally accomplished by the stripping of overburden.

Mechanical extraction methods normally require the greatest amount of developmental excavation to uncover and gain access to the mineral deposit. Overburden and rock adjacent to the deposit must be stripped and disposed of, some prior to and some during the exploitation stage. Open pit mines grow ever deeper and have to contend with more and more burden maintaining economic stripping ratios during mining requires careful planning and ore grade control (Section 5.4). Open cast mines must coordinate stripping and mining with some precision and often must contend with the recovery of several mineral seams simultaneously. Dimension stone quarries generally can afford to remove little overburden (or because of near-vertical walls, require less stripping), and auger mines by their nature remove none.

Aqueous extraction methods normally require overburden removal only in the case of placer mining, if at all. Stripping economically and technologically must be

kept to a minimum in either hydraulicking or dredging. In solution mining, stripping is not necessary for borehole extraction and only modestly employed in leaching, if at all.

8.3 CYCLE OF OPERATIONS

The basic cycle of operations for surface mining varies widely. It varies with the material handled (consolidated vs. unconsolidated, as well as overburden vs. ore, dry vs. wet, etc.) but mainly with the mining method. Both production operations and auxiliary operations in surface mining are heavily mechanized and closely identified with specific equipment. The two major methods— open pit and open cast mining— usually employ a so-called standard cycle (drill + blast + load + haul), while the other, more specialized methods utilize modified and somewhat unique operations.

Production Operations

Table 8.1 summarizes the variety of unit operations, both standard and modified, which comprise production cycles in surface mining (equipmentwas pictured in Figs. 5.8a-e.) Common cycles of operation for the principal surface methods consist of the following combinations (see symbols in Table 8.1):

MECHANICAL EXTRACTIO N

Open pit mining	Ala + A lb + A le + A2a + A2b
	B1 + A2a + A2b
Quarrying	B2a + B2b + A2c + A2b

AQUEOUS EXTRACTION

Hydraulicking	B 5(slurry)
Dredging	B 5 (slurry)
Borehole mining	B5 (slurry, solution, fusion, solvent
Leaching	B5 (solvent)

(Note : In the open pit and open cast methods, stripping and mining may be performed by different cycles of operation. To a much lesser extent stripping may be required for quarrying, hydraulicking, and dredging Stripping is not necessary for other surface methods.)

Auxiliary Operations

Auxiliary tasks performed in surfacc mining arc listed in their entirety in System 6-2. Those peculiar to or most important in surface mining include the following:

1. Reclamation
2. Slope stability
3. Haul road construction and maintenance
4. Equipment maintenance

5 . Pit drainage
6. Communications
7 . Power distribution
8. Dust control, noise control, safety, etc.

8.4 GEOLOGIC AND NATURAL CONDITIONS

A summary comparison of the geologic, spatial, and natural conditions conducive to the use of a particular surface mining method is given in Table 8.2. This is not an exhaustive list, nor is it definitive (see the more complete discussion of individual methods in Chapters 6 and 7). It is useful, however, in establishing the range of conditions for which surface mining is applicable as well as comparing differences for individual methods. Certain general observations are appropriate with regard to the applicability of surface methods:

1. Surface mining is versatile, especially with regard to ore and rock strengths, deposit dip, and ore grade, but it is restrictive with regard to deposit shape and size, ore uniformity, and depth (both absolute and based on stripping ratio).
2. It is ideally suited for large, flat-bedded (or massive) deposits that are laterally extensive and thick (or moderately so) and that occur near the surface (within depths of a few hundreds of feet or meters) 3. It is not well suited for deposits that are small, thin, or nonuniform, plunge steeply, or occur at great depth (except in borehole mining).
4. Mechanical extraction methods are the more conventional, widely applicable, easily initiated, and flexible to modify.
5. Aqueous extraction methods are cheaper and more suitable for small deposits and variable grades, but are limited seriously in application to deposits that are susceptible to aqueous attack and for which a copious supply of water is economically available. Because the aqueous methods are so highly specialized, they commonly do not compete with the mechanical extraction methods in general application-and, for that matter, mechanical methods do not often compete with one another.

8.5 ADVANTAGES AND DISADVANTAGES

Characteristics, both advantages and disadvantages, of the various surface mining methods are compared in Table 8.3. The terms employed in the table intentionally are relative, not absolute (for more definitive statements, sec Chapters 6 and 7). Elaboration of the characteristics that were selected for comparison follows:

1. *Mining cost:* unit, overall mining cost on a relative basis; low in surface mining (except quarrying)
2. *Production rate:* ranges from small-scale to large-scale
3. *Productivity:* degree of labor intensity, measured in output per employee per unit of time; high, except in quarrying

4. *Capital investment:* disadvantageous if large, but also indicative (with item 3) of suitability for mechanization, remote control, and automation
5. *Development rate:* ease and speed of development; advantageous in obtaining economic return on investment
6. *Depth:* only borehole mining unlimited (practically) by depth (oil wells reach 6 mi, or 10 km); other methods face ultimate economic (stripping ratio) or technological (absolute depth) limits
7. *Selectivity:* ease of rejecting unprofitable material during mining (leaving in place, sorting and hauling to waste, etc.); may achieve but at unacceptable cost
8. *Recovery:* generally high and readily controllable in mechanical extraction. less so in aqueous methods
9.Dilotion: tendency for dilution to increase with recovery in surface mining, can manage in mechanical extraction by selectivity
10. *Flexibility:* some adaptability and variability in method highly desirable, as conditions change; low to moderate range
11. *Stability of openings:* in surface mining, evident in pit slopes (goal is steep slopes that stand without maintenance over life of miner moderate problem compared to underground mining
12 . *Environmental risk:* reflects concern over air, water, and solids pollution, as well as surface restoration and land reclamation; may be most venous problem in surface mining
13 . *Waste disposal:* balance against selectivity; excessive waste generation costly, difficult to dispose of, and detrimental to environment (may require rehabilitation)
14. *Health an d safety:* final and most critical criterion in selecting method; all surface methods inherently safer than underground methods
15. *Other:* miscellaneous characteristics or features of various methods

8 6 IMPORTANCE AND SUMMARY

We have previously stressed the importance of surface mining in U.S. mineral production (it accounts for about 85% of all marketable tonnage produced. as well as *95%* of all material excavated, ore and waste). When viewed against the decisive criteria of safety, technology, and economics, surface mining appears the easy winner over underground mining. All other considerations being equal, surface methods arc the logical choice to mine any suitable ore body or mineral deposit. The key word in such a sweeping conclusion is *suitable.* Unfortunately, not all deposits arc suitable for surface mining, (.imitations which singly or jointly may preclude surface methods are excessive depth, high capital investment. poor selectivity, high environmental risk or land reclamation costs, or prohibitive waste-disposal requirements. Other factors which may contribute to an adverse decision regarding surface mining are insufficient reserves, unacceptable dilution, adverse climate or precipitation, or excessive water requirements.

Nonetheless, surface mining has established an unequaled record of accomplishment. In one year, Bingham Canyon open pit copper mine produced 32 million tons (29 million tonnes) of ore and S4 million tons (76 million tonnes) of

waste; at rated capacity, it produces 463.000 tons/day (420.000 tonnes/day) total material. The current grade is a lean 0.58Я (11.6 lb ton, or 5.3 kg/tonne) copper; the pit depth exceeds 0.5 mi (0.8 km), and its maximum horizontal dimension is 2.3 mi (2.7 km). In surface coal mines epths of 300 ft (90 m) are not uncommon, the deepest-the Pittsburg &Midway mine al Kemmerer, WY-having reached 700 ft (210 m). Stripping *ratios now attain* 5 yd^3/ton (4 m^3/tonne) in metal mines and a staggering 45 yd^3/ton *(38 m^3/tonne) in coal* mines. Recovery can approach *100%* and dilution 0% *in surface mining, with proper* grade-control measures. Productive ties *considering waste* as well as ore output, range from 100 to 400 tons (90- 360 tonnes) per *employee-shift in open pits (metal* and nonmetal) and from 500 to 1000 tons (450-900 tonnes) in open cast mines (coal). And direct *mining costs with the cheapest* aqueous extraction methods are measured in cents per ton (tonne).

For the present, in the interest of fairness as well as correctness, we must withhold judgment in selecting the mining method best suited for a given ore, coal, or stone deposit. Only after we have examined underground mining in equivalent detail can we make an impartial, accurate decision in weighing all candidate methods, surface as well as underground.

8.7 SPECIAL TOPIC:COSTESTIM ATION OF MATERIALS-HANDLING EQUIPMENT

The procedure presented here for cost estimation is applicable to materials-handling equipment as well as to other equipment in the production cycle. both surface and underground. We demonstrate it for the same types of machines selected in Section 7.6—power shovels and trucks— and in continuation of the numerical examples presented there.

Our cost estimation procedure is not a precise calculation but a close estimate of expected costs. It is intended to provide a valid comparison between candidate machines being considered for a mine task, tentatively chosen by the equipment selection process employed in Section 7.6. Our objective is to determine the unit cost in S/ton ($/tonne) of materials handling—excavation and haulage—but to do that, we develop costs per unit time, usually in $/hr. for both operating and ownership. *Ownership costs* are fixed and consist of depreciation, interest, tax. insurance, and storage. *Operating costs* are variable (although some are calculated as though they are fixed) and include tires (if a rubber-tired vehicle), maintenance, fuel or power, lubrication, and labor. As is customary when dealing with individual operations of the production cycle, direct costs only are considered.

The cost estimation procedure employed is based on that developed by *U.S.* manufacturers *of* earth-moving and materials-handling equipment and detailed in their publications (e.g.. Anon., 1976b. 1981a). A form is utilized to simplify calculations that has been modified to apply to a variety of mining equipment (Table 8.4). Note that ownership costs are fixed because they accrue whether the machine is used or not, whereas operating costs vary with the service (time period, type of use, operating conditions, ets.). We can understand the cost estimation

procedure by following the use of the form for each machine and then take up numerical examples that are continuations of Examples 7.2 to 7.4.

Excavator (power shovel)

A.Ownership costs
1.Depreciation

a. Purchase price: Estimats as \$110,000/yd^3(\$145,000/m^3) of bucket capacity.
b. Salvage valui Recoverable; estimate as 15% of purchase price.
C. *Freigh t: Estimate shovel weight* as 54.000 lb/yd ^{3}(32.000 kg/ *m ^{1} o f bucket capacity. Estim ate typical freight chargc as* \$4.00/ *cwt (per* hundredweight) (\$8.80/100 kg).
d. *Unloading and moving cost:* Estimate as *10% of freight* cost.
e. *Delivered price:* Sum *of purchase price, freight* cost, and unloading cost, *less salvage* value (rounded *off* to nearest \$1000).
f. *Operating period:* Assume 2000 hr/yr for 1 shift/day operation (K *hr/shift x* 5 day/wk x 50 wk/yr = 2000 hr/yr).
g. *Economic life:* Based on Internal Revenue *Service* guidelines, the following economic lifespans of excavators may be used (Anon., 1976b): Dragline l—4| ydJ 9.000-19,000 hr (>3.8)

The values reflect a range of conditions from unfavorable to favorable. To find *n*, the economic life in yr, divide life in hr by operating period in hr/yr.
h. *Depreciation:* Calculate from delivered price and economic life; units: \$/yr.
2. *Interest, taxes, insurance, and storage*
a. *Rate:* Sum estimated fixed rates (interest based on prevailing cost to borrow money); use overall rate *8%* low to 18% high.
b. *Average annual investment rate:* Allow for effect of compound interest in borrowing money by using this equation:

$$\text{Rate} = \frac{(n + 1)}{2n}\ \% \qquad (8.1)$$

c. *Average annual investmen t:* Calculate from delivered price and average annual investment rate.

d. *Annual fixed charge:* Calculate from average annual investment and fixed rate (interest, taxes, etc.).
e. *Fixed charge:* Calculate from annual fixed charge and operating period.
3. *Total ownership costs.* Sum of items Ih and 2e (depreciation and fixed charge).
B . *Operating costs*
1. *lire replacement cost.* N/A (no tires on shovel).
2. *Tire repair cost.* N/A.

3. *Rep airs, maintenance.* Repair cost is computed as though it is a fixed cost, a percentage of depreciation cost (item 1h). Percentage factors are based on type excavator and type service (Anon 1976b):

CONDIT IONS	DRAGLINE REPAIR FACTOR	SHOVEL R E PAIR FACTOR
Favorable	60%	65%
Average	65	70
Unfavorable	70	75

4. *Fuel or power.* Most mining shovels are electric; therefore, this calculation is a power cost, computed from the shovel power rating, load factor, and unit power cost. Estimate shovel power as 55 hp/yd3, converted to kW (54 kW/m3). Select load factor based on operating conditions (Anon., 1976b):

CONDIT IONS	SHOVEL R E PAIR FACTOR
Favorable	70%
Average	60
Unfavorable	50

5. *Lubrication* . Estimate as 1% of fuel consumption. Calculate as though excavator is diesel-powered, and assume fuel consumption is 0.059 gal/hp-hr (0.30 L/kW-hr) for 100% load factor. Base on cost of lubrication oil in $/gal ($/L)
6. *Auxiliary fu e l.* N/A (for auxiliary engines).
7. *Lab o r.* Base on wage rates for operator, oiler, and helper (assume crew of 2 on excavators <50 yd^3, or 38 m^3, and 3 on excavators >50 yd3, or 38 m3), and assume 35% fringe benefits.
8. *Total operating costs.* Sum of items 1-7.
C. *O overall ownership and operating costs.* Sums of items A and B.
D. *U nit cost.* Calculate from total hourly cost divided by hourly production

EXAMPLE. For the 15-yd-^{3} (11-m^3)power shovel selected in Example 7.2. estimate the total hourly cost and unit cost. Assume operating conditions average Power cost =7f/kW -hr. lubrication oil = $4.00/gal ($1.0 6 / L) operator *wage =$l6.00/hr, and oiler wage m $l2.00/hr.*
SOLUTION . See *Table* 8.5. *Follow the* procedure *outlined* above. Calculations requiring *explanation arc summarized below (coded to Tables 8.4 and 8.5).*
A. Ownership costs
la . Purchase price =5 yd^3 x *$110.000/yd' = $1.650,000*
b. Salvage value =0.15 x *$1.650.000 - $247.500.*
c. Weight=15yd^3x54,000 lb/yd^1=810,000 lb. Freight=810,000
Ib x *($4.00/100 lb)* = $32.400.
d. Unloading and moving cost = 0.10 x $32.400 - $3240.
c.Delivered price - a - b + c + d - $1.438.140. say $1.438.000.

f .For operating period of 1 shift/day, use 2000 hr/yr.
g. For average conditions, use economic life of 18.000 hr +2000 »
9 yr.
2. All calculations arc self-explanatory.
B. Operating costs
3. Select repair factor = 70% (average conditions). Repair cost *
0.70 x $79.89 = $55.92/hr.
4. Power rating = *15 yd* ' x 55 hp/yd- = 825 hp x 0.746 = 615 kW.
Select load factor =60% (average conditions). Power use = 615
kW x 0.60 = 369 kW. Power cost = 369 kW x $0.07/kW-hr =
$25.85/hr.
5. If diesel, fuel oil consumption = 0.059 gal/hp-hr x 0.60 x 825 hp =
29 gal. Lubrication oil cost =29/100 gal/hr x $4.00/gal = $1.16/hr.
C. *Overall ownership and operating costs* = A + B = $272.58/hr
D. *Unit cost:* hourly production = 10.000 tons/shift + 8 hr = 1250 tons/
hr: unit cost = ($272.58/hr)/(1250 ton/hr) = $0.22/ton ($0.24/tonne)
Haulage Unit (truck)
A. *Ownership costs*
I. *Depreciation*
a. *Purchase price* : Estimate as $7650/ton ($8430/tonne) of truck
capacity (live load).
b. *Salvage value:* Same as for excavator.
c. *Freight:* Estimate truck weight (dead load) as 1400 lb/ton (700
kg/tonne) of truck capacity (live load). Use freight charge as
$4.00/cwt ($8.80/100 kg).
d. *Unloading and moving cost:* Same as for excavator.
e. *Delivered price:* Same as for excavator.
f. Operating period Same as for excavator, but subtract tire cost before rounding off (see item B1
g. Economic life; Base on type machine and type service, in hr (Anon.,1981a)

FRONT-END CONDITIONS	*DOZER*	*SKRAPER*	*LOADER*	*TRUCK*
Favorable	*15,000*	*16,000*	*12,000*	*25,000*
Average	*12,000*	*12,000*	*10,000*	*20,000*
Unfavorable	*10,000*	*8,000*	*8,000*	*15,000*

Calculate n for excavator.
h. depreciation; same as for excavator.
2. Interest, taxes, insurance, and storage; Same as for excavator.
3. Total ownership costs; Same as for excavator
B. Operating costs
1. Tire replacement cost. Estimate tire cost as 5% of truck purchase price. Select tire life on operating conditions, in hr (Anon., 1981 a)

FRONT-END TRACTOR-

CONDITIONS	*SKRAPER*	*LOADER*	*TRAILER*	*TRUCK*
Favorable	*5500*	*4000*	*8000*	*4000*
Average	*3500*	*3000*	*5000*	*3200*
Unfavorable	*2500*	*2000*	*3500*	*2500*

2. Tire repair cost. Base on tire cost and operating conditions. Select tire repair factor as follows (Anon., 1981a):

CONDITIONS	T IRE	REPAIR	FACTOR
Favorable	12%	12%	
Average	15	15	
Unfavorable	17	17	

3. *Repairs, maintenance.* Base on depreciation cost and repair factor, selected for operating conditions (Anon., 1981a):

FRONT-END CONDITIONS	*TRACTOR-DOZER*	*SKRAPER*	*LOADER*	*TRAILER*	*TRUCK*
Favorable	70%	42%	45%	30%	*37%*
Average	110	50	55	35	*45*
Unfavorable	150	62	70	45	*60*

Correct for economic life of machine if other than 10,000 hr. 4 *Fuel or power* Estimate haulage-unit power as 10 hp/ton (8 2 kW/ tonne) of truck capacity. Select fuel consumption, in gal/hp-hr (L/kW-hr), based on operating conditions (Anon., 1981a):

FRONT-END CONDITIONS	*TRACTOR-DOZER*	*SKRAPER*	*LOADER*	*TRAILER*	*TRUCK*
Favorable	0.034	*0.025*	*0.020*	*0.017*	*0.014*
	(0.173)	*(0.127)*	*(0.102)*	*(0.086)*	*(0.071)*
Average	0.040	*0.030*	*0.026*	*0.023*	*0.020*
	(0.203)	*(0.152)*	*(0.132)*	*(0.017)*	*(0.102)*
Unfavorable	0.046	*0.035*	*0.031*	*0.029*	*0.026*
	(0.234)	*(0.178)*	*(0.158)*	*(0.147)*	*(0.132)*

5. *Lubrication.* Base on fuel cost, selecting lubrication oil factor for operating conditions (Anon., 1981a):

CONDITIONS	LUBRICATION	FACTOR
Favorable 4	1	
Average i	1	
Unfavorable i	1	

6. *Auxiliary fuel.* Same as for excavator.
7. *Labor.* Assume 1 operator/truck. Base on wage rate and 35% fringe benefits.
8. *Total operating costs.* Same as for excavator.
C. *Overall ownership and operating costs.*
1). *U nit cost.* Same as for excavator, except that cost of active units in fleet is based on item C (ownership + operating costs), and cost of standby units is based on item A (ownership cost).

EXAMPle 8.2. For the five 85-ton (77-tonne) trucks selected in Example 7.3, estimate the total hourly cost and unit cost. Assume operating conditions average, except truck and tire life unfavorable. Diesel fuel cost = $1.00/gal (S0.26/L) and operator wage = $ 12.00/hr.
SOLUTION . See Table 8.6 . Follow the procedure outlined above. Calculations requiring explanation arc summarized below (coded to Tables 8.4 and 8.6).
A. *Ownership costs*
la. Purchase price = 85 tons x $7650/ton = $650,250.
b. Salvage value = 0.15 x $650,250 = $97,538.
c. Weight = 85 tons x 1400 lb/ton = 119,000 lb.
Freight =119,000 lb x $\frac{\$4.00}{100}$ = $4760.
d. Unloading and moving cost = 0.10 x $4760 = $476.
e. Tire cost = 0.05 x $650,250 = $32,512 (enter B l).
Delivered price = a - b + c + d – B1
= $525,436, say $525,000.
f. For operating period of 1 shift/day, use 2000 hr/shift.
g. For unfavorable conditions, use economic life of 15,000 hr -*• 2000 = 7.5 yr.
2. All calculations are self-explanatory.
B . *Operating costs*

1. Select tire life = 2500 hr (unfavorable conditions Tire cost = = $13.00/hr. 2500 hr
2. Select tire repair factor = 17% (unfavorable conditions). Tire repair cost = 0.17 x $13.00 = $2.21/hr.
3. Select repair factor = 45% (average conditions). Repair cost = 0.45 x $35.00 x = $23.63/hr. 10,000 hr
4. Power rating = 10 hp/ton x 85 tons = 850 hp. Fuel consumption = 0.020 gal/hp-hr x 850 hp = 17.0 gal/hr. Fuel cost = 17.0 gal/hr x $1.00/gal = $17.00 hr.
5. Select lubrication oil factor = J (average conditions). Lubrication cost = i x $ 17.00/hr = $5.67/hr.
C. *O verall ownership and operating costs* = A + B = $ 139.50/hr
D. *U nit cost.* Total costs:
Fleet in use: 5 trucks @ $139.50 = $697.50/hr Standby: I truck @ 61.79 = 61.79
Total haulage cost: 5759.29/hr <i>759 29
Unit cost = p 5Q t0nS'hr = ^ ^ ton ($0.67/tonne).
PR O BLEM S

8.1 Estimate the overall costs in dollars per hour of owning and operating the excavating and haulage equipment selected in Problem 7.3. Also determine the combined materials handling unit cost in dollars per ton. Following are additional data needed for cost estimation:
General
Operating period
Freight
6000 hr/yr, 3 shifts/day
$4.00/cwt ($8.80/100 kg) Tire cost = = $13.00/hr. 2500 hr
2. Select tire repair factor = 17% (unfavorable conditions).
Fuel consumption = 0.020 gal/hp-hr x 850 hp = 17.0 gal/hr.
Tire repair cost = 0.17 x $13.00 = $2.21/hr.
3. Select repair factor = 45% (average conditions).
Repair cost = 0.45 x $35.00 x = $23.63/hr. 10,000 hr
4. Power rating = 10 hp/ton x 85 tons = 850 hp.

Fuel cost = 17.0 gal/hr x $1.00/gal = $17.00 hr.
5. Select lubrication oil factor = J (average conditions).
Lubrication cost = i x $ 17.00/hr = $5.67/hr.
C. *O verall ownership and operating costs* = A + B = $ 139.50/hr
D. *U nit cost.* Total costs:
Fleet in use: 5 trucks @ $139.50 = $697.50/hr
Standby: I truck @ 61.79 = 61.79
PROBLEMS
8.1 Estimate the overall costs in dollars per hour of owning and operating the excavating and haulage equipment selected in Problem 7.3. Also determine the combined materials handling unit cost in dollars per ton. Following are additional data needed for cost estimation:

General	
Operating period	
Freight	6000 hr/yr, 3 shifts/day
	$4.00/cwt ($8.80/100 kg)
Excavator	
Power rating	600hp (448kw)
Power cost	*6/kW-hr*
Load factor	*60%*
Lubrication cost	*$4.20/gal ($1.12/L)*
Labor cost (wages)1 operator	*$15,50/hr*
1oiler	*$11.20/hr*
Haulage Unit	
Tire cost (4)	*$22,000*
Power rating	*450 hp(336k W)*
Fuel cost	*$1.05.gal ($0.28/L)*

Labor cost (wages); 1 driver	*$12,50/hr*

8.2 Estimate the overall costs in dollars per hour of owning and operating the excavating and haulage equipment selected in Problem 7.4. Also determine the combined materials *handling* unit cost in dollars per ton. Following are additional data needed for cost *estimation:*

General	
Operating period	4000 hr/yr, 2shifts/day
Freight	*$* 4.00/cwt (*$8.80/100kg*)
Excavator	
Power rating	1375 hp (1026 kW)
Power cost	4/k W-hr
Load factor	65%
Lubrication cost	*$3.90/gal ($1.04/L)*
Labor cost (wages)1 operator	*$16,30/hr*
1oiler	*$12.15/hr*
Haulage Unit	
Tire cost (4)	*$76,000*
Power rating	*2000 hp(1492k W)*
Fuel cost	*$1.15.gal ($0.30/L)*
Labor cost (wages); 1 driver	*$13,40/hr*

9

UNDERGROUND MINE DEVELOPMENT

9.1 NATURE AND SCOPE OF TASK

Role of Underground Mining

If the appeal of surface mining lies in its mass production and minimal-cost capabilities, then the attraction of underground mining stems from the variety and versatility of its methods to meet conditions too demanding and extreme for surface exploitation. True, underground mining cannot compete with surface mining today in its share of U .S. mineral production (15% vs. 85%). Underground methods account for only 4% of the nonmetallic ores, 13% of the metallics, and 42% of the coal being produced (review Table 5.1). But the United States depends heavily on underground mining for certain strategic minerals: All or most of its fluorspar, lead, potash, trona, and zinc come from underground mines, plus a significant part of its bituminous coal, gold, molybdenum, salt, and silver. Regardless, then, of present status and past trends, it seems safe to conclude that (1) underground mining still occupies an essential role in mineral exploitation, and (2) no diminution in application is forseeable.

While it is risky to attempt to predict trends and the future, indications in fact seem to favor a return of underground mining to the prominence it formerly held (Section 3.6). Reasons include (1) increasing deposit depths, (2) reduced mobility of large surface machines, (3) ever-tightening environmental constraints, and (4) promising advances in underground rock-boring and continuous mining equipment. W e have only to remind ourselves that the ultimate technological limit in all mining is depth, and that underground exploitation effectively postpones the inevitable. (Economics, of course, may impose a shallower limit than technology but never a dapeer one.)

Uniqueness of Underground Access

In preparing a mineral deposit for exploitation, development in underground mining requires certain considerations that surface-mine development does not (Section5.1). A review of the governing factors discussed in Section 3.2 indicates the least concern for locational factors (climate, in particular can almost be neglected, unless the mine requires heating or cooling). The most critical are natural and geologic factors: ore and rock strength, the presense of groundwater, and the rock-temperature gradient must be evaluated carefully (terrain in less important, because the surface plant is less extensive in underground mining). Social-economic-political-environmental factors can pose a plethora of problems in underground mining; a more skilled labor force must be recruited, financing may be more difficult because of the higher risk involved, and subsidence must be controlled.

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

Of the 11 steps comprising the general sequence of development listed in Section 3.2, all are broadly applicable to and usually performed in underground mining. One unique environmental feature— carried out as an auxiliary operation—is the necessity to provide an artificial atmosphere as a means of life support for the miners. The mine ventilation system utilizes access and production openings to distribute fresh air of the quality and quantity desired to all working places. Other than that requirement, underground development openings provide

access to the mineral deposit in the broad sense, permitting entry of miners and materiel (equipment, supplies, power, and water) as well as egress for the product mined and any attendant wastes.

On occasion, underground development openings double for exploration purposes, and vice versa. Those openings driven in advance of mining can provide valuable exploration information and afford suitable sites for exploration drilling and sampling. Likewise, openings constructed for exploration purposes sometimes can be utilized later as development workings; in Figure 2.13. for example, the exploration shaft and drifts would almost certainly serve subsequently to open up the deposit.

Types of Underground Openings

Underground development openings are ranked in three categories, by order of importance:

1. Primary: *Main openings* (e.g., shaft, slope)
2. Secondary: *Level or zone openings* (e.g., drift, entry)
3. Tertiary: *Lateral or panel openings* (e.g., ramp, crosscut)

Generally, they are driven in this same rank order to connect the surface with exploitation (production) openings located underground; that is, the progression is from (1) main development to (2) level or zone to (3) lateral or panel. Geographic boundaries are not sharply delineated, and the rankings may overlap in an operating mine.

Even more than surface mining, underground mining employs a distinctive nomenclature and lexicon. Grouping terms in categories assists us in learning them. Following are listings of the terms most commonly used to describe underground workings both for development and exploitation (Lewis and Clark, 1964; Thrush, 1968; Hamrin, 1982).

Deposit and Spatial Terms

1. *Back:* Roof, top, or overlying surface of an underground excavation
2. *Bottom :* Floor or underlying surface of an underground excavation
3. *Capping:* Waste material overlying the mineral deposit

4. *Country rock:* Waste material adjacent to a mineral deposit

5. *Crown pillar:* Portion of the deposit overlying an excavation and left in place as a pillar

6. *Dip :* Angle of inclination of a deposit, measured from the horizontal; also *pitch* or *attitude*

7. *Floor:* Bottom or underlying surface of an underground excavation

8. *Footwall:* Wall rock under the deposit

9. *Gob:* Broken, caved, and mined-out portion of the deposit

10. *Hanging wall:* Wall rock above a deposit

11. *Pillar:* Unmined portion of the deposit, providing support to the roof or hanging wall

12. *Rib:* Side wall of an excavation; also *rib pillar*

13. *Roof:* Back, top, or overlying surface of an excavation

14. *Sill pillar:* Portion of the deposit underlying an excavation and left in place as a pillar

15. *Strike:* Horizontal bearing of a tabular deposit at its surface intersection

16. *Wall rock:* Country rock boundary adjacent to a deposit

Directional Terms

1. Breast: Advancing in a near-horizontal direction; also the working face of an opening[1]
2. Inby: Toward the working face, away from the mine entrance
3. Outby: Away from the working face, toward the entrance
4. Overhand: Advancing in an direction[1]
5. Underhand: Advancing in a download direction[1]

Excavation Terms

*1.Adit:*Main horizontal or near-horizonal underground opening, with single access to the surface

2. Bell: Funnel-shaped excavation formed at the top of a raise to move bulk material by gravity from a stope to a drawpoint

3 *Bleeder:* Exhaust venlilalion lateral

4.*Chule: O*pening from a drawpoint. utilizing gravity flow to direct bulk material from a bell or orepass to load a conveyance

5 *Crosscut :* Tertiary horizontal opening, often connectmg drifts entries or rooms; oriented perpendicularly to the stike of a pitching deposit; also *breakthrough*

6 Decline- Secondary inclined opening, driven downward to connect levels, sometimes on the dip of a deposit; also *declined shaft*

7 *Drawpoint:* Loading point beneath a stope, utilizing gravity to move bulk material downward and into a conveyance, by a chute or loading machine; also *boxhole*

8. *Drift:* Primary or secondary horizontal or near-horizontal opening; oriented parallel to the strike of a pitching deposit

9. *Entry:* Secondary horizontal or near-horizontal opening; usually driven in multiples

10. *Finger raise:* Vertical or near-vertical opening used to transfer bulk material from a stope to a drawpoint; often an interconnected set of raises

11. *Grizzly:* Coarse screening or scalping device that prevents oversized bulk material from entering a material transfer system; constructed of rails, bars, beams, etc.

12. *Haulageway:* Horizontal opening used primarily for materials handling

13. *Incline:* Secondary inclined opening, driven upward to connect levels, sometimes on the dip of a deposit; also *inclined shaft*

14. *Laterial:* Secondary or tertiary horizontal opening, often parallel or at an angle to a haulageway, usually to provide ventilation or some auxiliary service

15. *Level:* System of horizontal openings connected to a shaft; comprises an operating horizon of a mine

16. *Loading pocket:* Transfer point at a shaft where bulk material is loaded by bin, hopper, and chute into a skip

17. *Longwall:* Horizontal exploitation opening several hundred feet (meters) in length, usually in a tabular deposit

18. *Manway:* Compartment of a raise or a vertical or near-vertical opening intended for personnel travel between two levels

19. *Orepass:* Vertical or near-vertical opening through which bulk material flows by gravity

20. *Portal:* Opening or connection to the surface from an underground excavation

21. *Raise:* Secondary or tertiary, vertical or near-vertical opening, driven upward from one level to another

22. *Ramp:* Secondary or tertiary inclined opening, driven to connect levels, usually in a downward direction, and used for haulage

23. *Room :* Horizontal exploitation opening, usually in a bedded deposit

24. *Shaft:* Primary vertical or near-vertical opening, connecting the surface with underground workings; also *vertical shaft*

25. *Slope:* Primary inclined opening, usually a shaft, connecting the surface with underground workings

26. *Slot:* Narrow vertical or inclined opening excavated in a deposit at the end of a stope to provide a bench face

27. *Stope:* Large exploitation opening, usually inclined or vertical, but may also be horizontal

28. *Sublevel:* Secondary or intermediate level between main levels or horizons, usually close to the exploitation area

29. *Transfer point:* Location in the materials-handling system, either haulage or hoisting, where bulk material is transferred between conveyances

30. *Tunnel:* Main horizontal or near-horizontal opening, with access to the surface at both ends

31. *Undercut:* Low horizontal opening excavated under a portion of a deposit, usually a stope, to induce breakage and caving of the deposit; also a narrow *kerf* cut in the face of a mineral deposit to facilitate breakage

32. *Winze:* Secondary or tertiary vertical or near-vertical opening, driven downward from one level to another

Classification of General Purpose Openings. General-purpose development openings may be classified by their orientation and whether they are used primarily in coal or metal/nonmetal mining:

INCLINATION	M E T A L/N O N M E T A L	COAL
Vertical and near-vertical	Shaft (I) Raise (3) Winze (3)	Shaft (I)
Horizontal and near-horizontal	Adit (I) Tunnel (I) Drift (2) Crosscut (3)	Drift (1) Entry (2) Crosscut (3)
Inclined and steeply inclined	Decline (2) Incline (2) Ramp (3)	Slope (1)

Figures in parentheses refer the rank order importance of the openings. To illustrate the purpose and application of development openings in underground mining, many of the more common are depicted in the idealized layout of a noncoal mine Figure9.1. (Another metal mine layout showing principal development openings appeared in Fig. 3.1.) Compare Figure 9.1 with Figure 9.2. showing three alternative types of main development openings for a coal mine.

Finally, we may note the frequency of use of various development opening from statistics compiled by the U.S. Bureau of Mines (Martens, 1982). In a recent year (1980), 1.036.000 ft (316 km) of underground openings were driven in metal/nonmetal mines. Of this amount. 82% were horizontal openings(drifts, crosscuts, adits, or tunnels), and 18% were vertical or steeply inclined (shafts, ramps, or raises, of which 16% were raises).

9.2 MINE DEVELOPMENT AND DESIGN

Because of the complexity and expense of underground mining, extreme care must be taken with decisions during development that may also affect subsequent production operations. There are several crucial matters that require early consideration, and these largely determine the nature of mine development and mine layout preparatory to actual exploitation.

Mining Method

Once the decision has been made to use underground mining, attention focuses on the selection of an exploitation method. Development cannot proceed until a mine production plan has been adopted, and the first step is

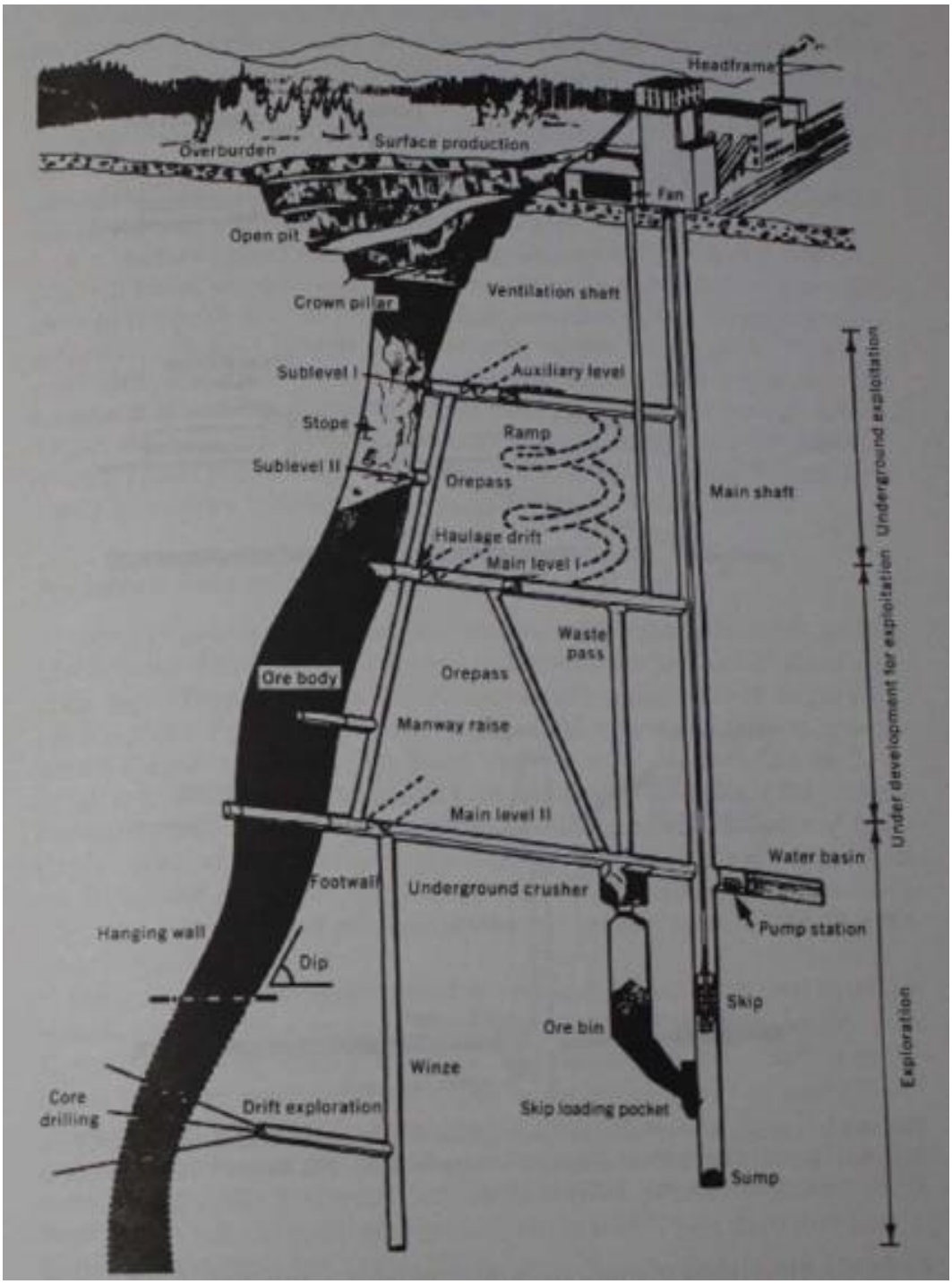


Figure 9.1. Layout of an underground noncoal mine, identifying openings, working places, and stages in the life of the mine. A surface mine that underwent transition to underground is also shown. (After Hamrin. 1982. By permission from Society of Mining Engineers, Inc., Littleton. CO.)

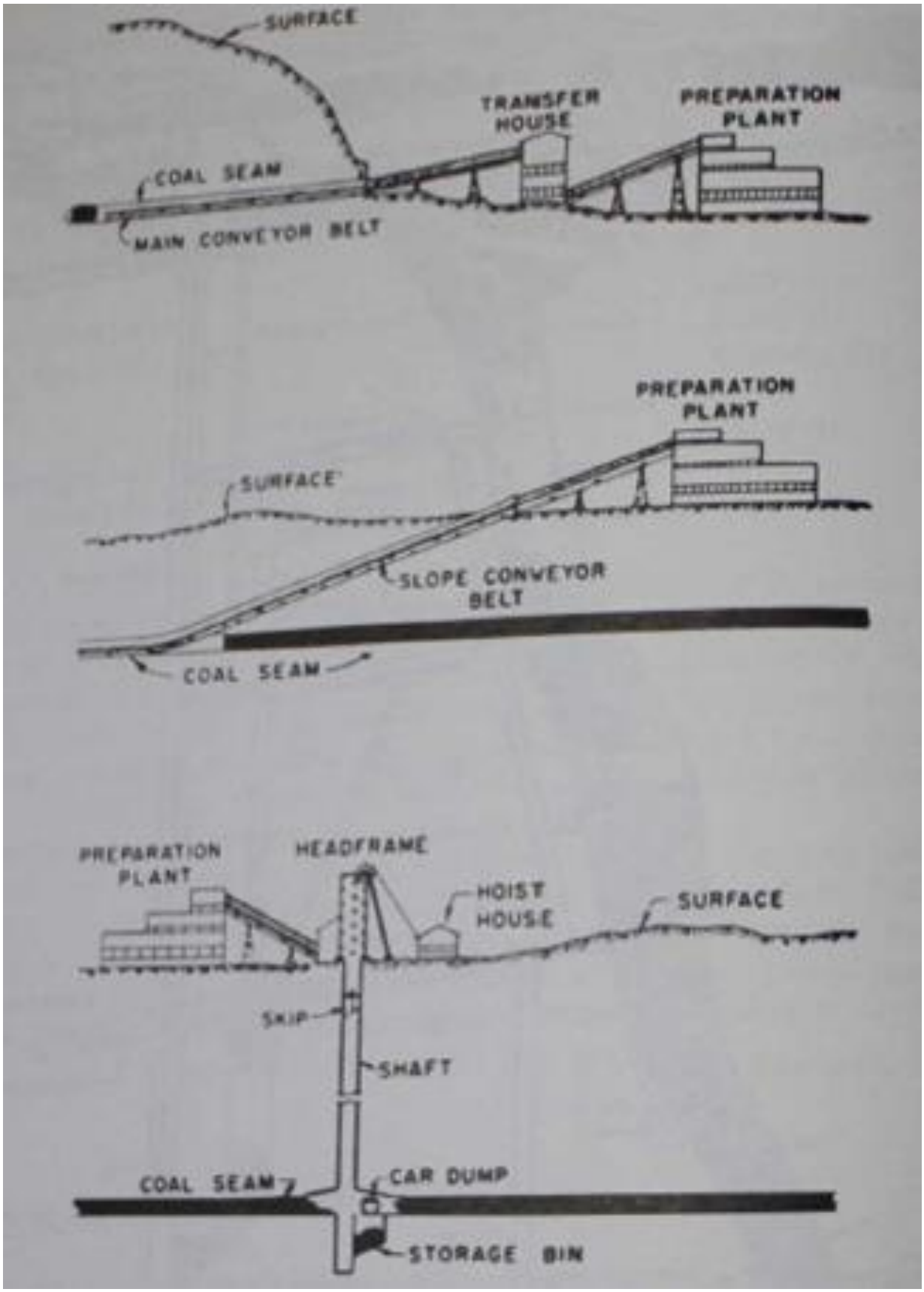


Figure 9.2. Layouts of main access openings for an underground coal mine. *(Top)* Drift.(*Center*) Slope. (*Bottom*) Shaft (After Stefanko and Bise. 1983. By permission from Society of Mining Engineers. Inc.. Littleton. CO.)

to decide which class of methods is most suitable: unsupported, supported, or caving (Morrison and Russell. 1973). As we have discussed (Section 3.7) and shall confirm again, the method selection process hinges largely on natural and geologic conditions related to the mineral deposit and on certain other factors. Selection of at least the class if not the method itself is necessary at this point to allow development planning to proceed.

The reason that the choice of method is so crucial is that it largely governs the placement of the primary development openings. If disturbance of the surface due to subsidence is anticipated—which is inevitable with caving methods and may occur with other classes of methods— then all main access openings must be located outside the zone of fracture and collapse bounded by the *angle of draw.* If the integrity of the ground overlying the active mining area can be assured for the life of the mine, then the primary openings can be located more centrally above the deposit.

A secondary consideration involving mining method is whether exploitation will occur on the *advance* or *retreat;* that is. production commences close to the point of access and advances outward, or it originates near the property or deposit boundary and retreats to the access point. This is a particularly important decision in mining a flat deposit of large lateral extent because it allects the placement of the main development openings (Obert, 1973c). Advance is favored for economic reasons (there is a faster inflow of revenues from production), but retreat is preferred for safety reasons (it is nearly mandatory with longwall coal mining in the United States).

Production Rate and Mine Life

A variety of geologic and economic conditions determines the optimum rate of production from a mineral deposit of known reserves, and hence the life of the mine. They include market conditions and selling price of the product, mineral grade, development time, mining costs, means of financing, governmental support and taxation policies, and a number of lesser factors.

Basic to all modern mine evaluations and design concepts is the requirement to optimize the *net present value*, that is. to operate the mineral property in such a way that the maximum internal rate of return is generated from the discounted cash flows (Bullock, 1982a). The optimum production rate is selected on the basis of maximizing the net present value of the after-tax cash flows.

Consider a mine being developed in a mineral deposit, with total costs and revenues plotted for a range of assumed daily production rates as shown in Figure 9.3. The optimum rate cannot be calculated directly, so it is determined from the resulting curves. The question is: Which is appropriate? Three cost curves and a revenue curve arc plotted in the graph. The correct curve to use is the net present value (revenue minus operating cost), because it not only considers revenue but the time value of money (Dowis, 1982). From Figure 9.3, the optimum production rate is 1150 tons/day (1040 tonnes/day), not 1050 (950) nor 1300 (1180).

All other things being equal, the higher the mine production rate, the shorter the mine life. Formerly, life spans were measured in decades. With the higher cost of borrowed money and greater investment risks, mine lives today are often limited to years. While low-grade, high-tonnage deposits are still financed with a life expectancy of 15-30 years, it is not uncommon for a high grade, low-tonnage deposit (e.g., gold or silver) to be designed for a two- to four-year life span (Glanville. 1984). Because of the relatively high cost of labor and borrowed money, however, the trend in the industry still is toward large mines and high production rates. Nilsson (1982b) explores the reasons and concludes that the trend will continue.

Main Access Openings

Several initial decisions related to the mode of primary development must be nude They concern (I) the type, location, number, shape, and size of main openings), as well as (2) the design of the materials-handling system. The two matters are interrelated, and a satisfactory resolution requires that they be considered together. Factors influencing the decisions are the depth, shape, and size of deposit; surface topography; natural and geologic conditions of the ore (or coal or stone) and

overlying waste; mining method; and production rale (Peete, 1941). The process is simplified somewhat by the fact that the choices are not unlimited. However, wise decisions made at the outlet avoid later changes in development openings, changes that are always disruptive and expensive.

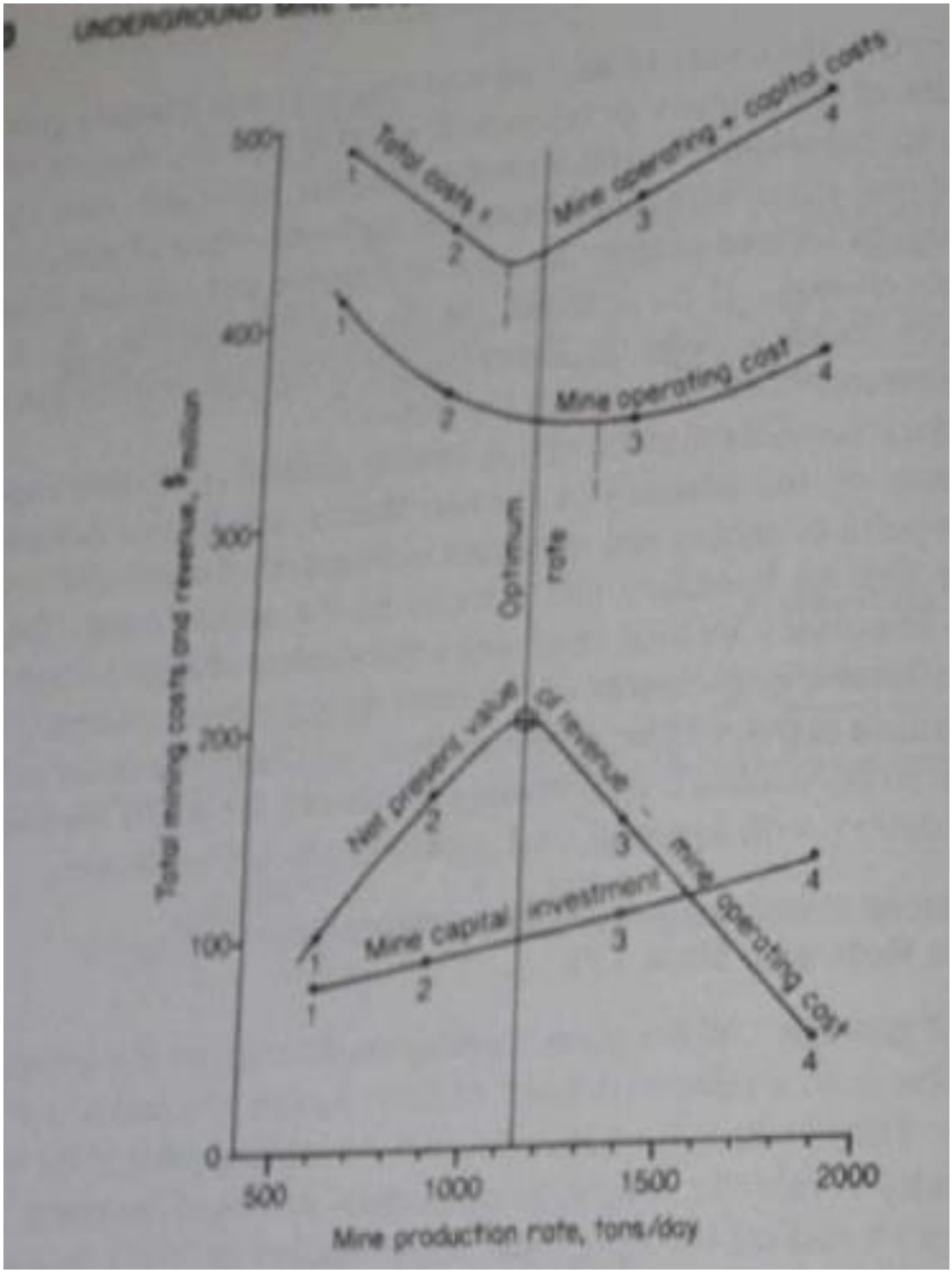


Figure9.3 Determination of optimum mine production rate from plot of net present value of revenue minus operating cost in 1900 dollars (After Dowis, 1982 By permission from Northwest Mining Assoc Spokane)

Type of Openings. Concerning type of openings, there arc only three c h o i c e s : vertical shaft, slope (inclined shaft), and drift (adit). They are shown in figures 9.2 and 9.7. A shaft is preferred for large, deep, or flat deposits, for intermediate to poor natural conditions, or for high production rales, and is ncariy always used in conjunction with skip hoisting A slope or decline is a frequent choice for shallow, flat deposits, especially coal, for which belt-conveyor or truck haulage throughout the mine is planned (asteeply inclined shaft is not often used because hoisting is slow and inefficient).

A drift or adit is employed when the deposit outcrops, usually in an area of high relief and shallow cover, or dips steeply and can be intersected at relatively shallow depth. If the situation permits consideration of more than one type of opening, then a decision is based on costs, operating as well as capital, and development time. While a shaft is most expensive to construct per foot (meter) of advance and a drift the cheapest, a slope must be three to five times longer to attain the same depth as a shaft (a drift is not a candidate if any depth must be traversed). In one study, construction costs per vertical foot (meter) were calculated and compared over depth for a similarly sized shaft and slope in Figure 9.4; while unit costs were less for the slope, unit costs per foot (meter) of depth and total costs were less for the shaft for depths greater than 580 ft (170 m). or roughly 600 ft (200 m). A complete analysis requires consideration of operating costs as well, and while the crossover point may shift, the use of overall generally does not alter the outcome (Stefanko and Bise,1983). Secondary considerations include construction time (usually less for a shaft) ahnd natural conditions (if adverse, a shft is faroved).

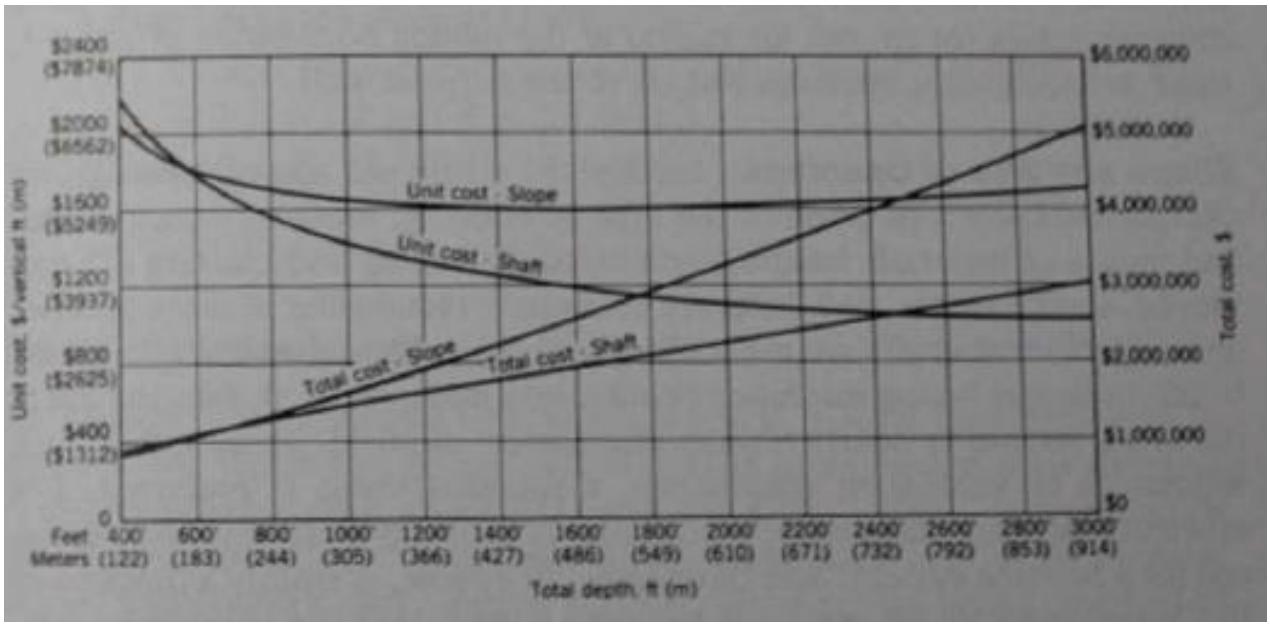


Figure 9.4. Comparison of calculated construction costs, unit and total, for shaft and slope Shaft 20 ft. (6.1 m) diameter. Slope: 240 ft* (22 m*) area. 15* inclination. (After Bullock 1982a. By permission from Society of Mining Engineers, Inc.. Littleton, CO.)

Location of openings. All other things being equal, the location of the primary development openings should be central for the deposit being mined. An old concept is to locate the main opening so it intersects the deposit at its centroid to minimize haulage costs, but Thomas (1978) stresses that other factors (dip, ground, conditions, number of working places, ets.) may outweight the importance of balanced haulage distances. Surface topography, transport, drainage, and property lines are also constraints; they may favor a particular locations for the surface plan the hence for the collar (opening) of the primary access (Obert,1978 c). Outweighing all considerations, however, as discussed initially, is the choice of mining method and whether caving and subsidence is expected-if so, protection of the main development opening becomes the overriding concern, and it will have to be located outside the affected area. In any event, a protective, unmined zone (pillar) around the shaft or main opening is always maintained.

Number of Openings. The number of openings is a function primarily of legal requirements for safety and ventilation and secondarily of convenience of access to underground workings (if scattered), production rate, and spatial relation of the mineral deposits (if more than one). At least two major, separate access openings must be provided for safety purposes in U.S. mines, according to *CFR* Title 30 (Anon., 1984a). For adequate ventilation, at least two openings arc also required; ideally, they should be widely separated and located at the extremities of the property to permit unidirectional airflow, the most efficient arrangement (Hartman, 1982a). A common practice in large mines. especially those that are extensive horizontally, is to drive air raises (or to sink air shafts) at the mining boundaries as needed; these are secondary openings and serve the purpose well.

Shape and Size of Openings. Finally, the shape and size of openings are characteristically a function of the type of opening, mode of construction, and means of materials handling employed. If drilling and blasting arc employed. shafts, slopes, and drifts are commonly rectangular in cross section, although circular shafts are more often sunk now. If mechanical excavation (shaft- or tunnel-boring machine or continuous miner) is used, then the opening cross section is

nearly always circular or elliptical. From either rock mechanics or ventilation standpoints, a rounded shape is preferred. For efficiency of space utilization, rectangular is probably best; it is also preferred for a hoisting system. Size of opening, of course, is mainly a function of the *design* capacity *tor* materials handling— the higher the production rate, the larger the opening must be (Bullock, 1982a). Ventilation and drainage needs may require additional opening capacity. Here again as with other design features, overall cost, the sum of capital and operating costs, is the final determinant of shaft size. A recent survey of shafts constructed in U.S. noncoal mines revealed the following practice (Anon., 1974):

Shafts, vertical		96%
Sunk, circular (12-23 ft, or 3.7-7.0 m)	44	
Sunk, rectangular (6 x 8 to 15 x 21 ft, or 1.8 X 2.4 to 4.6 x 6.4 m)	16	
Bored, circular (5-16 ft, or 1.5-5.0 m)	36	
Shafts, inclined (declines) (12 x 14 to 12 X 20 ft, or 3.7 x 4.3 to 3.7 x 6.1 m)		4%
		100%

Nearly all the sunk shafts were concrete-lined, and 58% of the bored shafts were lined; about 70% of all shafts were constructed by contractors. Some 72% of the sunk shafts were for production (hoisting), and 75% of the bored shafts were for ventilation. Rectangular vertical shafts and declines arc infrequently used in noncoal mines today. In coal mines, small bored shaftsand slopes arc both widely used as main access openings.

With other development openings, the following practice prevails (Bullock, 1982a):

Coal mines (height of opening is approximate height of seam)

Entries, crosscuts (width 14 -20 ft, or 4.2-6.0 m)

Rooms, crosscuts (width 15-24 ft, or 4.5-7.2 m)

Noncoal mines

Drifts, crosscuts (9 x 14 to 14 x 20 ft, or 2.7 x 4.2 to 4.2 x 6.0 m)

Ramps, declines (similar to drifts)

Raises, rectangular (one or two compartments, 4 x 4 to 6 x 8 ft, or 1.2 x 1.2 to 1.8 x 2.4 m ca.)

Raises, circular (3-10 ft, or 0.9-3 m)

Interval Between Levels

If a deposit is to be mined with multiple levels, a level interval should be selected that yields the lowest overall mining costs for the mine development-exploitation plan chosen (Lewis and Clark, 1964). Factors which affect these costs are the geologic and natural conditions of the mineral deposit and country rock, mining method, development layout, method of driving openings, life of openings, production rate, mine life, and various financial considerations (Peele, 1941; Thomas, 1978).

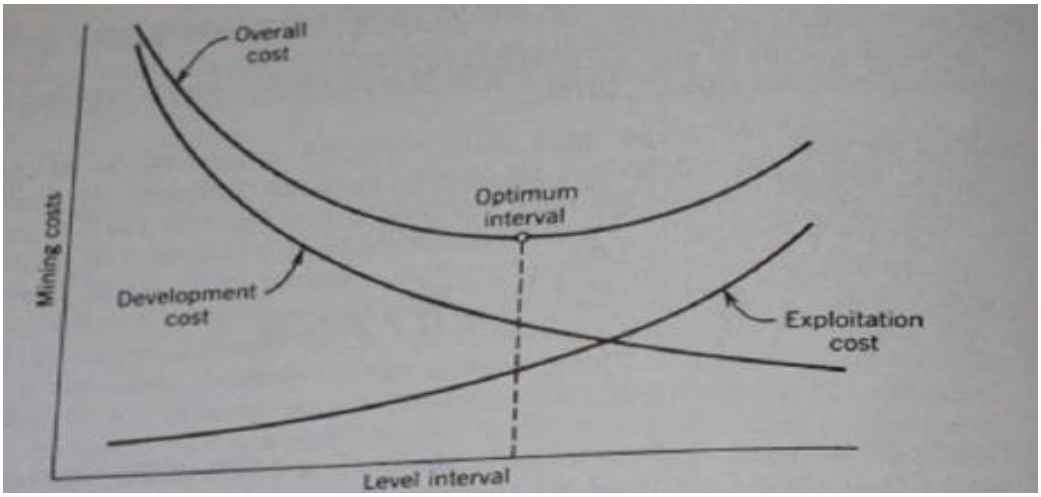


Figure 9.5. Determination of the optimum interval between levels for a hypothetical multilevel mine.

Unless thc configuration of a mineral deposit or the type of mining method takes precedence in locating and spacing levels, it is not possible to determine the optimum level interval without estimating costs. Generally, development costs (constructing, equipping, and supporting openings and installing materials-handling facilities) increase in proportion to the number of main levels required. In contrast, exploitation costs (producing and handling ore and providing auxiliary operations such as ventilation, drainage, and power supply) as well as convenience of access for the miners decrease with the number of levels (Lewis and Clark. 1964: Nilsson, 1982b).

If costs can be estimated with some confidence, then the most expeditious way to arrive at the optimum level interval is to calculate development, exploitation, and overall mining costs (unit or total) for three or four assumed intervals, plot them, and select the interval corresponding to the lowest overall

cost. Figure 9.5 illustrates the procedure. In this hypothetical but typical case, development costs are assumed to be higher and therefore exert the greater influence on level interval (i.e., tending to increase it). The current trend with mechanized, high-production methods is to fewer levels, spaced farther apart, and supplemented by minimum-cost sublevels as required. Level intervals in small mines range from 100-300 ft (30-90 m) and from 300-800 ft (90-240 m) in large mines (Nilsson, 1982b).

SecondaryDevelopment Layout

Construction of some secondary (level or гопе) and many tertiary (lateral or panel) development openings can be postponed until the exploitation stage is reached . Planning for these openings, however, as a part of the general mine layout must take place during the conceptual phase of the development stage.

The openings involved and idealized layouts for noncoal mines may be seen in Figures 3.1 and 9.1 In choosing a configuration of openings on each level and between levels, the major considerations are materials handling of ore, access for the miners and for materiel, and circulation of the ventilating air (Stoces, 1954). On the level, a minimum number of drifts and crosscuts is driven for access, haulage and ventilation, locating them in the footwall rather than the ore body if ground control is a problem (Bullock 1983a) Raises, winzes, ramps, and special-purpose openings (orepasses. Ventilation raises, manways. etc.) are similarly located between levels as required. In addition, such stope development (drawpoints, finger raises, bells, undercuts. sublevels, etc.) as necessary is commenced. (Clearly, a decision regarding the mining method has had to be made at the onset of secondary development, if it has not been made previously.) In a large mine, computer simulation is useful in level layout to achieve the most cost-effective haulage as well as transfer between levels (Nilsson, 1982).

Deepening a mine in later life can be accomplished by sinking (or raising) an interior shaft or decline, should extending the main shaft not be practical or interrupt production.

Coal mine development normally presents a different and simpler situation

if a single, flat (or nearly flat) seam is being exploited (Obert, 1973c). An idealized development plan for a large coal mine appears in Figure 9.6, showing primary and subsequent openings. The main shaft or slope is used only for materials handling and materiel supply and is centrally located; a separate shaft for personnel is placed close by (both would have to be protected from ground movement by a sizable barrier pillar). Horizontal development for access and haulage is carried out by driving multiple headings, the principal ones being main entries; they are interconnected by crosscuts, required by law and necessary for good ventilation. (Multiple openings are less used in hard-rock mines because of the higher cost of excavation and less gassy conditions.) To minimize haul distances, the main entries could be driven diagonally, provided the cleat structure of the coal is not too prominently developed. After completing the mains, the deposit is then divided into large exploitation zones by panel entries (submains may also be needed in a very large property) and into smaller panels by room entries. As shown, this mine is operated on the retreat; entries are driven to the extremities of the property and exploitation carried out on the retreat. Most mines employ a combination advance-retreat system to reduce development time. Ventilation in Figure 9.6 is unidirectional: Air enters the mine through the main shaft and man shaft and exhausts through two (or more) air shafts. (The property is bounded by bleeder entries that serve as ventilation exhaust laterals.)

The mining plan shown in Figure 9.6 is intended for either of the two common U.S. coal mining methods, room and pillar or longwall. It is also suitable for mining horizontal, tabular deposits of metals and nonmetals by the room and pillar or stope and pillar method. We may note that an effective way to operate a mine with conventional haulage methods in a pitching (inclined) seam or bed is to drive all openings cross-pitch, that is, at an angle to the strike of the deposit to reduce, in effect, the grade of the openings. Driving up-pitch is preferred when there are water problems.

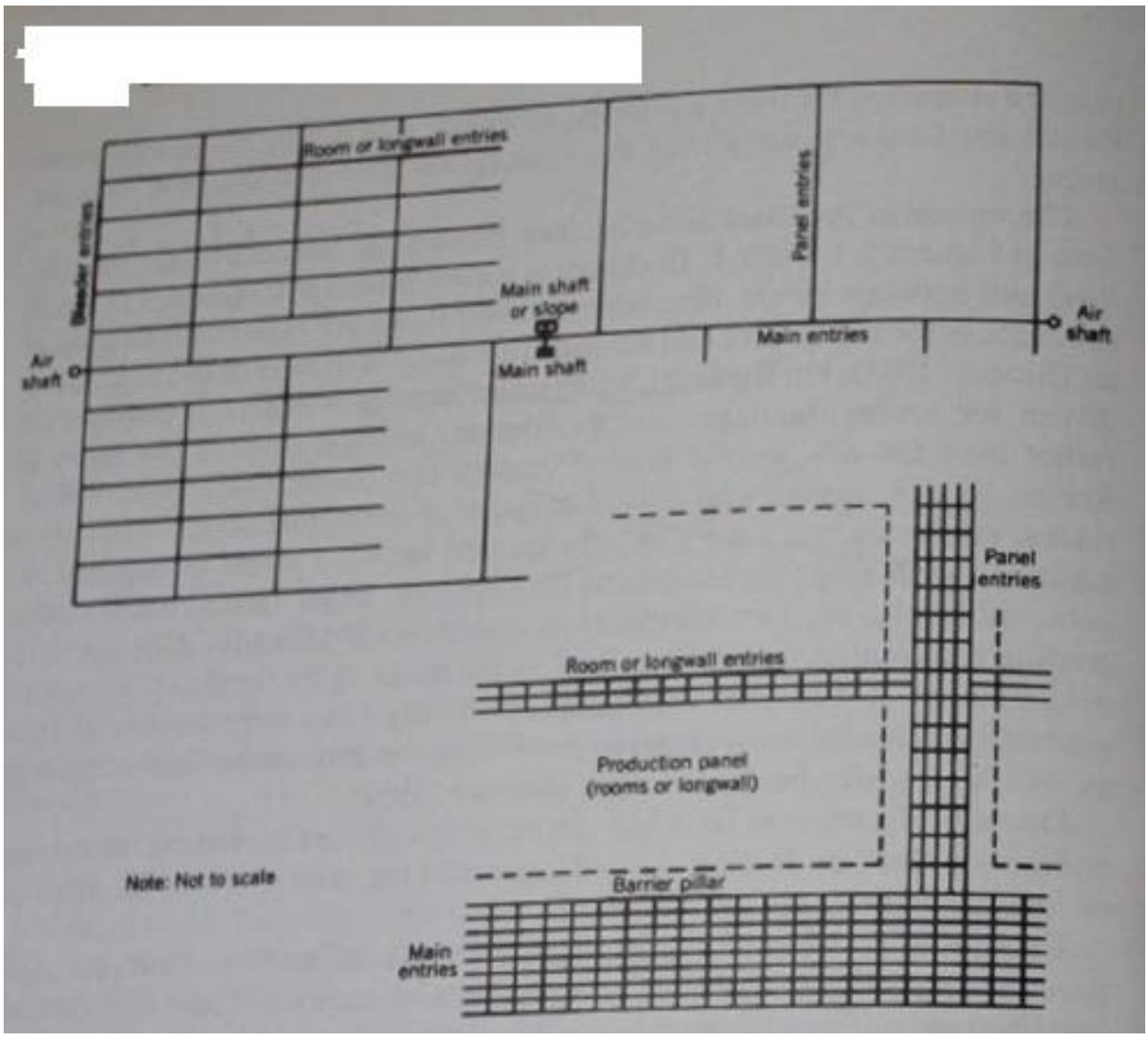


Figure 9.6. Layout of an underground coal mine, showing development openings and production panels.

9.3 MINE PLANT LAYOUT

The physical plant that must be constructed to service subsurface mining operations is made up of three groups of components, distinguished by their locations. They are the surface plant, shaft plant, and underground plant. Construction of the mine plant is essentially completed during the development stage and, because time is of the essence, in accordance with strict imposed by a CPM or PERT schedule (review Section 3.4 and Fig.3.3). We examine some critical aspects of the three plant components, focusing mainly on mine access by shaft, the most common means.

Surface Plant

We can assume that all the normal tasks of mine development have been carried on at the surface, including the proper location and erection of the general *surface plant* (Section 3 2). These elements consist of access roads and parking, transportation facilities, power supply and utilities (including water supply), service and maintenance buildings, administration and personnel buildings, mineral processing plant, bulk storage, and waste-disposal facilities for air, water, and solids. It is with the portion of the plant designed especially for underground mining that we are now concerned.

The surface facilities that arc unique are the shaft collar (portal) and enclosure, headframe, bins, and hoist house. In coal mining, the entire assembly is olten referred to as the *tipple,* sometimes including the mineral-processing facilities as well. The headframe is the dominant structure, toweringover the rest of the surface plant, and it houses the drive or idler sheaves around which the hoist ropes pass, connecting the shaft conveyan cesto the hoist. Storage bins and occasionally the hoist arc also mountedin the headframe. Design of the hoisting system is an engineering task that is covered later in this chapter.

Facilities for slope or decline access differ little if hoisting is used for materials handling. If, however, as is usually the case, a belt conveyor or trucks provide transport from the mine, then hoisting is eliminated and the surface plant is much simplified. Drift access is essentially horizontal and also dispenses with hoisting.

Shaft Plant

In addition to the main access opening itself, the *shaft plant* consists of the facilities installed for materials handling of the ore, coal, or stone and associated waste and the means of transport for the miners and materiel. Also arrangements for ventilation, drainage, power supply, and communications must be provided.

Since at least two access openings to an underground mine are required by law, certain of the functions identified above can be divided between the two openings. Initially, in the shaft-sinking stage and the period of horizontal

development underground before connection to the second access opening can be made, it is necessary that the main access serve all functions, including providing fresh air and discharging exhaust air. The ventilation requirement is met by installing vent tubing in the shaft, connected to a fan on surface. Occasionally, the shaft is compartmented and bulkheaded to divide the airflow.

When connection between the two surface accesses has been completed, then permanent arrangements can be made to meet all the development and exploitation requirements plased on the shafts. Normally, the production shaft is compartmented if hosting is used; conveyances called skips are suspended in the shaft on wire rope, and to are operated in balance with one another (one descends when the other ascends). The ropes are connected through the head frame to the hoist on the surface. Guides or tracks are installed in the shaft to facilitate fast travel of the conveyances. Personnel and materiel era transported in cages or elevators that operate in tandem with the skips or independently in separate compartments or another shaft (the trend today is separate shafts for safety and cost, reasons). Power and water lines are suspended in unused space or in a personnel escapeway compartment of the opening.

If another type of opening or a different materials-handling system is employed, then some modifications to the arrangements just outlined are necessary. For example in conveyor or truck transport is use in a slope or drift in place of shaft hoisting, then the system is much simplified. Separate facilities for hoisting of personnel and materiel are usually still necessary, however.

Some of .he alternative shaft, slope, and drift access arrangement for different occurrences of mineral deposits are shown m Figure 9.7. Reviewing the discussion of the previous article, we may summarize the application of each of the four cases as follows:

a. *Vertical shaft with hoisting.* For deep horizontal (<30°). vertical, or steeply inclined (>70°) deposit: bad natural conditions; high production long life

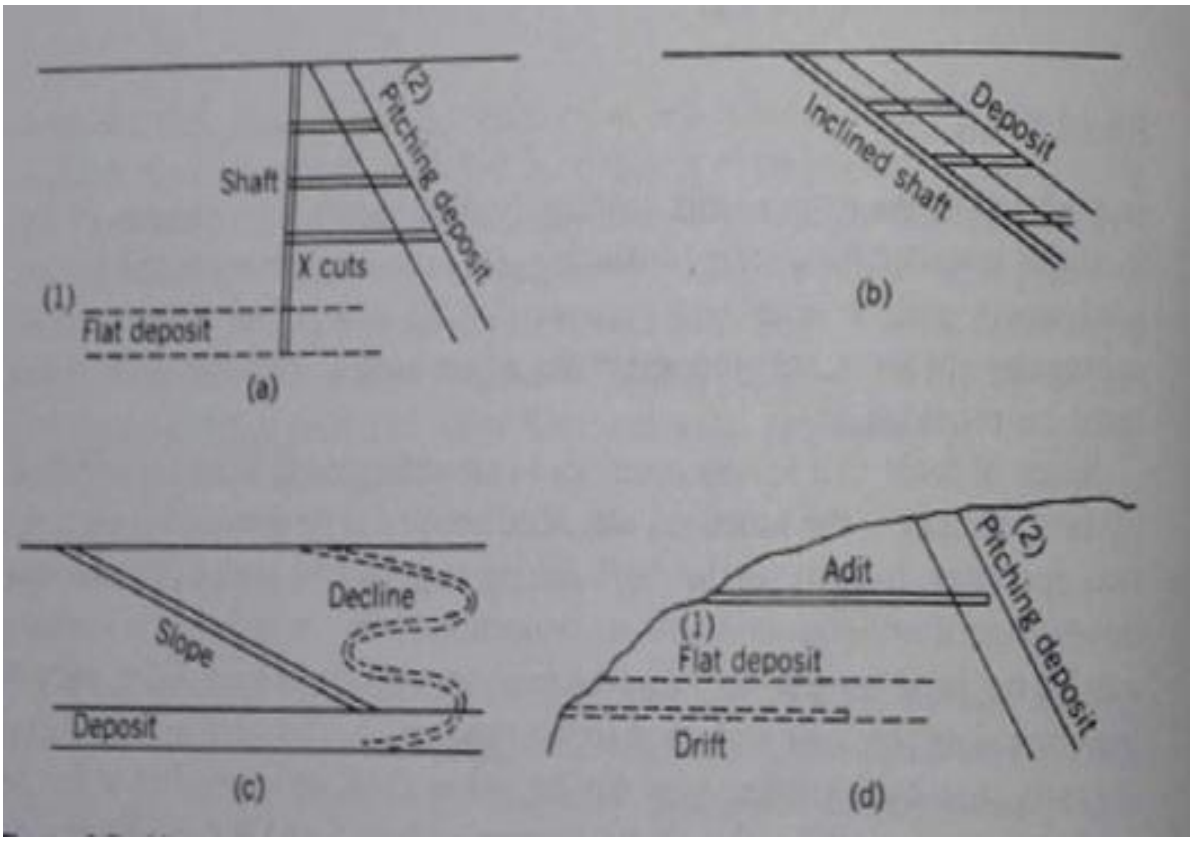


Figure 9.7. Alternative placement of main access openings for different occurrences of mineral deposits, (a) Shaft with hoisting, (b) Inclined shaft with hoisting, (c) Slope or decline with haulage (d) Adit or drift with haulage.

b. *Inclined shaft with hoisting.* For moderately inclined (30-70°) deposit; moderate conditions; low to moderate production and life; shortens horizontal development, and allows exploration during sinking

c. *Slope or decline with haulage* (conveyor or truck). For shallow horizontal deposits; good to moderate conditions; moderate to high production; long life; can install rope hoisting and use with rail haulage; limited to 12° with trucks and 20° with conveyor (unless high-angle)

d. *Drift or adit with haulage* (conveyor, truck, rail, etc.). For shallow, outcropping, horizontal deposit or steeply inclined deposit in area of high relief; varied conditions; high production; long life

Underground Plant

The secondary and tertiary underground development openings that are constructed, in and adjacent to the deposit have already been discussed. These are necessary to complete the network of openings required in mining, both for

development and exploitation purposes. In addition there is a group of specialized openings, the *underground plant,* required at the shaft where (I) horizontal openings intersect with vertical ones, and (2) a transition from haulage to hoisting occurs. This location is called a *shaft station*.

At the junction of two different materials-handling systems, arrangements must be made to accomplish fast, efficient, safe transfer. Material is taken from the haulage system at a dump, collected in a storage bin to provide surge capacity, crushed if necessary, and then measured in a loading pocket for transfer to skips in the shaft (Lucas and Haycocks, 1973). Facilities must also be proviided on each working level at the shaft station to handle personnel and materiel.

Different arrangements are made if the main production opening is a slope or drift. If hoisting is not used and the same haulage method serves the mine from production face to surface, no shaft station is necessary, and a simple transfer point suffices. Again separate transport facilities for personnel and materiel are necessary.

9.4 CONSTRUCTION OF DEVELOPMEN TO PENINGS

The construction of underground development openings is specialized and expensive. Consequently, it has in recent years become increasingly mechanized and efficient in a necessary effort to reduce costs. By its very nature, however, development in noncoal mining remains several times more expensive than exploitation (the latter has the decided advantage of multiple free faces to break to and often can utilize gravity to assist materials handling). In coal mining, w hen using the room and pillar method, there is less distinction between the two stages of mining because of the similarity of the openings, although the longwall method exhibits a marked contrast between development and exploitation both in type of opening and cost.

Cycle of Operations

The different production cycles of operations and equipment used to drive openings in underground mining-both development and exploitation, coal and noncoal, and cyclic and continuous-are summarized in Table 9.1. We discuss now

some of the unique aspects of construction for several openings, nothing that continuous methods in general are applicable only to soft or medium-hard rock.

Shaft. Shaft sinking and boring procedures are applicable in some cases to steeply inclined declines and winzes as well as vertical shafts. Because of the importance and urgency in completing the installation of the first opening, usually a shaft, highly specialized equipment and procedures have been developed. Examples of cyclic and continuous methods are shown in Fig.9.8. Since only one cyclic operation can be carried on at the face at one time, other equipment must be suspended in the shaft; lining of the shaft for ground and water control purposes, however, can proceed simultaneously if a *sinking stage* (dected platform) is legal to use (e.g., in South Africa). Mc.Kinstry (1983) describes deep shaft sinking operations in detail.

Loading (called *mucking* in metal mines) is carried out by specially fitted machines, sized approximately as follows (Russell. 1982):

m a c h in e	CAPACITY. FT^3 (m^3)	O U T PU T , t o n s /h r (t o n n e s /h r)
Clamshell	10 (0.3)	30 (27)
Fixed-boom clamshell	10-20 (0.3-06)	25-80 (23-73)
Overhead loader	5-9 (0.1-0.3)	50-100 (45-91)
Cactus grab	15-30 (0.4-0.8)	100-200 (91-181)

Specifications are not available for a fifth machine, the backhoe, but it is somewhat comparable to the overhead loader.

The most important auxiliary operation in a conventionally sunk shaft is ground control (usually a reinforced concrete lining, shotcrete, steel sets or arches, cast iron tubbing, or bolts and steel mesh are used), with control of groundwater a close second (grouting or freezing as well as water rings and concrete lining may be necessary in extremely wet conditions). Bored shafts usually present no support problem but may have to cope with water, and for that

reason arc sometimes concrete-lined or shotcreted. Ventilation is mandatory during the sinking of a shaft if miners must work at the face or the quality of the atmosphere must be maintained.

TABLE 9.1 Equipment for Production Cycle of Operations in Underground Mining

Opening	Continuity of Operation	Cycle of Operations: Drilling	Blasting	Loading	Haulage
Development					
Shaft	Cyclic	Percussion jumbo	Dynamite, slurry	Clamshell, loader, cactus grab, backhoe	Bucket, skip
	Continuous	Shaft-boring machine (full face or pilot hole + ream)			
Slope, decline, ramp, winze	Cyclic	Percussion jumbo	Slurry, ANFO	Shovel loader, slusher, LHD	Skip, truck, LHD, conveyor
	Continuous	Tunnel-boring machine			
Raise, incline	Cyclic	Stoper, jumbo, mobile platform	ANFO, slurry	Gravity, chute, loader	Truck, mine car,conveyor
	Continuous	Raise-boring machine (full face or pilot hole + ream)			
Drift, crosscut, adit, tunnel	Cyclic	Percussion jumbo, airleg	ANFO, pneumatic loader	Shovel loader, slusher, LHD	Truck, LHD, mine car, conveyor
	Continuous	Tunnel-boring machine, roadheader			
Entry, crosscut (coal)	Cyclic	Cutting machine, rotary auger drill	Permissible AN dynamite	Gathering arm loader	Shuttle car, mine car, conveyor
	Continuous	Continuous mining machine			
Exploitation					
Stope	Cyclic	Jumbo, stoper, airleg, downhole	ANFO or slurry	Gravity, slusher, shovel loader, LHD	Truck, LHD, mine car, conveyor
	Continuous	N/A (use continuous miner if stope and pillar and soft rock)			
Room, crosscut (coal)	Cyclic	Cutting machine, rotary auger drill	Permissible, airstick, cardox	Gathering arm loader	Shuttle car, mine car, conveyor
	Continuous	Continuous mining machine			
Longwall (coal)	Cyclic	N/A (no longer used)			
	Continuous	Shearer, plow			Conveyor, mine car

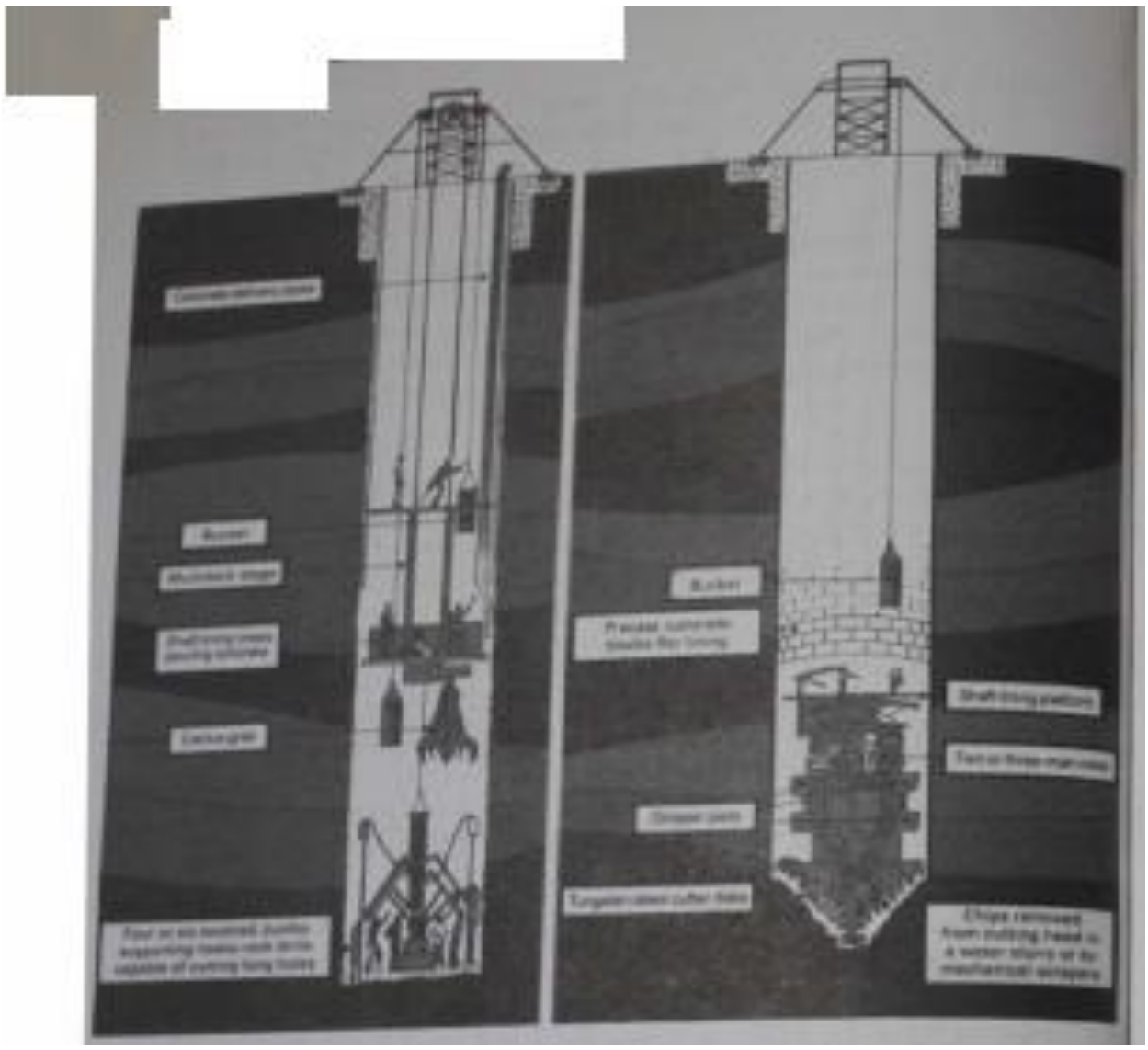


Figure 9.8. Shaft-sinking methods and equipment. *(Left)* Cyclic sinking utilizing sinking stage and drill rig (/umbo). *(Right)* Continuous advance with shaft-boring machine. (After Anon., 1983d. By permission from Chamber of Mines of South Africa, Johannesburg.)

Slope, Decline, Ramp, or Winze. Usually a slope or similarly declined opening is sunk at a low enough angle (<20°) that it can be driven or bored like a horizontal opening. If not, then it is sunk or bored like a shaft, as described above. Difficulties are experienced due to the inclination, but they are usually not serious.

Raise or Incline. Secondary openings that are driven vertically or near vertically employ specialized cyclic or continuous operations. Because the direction of mining is overhand, some unique circumstances are encountered. For example, once advance has progressed beyond the level, a working platform must be erected close enough to the face to permit miners to reach the face; after each round of face advance, the platform must also be advanced. Ground support is usually necessary,

and bolting and timbering is often used (it also supports the platform). On the other hand, the muck falls from the raise by gravity and can be loaded directly through a chute into the conveyance on the level below. Ventilation is a critical auxiliary operation, especially if strata gas is encountered.

To facilitate raise development, a mobile platform, called raise climber, may be installed. It is mounted on rails and provides a movable platform for scaling, drilling, charging explosives, and installing rock bots or other support. A raise-bonrig machine offers advantages where rock handness permits, but it generally requires a pilot hole between levels which is then reamed to the desired diameter. The borer may be mounted on either level and operate overhand or underhand.

Drift, Crosscut, Adit, or Tunnel. Development and exploitation in a horizontal direction (breast mining) is the most easily carried out and the technology the most advanced in mining. Much of it is still cyclic in nature, but the mechanization of drilling, charging holes with explosives mucking and haulage contribute to a highly efficient cycle. If rock hardness permits a TBM or roadheader can be employed for faster, cheaper advance in drift development (Robbins. 1984: Douglas. 1985). Ground support is usually provided by rock bolts, timber or steel sets, or shotcrete.

Entry and Crosscut (Coal). Horizontal development openings in coal differ from those in hard-rock mines in two significant ways: The material being mined is comparatively soft, lending itself to mechanical excavation and the headings are advanced in multiples (2-12 or more). This permits faster cheaper development, approaching exploitation in its performance Continuous mining machines are now utilized for the preponderance of development in all coal mines and are gaining acceptance for the softer ores in noncoal mines. Ground support is principally by roof bolts and timbering.

Exploitation Openings. Exploitation openings arc included here for the sake of completeness since the principle of the cycle of operations is basically the same and the type of equipment is similar in development and exploitation.

Stopes in noncoal mines are mined by overhand, underhand, or breast methods. If breast mining is employed, the methods resemble those for drift development. If overhand or underhand, then specialized methods, cycles, and equipment are employed. Because the hardness of the rock usually requires drilling and blasting, continuous mechanical mining is not feasible with today s technology. We shall study the particulars of the various stopping methods in succeeding chapters.

There two underground exploitation methods in use today for coal deposits. If room and pillar is the choice, then the rooms and crosscuts are driven exactly as the entries and crosscuts during the development stage (by cyclic or continuous methods). If longwall mining is selected, the longwall face is advanced continuously by a plow or shearer (cyclic methods are no longer used). These methods will also be discussed in detail later.

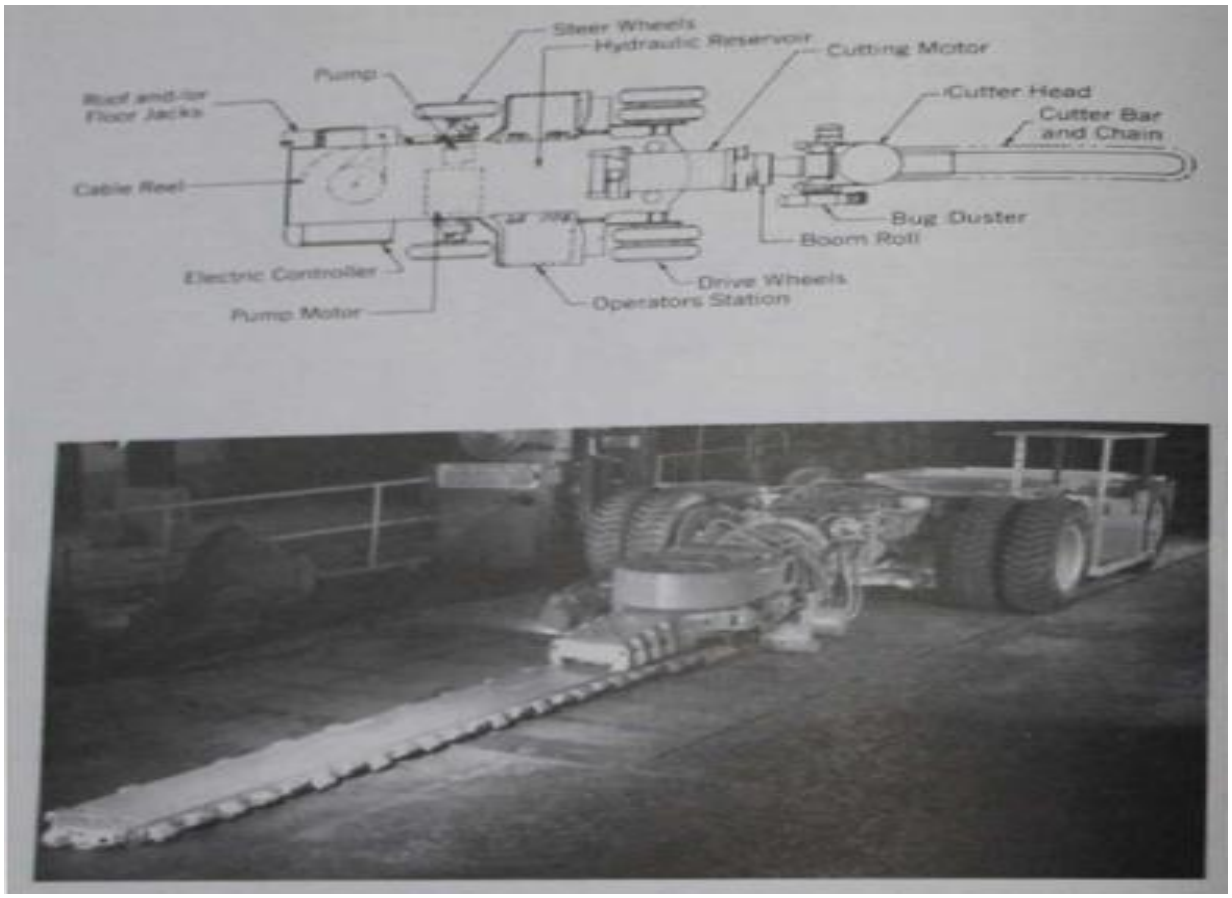


Figure 9.9a. Unit operations and equipment for development and exploitation in underground mining cutting *(Top,* after Russell, 1982. By permission from Society of Mining Engineers. Inc.. Littleton. CO. *Bottom,* by permission from Joy Manufacturing Co., Franklin. PA)

Equipment Selection

The factors involved in selecting the equipment most suitable for the existing conditions in a particular mining situation were discussed in Section 4.4. Because o f the specialized nature of equipment for underground mining some o f the machines used for development and exploitation arc illustrated for individual unit operations in Figures 9.9a-j. For comparison, performance characteristics are also included for certain equipment. More detailed performance and cost data are provided by Lucas and Haycocks (1973),Wilson et al. (1973), and Russell (1982). Specific equipment selection is made with the aid of manufacturers' specifications and performance tables.

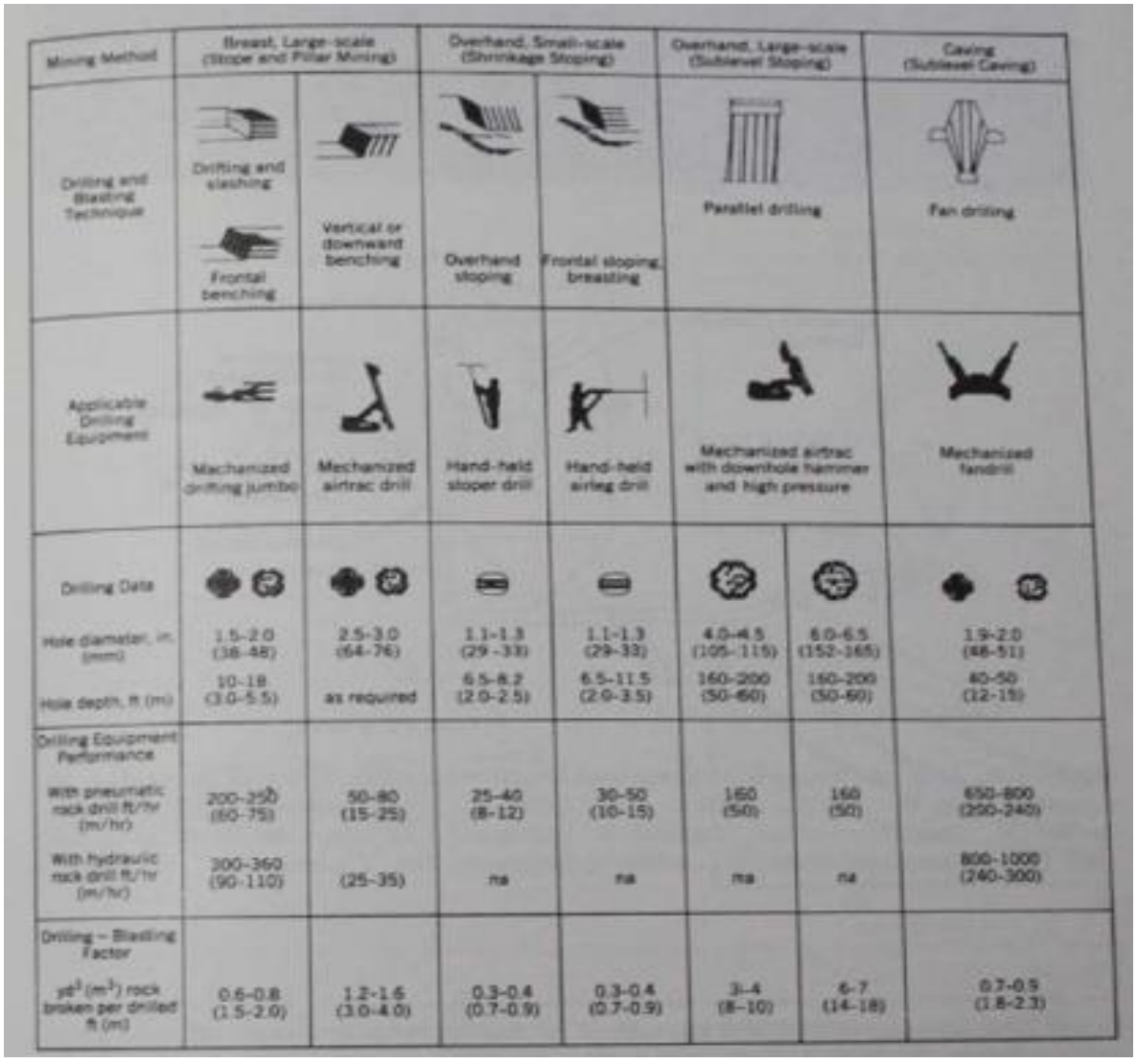


Figure 9.9b. Unit operations and equipment for development and exploitation in underground mining: drilling. Perform ance data included. (After Hamrin, 1982. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

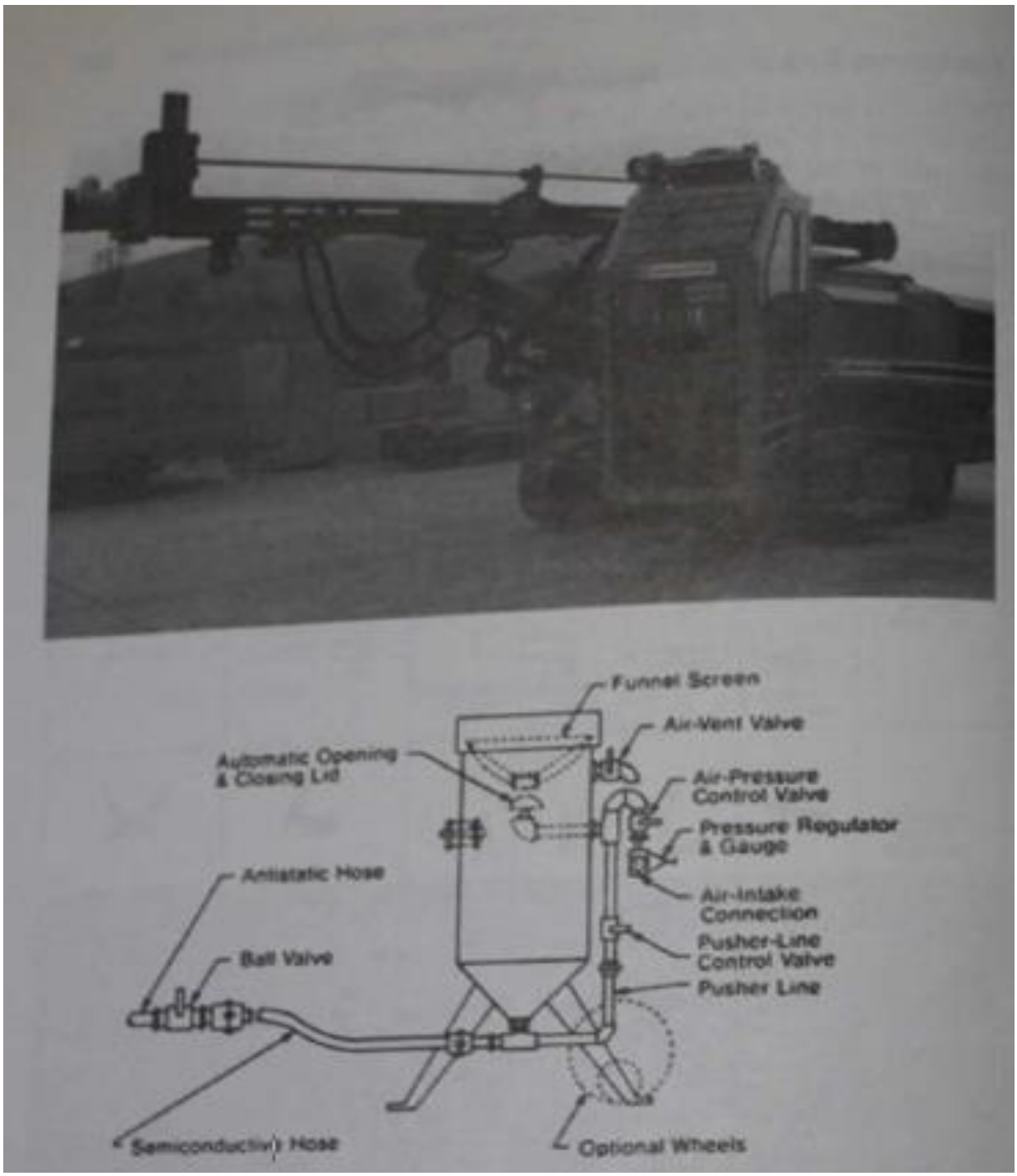
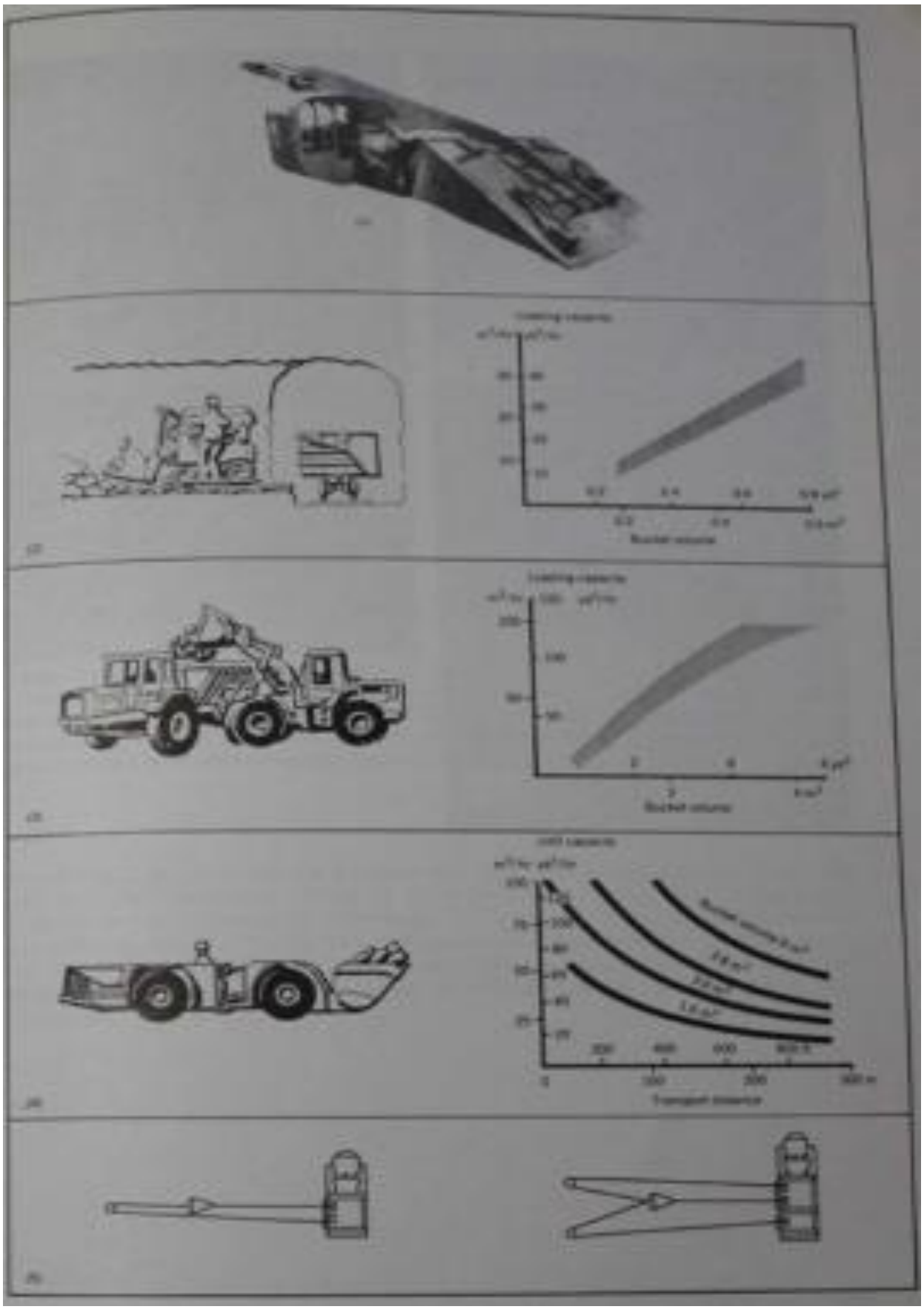


Figure 9 9c. Unit operations and equipment for development and exploitation in underground mining: drilling and blasting (Top) Drill jumbo. (By permission from Gardner-Denver Co Cooper Industries. Roanoke. VA) *(Bottom)* Explosives loader (After Russell. 1982 By permission from Society of Mining Engineers. Inc . Littleton. CO.)

Figure 9.9d. Unit operations and equipment for development and exploitation in underground mining loading (1) Gathering-arm loader. (By permission from Jo y Manufacturing Co.. Franklin. PA.) (2) Overhead loader. (3) Front-end loader. (4) LHD unit. (5) Scraper hoist or slusher: *(left)* two-drum, *(right)* three-drum. Some performance data included. (After Hamrin. 1982: Russell. 1982 By permission from Society of Mining Engineers. Inc.. Littleton. CO.)



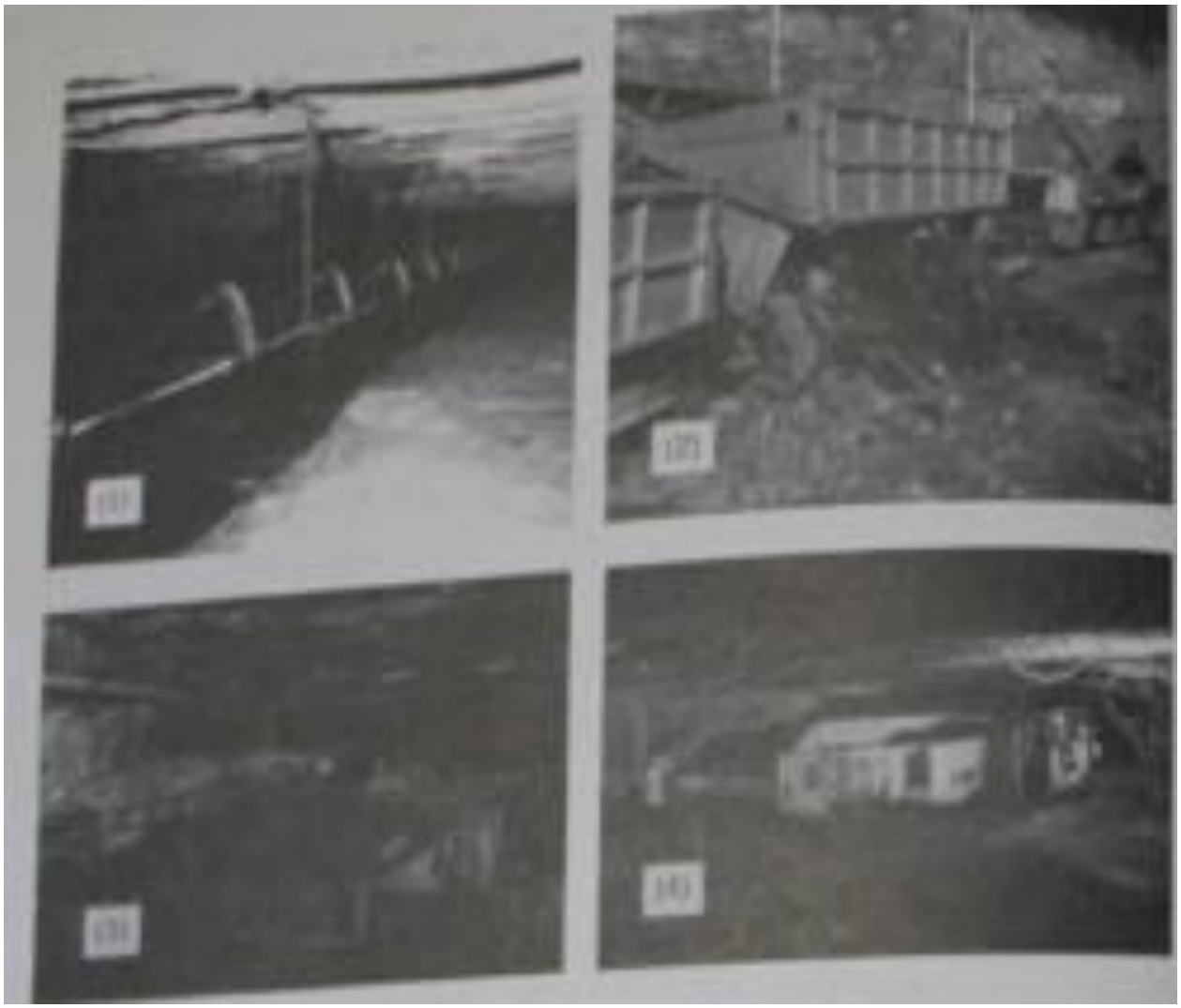


Figure 9.9e. Unit operations and equipment for development and exploitation in underground mining haulage. (1) Wire-rope belt conveyor. (By permission from Continental Conveyor &Equipment Co., Winfield, AL.) (2) Trolley locomotive and mine cars. (By permission from Marmon Transportive, Knoxville, TN.) (3) Articulated truck. (By permission from Ingersoll-Rand Co., Bristol, VA.) (4) Shuttle car. (By permission from National Mine Service Co., Pittsburgh.)



Figure 9.91. Unit operations and equipment for development and exploitation in underground mining: continuous mining. (1) Continuous miner, boring type (By permission from National Mine Service Co.. Pittsburgh.) (2) Continuous miner. (By permission from joy Manufacturing Co.. Pittsburgh.) (3) Roadheader. (By permission from Voest-Alpmeinti Corp San Antonio, TX.) (4) Longwall plow (After Brezovec. 1986. Copyright % 1986. McGraw-Hill, Inc.. New York.) (5) Longwall shearer. (By permission from Eickhoff Corp., Pittsburgh.).

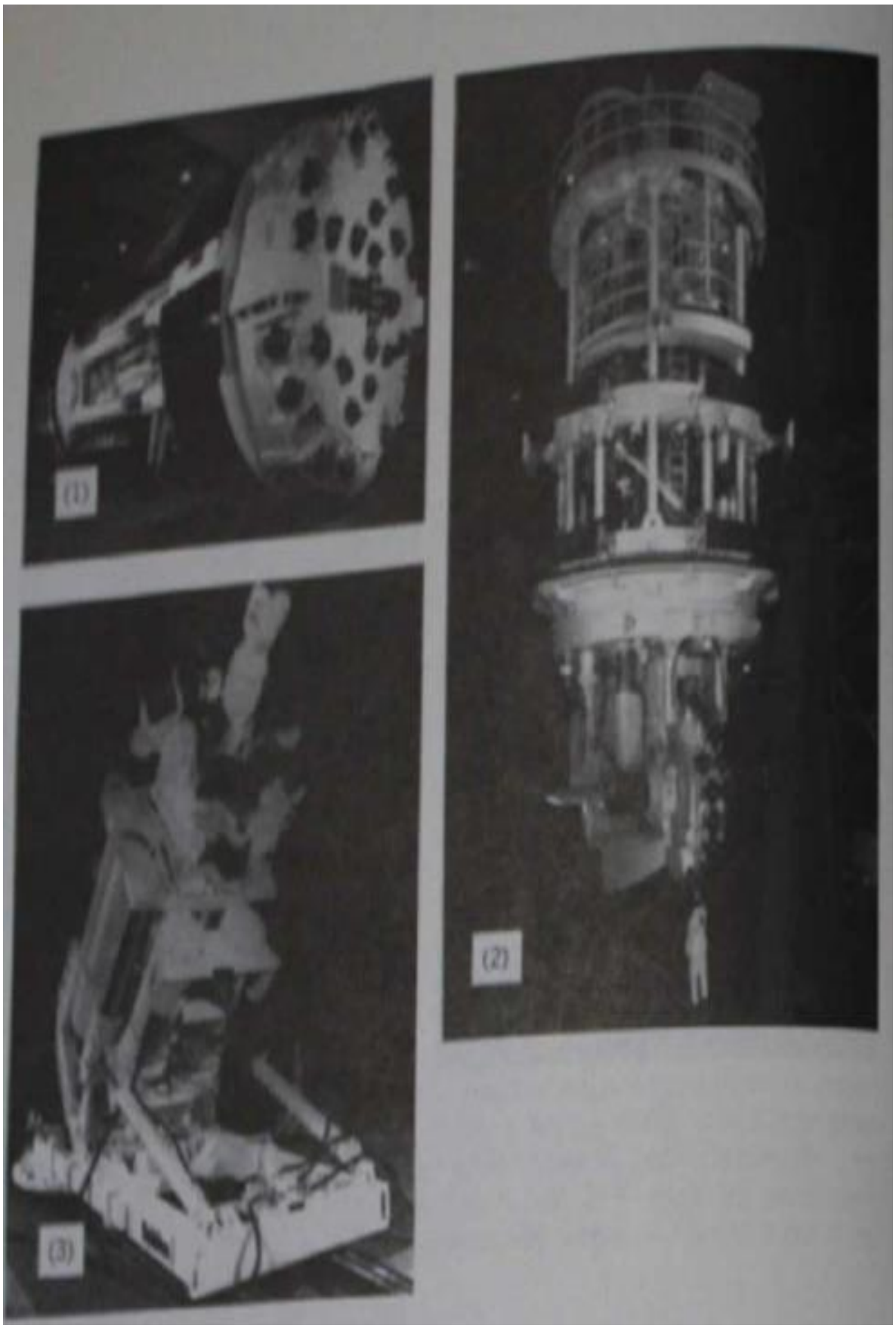


Figure 9.9g. Unit operations equipment for development and exploitation in underground mining: boring machines. (1) Tunnel borer. (2) Shaft borer. (3) Raise borer. (By permission from the Robbins Co., Kent, WA.)

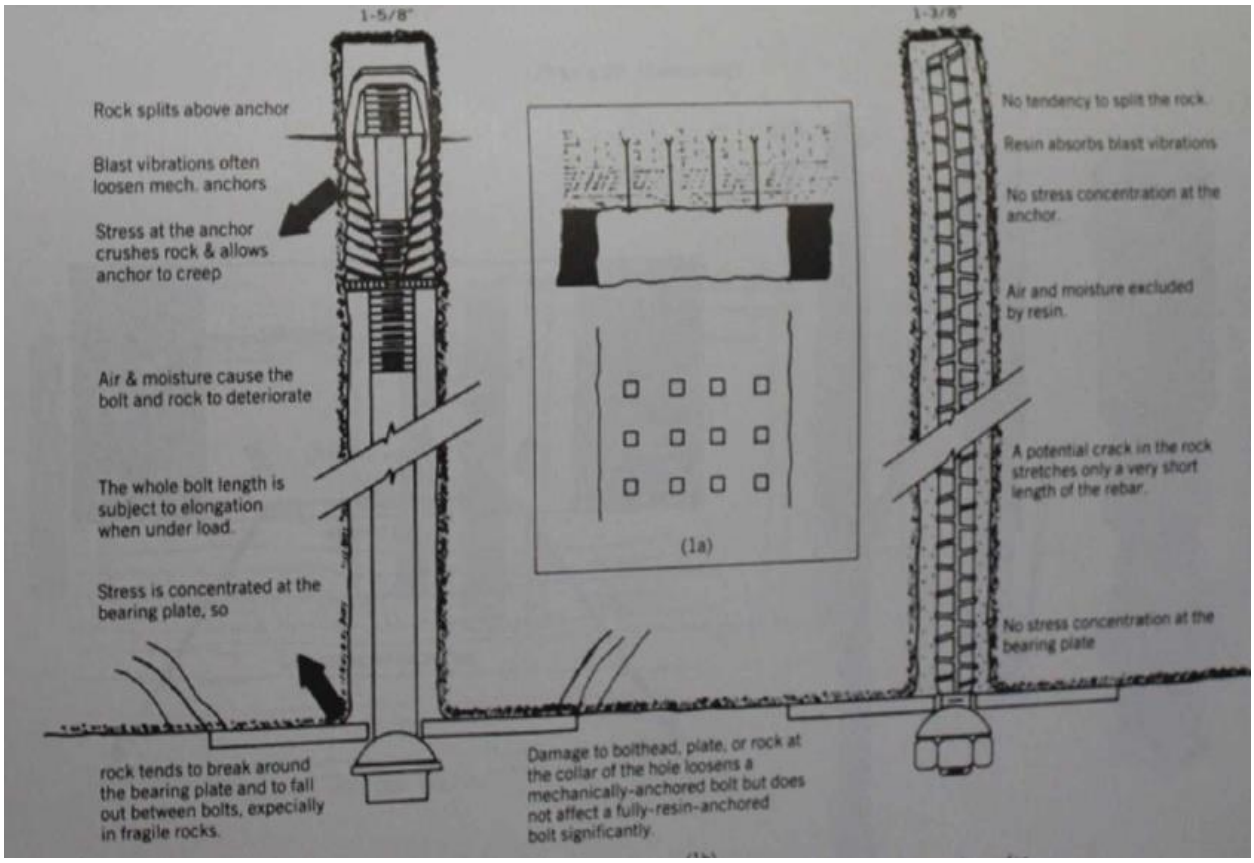


Figure 9.9h. Unit operations and equipment for development and exploitation in underground mining: root and ground support. (1a) Roof-bolt pattern. (1b) Bolt anchors: (left) mechanical, (reight) resin. (2) Bolt truss (3) Hydraulic jacs. (4a) Timber set. (4b) Steel set. (5) Yieldable atch.(After Lucas and Adker, 1973. By permission from Society of Mining Engineers, Inc., Littleton, CO)

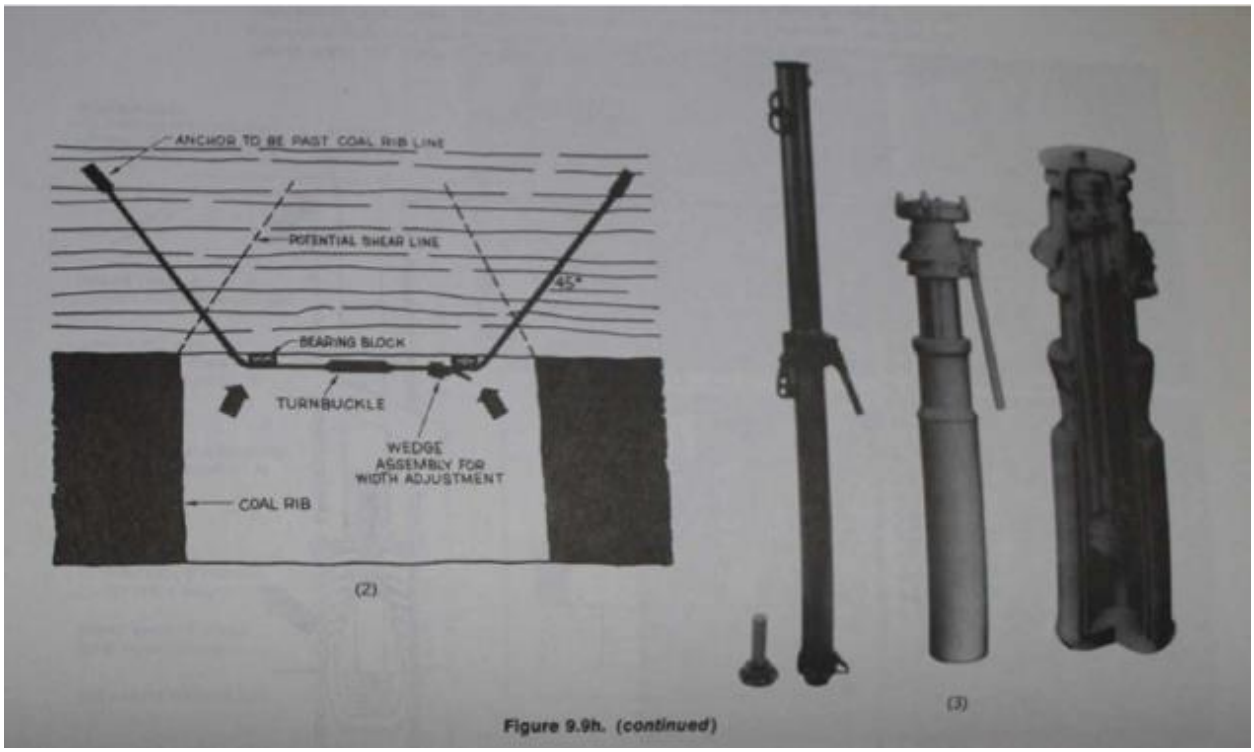
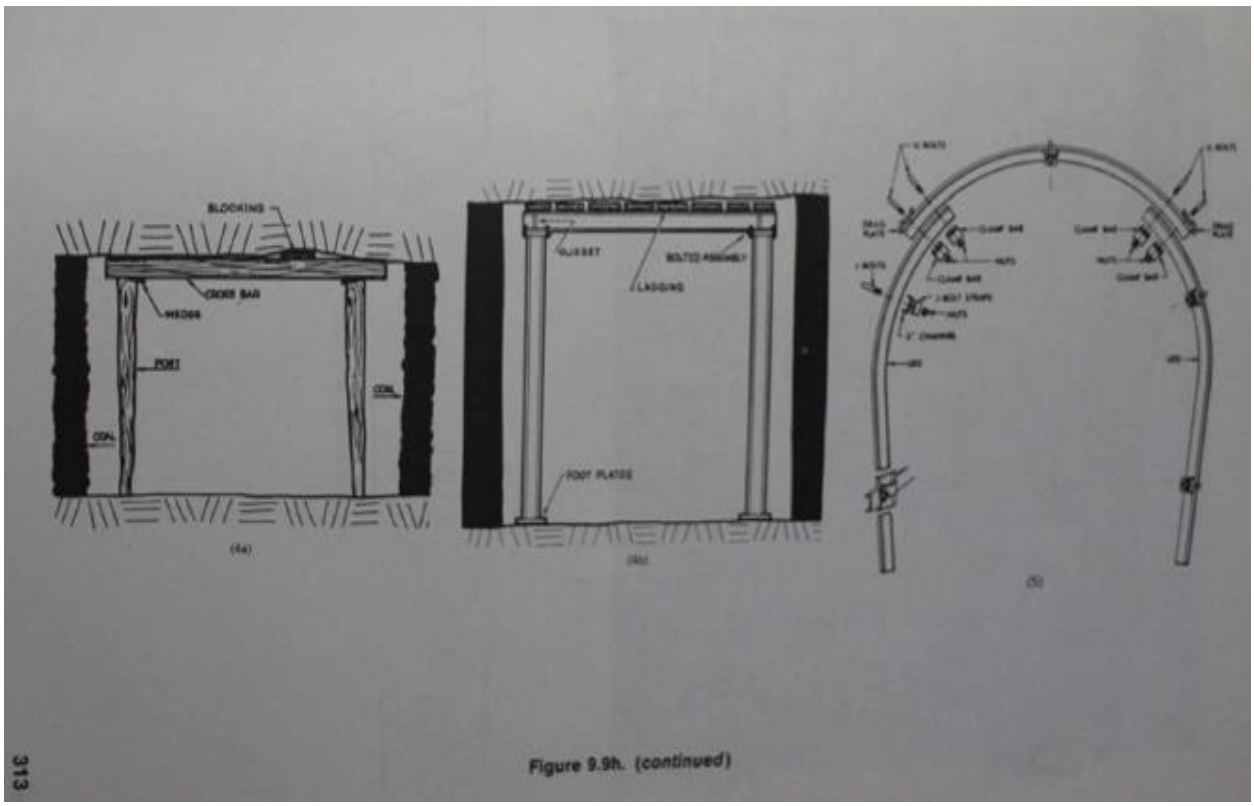


Figure 9.9h. (*continued*)



313

Figure 9.9h. (continued)

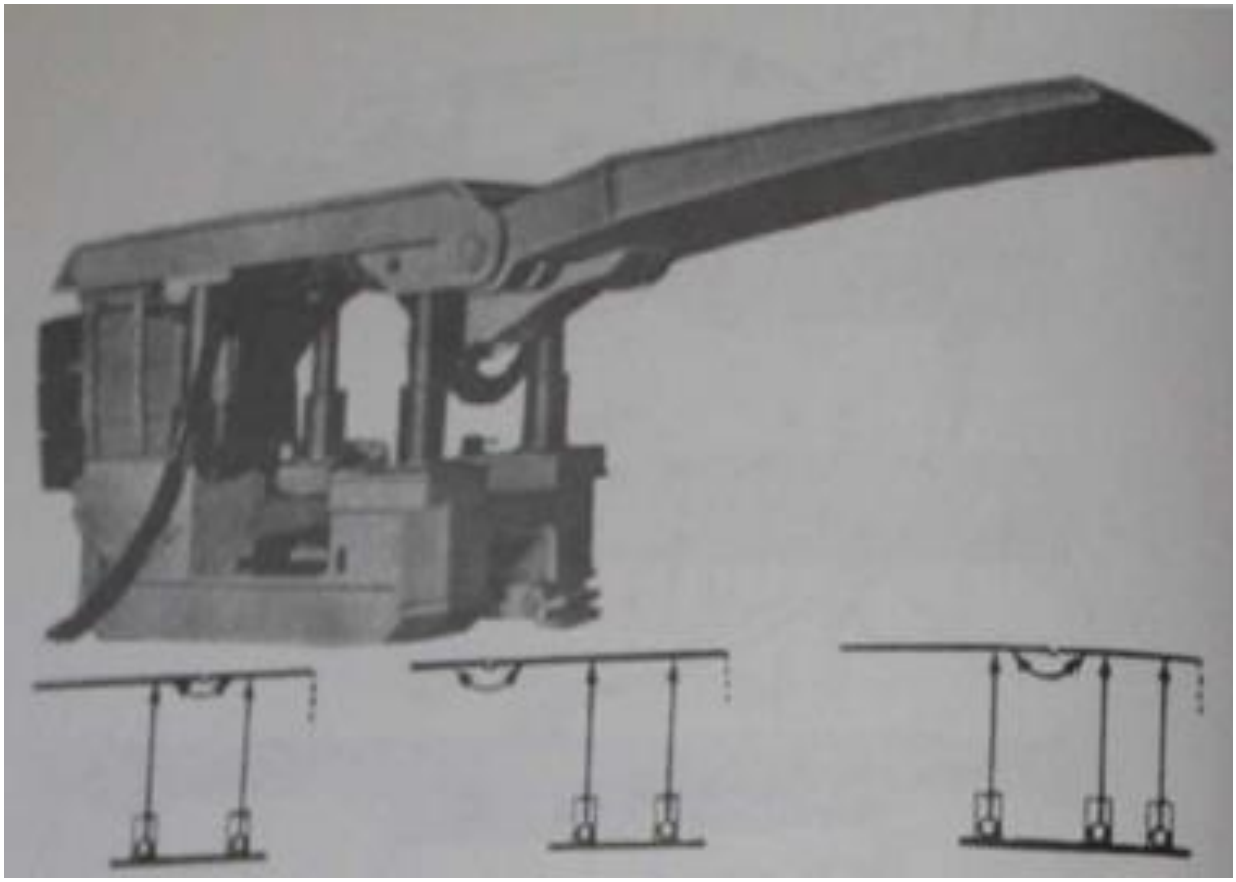
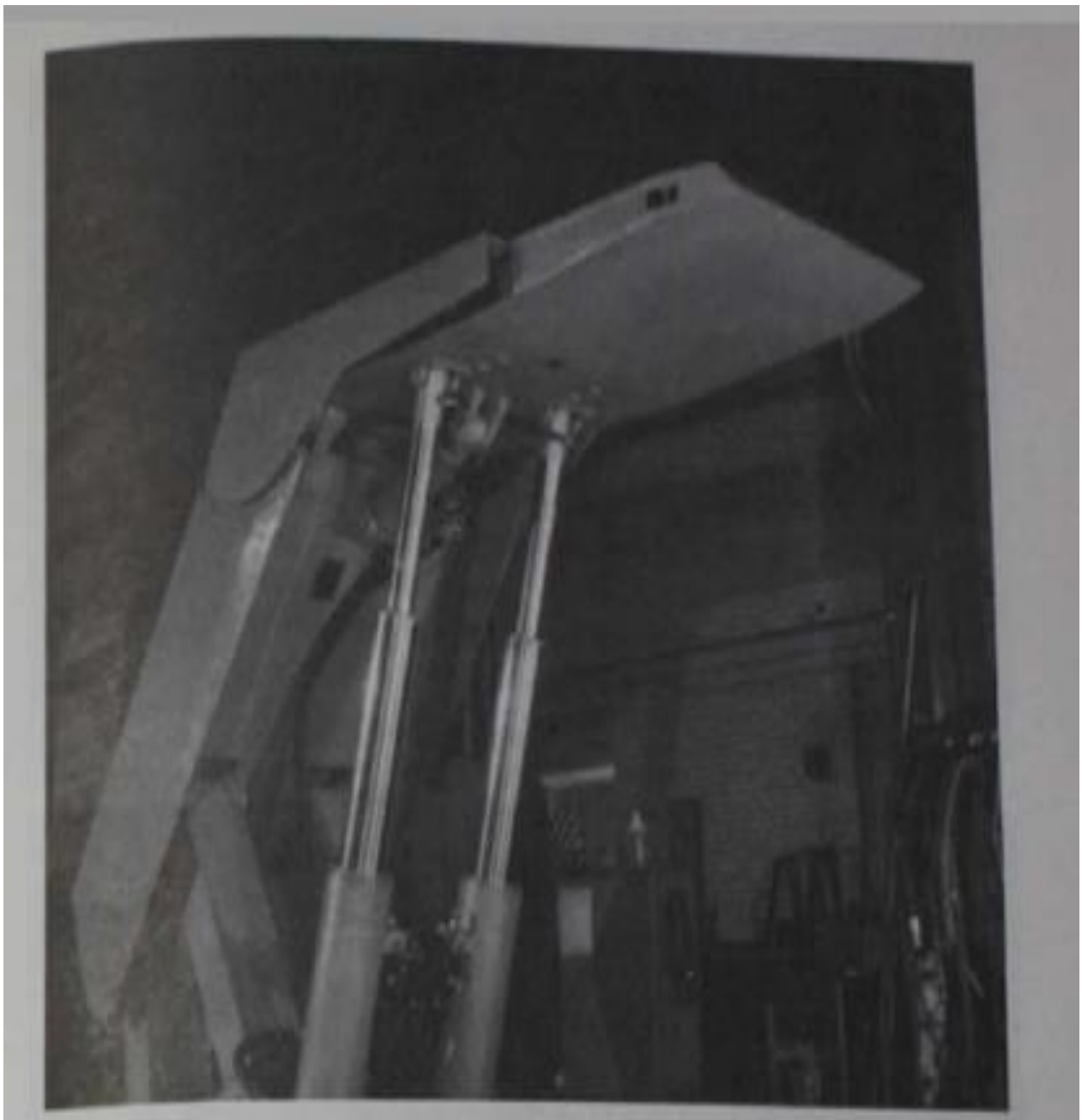


Figure 9.9i Unit operations and equipment for development and exploitation in underground mining longwall powered roof supports (Above) Chocks. (Next page) Shields after Peng end Chiang 1984 Copyright © 1984 John Wiley & Sons New York Shield, by permission from Joy Manufacturing Co.,Franklin. PA)



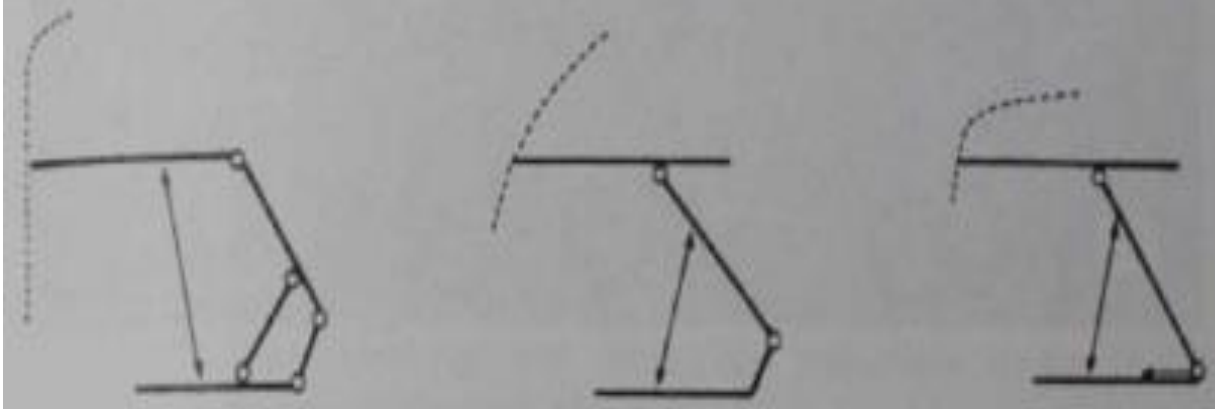


Figure 9.9i. (*continued*)

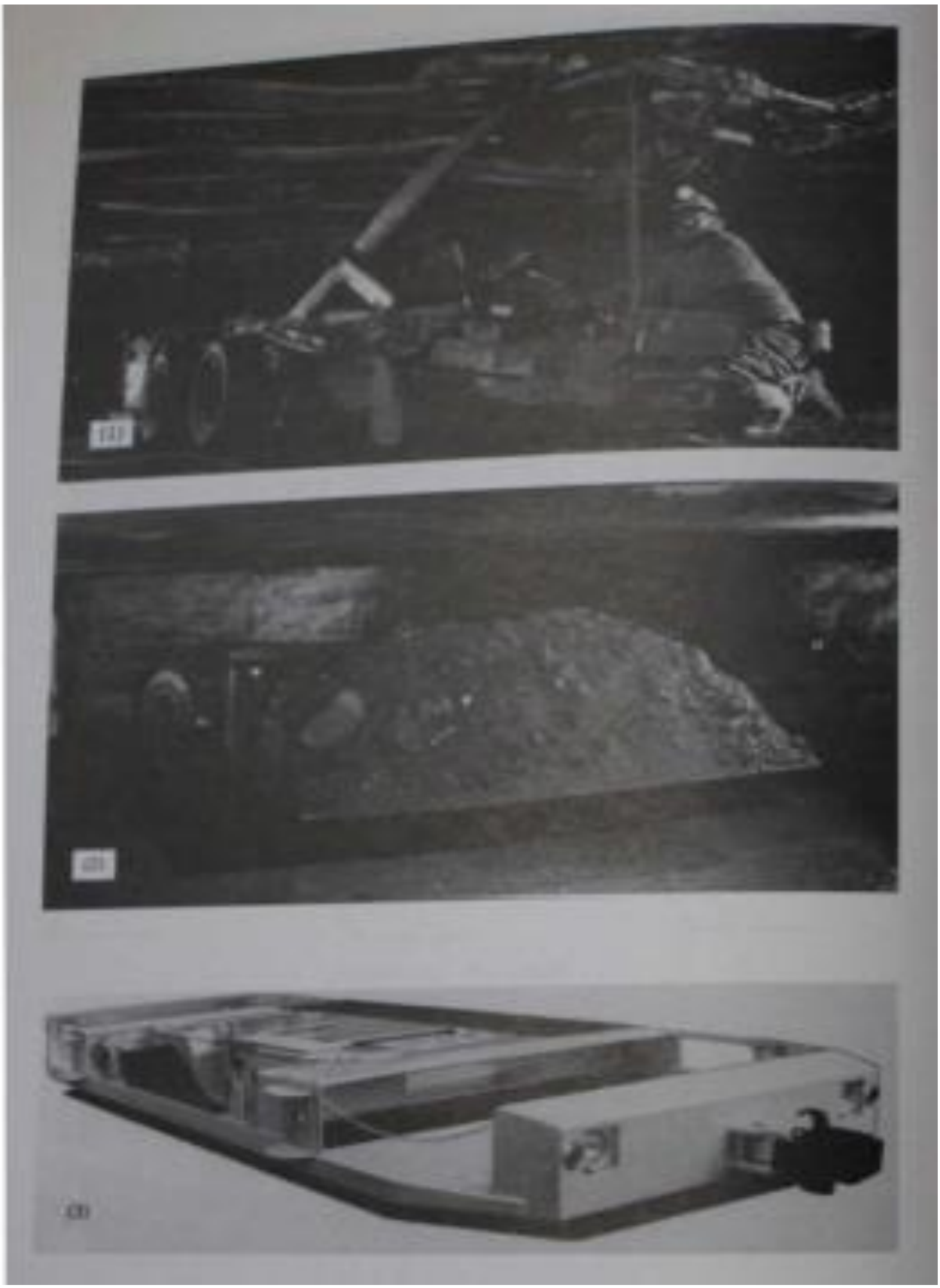


Figure 9.9j. Unit operations and equipment for development and exploitation in underground mining: auxiliary operations. (1) Roof bolter. (2) Scoop tram. (By permission from Ingersoll Rand Co.. Bristol. VA.) (3) Personnel carrier. (By permission from National Mine Service Co.. Pittsburgh.)

Development Costs

To compare the economics of constructing main development openings by cyclic methods, a few approximate unit costs are given here (Hoskins. I982):

Shaft, small	$ 120/ft	($ 400/m)
Shaft, large	300-1200	(1000-4000)
Decline or ramp	180-300	(600-1000)
Drift or crosscut	90-120	(300-400)
Raise	90	(300)

9.5 S HAFT HOISTING SYSTEMS

The hoist together with its associated plant for an underground mine is the single most important and expensive clement of the mine plant (Butler and Schneyderberg, 1982). Because it is also the most sophisticated part of the entire plant, it is not feasible to build in redundancy (except for the prime mover), yet it must perform with close to 100% reliability. Its cost may be of the order of 5-10% of the entire development budget. Obviously, wise engineering judgments are called for in the design of the hoist plant.

Components of Hoist Plant

The *hoist plant* consists of all those components of the mine plant that are necessary to elevate ore, coal, stone, or waste and to raise and lower personnel and materiel in the mine. It is made up of some of the constituents of the mine plant which we discussed in Section 9.3. Classified by location, they consist of the following:

1. Surface plant
   a. Hoist room (headframe- or ground-mounted)
   (1) Hoist drum or sheave (imparts motion to rope)
   (2) Hoist electrical and mechanical equipment (prime mover, brake, clutch, controls)
   (3) Hoist ropes (steel wire strands, woven in a pattern or lay)
   b. Headframe (tower or A-frame, steel or reinforced concrete)
   (1) Idler sheaves

(2) Storage bins (ore and waste)

(3) Skip dump mechanism (overturning or bottom dump)

2. Shaft plant

a. Skips (hulk transport)

b. Cages, elevators (personnel, materiel)

c. Shaft guides (tracks for skips and cages)

3. Underground plant

a. Dump and storage bin

b. Crusher (if size reduction required for hoisting)

c. Loading pocket

d. Personnel and materials-handling facilities

Figure 9.10 assists us in visualizing these components o f the hoist plant assuming a vertical shaft as the main access opening. With little modification, the same layout would also suffice for an Ticc for an inclined shaft using (a slope, ramp, or drift mine differs, presumably because hoisting would not be used.

Hoisting System

It is s with the *hoisting system* it self— these components of the hoist plant located in the hoist room-that engineering design is mainly needed. Three key factors govern hoist selection:

1.Production rate, or tonnage to be hoisted per unit of time

2.Depth of shaft

3. Number of levels to be accessed

There are basically only two hoisting methods plus some modifications in use today: drum and friction. The *drum hoist* stores the rope not exlended in the shaft. The *friction-sheave hoist* passes the rope (or ropes) over the drive wheel but does not store it. (Sometimes the method is called *Koepe hoisting,* after its inventor.) A minor method that has fallen into disuse is *reel hoisting,* in which a single width of rope is wrapped in many layers. A new method devised in South Africa for very deep shafts is *multidrum (Blair) hoisting* using multiple ropes.

Applications of the drum, friction-sheave, and multidrum hoisting methods are summarized in Table 9.2.Since there are as yet no applications of multidrum hoisting in the United States, we will direct our attention only to the drum and friction-sheave methods.

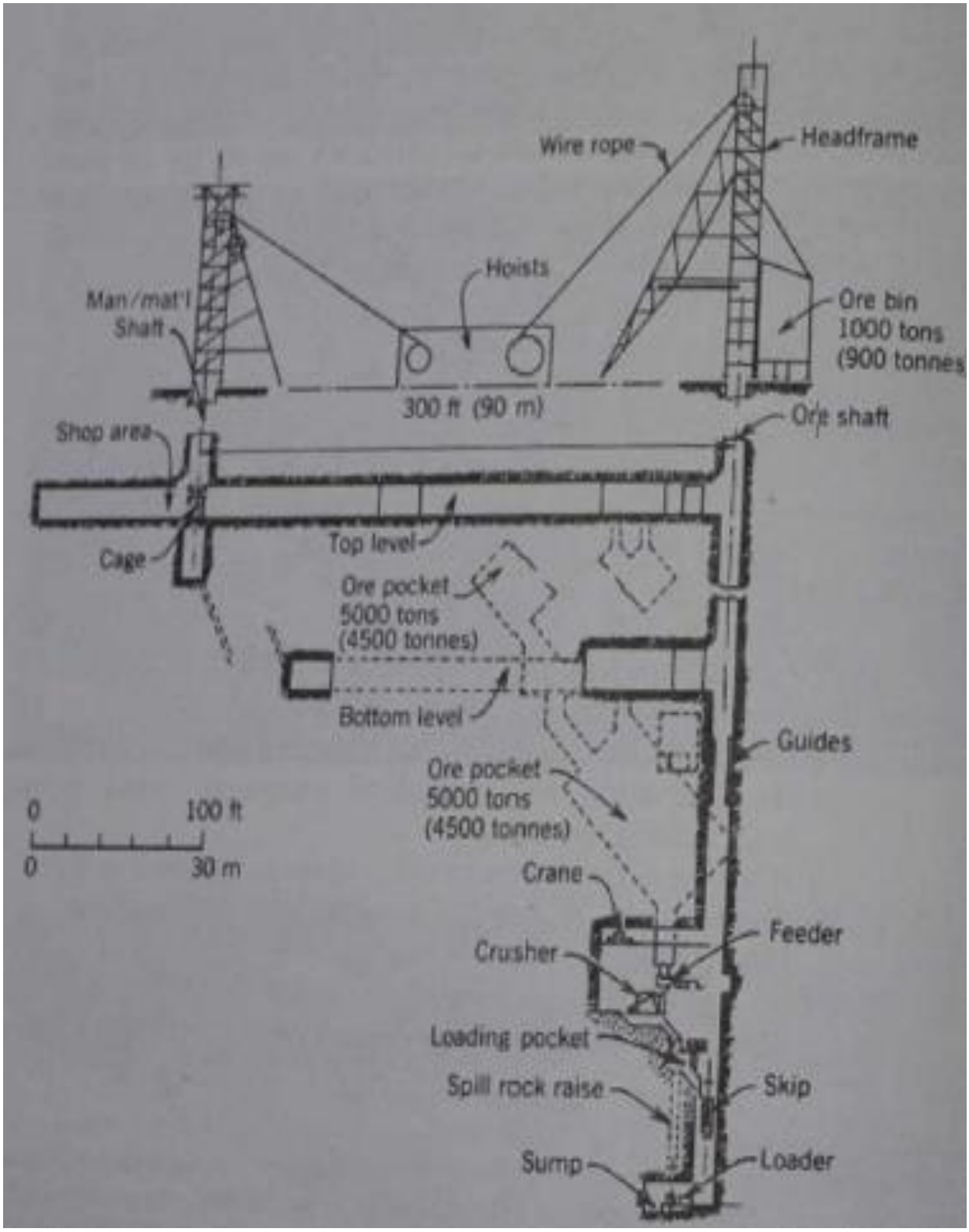


Figure 9.10. Components of the hoist plant installed with a vertical shaft. (After Lucas and Haycocks. 1973. By permission from Society of Mining Engineers, Inc., Littleton. CO.)

TABLE 9.2 Applications of Hoisting Methods

	Drum	Friction Sheave	Multidrum
Optimum depth, ft (km)	<6000 (1 .8)	<3000 (0.9)	>6000 (1 .8)
Maximum skip capacity* tons (tonnes)	28 (25)	85 (77)	56 (51)
Maximum output,-* tph (tonnes/hr)	900 (820)	2800 (2540)	1800 (1630)
Features	Single rope Multilevel Medium depth W ide use in U.S.	Multirope Single level Limited depth High production Best efficiency Wide use outside U.S.	Multirope Great depth

Drum hoisting. The simplest drum system is a single cylindrical drum, plus attached electrical and mechanicals. With two skips, it is suitable only for single-level hoisting, typical of coal mining. For multilelvel hoisting, a double drum hoist is used, clutched to permit one skip or cage to stop while the other is in motion. With the drum method, two skips or cages travel in balance: occasionally, a counterweight replaces one conveyance (e.g., with a single drum to permit multilevel hoisting). Drums must be ground-mounted; idler sheaves are located in the headframe to support the hoist ropes. Other drum shapes that are employed to improve power characteristics are the conical and cylindroconical. A double-clutched, double-drum hoist is shown in Figure 9.11 (top).

Friction-Shave-Hoisting. The friction-sheave system, at its name implies, operates on the principle that a hoist rope passing over a drive sheave is prevented from slipping even though there is a considerable difference in tension in the two segment of rope (Fig. 9.11. bottom). The tread material on the sheave is usually polyurethane; using a proper lubricant with the rope improves its nonslipping qualities. Like the drum method, two conveyances (or one and a counterweight) are used in balance. Unlike the drum method, a tail rope is preferable (it equalizes the rope weight and reduces slippage). Since the angle of wrap around the friction sheave also influences slippage, it can be increased by a deflection sheave or by ground mounting of the friction sheave (although headframe mounting is the

preferred location constructionwise). Intended for single-level hoisting, the friction sheave method must use a counterweight for multiple levels. Multiple (usually four) ropes are normally used; several ropes have better flexing qualities than a single rope and permit the use of a smaller sheave.

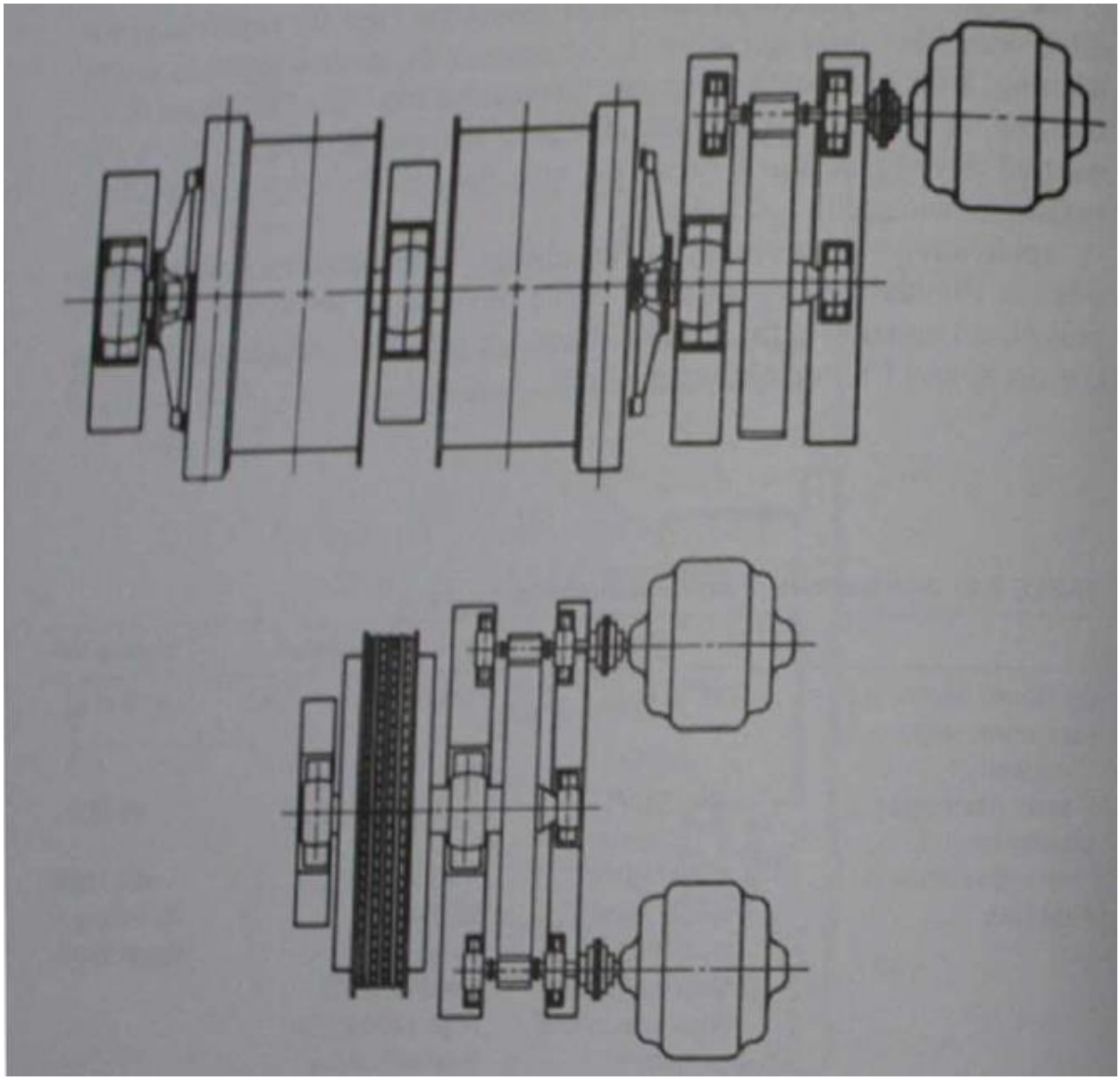


Figure 9.1 1 . Diagrams of mine hoists (Top) Double-clutched, double-drum hoist. (Bottom) Multirope friction-sheave hoist. (After Russell. 1982. By permission from Society of Mining Engineers. Inc., Littleton. CO.)

Performance Characteristics. With their different principles, shapes, and numbers and arrangements of ropes, the various hoisting systems exhibit quite different performance characteristics. In comparing operating costs, the parameter

that determines energy consumption and hence cost is the *duty cycle* of the hoist. It is a plot of the instantanecous power requirements vs. hoisting time. In Figure 9.12., rope-velocity diagrams and duty cycles for the principal hoisting systems are shown. The main differences are associated with drum shape and the use or absence of a tail rope.

Usually the most attractive system for high-speed, high-production low-energy skip hoisting from a single level at moderate depths, s the friction-sheave method. Using counterweights, it is also adaptable to multilevel hoisting, and with ground mounting of the sheaves can he used at depths to 5000 ft (1.5 km) or more. Details of the installation of a friction-sheave hoist and its hoist room and headframe appear in Figure 9.13.

9.6 SPECIAL TOPIC: DESIGN OF HOISTING SYSTEM

The design process for a mine hoisting system should be understood by the mining engineer, even though the design and installation are contracted to an engineer-constructor firm and the equipment bid to a hoist manufacturer. Typically, the mining company developing the mine assigns its own engineering department to monitor the entire process, including both the planning and construction of the surfacc hoist plant. The design process will now be examined in detail and illustrated by a numerical example.[2]

1. *Balanced hoisting.* All mine hoisting systems are operated in balance to reduce moments, torque, and power demand on the hoist. Generally, two conveyances (skips and/or cages) are suspended from one hoist; sometimes, when more than one level is to be serviced, a counterweight replaces one

[2]The design procedure, equations, and discussion are taken in large part from Harmon (1973).

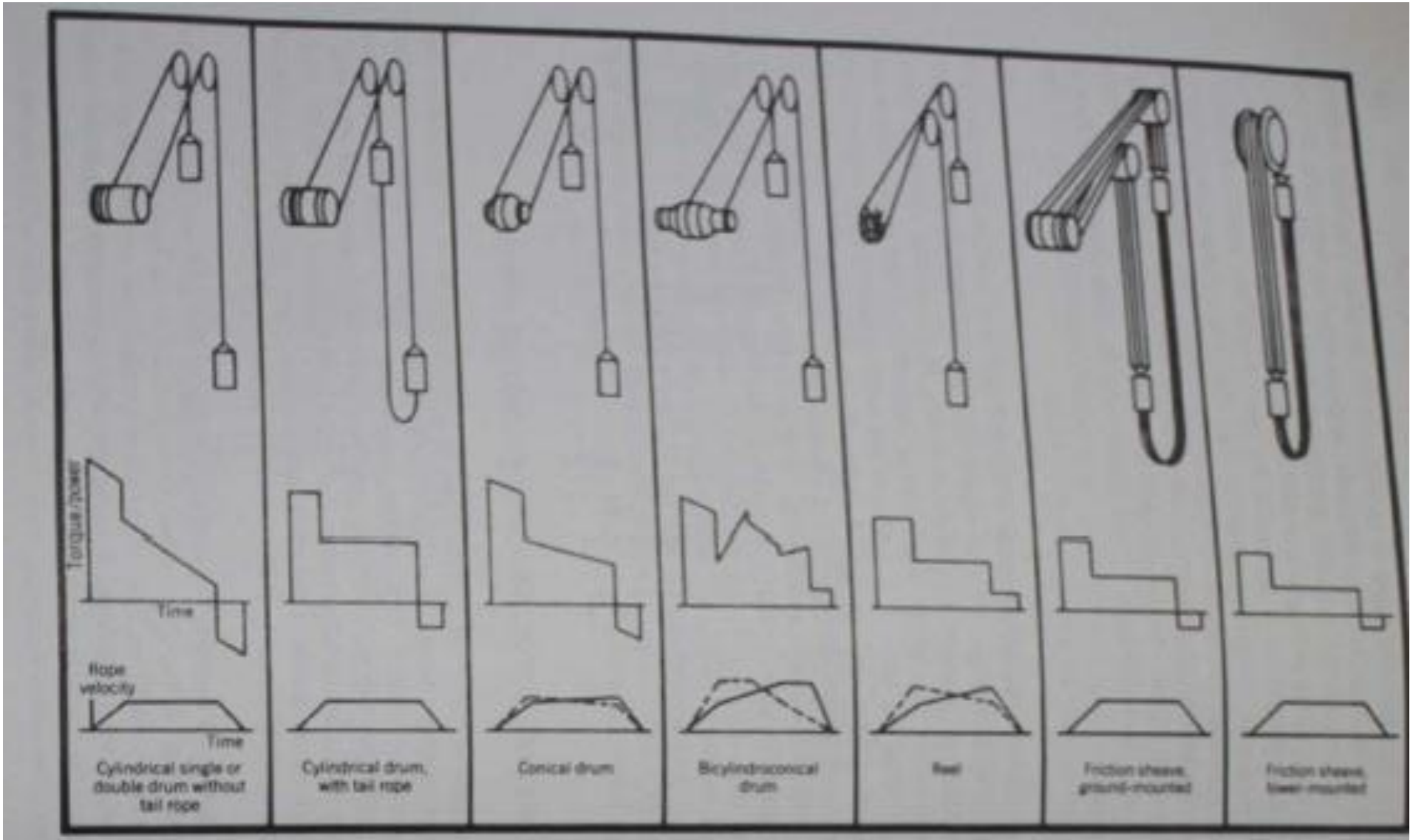


Figure 9.12. Hoist arrangements and duty cycles for common hoisting systems. (After Brucker, 1975. By permission from ASEA, Inc., Montreal, Canada.)

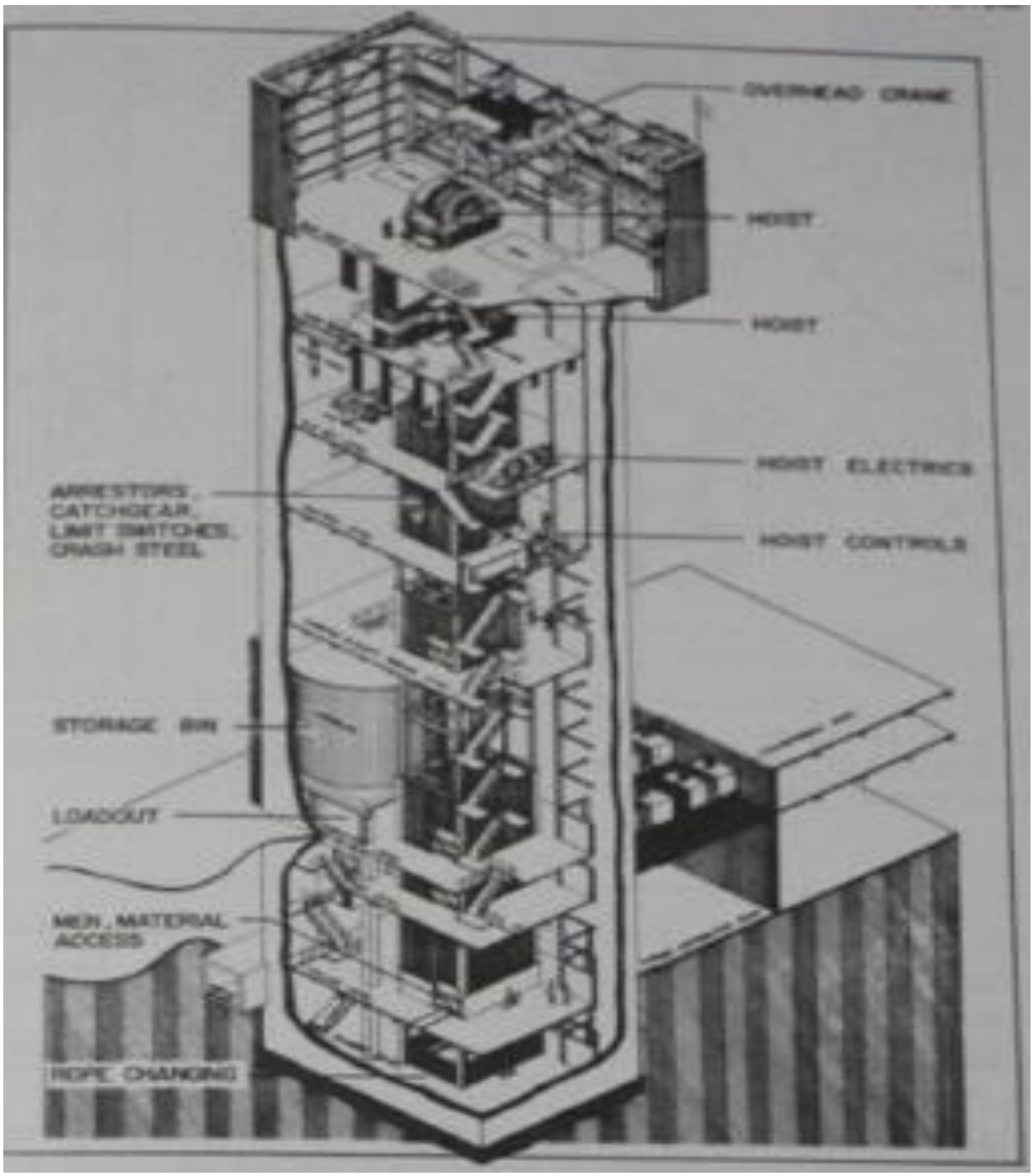


Figure 9.13. Multirope, friction-sheave hoisting system, mounted in headframe (After Butler and Schneyderberg. 1982. By permission from American Mining Congress. Washington, D C)

conveyance. It is designed with a weight equal to the dead load of the skip or cage plus one-half the live load. To further balance the loads, a tail rope can be installed; although rare with a drum hoist, it is normally used with a friction-sheave hoist to reduce slippage as well as moments.2. *Slippage in friction sheave hoisting.* Slippage occurs in a friction-sheavehoist if the ratio of the rope tensions exceeds a theoretical limit (Fig.9.14). The relationship is

$$\frac{T_1}{T_2} \leq e^{\mu\theta} \qquad (9.1)$$

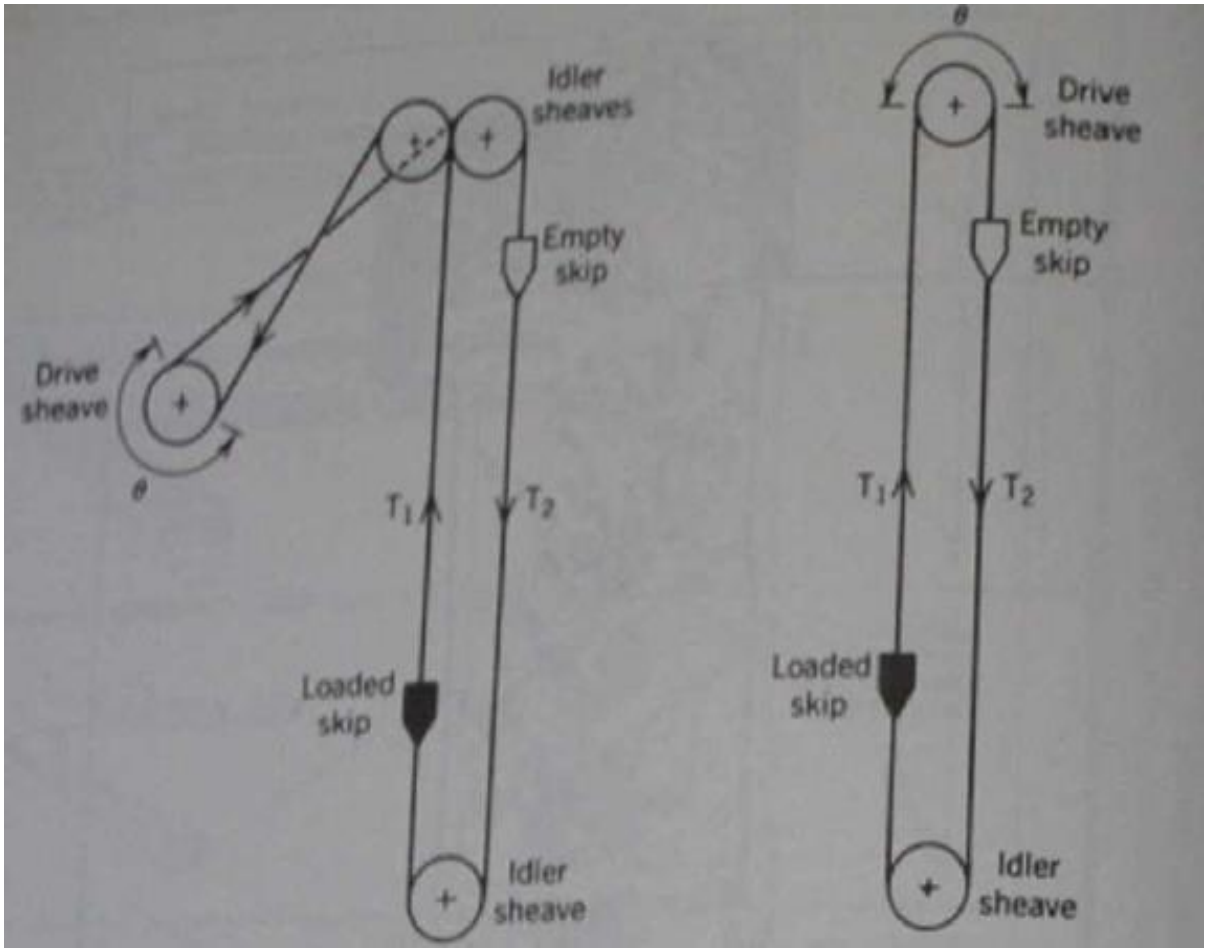


Figure 9.14. Analysis of friction-sheave hoist to determine If rope slippage occurs. *(Left)* Ground-mounted hoist (*Right*) Headframe-mounted hoist.

where T is rope tension. the subscripts I and 2 refer to the loaded and empty skips, respectively, e is the natural logarithmic base ,μ is coefficient of friction =0.45-0.50. and θ is angle of wrap from π radians (180°) for a headframe-mounted hoist to 1.3 π radians (240°) for a ground-mounted hoist. In Eq. 9.1, the limiting ratio is 1.5-1.6 for a headframe-mounted hoist and 1.8-1.9 for a ground-mounted hoist. To increase θ, a deflection sheave can be used. To increase μ. the sheave lining or rope lubricant can be varied.

3. *Wire rope size.* Wire rope has a complex structure; several of the more common types of hoist rope arc shown in Figure 9.15. Basically, there arc three types used for mine hoisting: round-strand, flattened-strand, and locked-coil. In general,

round-strand rope is used with drum hoists, flattened- strand with friction-sheave hoists, and locked-coil for any system at depths over 3200 ft (0.96 km).

In designing a hoisting system, the two properties of wire rope that are most important are weight per unit length and breaking strength. These properties are given for the three popular hoist ropes in Table 9.3. Note that properties for two qualities of steel are included for round-strand and flattened-strand rope.

In selecting the proper size of wire rope for a hoisting application, the most critical consideration is the factor of safety (Fig. 9.16). Values in the United States are established by MSHA and vary with the type of hoist,

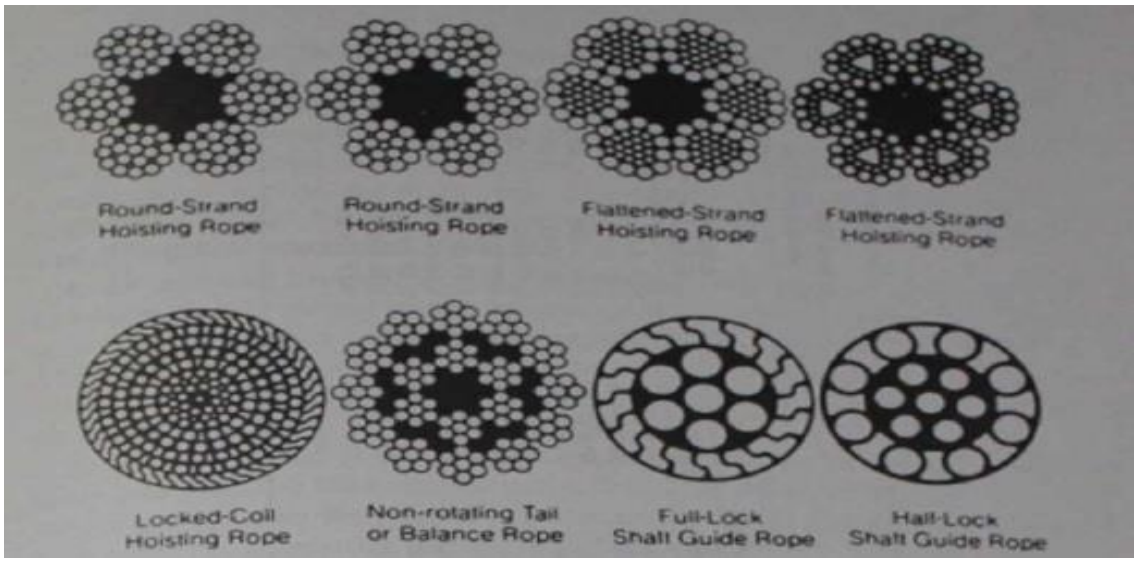


Figure 9.15. Construction of wire rope. The three types used most commonly in mine hoisting are round-strand, flattened-strand, and locked-coil. . (After Russell. 1982. By permission from Society of Mining Engineers. Inc., Littleton. CO.)

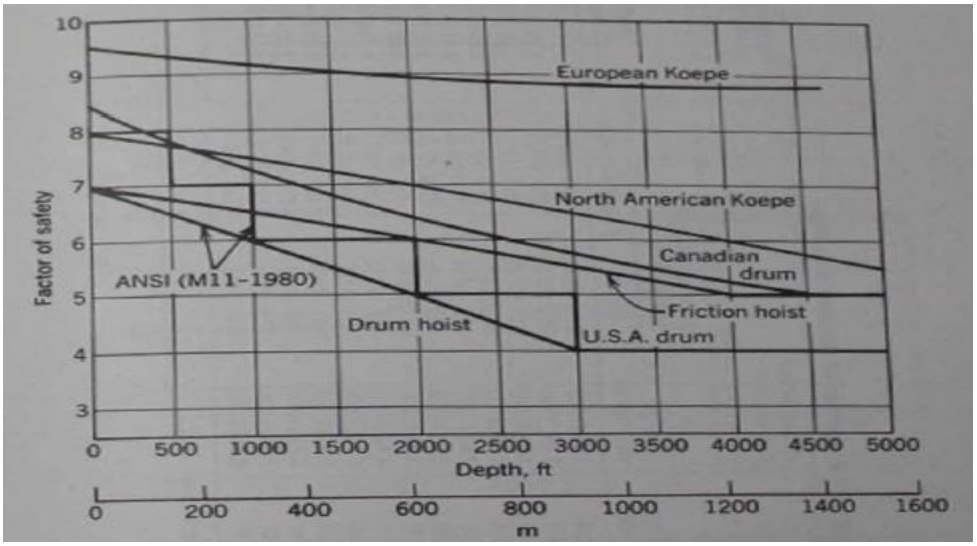


Figure 9.16. Required factors of safety for wire rope in mine hoisting. (By permission from Nordberg Hoists, Rexnord. Inc., Milwaukee.)

TABLE 9.3 Properties of Hoisting Wire Rope

Diameter, in. (mm)	Round Strand 6 × 19 Class Improved Plow Steel			Flattened Strand 6 × 27 Type H, 6 × 30 Type G			Locked Coil Improved Plow Steel	
	Weight, lb/ft (kg/m)	Breaking Strength, tons (tonnes)		Weight, lb/ft (kg/m)	Breaking Strength, tons (tonnes)		Weight, lb/ft (kg/m)	Breaking Strength, tons (tonnes)
		Normal	High-Strength		Normal	High-Strength		
⅜ (19.0)	0.95 (1.41)	23.8 (21.6)	26.2 (23.8)	1.01 (1.50)	26.2 (23.8)	28.8 (26.1)	1.37 (2.04)	35.0 (31.8)
½ (22.2)	1.29 (1.92)	32.2 (29.2)	35.4 (32.1)	1.39 (2.07)	35.4 (32.1)	39.0 (35.4)	1.87 (2.78)	46.0 (41.7)
1 (25.4)	1.68 (2.50)	41.8 (37.9)	46.0 (41.7)	1.80 (2.68)	46.0 (41.7)	50.6 (45.9)	2.43 (3.62)	61.6 (55.9)
1⅛ (28.6)	2.13 (3.17)	52.6 (47.7)	57.9 (52.5)	2.28 (3.39)	57.9 (53.5)	63.7 (57.8)	3.30 (4.91)	76.1 (69.0)
1¼ (31.8)	2.63 (3.91)	64.6 (58.6)	71.0 (64.4)	2.81 (4.18)	71.0 (64.4)	78.1 (70.9)	3.75 (5.58)	92.0 (83.5)
1⅜ (34.9)	3.18 (4.73)	77.7 (70.5)	85.4 (77.5)	3.40 (5.06)	85.5 (77.6)	94.0 (85.3)	4.78 (7.11)	115 (104)
1½ (38.1)	3.76 (5.63)	92.0 (83.5)	101 (91.6)	4.05 (6.03)	101 (91.6)	111 (101)	5.65 (8.41)	135 (122)
1⅝ (41.3)	4.44 (6.61)	107 (97.1)	118 (107)	4.75 (7.07)	118 (107)	130 (118)	6.86 (10.24)	155 (141)
1¾ (44.4)	5.15 (7.66)	124 (112)	136 (123)	5.51 (8.20)	136 (123)	150 (136)	7.56 (11.25)	182 (165)
1⅞ (45.3)	5.91 (8.80)	141 (128)	155 (141)	6.33 (9.42)	176 (160)	191 (173)	9.00 (13.39)	212 (192)
2 (50.8)	6.72 (10.00)	160 (145)	176 (160)	7.20 (10.71)		215 (195)	9.77 (14.54)	240 (218)
2¼ (54.0)	7.59 (11.30)	179 (162)	197 (179)	8.13 (12.10)		240 (218)		
2½ (63.5)	8.51 (12.66)	200 (181)	220 (200)					

Source: Russell, 1982.

depth, and Whether personnel are being hoisted; the stepped curve gives the minimum allowable values to ho.st personnel. Standards set by the American National Standards Institute (ANSI) are the maximum load for personnel, or materiel. Since die weight of the rope must be taken into account in determining the total load on the hoist, the design process in selecting rope size becomes a trial-and-error one. Eventually, depth imposes a limit in single-lift hoisting, because rope weight increases with depth exceeding rope strength at some critical depth.

4. *Sheave and drum dimet r* . To minimize flexing and stressing of the wire rope as it is wound over a sheave or drum, a recommended minimum ratio of drum or sheave diameter to rope diameter should be observed (Fig. 9.17). Since the cost of wire rope is modest, there may be occasions, especially in shallow shafts, where it is more cost-effective to select a smaller drum diameter and replace the hoist rope oftener.

5. *Rope fleet angle* . This is the angle subtended by the hoist rope and the centerline from the idler sheave to the drum. To reduce rope abrasion in the sheave groove, the fleet angle is restricted to 1-l,5^0. The principal effect of such a limit is to restrict the width of the drum.

6. *Skip size vs . hoisting velocity* . To achieve a desired production rate in a shaft, the design engineer seeks a balance between skip size and hoisting velocity. The ultimate limit on skip size is rope strength, and on hoisting velocity it is energy consumption. As a compromise, it is generally advantageous to hoist the largest skip load possible at the lowest possible rope velocity.

7. *Hoisting cycle.* The relationship of time to distance in hoisting is referred to as the *hoisting cycle.* Calculation of time and distance elements is accomplished with the following formulas:

a. *Acceleration time* $t_a = \frac{V}{a}$ (9.2)

Where V is hoisting velocity and a is acceleration rate; generally, acceleration rate and time equal retardation rate time t_r , respectively.

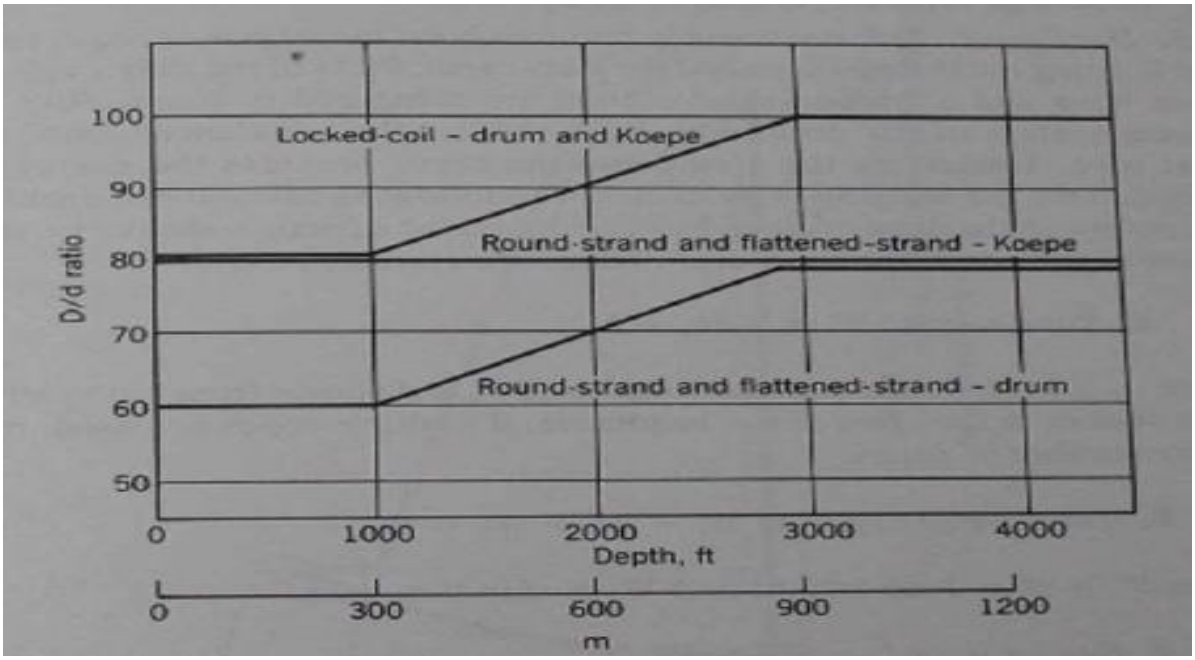


Figure 9.17. Suggested minimum ratio of drum-to-rope diameter for varying hoisting depths. (By permission from Nordberg Hoists, Rexnord, Inc.. Milwaukee.)

b. *Acceleration distance* $h_a = \frac{1}{2} a t_a^2$ - *iota*

= retardation distance h_r (9.3)

c. *Constant-velocity distance* $h_v = h_t - h_a - h_r$. (9.4)

where h_t is total hoisting distance (from loading pocket to headframe bin).

d. *Constant-velocity time* $t_v = \frac{h_v}{v}$ (9.5)

e. *Cycle time* $t_r = 2(t_a + t_v + t_r + t_d)$ (9.6

per round trip, where t_d is load or dump time.

8. *Duty cycle.* The relationship between hoist motor power requirements and hoisting cycle times is called the *duty cycle.* Plots of the duty cycle for a drum hoist and a friction-sheave hoist are compared in Figure 9.18. The sloping section of the drum duty hoist reflects the unbalanced load of the hoist rope. Integrating the area under the curve provides the energy consumption for the hoisting-duty cycles. The following calculations enable the key points on the duty cycle to be determined for a friction-sheave hoist (the power equations differ for a drum hoist; sec Harmon. 1973):

a. *Rope weight* $W_r = w_r (h_1 + h_h)$ (9.7)

where w_r is rope weight per unit length and *hh* is distance from bin to idler or drive sheave at the apex of the headrame; if multiple ropes are used, multiply by number of ropes.

b. *Total weight o f load* $W_1 = (\backslash W_r + W_s + W_o$ (9.8)

where W_s is skip dead weight and W_o is skip live load.

b. Design load L = FS x W_l (9.9)

where FS is factor of safety.

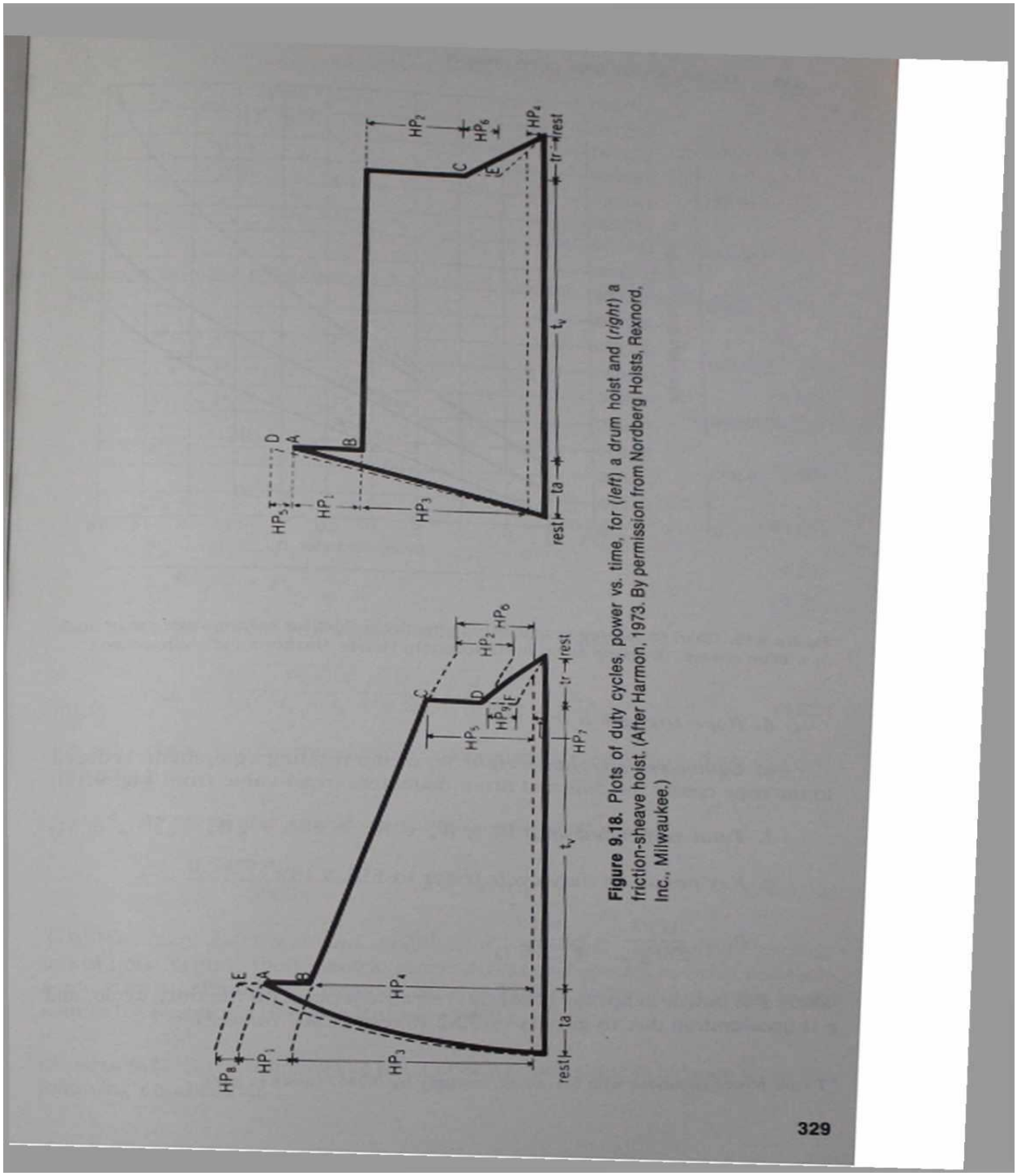


Figure 9.18. Plots of duty cycles, power vs. time, for (*left*) a drum hoist and (*right*) a friction-sheave hoist. (After Harmon, 1973. By permission from Nordberg Hoists, Rexnord, Inc., Milwaukee.)

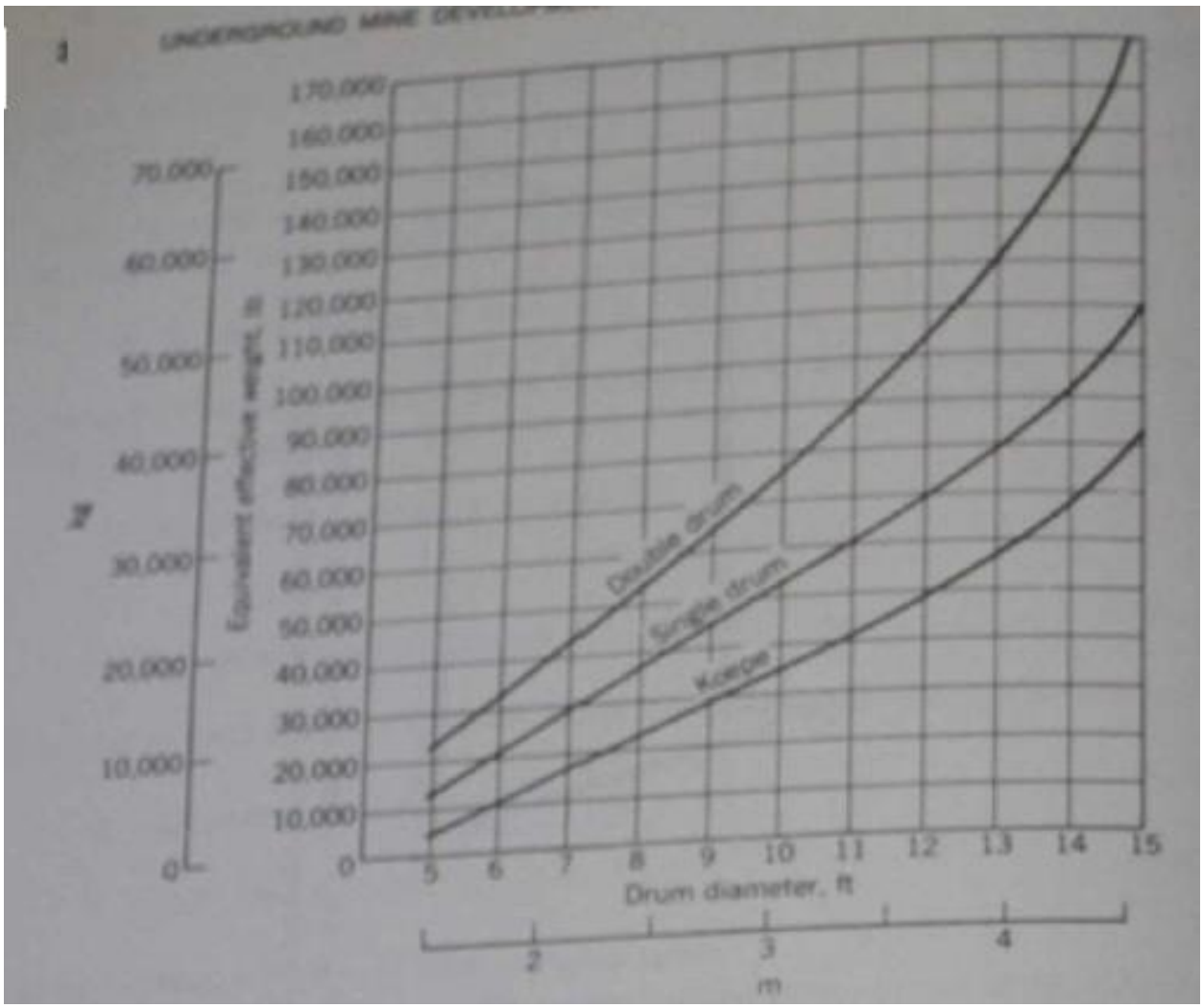


Figure **9.19**. Chart to determine equivalent effective weight for varying diameter of drum of friction sheave. (By permission from Nordberg Hoists, Rexnord, Inc.. Milwaukee.)

d. *Rope strength* $S \geq L$ (9.10)

e. *Equivalent effective weight* W_e of the rotating equipment, reduced to the rope center for different drum diameters (read value from Fig.9.19).

f. *Total suspended load* $W\text{-}W_e + W_o + 2W_s + 2W_r$ (9.11)

g. *Keys points on duly cycle* (refer to Fig. 9.19):

$$P_1 = \frac{wv^2}{550gt_a} = \frac{wv^2}{17{,}700t_a}, \text{ hp} \quad (9.12)$$

where P is power in hp. the subscript refers to a point on the duty cycle, and g is acceleration due to gravity - 32.2 ft/ sec^2 (9.807 m/ sec^2).

$$P_2 = -\frac{wv^2}{17{,}700t_e} \quad (9.13)$$

$$P_3 = \frac{wv}{550} \quad (9.14)$$

[3] To use power equations with S.I. units, multiply by 0.7457 (units in kW).

$$P_4 = \frac{W_v V}{550}\left(\frac{1-\eta}{\eta}\right) \quad (9.15)$$

η is hoist efficiency as a decimal; assume 90% for friction-sheave

$P_a = P_1 + P_3 + P_4$ (9.16)

$P_B = P_3 + P_4$ (9.17)

$P_c = P_2 + P_3 + P_4$ (9.18)

$$P_5 = \frac{1.2(0{,}75P_A)}{t_a} = \frac{0{,}9P_A}{t_a} \quad (9.19)$$

$$P_6 = \frac{-0{,}9P_A}{t_r} \quad (9.20)$$

$P_D = P_A + P_5$ (9.21)

$P_E = P_C + P_6$ (9.22)

h. RMS *(root mean square) power for ac motor:*

$$P_{rms} = \sqrt{\frac{P_D^2 t_a + P_B^2 t_v + P_E^2 t_r}{0{,}5t_a + t_v + 0.5t_r + 0.25t_d}} \quad \text{hp}$$

i. *Approximate energy consumption for duty cycle:*

$$E = \frac{0.7457\, P_B\,(t_a + t_r)}{3600\eta} \text{ kw-hr/trip}$$

9. *Auxiliary electrical and mechanical equipment.* For further particulars of hoist drums, their electric motor drives and control devices, and their associated mechanical equipment such as brakes and clutches, refer to accounts by Harmon (1973) or Russell (1982).

example 9.1. A friction-sheave hoist operates two skips in balance under the following conditions:

Shaft depth	1000 ft (305 m)
Skip live load	5 tons (4.5 tonnes)
Skip dead-load/live-load ratio	1,2
Hoist ropes	4 @ 1 in. (25,4 mm) flattened strand, normal strength
Sheave diameter	10,3 ft (3.14 m)
Hoisting velocity	20 ft/sec (6.1 m/sec^2)

Hoisting cycle	t_a=10 sec
	t_b= 39.75 sec
	t_r= 8 sec
	t_d=10 sec
Hoist efficiency	90%

Check for rope slippage, and determine the rms rating of ac electric motor to drive the hoist and the approximate energy consumption per trip,

SOLUTION. *Use Eq.* 9.6 to find the cycle time:

t_r=- 2(10 +39.75 + 8 + 10) =135.5 sec

Skip dead load W_s= 1.2(5) =6.0 tons = 12,000 lb

Rope *weight (Eq . 9.7)* W_r= 4(1.80X1000) = 7200 lb

using *Table* 9-3 to obtain the rope weight per unit length.

Total weight of load (Eq . 9.8) W_l = (7200 + 10.000 + 12.000) = 29,200 lb

Check rope *slippage (Eq.* 9.1):

Loaded *skip* T_1= W_1 = 29.200 lb

Empty skip T_2=: W_s +W_r= 12.000 + 7200 = 19.200 lb

$$\frac{T_1}{T_2} = \frac{29{,}200}{19{,}200} = 1{,}52 < 1{,}6$$

which is satisfactory for a headframe-mounted hoist.

For sheave diameter = 10.3 ft. read equivalent effective weight (Fig. 9.19)

W_e= 37.000 lb.

Total suspended load (Eq. 9.11) W = 37.000 + 10.000+24.000 + 14.400= 85.400 lb

Calculate points on duty cycle (Eqs. 9.12-22):

$$P_{1`} = \frac{(85{,}400)(20)^2}{17{,}700(10)} = 193 \text{ hp}$$

$$P_2 = -\frac{(85{,}400)(20)^2}{17{,}700\ (8)} = -241 \text{ hp}$$

$$P_3 = \frac{(10{,}100)(20)}{550} = 364 \text{ hp}$$

$$P_4 = \frac{(10{,}100)(20)}{550}\left(\frac{1-0{,}9}{0{,}9}\right) = 40 \text{ hp}$$

P_A=193 + 364 + 40 = 597 hp

P_B = 364 + 40 = 404 hp

P_C-=-241 + 364 + 40 = 163 hp

$P_5 \frac{0,9(597)}{10} = 54$ hp

$P_6 = -\frac{0,9(597)}{8} = -67$ hp

P_D = 597 + 54 = 651 hp

P_E = 163 - 67 = 96 hp

Using these values, the duty cycle can be plotted, similar to Figure 9.18. Calculate rms power for an ac motor (Eq. 9.23):

$$P_{rms} = \sqrt{\frac{(651)^2(10)+(404)^2(39,75)+(96)^2(8)}{0,5(10)+39,78+0,5(8)+0,25(10)}} = 459 \text{ hp (342 kw)}$$

Calculate approximate energy consumption (Eq. 9.24):

$$E = \frac{0,7457(404)(10+39,75)}{3600(0,90)} = 4,63 \text{ kw-hr/trip}$$

PROBLEMS

9.1a. Calculate the daily production of a shaft equipped with a balanced friction-sheave hoisting system, given the following:

Shift time	7.2 hr
Shifts/day	3
Skip capacity	12 tons (11 tonnes)
Cycle time, 1 trip	85 sec/skip

b. Also calculate the approximate energy consumption per skip hoisted, if the average power consumed is hp (730 kw), the hoist efficiency is 85%, the acceleration time is 6,5 sec, and the constant velocity time is 63,5 sec

9.2a. A hoisting cycle consists of the following elements:

Constant-speed travel time	80 sec
Acceleralion=retardation time	6 sec
Load=dump time	8 sec
Shift time	7.5 hr
Shifts/day	3

Skip capacity	12 tons (11 tonnes)

For a balanced hoisting system,calculate the expected daily production from the shaft.

b. Also calculate the approximate energy consumption per skip hoisted if the average power consumed is 12oo hp (800 kw) and the hoist efficiency is 87%.

9.3 Select the proper size of friction-sheave hoist, calculate the hoisting and duty cycles, and determine the approximate unit energy consumption and cost for the following hoisting system:

Two-balanced skips and cages, sheave in headframe

Production rate	6300 tons/day (5715 tonnes/day)
Working time	3 shifts/day. 7 hr/shift
Shaft depth (I level)	2250 ft (686 m)
Headframe height	100 ft (30 m)
Bin height	50 ft (15 m)
Hoist rope	4 round strand
Skip + age dead load	0.67 live load (skip capacity)
Load time = dump time	6.0 sec
Hoisting speed	2250 ft/min (11.43 m/sec)
Acceleration = retardation	7.5 ft/sec^2 (2.29 m/sec^2)
Power cost	5¢/kW-hr

Determine the following design elements:

Components and total cycle time (I round trip), sec

Skip capacity, tons (tonnes) (to nearest 0.1 ton or tonne)

Rope size. in. (mm) (standard size, normal strength)

Rope slippage (if unsafe, recommend design change but do not recalculate)

Sheave diameter, ft (m) (to nearest ft or m)

Hoist duty cycle (plot of hp or kW vs. time)

RMS power rating of ac hoist motor, hp (kW)

Unit energy consumption. kW-hr/trip and kW-hr/ton (kW-hr/tone)(approximate)

Unit energy cost, ¢ /ton (¢/tonne) (approximate)

Compare with a drum hoist requiring a 1650-hp (1230 kW) motor, 3 kW-hr/ton (3.3 kW-hr/tone), and 15¢/ton (16,5¢/tone) and explain the difference.

9.4 Select the proper size of friction-sheave hoist, calculate the hoisting and duty cycles, and determine the approximate unit energy consumption and cost for the following hoisting system:

Two-balanced skips and cages, sheave in headframe

Production rate	7500 tons/day (6804 tonnes/day)
Working time	3 shifts/day. 7,5 hr/shift
Shaft depth (I level)	1850 ft (564 m)
Headframe height	150 ft (46 m)
Bin height	100 ft (30 m)
Hoist rope	4 flattened standart
Skip + age dead load	0.8 live load (skip capacity)
Load time = dump time	8.0 sec
Hoisting speed	1800 ft/min (9.14 m/sec)
Acceleration = retardation	6,0 ft/sec^2 (1,83 m/sec^2)
Power cost	4¢/kW-hr

Determine the following design elements:

Components and total cycle time (I round trip), sec

Skip capacity, tons (tonnes) (to nearest 0.1 ton or tonne)

Rope size. in. (mm) (standard size, normal strength)

Rope slippage (if unsafe, recommend design change but do not recalculate)

Sheave diameter, ft (m) (to nearest ft or m)

Hoist duty cycle (plot of hp or kW vs. time)

RMS power rating of ac hoist motor, hp (kW)

Unit energy consumption. kW-hr/trip and kW-hr/ton (kW-hr/tone)(approximate)

Unit energy cost, ¢ /ton (¢/tonne) (approximate)

Compare with a drum hoist requiring a 1650-hp (1230 kW) motor, 3 kW-hr/ton (3.3 kW-hr/tone), and 15¢/ton (16,5¢/tone) and explain the difference.

10

UNDERGROUND MINING: *UNSUPPORTED* METHODS

10.1 *CLASSIFICATI N OF METHODS*

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

Reflecting the importance of ground support, underground mining methods are categorized in three classes on the basis of the extent of support utilized (Table 3.1). They are unsupported, supported, and caving. The unsupported class, the subject of this chapter, consists of those underground methods which are essentially self-supporting and require no major artificial system of support to carry the superincumbent load, relying instead on the walls of the openings and natural pillars. (The superincumbent load is comprised of the weight of the overburden and any tectonic forces acting at depth.) This definition o f unsupported methods does not preclude the use of rock or roof bolts or light structural sets of timber or steel, provided that such artificial support does not significantly alter the load-carrying ability of the natural structure (Morrison and Russell, 1973).

Theoretically, the unsupported class of methods can be used in any type of mineral deposit (except placer) by varying the ratio of span of opening to width of pillar to achieve the desired mine life expectancy. Since the stable size of opening is determined by the depth and the strength properties of the ore and overlying rock, the safe span conceivably could range from a few feet (meters) to over 100 ft (30

m). Practically, the unsupported methods are not universally applicable and are not universally applicable and are limited to deposits with favorable characteristics. The unsupported class, however, is still the most widely used underground, accounting tor over 80% of the U.S. mineral production from subsurface mines.

When exploitation workings cannot be held open for the required life expectancy without the substantial use of artificial support systems, supported methods are used. When the deposit and overlying rock are sufficiently weak and subsidence is tolerable, then support is withheld, undercutting is carried out as necessary, and a caving method is employed. Supported methods are little used today, while caving is growing in popularity.

Other factors, of course, enter into the selection of a method (Section 3.7). Their influence is considered in the discussion of each method and also in the summary for underground mining methods (Chapter 13),as was done with surface methods. Although method selection remains largely an empirical art, the evaluation of all determining factors must be done as objectively as possible (Boshkov and Wright. 1973).

While there is a lack of agreement among the various schemes to classify underground methods, we shall employ the generic one presented in Table 3.1 . In that classification, the following are considered unsupported methods:

1. Room and pillar mining
2. Stope and pillar mining
3. Shrinkage sloping
4. Sublevel sloping

Unlike surface mining, there is little distinction in the cycle of operations for the various underground methods (except in coal mining), the differences occurring in the direction of mining (vertical or horizontal), the ratio of opening-to-pillar dimensions, and the nature of artificial support used, if any. Of the unsupported methods, room and pillar mining and stope and pillar mining employ horizontal openings, low opening-to-pillar ratios, and lightto-moderate support in all

openings. Shrinkage and sublevel stopping utilize vertical or steeply inclined openings (and gravity for the flow of bulk material)high opening-to-pillar ratios, and light support mainly in the development openings.

10.2 ROOM AND PILLAR MINING

In room *an d pillar mining* openings are driven orthogonally and regular intervals in a mineral deposits-usually flat-lying (or nearly so), tabular, and relatively thin—forming rectangular or square pillars for natural support. If the deposit and method are very uniform, the appearance of the mine in plan view is not unlike a checkerboard or the interesting streets and rectangular blocks of a city. As discussed in Sections 9.3 and 9.4 both development openings (entries) and exploitation openings (rooms) closely resemble one another; both a driven parallel and multiple, and when connected by crosscuts, pillars are formed. Driving several openings at one time increases production and efficiency by providing numerous working places and improves ventilation and transportation as well.

By its very nature, room and pillar minings is ideally suited to the underground production of coal and several nonmetallic and few metallic minerals. Largely an American method, it matured with the coal industry, and today still accounts for nearly 85% of U.S. underground coal production and about 62% of U.S. underground mineral production. Properly considered a large-scale method, it faces increasingly stiff competition from longwall, a caving method which is highly productive and responsible for the balance of the coal produced below ground. Our discussion here is mainly of the application of the room and pillar method to the mining of coal. The term, coal refers to bituminous, although it may include lignite and anthracite as well. See again Table 1.5 for salient industry statistics.)

A generalized representation of room and pillar mining appears in Figure 10.1 comparing conventional and continuous equipment. So-called *conventional mining* is cyclical, employing mobile, mechanized equipment to carry out the production cycle of operations. As shown it requires at least 5 working places for a smooth cycle and as many as 8-12 (to allow for delays) for high efficiency (Lucas

and Haycocks, 1973). With *continuous mining*, separate unit operations arc eliminated or performed by a single high-performance continuous mining machine. One working place theoretically suffices for the miner with a second for the bolter, but again, for high efficiency, more places (4-6) are provided. Of today's coal production by the room and pillar method, approximately 25% is mined by conventional equipment and 75% by continuous, with the trend toward continuous mining. (Little if any coal is produced for the commercial market by hand methods; essentially all is now mined by mechanized equipment.) In general, conventional equipment is advantageous in hard scams, seams with hard-rock partings, gassy conditions, and variable scam height; continuous is superior in design as a noncyclic system and advantageous in thin seams, where there is a bad roof or a limited number of working places, and to produce a fine product (Stefankoand Bise, 1983).

In addition to number of working places, there are several other design parameters to be selected in room and pillar mining, the most important of which are the dimensions of the openings. The height of opening, of course, is equal to the thickness of the bed or deposit, unless the bed is too thin to permit mechanized operations, in which case some rock in the roof or floor is mined along with the coal (typically, U.S. coal seams that arc minable underground are 2.5-15 ft, or 0.8-4.5 m, thick). The width of opening (span) for production or recovery reasons should be a maximum, but for roof control and safety reasons, it is limited by M SH A (to 20 ft, or 6 m, if roof bolts are used, or to 30 ft, or 9 m, if other support is used as well) (Stefanko and Bise, 1983; Anon., 1984a). The spacing (centers) of openings, entries or rooms, is sufficiently close that the stress distribution around one opening may affect that around an adjacent opening (Morrison and Russell, 1973); it varies from 40-100 ft (12-30 m) in coal mines. Finally, the maximum spacing between crosscuts is limited by ventilation concerns and is usually specified in state laws (approximately 100 ft, or 30 m).

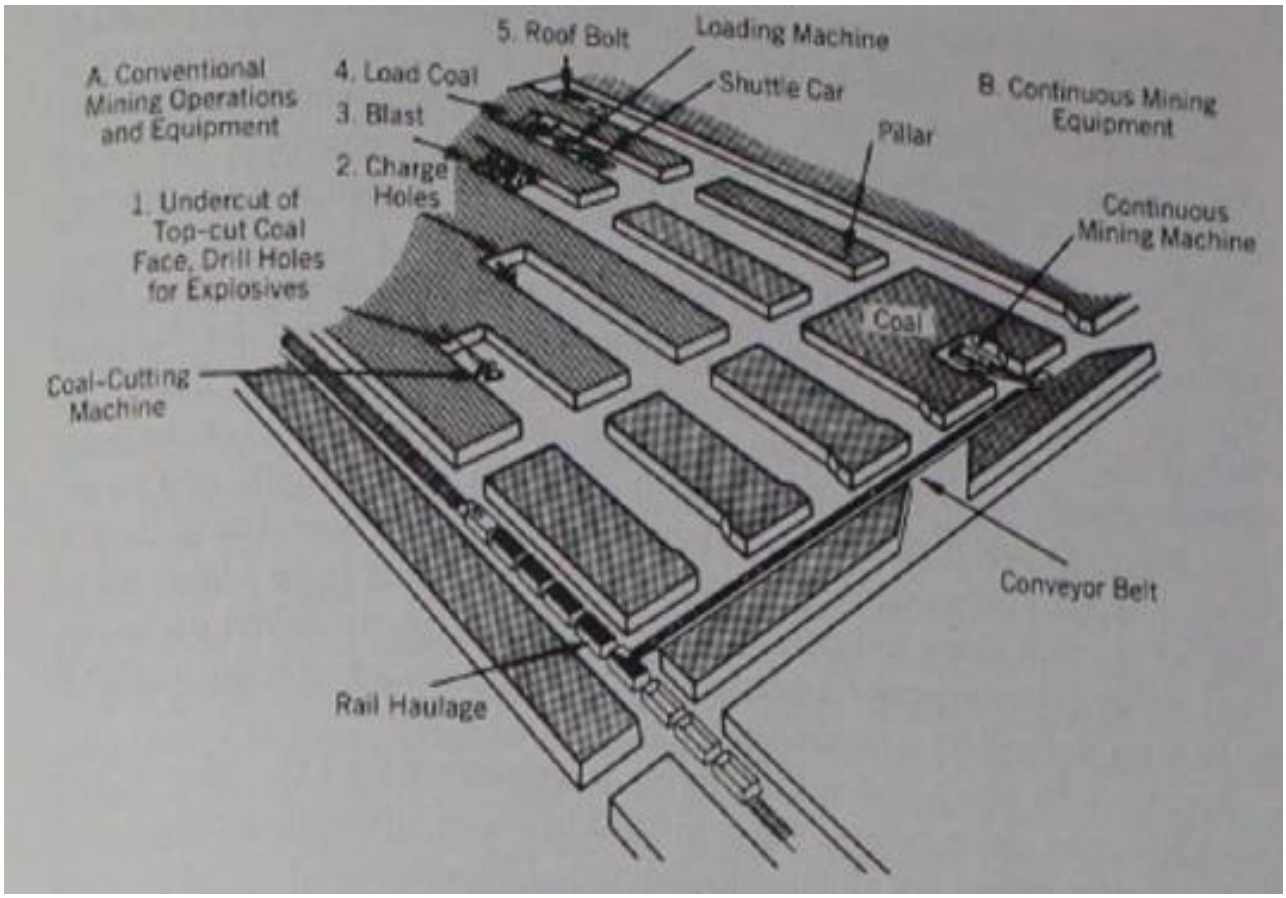


Figure 10.1. Generalized representation of room and pillar mining. Both conventional and continuous equipment are shown. (After Anon.. 1982a.)

In design practice, the selection of pillar size (or ratio of pillar-to-opening widths) and the nature and amount of support are carefully coordinated (Bullock, 1982b). When an opening is created, the unit weight of overburden above it is transferred by an arching action to the sides of the opening or adjacent pillars (see theory in Section 10.7). Because high stresses result, it may be necessary to design the width of the pillar to avoid stress superpositioning, especially in the case of long-life entries. To do so requires a pillar width equal to at least three times the opening width. For example, if entries are driven 20 ft (6 m) in width, their center spacing must be 80 ft (24 m) to prov.ide a pillar width of 60 ft (18 m), a ratio of 3:1. With such a design, a square roof -bolt pattern on- ft (1,2-1,5 m) centers would be likely. In exploitation, wider rooms (to 30 ft, or 9 m) would probably be driven on closer centers (say, 40-60 ft, 0r 12-18 m) because of their more temporary nature.

The layout of most room and pillar mines is remarkable similar, even under different conditions and equipment. Once the main and subsequent development has been completed (review Section 9.2 and Fig. 9.6), production panels of rooms are driven on one or both sides of the room entries. Panels range from 2000-4000 ft (600-1200 m) in length, and rooms vary from 300-400 ft (90-120 m) in length (Stefanko and Bise. 1983). Exploitation proceeds either on the advance or the retreat; Figure 10.2 illustrates a drift mine, with main and panel entries only, and rooms driven on the retreat. Since pillars are not recovered, rooms are driven a maximum width (30 ft, or 9 m) and pillars a minimum (10 ft, or 3 m).

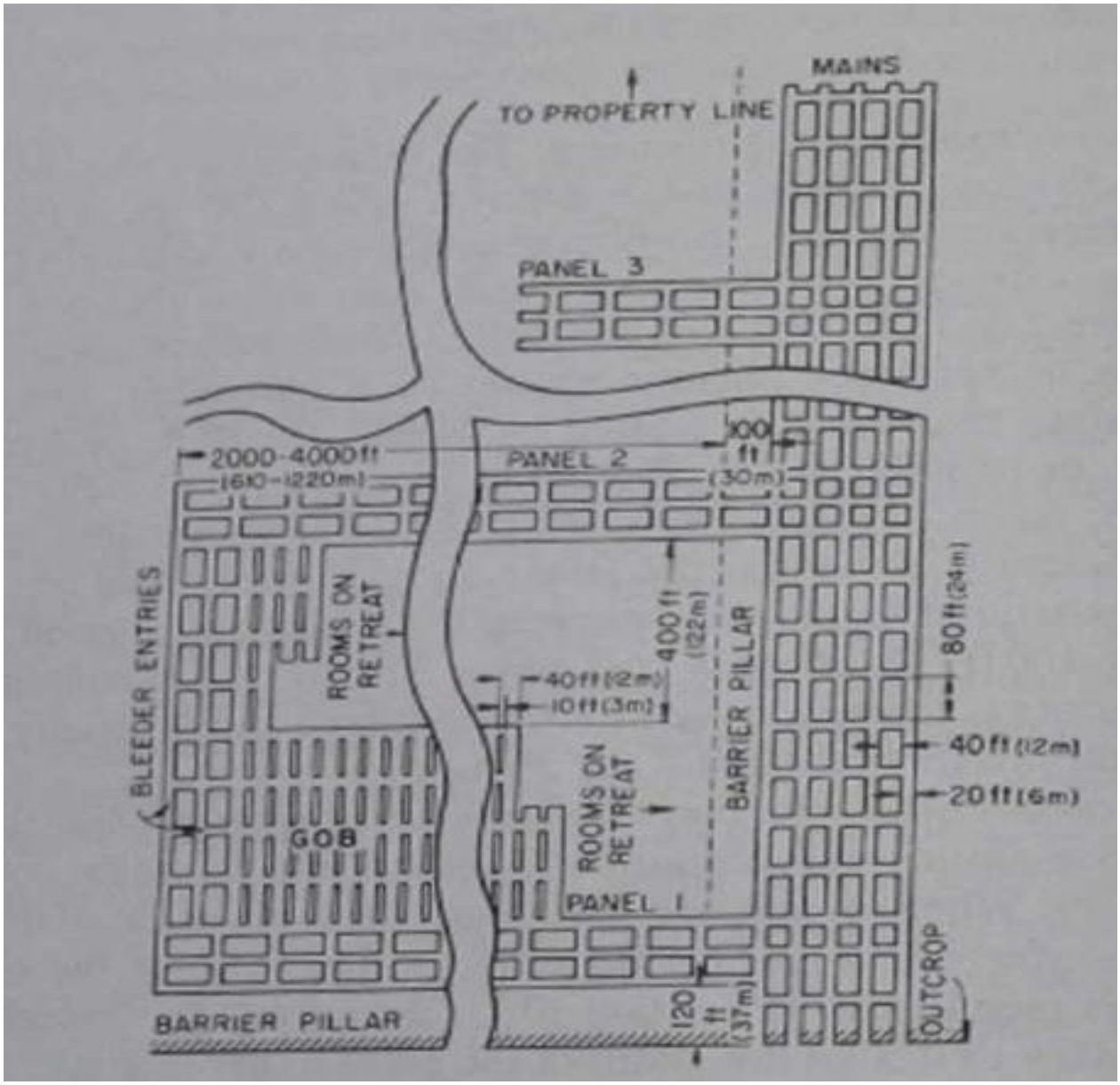


Figure 10.2. Room and pillar mining, driving rooms on the retreat without pillar recovery. (After Stefanko and Bise. 1983. By permission from Society of Mining Engineers, Inc. Littleton, CO.)

Recovery in room and pillar mining is enhanced by extracting pillars (normally only chain pillars, those formed between adjacent openings, are taken; barrier pillars are allowed to remain). This practice is referred to as *second mining* or *pillar recovery* (room driving is fermed *first mining*). Partial extraction recovers some pillars or portions of most pillars; wholesale roof collapse and subsidence usually do not occur. If more complete to total extraction is practiced, then caving is induced and surface eventually occurs. In that event, room and pillar mining ceases to be an unsupported method and becomes a caving method. Among the conditions which should prevail collectively for essentially complete pillar recovery are (1) an immediate roof that requires little support, (2) an overlying series of beds several times the thickness of the deposit which will cave readily, (3) a thick seam of high-quality coal, (4) moderate to deep cover, and (5) skilled personnel and supervision (Boshkov and Wright,1973; Stefanko and Bise, 1983).

Figure 10.3 *(top)* depicts a room and pillar coal mine in wich complete extraction of pillars (except bleeder chain pillars and barrier pillars) is practiced. . It is a shaft mine with nine main and five panel entries but without rooms (pillar recovery constitutes exploitation). Mining is carried out half-advance, half-retreat; that is. the panel entries are driven on the advance and pillars recovered on the retreat. This version of the room and pillar method is called the *block system* , because pillars are square (they a generally recovered). Details of pillar recovery for the same mine are shown in Figure 10.3 (bottom); the extraction technique is called pocket and wing. Other pillar recovery methods are open-ending, slabbing, pocket and stump, and splitting (Lucas and Haycocks, 1973). Additional temporary roof support (cribs, hydraulic jacks) is usually employed because of the greater risk involved. The pillar line in Figure 10.3 *(bottom)* is staggered; in continuous mining, it is usually parallel to the rooms or crosscuts while in conventional mining it is angled (ideally, 45^0) to provide more working places. Recoveries with and without pillar extraction vary as follows (Lucas and Haycocks, 1973);

No pillar extraction, conventional equipment 40-50%

continuous equipment 50-60

With pillar extraction 70-90

Pillar recovery as a practice is diminishing in coal mining but still far exceeds that in noncoal mines.

Room and pillar mining in the U .S. coal industry has been developed to a high degree of production efficiency. There is likewise a huge disparity in performance among mines, with the following selected as typical for comparable conditions (first mining in a 63-in., or 1.6-m, seam) with continuous and conventional equipment (Stefanko and Bise, 1983):

	CONVENTIONAL	CONTINUOUS
Production, tons(tonnes)/shift	504 (457)	446 (405)
Face crew	10	6
Productivity, tons(tonnes)/face employee-shift, clean coal	50 (46)	74 (67)

It is not unusual for conventional equipment to out produce continuous, but with a smaller crew, continuous is generally more productive.

Productivity data for the U.S. coal industry have followed a most revealing trend line. Prior to enactment of stringent health and safety legislation in 1969, productivity had risen steadily since World War II, it then declined rather precipitously and has only recently resumed its upward rise. Today, US underground coal productivity averages 11 tons (10 tonnes) per ployee-shift, based on processed (clean) coal, or about 20 tons (18 tonnes) based on run-of-mine (raw) coal.

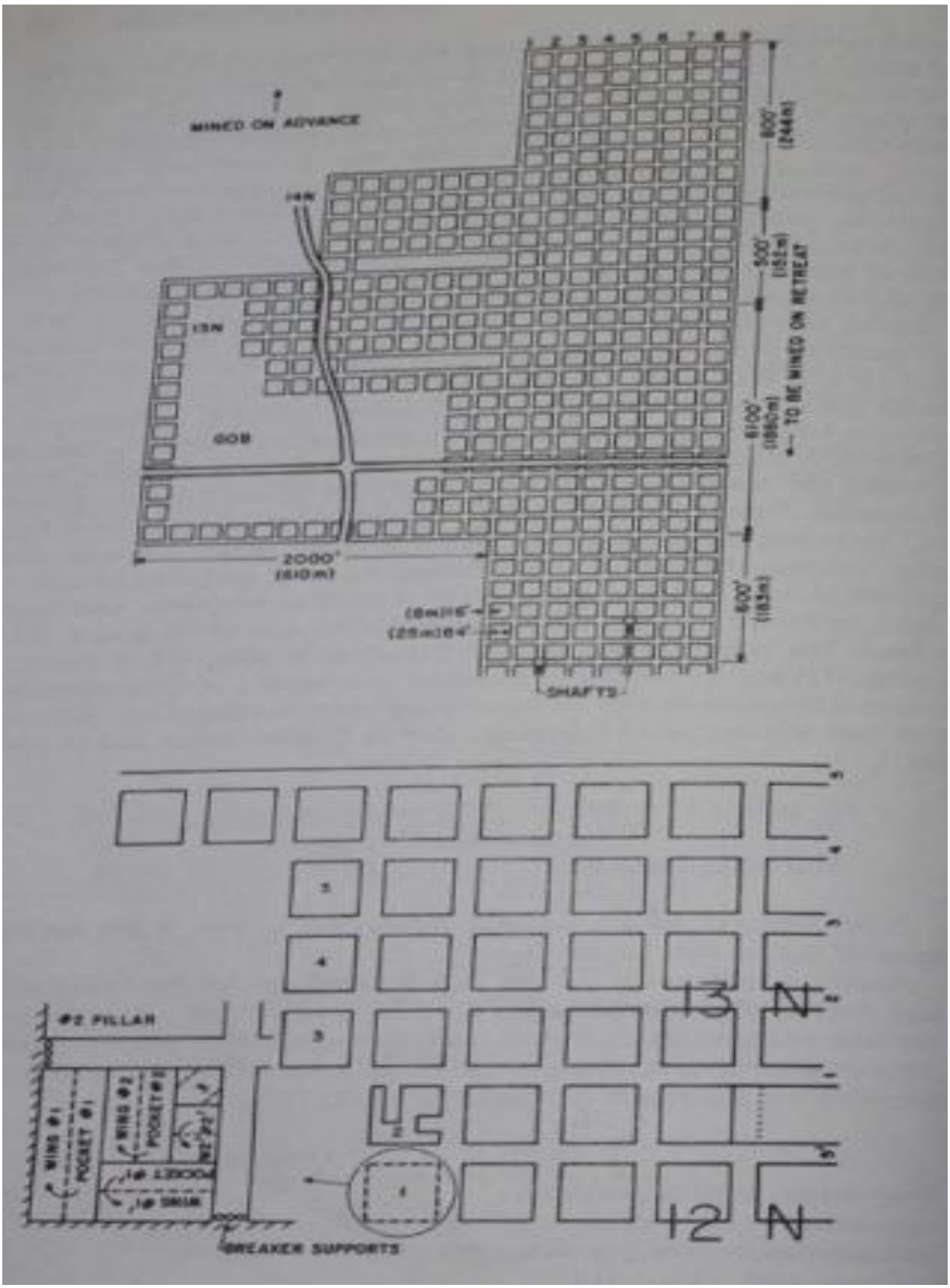


Figure 10.3. *(Top)* Room and pillar mining, block system, driving panel entries on the advance and recovering chain pillars on the retreat (half-advance, half-retreat). (*Bottom*) Details of pillar recovery. The pillar line is staggered across five pillars, with pillar recovery by the pocket-and-wing technique. Cuts are made in the sequence shown, driving a pocket and than a wing. (After Anon., 1976c. Copyright © 1976, Electric Power Research Inst., Palo Alto, CA. Reprinted by permission.)

Sequence of Development

Chapter 9 covered general underground development in some detail. The procedure for the development of flat-bedded deposits, to which room and pillar mining is applicable, was included (Sections 9.2-4). A complete mine layout with all development openings was shown in Figure 9.6; two modifications appear in Figures 10.2 and 10.3.

In sequence, main entries are driven in the deposit from the bottom of the shaft, slope, or outcrop (if a drift mine). Periodically, panel entries and then room entries are turned off at right angles, forming large rectangular blocks or panels in the mineral deposit in which production openings (rooms) are driven and pillars may be extracted. The same development procedure is used for both longwall and room and pillar mining.

The pattern in which multiple faces are advanced when driving entries or rooms is called the *cut sequence.* An echelon sequence, popular for advancing entries with a continuous miner in virgin coal, is shown in Figure 10.4. The cut sequences are numbered; all cuts numbered 1 arc advanced before those numbered 2, 2 before 3, and so forth. The law limits each cut to the length of supported roof, under which the machine operators work, which is approximately equal to the length of the machine (about 20 ft, or 6 m), unless a variance is granted (usually to 40 ft, or 12 m) (Anon., l9K4a). With conventional equipment, the advance averages 10 ft (3 m) and is limited by the length of the cutter bar.

Although it is almost always exceeded, the practical minimum number of parallel openings (entries or rooms) which can be advanced at one time in U.S. coal mining is three if conveyor haulage is employed. The restriction is imposed by the legal requirement that a belt conveyor be isolated on a neutral or separate split (circuit) of air and that at least one other opening be provided for fresh air and another for exhaust air (Anon.,1984).

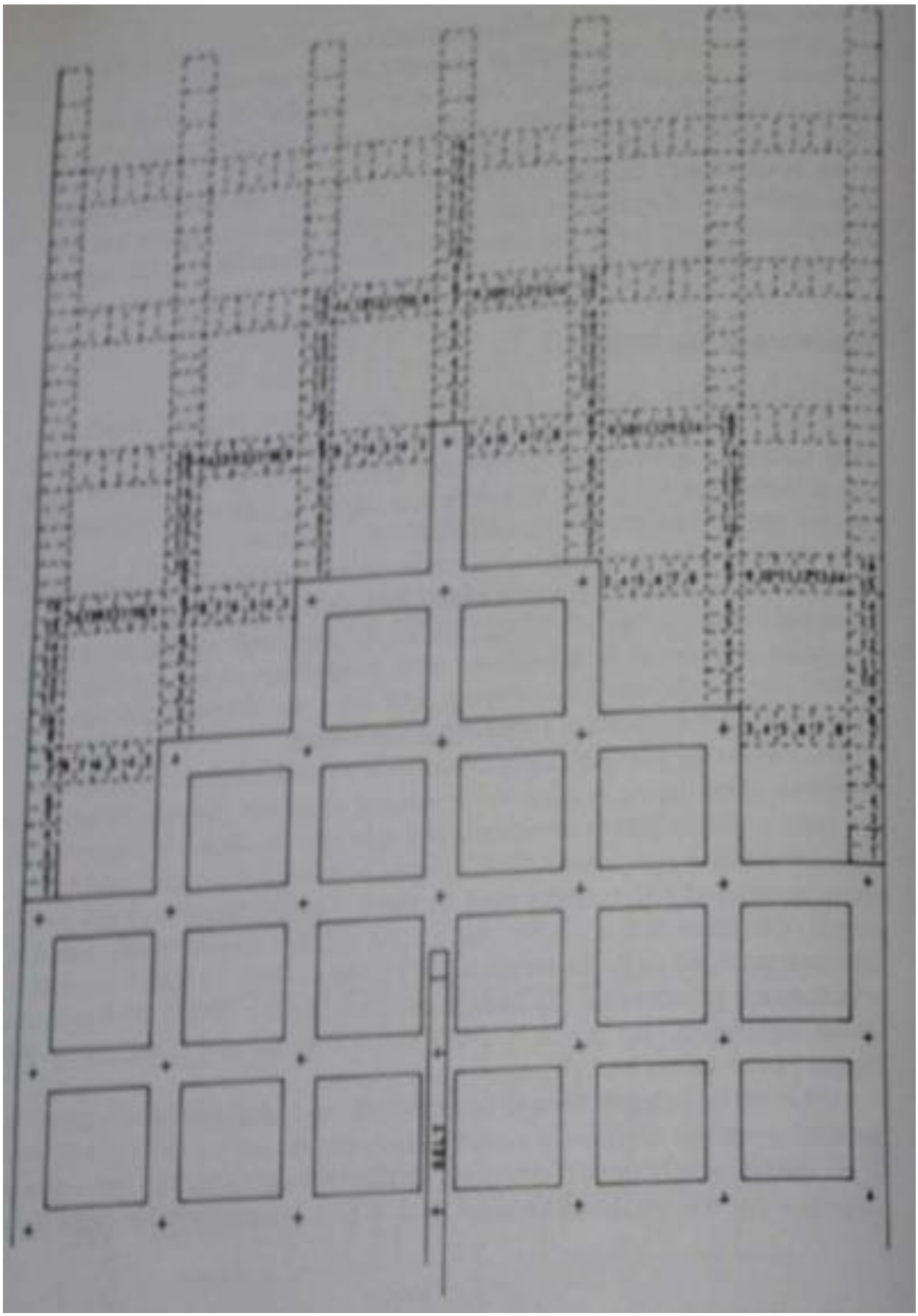


Figure 10.4 Cut sequence of continuous miner in seven-entry room and pillar mining. The echelon pattern advances the belt-conveyor entry first (After Bullock. 1982b.By permission from Society of Mining Engineers, Inc Littleton CO.)

Cycle of Operation

Convertional Mining Equipment. The production cycle of operations in the room and pillar mining of coal with conventional (cyclic) equipment is modified from the basic cycle by the insertion of cutting to improve breakage (section 4.1)

Production cycle = cut + drill + blast + load + haul

(The first three of these operation involved with coal breakage, comprise *fase preparation.*) To these productive unit operations are added certain auxiliary operations, including, as a minimum, roof support and ventilation. They are integral to the safe performance of mining by the room and pillar method and consequently are programmed into the cycle of essential operations. Cutting is also performed in the mining of softer nonmetallic minerals ((potash, salt, trona, etc.) but omitted in hard rock mining (iron, lead, stone, zinc, etc.).

Because of the highly cyclical, complex nature of room and pillar mining with conventional equipment, it lends itself to the application of mathematical optimization techniques. Computer programs using *simulation,* a systemaitic means of evaluating performance and operational factors, are available and ideally suited to optimize the cycle of operations in conventional mining Manula and Suboleski (1982) and Stefanko and Bise (1983) provide particulars.

Equipment employed in conventional mining (pictured in Fig. 9.9), and nearly all of it electric powered, consists of the following:

Cutting: shortwall (low-capacity, pulled by cable, undercutting only), universal (high-capacity, rubber-tired, cuts and shears) for coal; omitted in hard rock

Drilling: mobile auger (rotary drag-bit) drill rig (coal); percussion drill rig (hard rock)— drilling and cutting patterns in coal shown in Figure 10.5

Blasting {shooting): airdox (compressed air), cardox (compressed carbon dioxide): permissible nitroglycerin and ammonium nitrate explosives (coal). ANFO and slurry explosives (hard rock); loading by machine (bulk) or hand (cartridge), firing by electricity

Loading: mobile gathering-arm loader (coal); LHD, shovel loader, frontend loader, slusher (hard rock)

Haulage: rubber-tired shuttle car (electric or diesel), belt conveyor, rail (coal); truck, rail, convevor (hard rock)

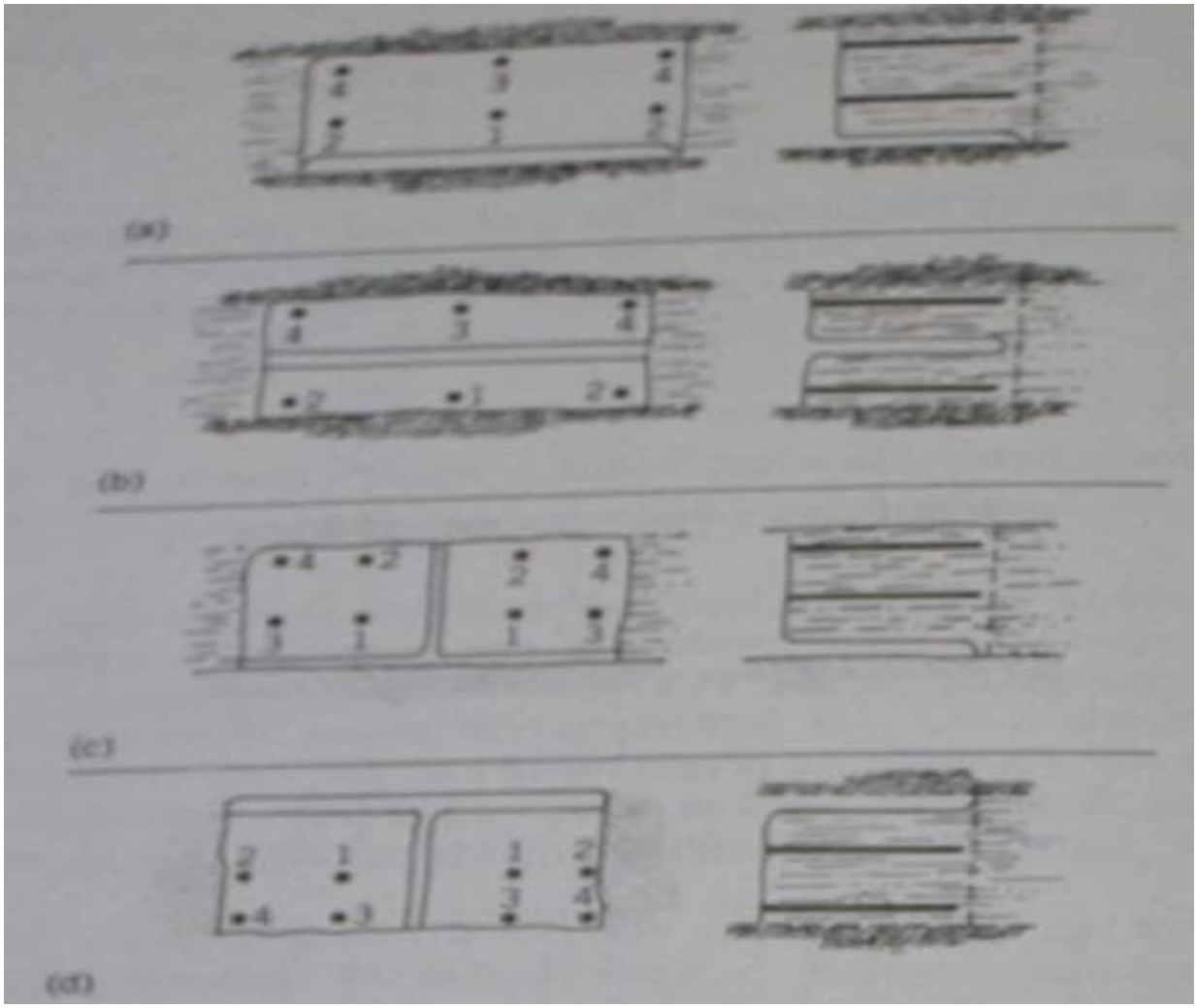


Figure 10.5 Drilling and cutting patterns for coal breakage. Holes are numbered in order of Firing (a) Undercutting. (b) Center cutting. (c) Undercutting and center shearing. (d) Top cutting and center shearing. (After Chironis,1983. Copyright © 1983, McGraw-Hill, Inc, New York.)

A unique feature of room and pillar mining, pioneered in the U .S. coal industry between the two world wars, is the now-total use of trackless equipment for all face operations. The practice has spread to hard-rock mining, especially with the stope and pillar method.

Continuous Mining Equipment. In continuous mining of coal, the basic cycle of operations is simplified as follows (Section 4.1):

Production cycle = mine + haul

(As in conventional mining, the production cycle is underground by auxiliary operations of roof control, ventilation, and cleanup.) A continuous miner breaks and loads the coal mechanically and simultaneously. It is not, however, as its name implies, truly a "continuously" operating machine; studies (Stefanko and Bise, 1983) have revealed its average productive time use to be only about 20%. Maintenance, moving and scheduling problems account for much of the inefficiency. While considerably more efficient than the conventional cycle, continuous mining, too, benefits from optimization, Simulation analyses are useful in improving its performance.

Continuous miners are also employed to exploit the softer nonmetallics; but in hard-rock mining, tunnel-boring machines must be used, and they are limited to development work in medium-hard rock.

The basic production cycle consists of the folloving operations and equipment (see fig. 9.9).

Mining(breaking and loading): continuous miner (ripper, borer. Milling head, auger, drum) (coal); tunnel boring machine, roadheader (hard rock)

Haulage: shuttle car. belt conveyor, rail (coal); truck, rail, convey or (hard rock)

Auxiliary Operations. Auxiliary operation in room and pillar mining are very similar for all mineral commodities and with all production equipment. The following from Table 4.1 are generally incorporated:

1. *Health and safety:* gas control (e.g.. methane drainage in coal), dust control (rock dust, water spray, dust collector), ventilation (line brattice,fan and vent tubing), noise abatement
2. *Environmental control:* flood protection, subsidence control, remote monitor to sense atmospheric contamination
3. *Ground control:* scaling, roof control (roof bolts, timber, arch, crib, hydraulic jack), controlled caving (Fig. 9.9h-j)
4. *Power supply and distribution:* electric substation, diesel service station

5. *Water and flood control:* pump station, drainage (ditch, pipeline,sump)

6. *Clean up and waste disposal:* scoop tram (Fig. 9.9j), storage, hoisting,dumping

7. *Materiel supply:* storage, delivery of supplies

8. *Maintenance and repair:* shop facilities

9. *Lighting:* stationary flood, equipment-mounted fluorescent

10. *Communications:* radio, telephone

11. *Construction:* haulage routes

12. *Personnel transport:* cage, trip, car (Fig. 9.9j)

As already mentioned, those considered essential to safety, such as ventilation and roof support, become an integral part of the production cycle in coal mining.

Conditions

The natural and geologic conditions of the mineral deposit wich are optimal or well suited for room and pillar mining are the following (Boshkov and Wright, 1973); Morrison and Russell, 1973, Hamrin, 1982):

1. *Ore strength:* weak to moderate
2. *Rock strength:* moderate to strong
3. *Deposit shape:* tabular
4. *Deposit dip:* low (<15^0), prefer flate
5. *Deposit size:*large areal extent, not thick (<15 ft, or <4,5 m) or bench with stope and pillar mining
6. *Ore grade:*moderate
7. *Ore uniformity:* fairly uniform
8. *Depth:* shallow to moderate (<1500 ft, or <450 m, for coal and <2000 ft, or 600 m, noncoal, although potash mined at 3000 ft, or 900 m)

Features

We continue to summarize method features as advantages and disadvantages (Boshkov and Wright, 1973; Stefanko and Bise,1983).

Advantages

1.Moderate high productivity (including both room and pillar and longwall, based on total work force. U.S average for clean coal 14 tons, or 13 tonnes, per

employee-shift; in room and pillar, for raw coal, face productivity 30-80 tons, or 27-72 tonnes, per employee-shift)

2. Moderate mining cost (relative cost 30%)

3. Moderately high production rate

4. Fair to good recovery with pillar extraction (70-90%), low to high dilution (0 40%)

5. Suitable for total mechanization, not labor-intensive

6. Concentrated operations (although multiple working places required for cyclic equipment)

7. Versatile for variety of roof conditions

8. Superior ventilation with multiple openings and if unidirectional (mining and ventilation methods compliment one another)

Disadvantages

1. Caving and subsidence occur with substantial pillar extraction
2. Method fairly inflexible and rigid in layout, not selective without waste disposal
3. Poor recovery (40-60%) without pillar extraction, fair (60-80%) with extraction
4. Ground stress and support loads increase with depth
5. Fairly high capital investment associated with mechanization
6. Extensive development required for coal deposit because of multiple openings
7. Potential health and safety hazards underground, especially in coal mines

Applications

Room and pillar is predominantly used in underground coal mining, (Brass, 1980; Mayercheck, 1983; Green. 1984) and to a lesser extent finds applikation in metallic and nonmetallic mining as well (Morrison and Russel,1973); borax (CA), fluorspar (KY), gilsonite (UT),iron (AL), lead (MO), limestone(AL,CA,MO,TN), potash (NM), salt (OH), trona (WY), and uranium (UT).

Case Study. Blue Creek No. 3 bituminous coal mine, Jim Walter Resources. Inc., Adger, A L.

General: among deepest U .S. coal mines (1500 ft. or 450 m), located in Warrior Basin in Appalachian coal field

Coal: steam and metallurgical grades (moisture 8%. volatiles 19.5%, fixed carbon 72.5%, ash 8%. sulfur 0.6%, calorific value 14.300 Btu/lb or 33.260 kJ/kg); Blue Creek scam 41-78 in. (1.0-2.0 m) thick

Production: 2.1 million tons/yr (1.9 million tonnes/yr)

Equipment: 8 continuous miners, shuttle cars, 3 longwall shearers, belt conveyors

Productivity: 10.7 tons (9.6 tonnes) per employee-shift

Mining methods: room and pillar, longwall

Variations

One important variation with the room and pillar method is *pitch mining.* Necessary when the deposit dip exceeds 15°, the obvious modification is to mine cross-pitch to reduce effective haulage grades. Other measures or methods can be employed when the dip exceeds 30^0 (Lucas and Haycocks, 1973). Very steep dips ($>45^0$) lend themselves to novel methods such as hydraulic mining (section 14.9)

Another variation occurs in the mining of multiple seams. Coal seams quite typically contain an impurity or rock parting called a middleman. If thin, it is loaded out with the coal; if thick, an attempt is often made to mine and load it separately or to gob it. When the parting becomes so thick than in effect two seams exist, then a decision must be made whether to mine them separately, together, or selectively (recover one, leave the other). Multiple-seam mining is not widely practiced in the United States. If adjoining beds are to be recovered, the preferred practices are to, (1) mine the upper bed in its entirety before starting on the lower bed, (2) mine the two in coordination so that the upper precedes the lower; or (3) mine two simultaneously. Lucas and Hayrocks (1973) discuss the procedures in detail.

Production and Cycle Time Estimates

In mine exploitation, it is often necessary to estimate the production rate. Room and pillar mining is a good method to use as an example because it is highly systematic and repetitive.

Example 10.1. Determine the daily and yearly production rate in a room and pillar coal mine using conventional equipment under the following conditions:

Working place	6 x 18 ft (1,8 x 5,4 m)
Working time	7 hr/shift. 2 shifts/day. 250 days/yr
Working sections	14
Advance/cut	10 ft (3.6 m)
Cuts/shift	12
Tonnage factor	24 ft^3/ton (0.75 m^3/ tonne)

SOLUTION

Volume mined/cut $V = (6)(I8)(10) = 1080\ ft^3$

Weight/cut $W = \frac{V}{TF} = \frac{1080}{24} = 45$ tons

Section production = (45 tons)(12 cuts/shift)= 540 tons/shift x 2 shifts/day = 1080 tons/day

Mine production = (1080 tons/day per section)(l4) = 15,120 tons/day (13,720 tonnes/day) x 250 days/yr = 3.78 million tons/yr (3.43 million tonnes/yr)

If recovery = 100%.

Surface area exploited A=(18)(10)(12)(14)(2)=60,480 ft^2/day+42,500 ft^2/acre=

=1,42 cares/day (0,63 hectares/day) ×250 day/yr=

355,5 acres/yr (159,0 hectares/yr)

In selecting equipment for a mechanized, sequential operation such as modern room and pillar mining, it is necessary to balance each operation within the production cycle. This calls for the estimation of cycle times for each unit operation and the calculation of the required output for each machine.

Example 10.2. Determine the cycle time per working place and the output rating of a gathering-arm loader for a section of the coal mine in example 10.1. given the following additional information m:

Place change time	3.5 min
Car change time	0.6 min
Delay/place	2.2 min
Shuttle car capacity	8 tons (7.3 tonnes)
Shuttle cars/place	2

SOLUTION.

Allowable cycle time/working place = $\frac{(7\text{hr/shift})\ (60\ \text{min/hr})}{12\ \text{cuts/sift}} = 35{,}0\ \text{min}$

Net available loading time = 35.0-(35+2.2) =29.3 min

Number of shuttle car loads/cut = $\frac{45\ tons/cut}{8\ tons/car} = 5.6\ say\ 6$

Load time/place = 29.3 - (6)(0.6) = 257 min

Required loader output = $\frac{45\ tons/cut}{25.7\ min/cut} = 1{,}8\ tons/\ \text{min}(1{,}6\ tonnes/min)$

Since this is a minimum rating based on average performance, a loader with a somewhat higher output, say, 3 or 4 tons/min would be selected to to allow for surges and contingencies.

10.3 STOPE AND PILLAR MINING

Strikingly similar to but displaying some unique differences from the room and pillar method, stope and pillar is the most widely used of all underground, hard-rock mining methods. *Stope and pillar mining* is the unsupported method in wich openings are drivenhorizontally in a mineral deposit in regular or random pattern to form pillars for ground support It is a large-scale (or intermediate) method, accounting for approximately 50% of U.S.underground noncoal production or nearly 15% of all mineral production. Together with room and pillar. these two similar methods are responsible for over 75% of all underground mining in the United States.

We will make considerable use of two current sources as we discuss noncoal mining methods, including all the remaining unsupported, supported. and caving

(except longwall mining) classes. Many of the references cited are from these two works. They are—

1. Steward, D.R., ed.,1981,*Design and Operation of Caving and Sublevel Stopping Mines*, Soc.Mng. Engr.— AIME, New York,843 pp.

2.Hustrulid, W.A., ed.,1982, *Underground Mining Methods Handbook*, Soc.Mng.Engr.-AIME, New York,1753 pp.

In addition, an older source is mentioned frequently:

3. Cummins, A.B., and I.A.Given, eds., 1973, SME Mining Engineering Handbook, 2 vols., Soc.Mng.Engr.-AIME, New York, 35 secs.

Generally, a method is classified as stope and pillar rather than room and pillar if it meets two of these there qualifications:

I. The pillar are irregularly shaped and sized and randomly located. The objective in placement of pillars is to locate them in in low-grade ore or waste rather than follow a systematic, invariant mining plan, so long as adequate roof support is provided. Because the pillars consist of rock, they are relatively strong, and fewer are required.

2. The mineral deposit is moderately thick to thick (> 20 ft, or 6 m). If it is sufficiently thick that it cannot be safely or technologically mined in a single pass then a benching or slabbing technique must be utilized.

3 The commodity being mined is a mineral other than coal. Although some noncoal deposits arc mined by the room and pillar method, no coal deposits are mined by the stope and pillar method.

Rule of thumb for naming method. When in doubt, specify room and pillar if coal and stope and pillar if noncoal. (Exceptions are noncoal mines with a very regular layout of openings and pillars that more properly might be called room and pillar mining.)

Other terms that have been applied to stope and pillar mining are *open sloping, breast sloping, pillar sloping,* and *bord and pillar mining,* but they have generally fallen into disuse.

Two representations of stope and pillar mining are shown in Figure 10.6.

A less mechanized, less productive version appears in Figure 10.6 *(top),*

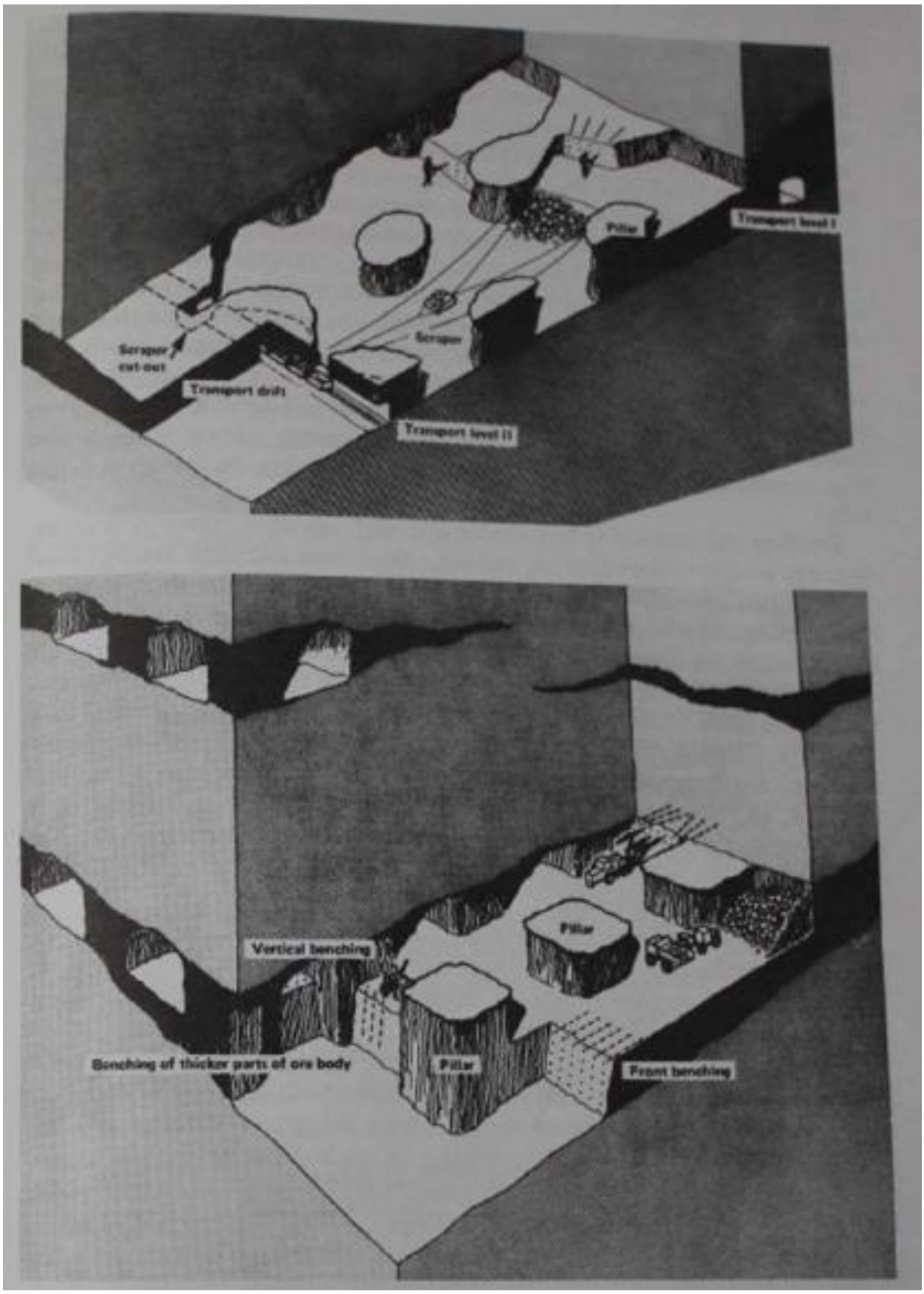


Figure 10.6. Stope and pillar mining by benching. *(T o p)* Intermediate method for pitching deposit using airleg drills, scrapers, and rail haulage. (Bottom) Large-scale method for flat deposit using drill rigs, front-end loaders, and trucks. (After Hamrin, 1982. By permission from Society of Mining Engineers. Inc., Littleton, CO.)

applicable to irregular pitching deporits and popular in the period following World War II. A more modern, more highly mechanized method is also depicted in Figure 10.6 (bottom), suitable for flatter deposits. Observe that development and production are difficult to distinguish, and, in fact, development openings are minimal.(For another view of mechanized slope and pillar mining, see Fig. 1.1.)

There are several reasons why the amount of development in stope and pillar mining can generally be sharply reduced over that in room and pillar mining. For one, the strict laws in coal mining regarding multiple openings and ventilation are not applicable to hard-rock mining (unless methane is detected and the mine is classified as gassy), and hence fewer openings are required and less development must be performed in advance (Anon.,1984a). For another, the deposits to wich stope and pillar mining are applied tend to be more irregularly shaped, and development openings cannot always be driven in ore: consequently, fewer are driven (e.g., multiple openings are unusual), and mining on the advance is the tule.

Because the method is less systematic and repetitive, there is less tendency to recover pillars in stope and pillar than room and pillar mining. Total pillar extraction is almost unheard of in a hard-rock mine, for theree reasons (1) the pillars are relatively small and more difficult to recover with safety; (2) the pillars are irregular in size and spacing and do not lend themselves to a systematic recovery operation; and (3) caving to the surface accompanied by subsidence is not usually practiced in noncoal mines (Morrison and Russell,1973; Hamrin,1982). Partial extraction (called *pillar robbing)* is sometimes practiced, especially if the pillars are larger than necessary to support the back, and some ore can be recovered safely. Rather than attempting to recover any of the pillars, it is likely that a stope and pillar mine would be designed initially with the maximum recovery possible (i.e., the highest possible stope-to-pillar ratio).

Design parameters are based mainly on (I) rock mechanics considerations especially of a ground-support nature; (2) economic factors, mainly cutoff grade and recovery; and (3) technological concerns, such as equipment maneuvering and gradeability limits. In addition to the ratio of the stope width to pillar width, a

critical parameter from the stress standpoint is the ratio of stope width to height (stress concentrations tend to rise with both ratios). Artificial support is used to supplement the pillars, although it is less common than in room and pillar mining. A typical design with a regular layout consists of 35-ft (10.5-m) slopes and 25-ft (7.5-m) pillars, which yields a pillar recovery of 85% (Boshkov and Wright, 1973). The opening height is generally equal to the thickness of the deposit, which may range from a few feet to hundreds of feet (2-100 m). If the height exceeds 20-25 ft (6-8 m), then it is customary to mine the deposit in *lifts* or benches— see the section **"Variations"** below (Bullock, 1982c).

Sequence of Development

The choices of main access openings for stope and pillar mining, a method restricted to relatively shallow or moderate depth, are similar to those for room and pillar mining. If skip hoisting is to be used for a relatively deep deposit, then a vertical shaft is installed;- or if a belt conveyor is planned for a deposit at intermediate depth, then a slope is sunk. There is an additional choice if rubber-tired haulage is projected throughout the mine and the depth is shallow, and that is a ramp or inclined shaft. Addition openings to the surface, certainly ventilation shafts or raises are located as required.

Depending on the geometry and attitude of the ore body, secondary openings are constructed on levels connecting the shaft with the production openings. If required by law or good mining practice, parallel drifts and connecting crosscuts may be driven to provide good ventilation. If the mineral deposits are discontinuous and occur on different horizons, then truck haulage and ramps may be selected to provide maximum flexibility

Cycle of O perations

Conventional Mining Equipment. Because the stope and pillar method is applicable only to noncoal mines, a basic cycle of operations (Section 4 I) is almost always indicated:

Production cycle = drill + blast + load + haul

Development and production equipment similar to that shown in Figure 9.9 is utilized (e.g., for drilling, see Fig. 9.9b. large-scale breast mining). If the mineral is sufficiently soft (e.g., salt) that trackless coal-mining equipment can be employed, then the modified cycle of Section 10.2 which includes cutting is used. If the ore instead is competent, typical of hard-rock mining, then the cycle is modified to include *secondary breakage or blasting*, especially hazardous in the vertical stopping methods to follow.

The conventional cycle of operations in hard rock consists of the following:

Drilling : pneumatic airleg or pneumatic- or hydraulic-powered, percussion or rotary-percussion drill rig (hard rock); electric-powered rotary drill rig (soft rock)

Blasting : ANFO , gels and slurries, N G dynamite explosives; charging by hand (cartridge) or by pneumatic loader or pump (bulk), firing electrically or by detonating fuse; blasting rounds shown in Figure 9.9b (Bullock. 1982d)

Secondary breakage: drill and blast, impact hammer, drop ball to fragment boulders

Loading: LHD, front-end loader, shovel, overhead loader, slushier (hard-rock); gathering-arm loader (soft rock); LHD equipment reportedly used in 75% of hard-rock mines (Bullock)

Haulage: truck, rail, LHD, belt conveyor, slushier (hard rock); shuttle cfr, belt conveyor (soft rock)

Continuous Mining equipment. The use of continuous mining with the stope and pillar method is infrequent because stope and pillar applications are in hard-rock mines. Tunnel boring machines and other mechanical excavators, hovewer, are used increasingly for development. In softer material, continuous miners are used The cycle of operations includes the following:

Mining (breaking and loading): tunnel-boring machine, roadheader, raise borer, shaft borer (development in soft to medium-hard rock); continuous miner (soft rock) (Bullock. I982d)

Haulage: belt conveyor, shuttle car, truck, rail

Auxiliary Operations. There are no significant differences in the auxillary operations that are performed with any underground mining methods. The list provided in Section 10.2 for room and pillar mining is therefore applicable, to stope and pillar mining as well. The most important auxiliary functions thet are performed are health and safety (dust control, face ventilation with fan and tubing noise abatement); ground control (scaling roof with boom lift if high opening, auxiliary support by rock bolts, timber, shotcrete, cable, steel sets and arches): power supply and distribution (compressed air, electric, diesel) and water and flood control (sumps, ditches, pipelines, pump stations) Both health and safety and ground control operations are integrated into the production cycle, but less often than in room and pillar coal mining. For particulars on ventilation, see Section 12.5 and Hartman (1982a) ; and on ground control Section 10.7 and Bullock (1982c) and Hamrin (1982).

Conditions

The following is based on Morrison and Russell (1973), Bullock (1982c), and Hamrin (1982):

1. *Ore strength:* moderate to strong
2. *Rock strength:* moderate to strong
3. *Deposit shape:* tabular, lens
4 .*Deposit dip:* preferably flat, low to moderate (<30°)
5 .*Deposit size:* any,preferably large areal extent, moderate thickness or bench if high (maximum <300 ft, or <90 m)
6. *Ore grade:* low to moderate
7. *Ore uniformity:* variable, leave lean ore or waste in pillars
8. *Depth:* shallow to moderate (2000 ft. or <900 m. in competent rock, although used to 3000 ft in very strong rock)

Features

The following is based on Morrison and Russell (1973), Lucas and Haycocks (1973),Bullock (1982c), and Hamrin (1982):

Advantages

1. Moderate to high productivity (U .S. average for noncoal mining 30- 50 tons, or 27-45 tonnes, per employcc-shift; maximum 50-70 tons, or 45-64 tonnes, per employce-shift

2. Moderate mining cost (relative cost: 30%)

3. Moderate to high production rate

4. High degree of flexibility; method easily modified; can operate on multiple levels simultaneously

5. Lends itself readily to mechanization; suitable for large, mobile equipment

6. Not labor-intensive; extensive skills not required, similar to surface mining jobs

7. Selective method, permitting waste or lean ore to be left in place; suitable for variable deposits

8. Early development not extensive

9. Multiple working places can be operated

10. Fair to good recovery without pillar extraction (range 60-80%, average 75%), low dilution (10-20%)

Disadvantages

1. Ground control requires continual maintenance of back, if rock and ore not competent; high back difficult to scale and support; ground stress on openings and pillars increases with depth

2. Large capital expenditure required for extensive mechanization

3. Some ore lost in pillars

4. Difficult to provide good ventilation for dilution of contaminants because air velocities low in large openings

Applications

Stope and pillar mining finds wide application in the exploitation of nonmetallic and some metallics; as stated, it is most popular of the underground noncoal methods. It is used in the mining of copper (MI-see Anderson.1981), iron (France-see Hoppe,1978),lead (MO-see Weakley,1982), limestone and marble (GA,PA),

oil shale (CO), uranium (UT, Canada-see White,1978), and zinc (TN-see Li,1978; White,1979).

Case study: Elmwood and Gordonsville zinc mines, Jersey Miniere Zinc Company, Carthage, TN.

General: first zinc mines in middle Tennessee (1974); prospecting began in 1964, development in 1972

Geology: mineralization occurs mainly as breccias ores in Kingsport formation of Knox group, located in the Nashville Dome

Ore: reserves 50-75 million tons (45-68 million tones), ore sphalerite, grade, 3,5-5,2% Zn

Production: Elmwood 3000 tons/day (2700 tonnes/day), Gordonsville 3000 tons/day (2700 tonnes/day), average grade 3,3% Zn, cutoff grade 1,5%

Mining method: stope and pillar

Equipment: percussion drill jumbos. ANFO, LHDs, rail

Variations

Thick deposits pose somewhat of a challenge in stope and pillar mining, requiring the adoption of *benching* or *slabbing* when the opening height reaches 20-25 ft (6-8 m) (Bullock. 1982c). Figure 10.7 compares the two methods. After breast faces have been advanced a sufficient distance through the deposit at the maximum minable height, then benching or slabbing commences to open up the slopes to their full height. The preferred benching method is shown in Figure 10.7 *{center* and *bottom*), in which the original face is driven to intersect with the intended back (roof) o f the stope; this permits benching to recover ore to the lower limit o f the deposit (Bullock 1982c). Drillholes may be drilled horizontally (Figure 10.7, *center)* or vertically (Figure 10.7, *bottom).* Vertical holes resemble the pattern in surface *mining*, permitting the extraction of very high benches on the second pass (Bullock. 1982d).

Pitch mining (or cross-pitch mining) is carried out if the deposit dip exceeds the gradeability of the mobile equipment in use (H am rin, 1982). Generally. this occurs if the pitch is >15° and is effective to a maximum limit of 30°. Above that

inclination, another mining method should preferably be used, although Hamrin proposes a method (step mining), which may extend the range o f stope and pillar mining to 45°. The angle of repose of broken ore in the stope (about 40-50°) is the usual lower limit for the application of vertical stopping methods, and stope and pillar is no longer competitive. A vertical method similar to stope and pillar mining in the supported class is stull stopping (see Chapter 11).

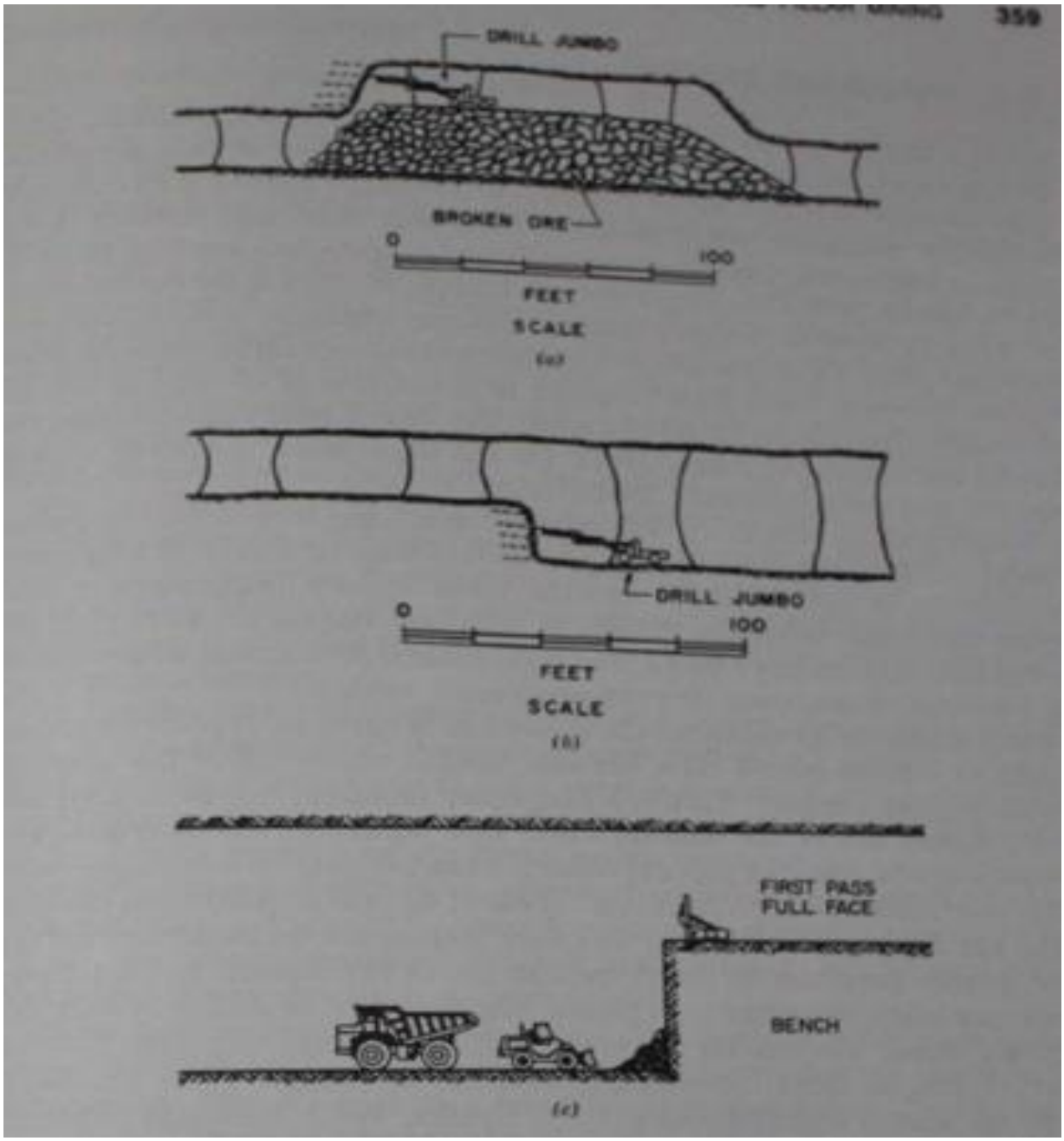


Figure 10.7. Methods of exploiting thick deposits by stope and pillar mining (Top). Breast stopping followed by overhand slabbing. *{Center)* Breast stopping followed by benching; horizontal drilling. (After Lucas and Haycocks. 1973. By permission from Society of Mining Engineers, Inc., Littleton. CO.) *(Bottom)* Breast stopping followed by benching; vertical drilling (Anon., 1974.)

10.4 SHRINKAGE STOPPING

We now ecounter the first of the so-called vertical stopping methods, those carried on essentially in a verrtical plane at an angle greater than the angle of repose of the broken ore. S*hrinkage stopping* is an overhand method in wich the ore is mined in horizontal slices and remains in the slope as temporary to the walls and to provide a working platform for the miners. Since the ore swells in breakage. 30-40% of the broken ore in the slope must be drawn off ("shrunk") during mining to provide sufficient working space. The remainder of the broken ore is recovered when the stope reachers its upper limit. This holdback in production of 60-70% of the ore represents a significant tie-up of capital, the cost of wich must be charged against the method (Lyman,1982). On the other hand, it affords storage capacity and the opportunity for blending.

Because of its simplicity and original small operating scale, shrinkage stopping was formerly a very popular method of nonocoal mining. Rising costs the scarcity of skilled labor, and the tred toward mechanization have largely displaced shrinkage stopping (Lucas and Haycocks, 1973). Its use to no more thay 1 % of U.S. mineral production, where it finds application as moderate or even small-scale method .

Some method classifications (Boshkov and Write,1973) place shrinkage stopping in the supported class because broken ore is left in the stope to provide ground support . This os a temporary measure, however, and , like pillars, makes use of the mineral itself. Few shrinkage stopes require the provision of any artificial support other than occasional rock bolts or timber stukks-see the section "Variations" below for use of pillars and backfill.

The key design parameters in shrinkage stopping are the dimensions of the stope largely governed by the shape and size o f the deposit. In a relatively narrow ore body, the slopes are placed longitudinally; in a large or wide ore body, the stopes are located transversely. Stope widths vary from 3-100 ft(1-30 m) , lengths from 150-300 ft (45-90 m), and heights from 200-300 ft(60-90 m) (Lucas and Haycocks. 1973; Lyman. 1982). Although rock mechanics is a consideration in

design, the stope opening itself is relatively small and not excessively stressed; and therefore the major concern is to maintain a manageable-sized stope that ensures a smooth flow of ore by gravity and effective draw control.

We consider shrinkage stopping a unsupported method because the stope opening itself generally has no pillars and very little artificial support. If the ore body is large, and several slopes adjoin one another, then it is customary to leave pillars between stopes for ground control. Rib pillars are left at the ends (sides) of the stope, a sill pillar at the bottom, and a crown pillar at the top. Sometimes these pillars are recovered in subsequent mining, and occasionally backfill is placed in the stopes after the ore is drawn down as a permanent means of ground control.

Sequence of Development

The nature of all vertical sloping methods is such that production operation are carried on over a considerable vertical distance. Consequently, several levels are required, the main or haulage production levels being spased generally 200-600 ft (60-180 m) apart (see section 9.2). If the stope height is less than the level interval, then sublevels may be constructed, connected by orepasses.

On each main level, a haulage drift is driven parallel to the strike of the ore body (Hamrin,1982). It is connected to the shaft (usually vertical because of the vertical distance involved) by haulage crosscut. If the stopes are transverse or drawpoints offset, than a haulage lateral or loading crosscuts to the drawpoints are constructed. To provide multiple access roules into each slope, but mainly to ensure through ventilation raise manways are driven between levels, preferably at the ends of each stope.

The two main tasks of vertical-stope preparation are to (1) construct a means of drawing ore in which muck flows by gravity , to the bottom oif the stope, and (2) undercut (horizontally) at the sill level or to slot (vertically) the stope. providing an opening into which the ore initially breaks and subsequently flows (Lucas and Haycocks,1973; Lyman 1982) Shrinkage requires undercutting.

To construct the draw system and undercut, finger raises are driven desired spacing to connect the haulage level with the sill sublevel. The tops of the finger

raises are connected by a small drift which runs the length of the stope; from it, crosscuts are driven above the raises to the walls of the stope. To form bells, slabbing of the finger raises begins at the top, diminishing toward the bottom of each raise, creating funnel-like openings. To form the undercut and provide working space for stopping (a desirable height of 6 ft or 1.8 m), (the pillars formed by the drift and crosscuts are slabbed off the broken ore falling into the just-formed bells

Three draw systems are in use today. The oldest, a small-scale method consists of bells and finger rises terminating in chutes through which haulage conveyances (usually mine cars drawn by a locomotive) are loaded directy (Fig. 10.8). Simple in concept, the gravity and chute system replaced hand loading, but blockages are frequent, and it is no match for drawpoint loading. A second system utilizes rope-drawn scrapers, installed in slusher (scram) drifts arranged longitudinally under the stope (Fig. 10.9). The modern and preferred system of drawpoints likewise eliminates the bottleneck of chutes and expanding on mechanization, uses loading machines operating at drawpoints in crosscuts under the stope (Fig. 10.10). It is capable of attaining moderate production rates.

In all draw systems, the spacing of chutes, slusher pockets (boxholes) or drawpoints ranges from 15-50 ft (5-15 m) (Lym an, 1982). Close spacing requires more development work but minimizes the tonnage of ore stranded on the sill after the stope is emptied and maintains a more even work platform of broken muck in the slope. Wide spacing economizes on development at thc expense of even draw control and ore loss on the sill (it may be acceptable if a lower stope recovers the sill pillar and stranded muck).

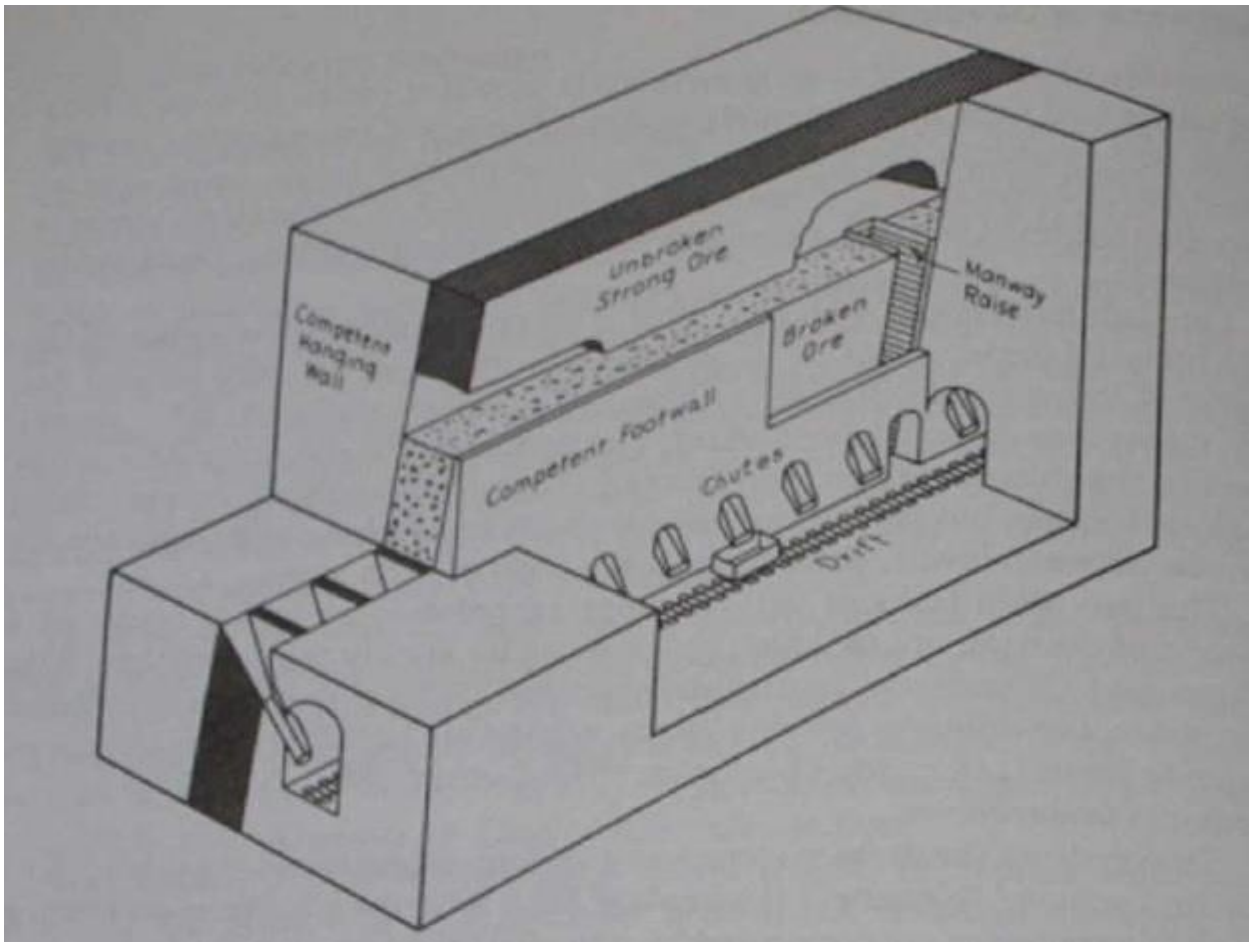


Figure 10.8. Shrimkage stopping using gravity draw and chutes to load mine cars. (After Lewis and Clark,1964. Copyright© 1964, John Wiley & Sons, New York.)

Cycle of Operations

Production Cycle. Working conditions in a shrinkage slope arc not ideal because of the uneven working floor of broken muck that periodically is disturbed and lowered by drawing operations in the haulage drift below. It is important to maintain an adequate and safe working space, neither too high nor too low, so that rock-breakage operations within the stope can be conducted properly.

Rock breakage is the principal activity in the stope itself. A bench face is established at the rib pillar and advanced across the face by horizontal drilling wilh airlegs or vertical drilling with stopers (see Section 9.4) (Lyman, 1982). Using longhole drifter drills mounted on columns at the access raise, it is possible to drill out the entire slope length from one setup— see the section "Variations," below'. After the holes arc charged but prior to blasting, drawing o f ore from the stope should occur. Before the cycle is repeated, any necessary ground control is carried

out, scaling followed by bolting (with wire mesh, if necessary) or timbering (with stulls, less frequently).

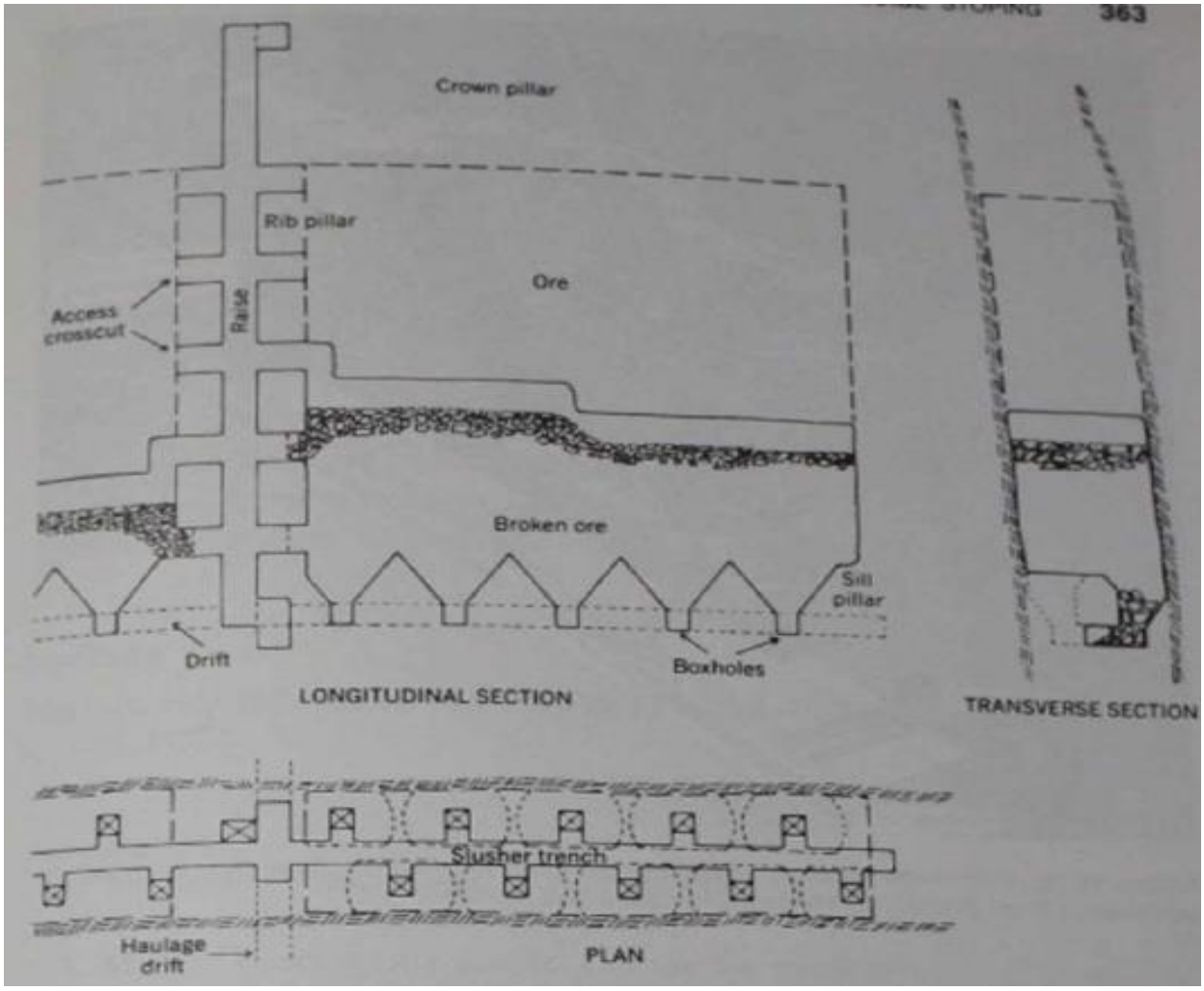


Figure 10.9. Shrinkage sloping using a scraper and slusher drift (After Handerson,1982, By permission from Society of Mining Engineers. Inc.. Littleton. CO.)

Production operations and equipment, all cyclic, in shrinkage stopping follow this basic cycle (Section 4.1):

Drilling : pneumatic airleg, stoper, or drifter percussion drill

Blasting: A N F O or slurry; charging by hand (cartridge) or by pneumatic loader or pump (bulk), firing electrically or by detonating fuse; blasting by bench rounds shown in Figure 9.9b for overhand small-scale mining

Secondary breakage: dynamite bomb, shaped charge, drill and blast to fragment boulders; impact hammer also used

Loading: gravity flow (through stope); front-end loader, overhead loader, LHD . slusher, chute (under stope)

Haulage: truck, LH D, rail, belt conveyor

Continuous extraction equipment is rarely used in vertical stopping methods because of the nature of the operations and the prevalence of hard rock in most such ore deposits. Its use is mainly limited to development (TBM and raise borer) when the rock hardness permits.

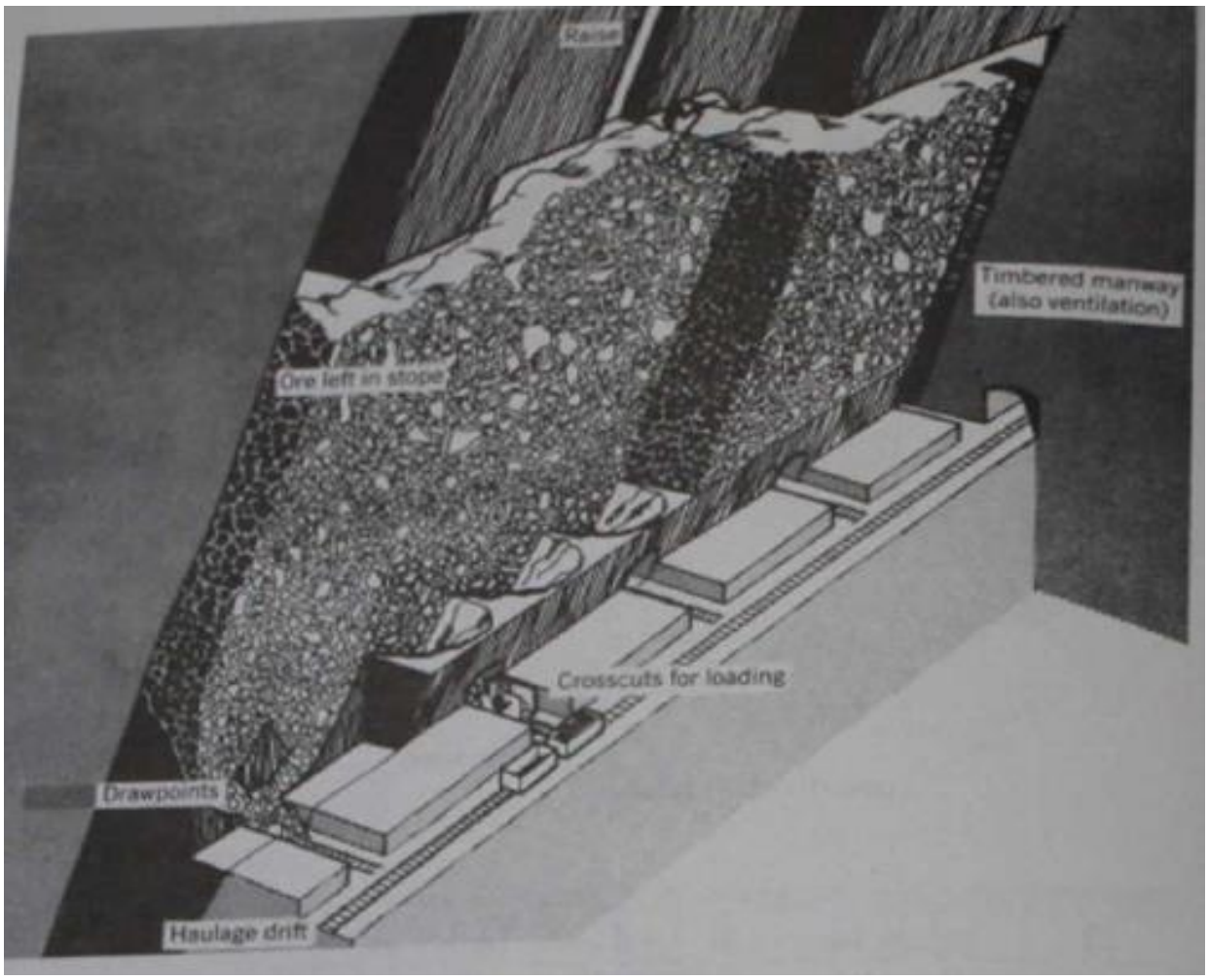


Figure 10.10. Shrinkage stopping using drawpoints and loaders. (After Hamrin. 1982. By permission from Society of Mining Engineers. Inc., Littleton, CO.)

Auxiliary Operations. The usual list of auxiliary operations for underground mining pertains— see the complete list for room and pillar mining in Section 10.2 and the abbreviated list for stope and pillar in Section 10.3.

Conditions

The following is based on Boshkov and Wright (1973), Lucas and Haycocks (1973). Morrison and Russell (1973),and Lyman (1982).

1.*Ore strtrength :* strong (other characteristics: should not pack oxidize or be subject to spontaneous combustion)

2. *Rock strength :* strong to fairly strong

3. *Deposit shape :* tabular or lenticular, regular dip and boundaries

4. *Deposit dip :* fairly steep (>45-50° or angle of repose, preferably 60-90°)

5. *Deposit size :* narrow to moderate width (3-100 ft. or 1-30 m) fairly large extent

6. *O re grad e :* fairly high

7.Ore uniformity : uniform

8.Depth : shallow to moderate (<2500 ft. or <750 m)

Features

The following is based on Morrison and Russell (1973), Hamrin (1982),and Lyman (1982):

Advantages

1. Moderate to small-scale production rate
2. Ore drawn down in stope by gravity
3. Method conceptually simple, can use for small mine
4. Low capital investment, little equipment required for basic method; lends itself to some mechanization
5. Little if any ground support in stope
6. Stope development moderate
7. Fairly good recovery (75-85%), low dilution (<10%)

Disadvantages

1. Low to moderate productivity (range 5-10 tons, or 5-9 tonnes, per employee-shift; maximum 10-15 tons, or 9-14 tonnes, per employeeshift)
2. Moderate to fairly high mining cost (relative cost: 50%)
3. Labor-intensive, difficult to mechanize
4. Rough footing, dangerous working conditions
5. Majority of ore tied up in slope (> 60%)
6. Ore subject to exploitation, packing, and spontaneous combustion in stope
7. Fair to moderate selective

Applications

Applications of shrinkage stopping are diminishing: this once popular method finds little use in modern mining. Current and recent examples are gold (Homestake, SD), iron (Pea Ridge, MO-see Irvine,1982), lead-silver (Idarado,CO), lead-zins-silver (South Bay, Ont.-see Jeremic,1984), limestone (Belleforne,PA), and nickel (Sudbury, Ont.).

Case study. Pea Ridge iron mine, St.Joe Minerals Corp.,Sullivan,MO.

General: only iron pellet producer in south-central United States, exploitation began in 1953, development in 1958 ($52 million), and exploitation in 1964

Geology: ore dposit occurs at depth of 1300 ff (390 m), extendes to 3000 ft (900 m); dike or veinlike structure lying nearly vertical in Precambrian rhyolite porphyry; mineralization: magnetite, hematite

Ore: proven reserves 102 million tons (92 million tonnes), average grade 56% Fe, cutoff grade 45%

Production: 1,9 million tons/y (1,7 million tones/yr) crude ore, 1,2 million tons/yr (1,1 million tones/yr) pellets

Mining methods: shrinkage stopping, sublevel stopping, sublevel caving

Equipment: airtracs, fan drills, downhole percussion drills; pneumatic cartridge loader or slurry pump; LHDs to belt conveyor or skip pocket

Variations

Several modification, of the basic gravity draw. shrinkage sloping method Shown in Figure 10.8 are in common use: they are the slusher and loader versions appearing in Figures 10.9 and 10.10 A new vanat.on us.ng *longhole drilling* from raises is in use at Pea Ridge and illustrated m Figure 10.11.It is an efficient, safe method of mining ore that ts no. strong enough for conventional shrinkage stopping. In other variations for weak rock, occasional p*illars* mas be left for ground support during mining, or *backfilling* may be introduced after mining to prevent caving and closing of the stope.

10.5 SUBLEVEL STOPPING

A principal distinction o f sublevel stopping is that it is the only patented mining method (more correctly, a modern version o f the method, *vertical*

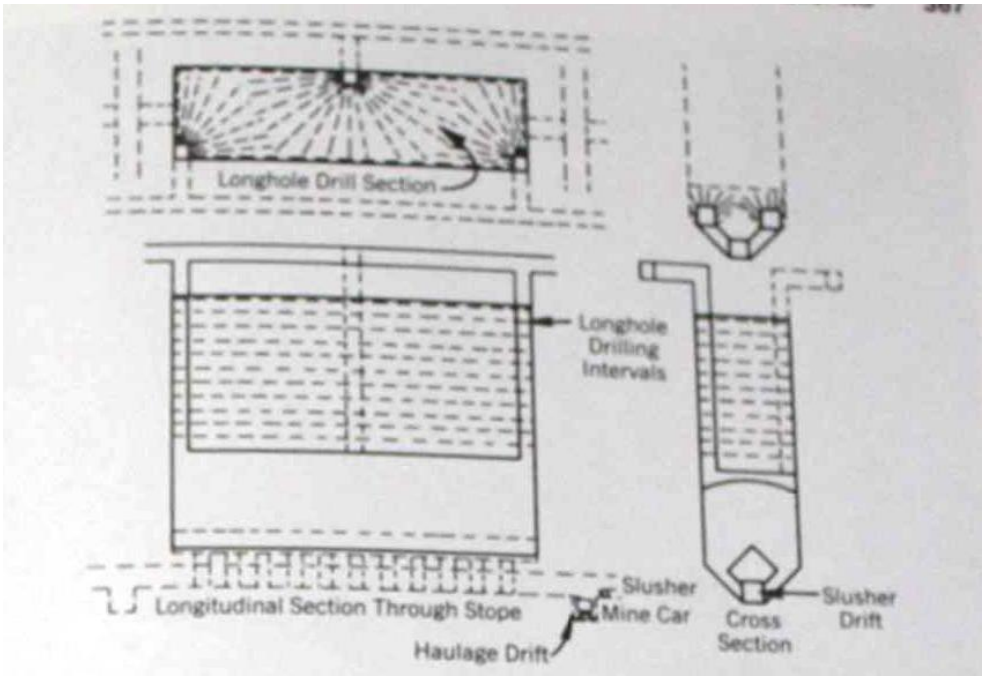


Figure 11.1 Modified shrinkage stopping using longhole drilling from run (After Irvine 1982. By permission from Society of Mining Engineers. Inc.. Littleton. CO.)

crater retreat, VCR, is patented). *Sublevel stopping is* an overhand, vertical stopping method utilizing longhole drilling and blasting carried out from sublevels to break the ore. The ore flows through the stope by gravity in the customary way and is drawn off at the haulage level. Always a popular method, sublevel stopping (also known as blasthole. longholc. open, or VCR stopping) is enjoying a revival in underground mining, presently having applicationfor about 99£ of U.S. noncoal production or 3% of all mineral production (Law rence. 1982). It is classified as a *large-scale* method.

Sublevel stopping, the last of the unsupported methods, employs even less temporary support in the stopes than the stope and pillar and shrinkage methods. The reason is that no personnel are exposed in the stope proper; drill and blast crews work in the protective cover of sublevel drifts and crosscuts, while loading and haulage crews labor in the security of the haulage drift below. If support is required in the sublevels, it is readily provided by rock bolts, wire mesh, cables, or shotcrete. Although the stopes are unsupported, pillars are usually left between stopes and occasionally within slopes.

The two common versions o f sublevel stopping are shown in Figures 10.12 and 10.13. Both employ longhole drilling, the one a ring pattern with small holes (Fig. 10.12) and the other parallel drilling with large holes (Fig. 10.13). In ring drilling, a vertical slot is opened, whereas either a slot or an undercut is used with parallel drilling (the V C R method requires parallel holes and an undercut). The unique feature of the V C R method is the application of blasting theory in the proper placement of explosives to yield improved fragmentation, reduced stope-wall damage, and increased production (Lang, 1983).

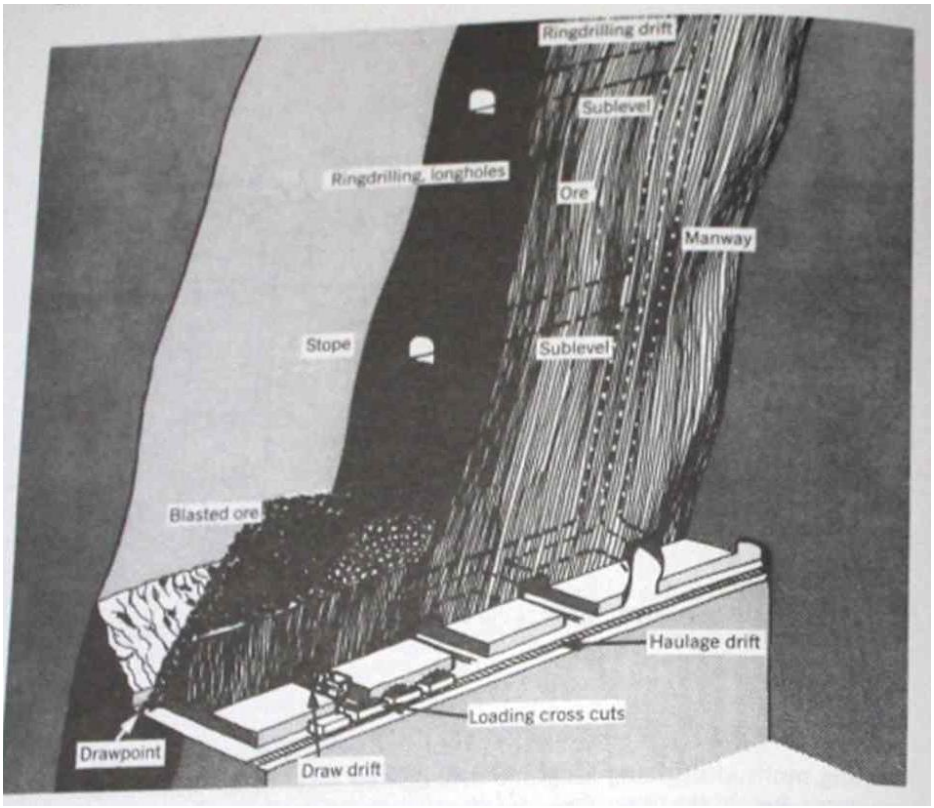


Figure 10.12. Sublevel stopping using ring drilling and blasting into a slot. (After Hamrin 1962. By permission from Society of Mining Engineers, Inc.. Littleton. CO.)

Ring drilling with blasting into a vertical slot (Fig . 10.12) was the original version o f sublevel stopping (Ham rin. 1982). The drillholes are relatively small (2-3 in., or 50-75 mm), bored with percussion rock drills mounted on a column and bar (drifter) or fandrill rig and with extension drill steel to a maximum length of 80-100 ft (24-30 m). To effect economies in rock breakage, the holes are loaded heavily and a ring (5-10 ft, or 1.5-3.0 m, in thickness) detonated simultaneously; the effect is not unlike that in bench blasting. Hole deviation is a serious problem, however, in which deflections o f several feet (meters) are not uncommon. The effect in blasting can be disastrous, because ring drilling requires accuracy in hole placement to obtain proper fragmentation. With the advent o f large-diameter ($7\frac{7}{8}$ -in., or 200-mm) rotary and downhole percussion drills, it became practical to adopt the parallel-drilling method of sublevel stopping (Fig. 10.13). Deviation is no longer a problem *(<2%)* with large, parallel holes, which can now be extended to a maximum of 300 ft (90 m), and sublevel spacing can be increased accordingly.

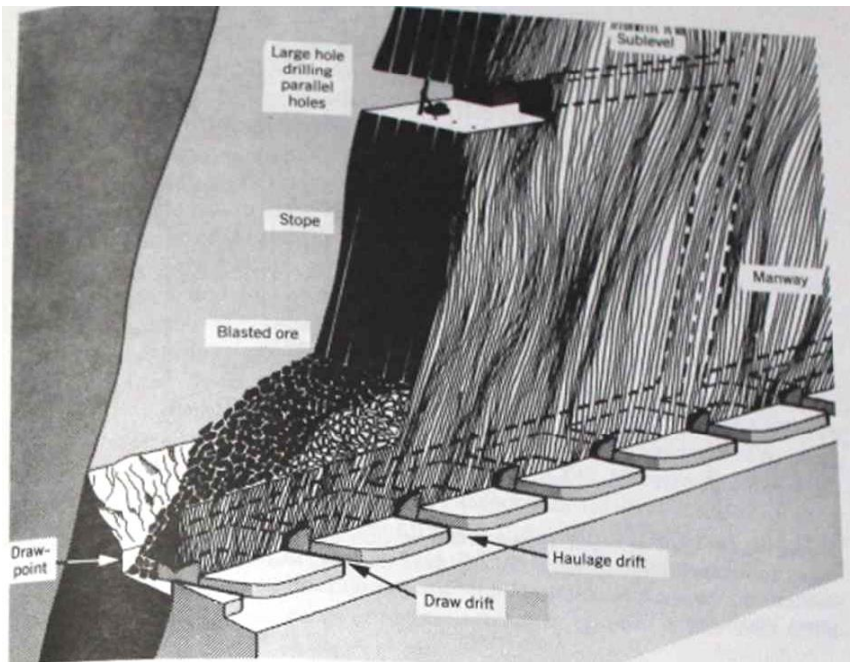


Figure 10.13. Sublevel stopping using parallel drilling and blasting into a slot. (After Hamrin. 1982. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

A further improvement has been realized in the last decade with the advent of the VCR method (Fig. 10.14). Large parallel holes arc used as in the parallel-drilling method, but a major innovation has been blasting horizontal slices of ore with near-spherical charges into the undercut (Mitchell, 1981; Lang, 1982, 1983). Spherical placement of explosives is the most efficient in terms of fragmentation and powder consumption (see Section 11.5). Holes are charged from the collar after plugging the opposite end; the size of charge is restricted to a length-to-diameter ratio of 6:1, which suffices in practice to simulate a spherical charge. All holes in a stope are detonated together. When the broken ore in the stope has been drawn down sufficiently, then the next slice of ore (15 ft, or 4.5 m, in thickness) is blasted. Because drilling is carried out from a sublevel and is usually complete before blasting commences, unit operations with the VCR method can be conducted with high efficiencies and productivities (Ham rin, 1982).

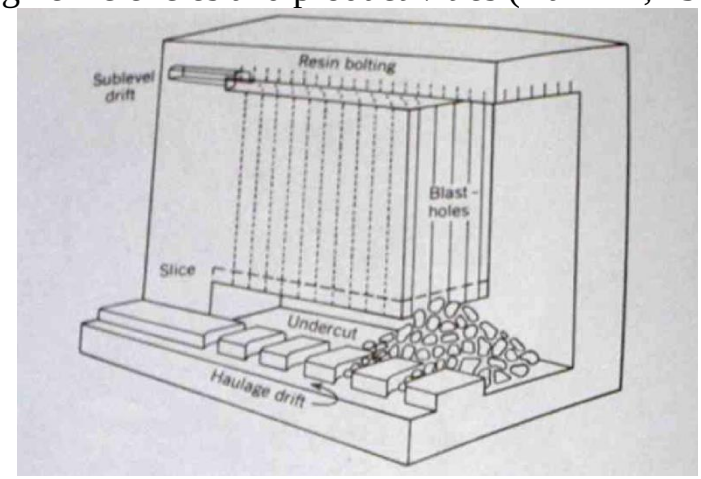


Figure 10.14. Vertical crater retreat (VCR) version of sublevel stopping. Large parallel holes are loaded with near-spherical explosive charges and horizontal slices of ore are blasted into the undercut (After Green. 1978 By permission from American Mining Con- grew. Washington, DC)

Another innovation in modem sublevel sloping is displayed in Figures 10.2-10.4. Development is simplified by eliminating bells, replacing them with draw drills or trenches. Ideally suited to high production with loading machines, draw drifts have been adapted to other bulk methods, such as sublevel caving and block caving.

Many of the design parameters in sublevel sloping involve the same factors and dimensions as in shrinkage stopping or any other form of vertical sloping. Slope width varies from 20-100 ft (6-30 m), maximum length 300 ft (90 m), and maximum sublevel height 300 ft (90 m) (Morrison and Russell, 1973; Mann. 1982). Boundary pillars are located similarly to those in shrinkage stopping. Rock mechanics, blasting, and materials-handling considerations are the major concerns in specifying dimensions. In addition, because of the unique reliance on longhole drilling and blasting, special attention in sublevel sloping must be paid to rock-breakage design: hole diameter and length, burden, explosive selection, powder factor, and so forth. Simulation models and cost analysis are profitably employed (Chatterjee and Just, 1981).

Sequenced Development

Sublevel stopping from many other vertical stopping and caving methods in utilizing one or more sublevels between main haulage levels. That distinction is less marked today, however, because other methods have introduced intermediate levels as stope heights have increased.

The general sequence of development in sublevel stopping parallels that in shrinking stopping and other vertical methods. A haulage drift, crosscut, and drawpoints or draw drifts and trench are driven for materials handling, together with interlevel raises for access and ventilation. Either an undercut or end slot is constructed to commence stopping is employed. If a slot, sublevel crosscuts are driven across the stope from the sublevel drift and a raise driven at the boundary. These are then blasted in a manner to excavate a slot, starting at the bottom.

In the ring – drilling version of sublevel stopping (Fig.10.12), only sublevel drifts must be driven for longhole drilling. In the other two versions, parallel drilling (Fig 10.13) and the VCR method (Fig. 10.14), a horizontal slot must opened across the ore body to provide room for the drill stations. This is constructed by driving a sublevel crosscut the width of the stope and then advancing it the length of the sublevel drift as mining progressed

Pillars (sill, crown, and rib) may be delineated at the stope boundaries and left permanently or recovered in a retreat operation, often after backfilling

Cycle of Operations

Production Cycle. I.ike many of the vertical sloping methods, rock breakage and materials handling are carried out in separate sections of sublevel slopes. Drilling

and blasting (all longhole now) are conducted in the slope proper in sublevel drifts, while the loading and haulage lake place underneath the slope in the drawpoints or draw drifts. Coordination is necessary, of course, but the two major groups of unit operations of the production cycle arc carried out largely independently of one another.

The cycle of operations follows the basic production cycle (Section 4,1):

Drilling: (1) longhole pneumatic percussion drill (small-hole) with coupled steel, drifter- or fandrill-mounted; (2) downhole pneumatic percussion drill (large-hole) on drill platform or rig; or (3) roller-bit rotary drill (large-hole) on drill platform or rig

Blasting: ANFO, slurries; charging by cartridge or bulk by pneumatic loader or pump, firing electrically or by detonating fuse; blasting by special longhole bench rounds (Fig. 9.9b) for overhand large-scale mining or by spherical charge in VCR method (Mitchell, 1981; Lang, 1982,

Secondary breakage: impact hammer, drill and blast, mudcap

Loading: gravity flow to drawpoints; LHD, front – end loader, shovel loader, slushier, gravity and chute (rare) (Hamrin, 1982)

Haulage: LHD, truck, rail, belt conveyor (rare)

Auxiliary Operations

See Sections 10.2. and 10.3 for lists of operations and equipment.

Conditions

The following is based on Boshkov and Wright (1973), Morrison and Russell (1973,) Mitchell (1981), Hamrin (1982), and Lawrence (l982):

25 ***Ore strength:*** moderate to strong, may be less competent than for slope and pillar

26 ***Rock strength:*** fairly strong to strong

3 ***Deposit shape:*** tabular or lenticular, regular dip and boundaries

6. ***Deposit dip:*** fairly steep (>45-50°, preferably 60-90°)

7. ***Deposit size:*** fairly thick to moderate width (20-100 ft, or 6-30 m**),** fairly large extent

8. ***Ore grade:*** moderate

9. ***Ore uniformity:*** fairly uniform to uniform

10. ***Depth:*** moderate (<4000 ft, or <1.2 km), deep (<8000 ft. or <2.4 km) at Homestake

Features

Advantages. The following is based on Morrison and Russell (1973), Mitchell (1981), and Hamrin (1982):

3. Moderate to high productivity (range 15-30 tons, or 14-27 tones, per employee-shift; maximum 30-40 tons, or 27-36 tones, per employee- shift)
4. Moderate mining cost (relative cost: 40%)
5. Moderate to high production rate
6. Lends itself to mechanization; not labor-intensive
7. Low breakage cost; fairly low handling cost
8. Little exposure to unsafe conditions; easy to ventilate
9. Unit operations can be carried on simultaneously
10. Fair recovery (75%); fair dilution (20%)

Disadvantages

1. Fairly expensive: slow and complicated development, high development cost
2. Inflexible and nonselective
3. Longhole drilling requires careful alignment
4. Large blasts may cause excessive vibration, air blast, and structural damage

Applications

Applications of sublevel sloping while most common in metal mines are frequent and varied in hard-rock mines in general. They include copper (Carr, Fork, UT-see Jackson, 1979; Crackel, et al., 1982; McArthur, 1984) copper – iron – zinc -sulfur (Copperhill TN - see Zimmerman and Verner, 1981), copper-lead (Mt. Isa, Australia—see Goddard, 1982; Hornsby and Sullivan, 1984), gold (Homestake, SD—see Mitchell, 1981), iron (Pea Ridge, MO-see Irvine, 1982), limestone (Bellefonte, PA), and nickel (Sudbury, Ont.—see Green, 1976: Monahan, 1984).

Case *Study*. Copperhill copper – iron – zinc - sulfur mine, Tennessee Chemical Company. Copperhill, TN.

General*:* only major underground primary producer of sulfur (as sulfuric acid): located in Copper Basin in southern Appalachians: mining dates to 1847
Geology: massive sulfide lenses in metamorphosed schist's and greywacke; mineralization: pyrrhotite, pyrite, chalcopyrite, sphalerite
Ore: mine ore bodies extending to at least 3000 ft (900 m) and up to 400 ft (120 m) in thickness; reserves 250,000-70,000.000 tons (230,00063,000,000 tonnes)
Production: 2.2 million tons/yr (2.0 million tonnes/yr) averaging 0 70% Cu, 20.0% S, and 0.50% Zn
Mining methods: open pit mining, sublevel sloping
Equipment (underground): drifter jumbos, raise borers, percussion fan- drills. rotary drill; LHDs; trucks, rail

Variations

The principal variation of sublevel stopping is the widely used vertical crater retreat method (Fig. 10.14), already discussed. Minor variations of the main method include *backfilling* when caving and subsidence must be prevented, primarily in pillar recovery, and leaving *pillars* in the slope on an occasional basis in sloughing ground.

10.6 SPECIAL TOPIC A: CALCULATION OF PERCENTAGE RECOVERY

During the planning phases of mining; estimation is often needed of the percentage recovery or extraction of ore, coal, or stone from a deposit when using a given mining method. Mathematically, it may be calculated as the ratio of mineral extracted to the total minable reserve in the deposit. If a method yields a high recovery, it may be simpler to calculate the unmined material left in pillars rather than that extraction. With a method as systematic as room and pillar mining in coal, it is not difficult to estimate the recovery in a panel or for the entire mine. For an approximation, and if the seam height is regular, it is acceptable to base the calculation on areas rather than volumes (or weights).

EXAMPLE 10.3. A portion of a panel in a room and pillar coal mine is shown in Figure 10.15. All openings are 20 ft (6 m) in width, and the mining height is regular. Rooms are driven on 60-ft (18-m) centers and crosscuts on 80-ft (24-m) centers. Calculate the percentage recovery in the panel (1) without *pillar* recovery and (2) with recovery of chain pillars. Disregard the effect of barrier pillars, and calculate for the smallest repetitive dimensions in the panel.

SOLUTION. (1) Without pillar recovery:

$$\text{Area of block} = (80)(60) = 4800 \text{ ft}^3$$
$$\text{Area of openings} = 20(80+40) = 2400 \text{ ft}^3$$
$$\text{Recovery} = \frac{2400}{4800} = 50.0\%$$

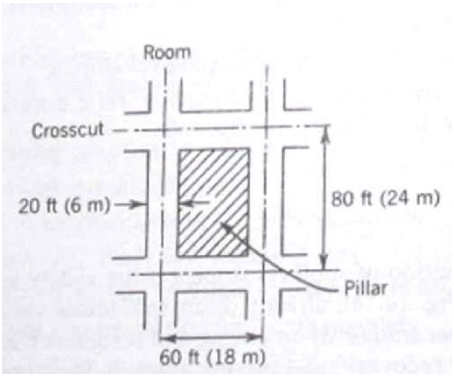


Figure 10.15. Portion of a panel in room and pillar mine. See Example 10.3.

(2) With pillar recovery:

$$\text{Area of openings} = \text{area of block} = 4800 \text{ ft}^3$$
$$\text{Recovery} = \frac{4800}{4800} = 100.0\%$$

The presence of barrier pillars along entries, usually required in any room and pillar mine, will reduce the recovery unless they can be extracted on the retreat. If second mining is not practiced, recovery in room driving can be improved by increasing the width of the openings and decreasing the size of the chain pillars, commensurate with safety in roof control. Note that the actual recovery realized during mining will differ from the calculated recovery because of losses of coal, pillar sloughs, and dilution by rock from roof falls.

10.7 SPECIAL TOPIC B: DESIGN OF MINE OPENINGS

Rock Mechanics

Rock mechanics is the study of the properties and behavior of rock the nature of the stresses about underground openings, and their relation in the design and support of mine workings and in the induced caving of rock in mine exploitation. All rock at depth is under stress due to the weight of the overlying rock (superincumbent load) and to possible stresses of tectonic origin. In addition, the presence of a mine opening induces or redistributes stresses in the rock surrounding the opening, and this rock (and the opening) will fail if the rock stress exceeds the rock strength (Obert et al.. 1960**).** Thus the problem of designing a stable mine opening reduces to determining (1) the maximum stress in rock surrounding the opening and (2) the strength of the rock in situ.

Rock mechanics is often defined more broadly. The aspect described above—that concerned with time rates of loading that are very long in duration—is referred to as ***static rock mechanics.*** A different aspect related to rock attack under rates of loading of short time duration and the corresponding behavior of rock is called ***dynamic rock mechanics.*** The latter includes rock penetration and fragmentation processes of all types, ranging from conventional means of drilling, blasting, and continuous mining to novel methods of applying energy to excavate rock such as fluid, thermal, and electrical attack (Section 4.2).

In this discussion, we shall be concerned with static rock mechanics only, because it is fundamental to a study of all rock mechanics and because the design, stability, and support of underground openings are fundamental to mining itself. We remind ourselves that the ultimate expression of depth as a constraint in mining lakes two forms and that one is the inexorable rise in rock stress (the other is the equally unrelenting climb in rock temperature).

Since our treatment of the subject of rock mechanics is abbreviated and restricted to the design of underground openings, a number of simplifying assumptions about rock prove helpful (Panck, 1951):

1. Rock is perfectly elastic (stress is proportional to strain).
2. Rock is homogeneous (there are no significant imperfections).
3. Rock is isotropic (its elastic properties are the same in all direction).

While never perfectly true, these assumptions apply reasonably well to many rocks (igneous best, sedimentary least) at moderate depth. Causes of departure *are the complex*, diversified, *and* heterogeneous nature of rock itself, the effects *of* high confining pressure and temperature at great depth, the presence o f water or solutions, and the effects o f geologic structures (bedding, fractures, fading, joint s. alteration, etc.). To a certain extent, uncertainties and departures from theory are compensated for in design by the use of factors of safety.
The following physical properties of rock are employed in many applications of rock mechanics, including the design of mine openings:

PROPERTY AND *DEFINITION*	RANGE
1.Young's modulus of elasticity = stress/strain	5-10 x 10^6 lb/in.^{2}
$E = \frac{S}{e}$	(34-69 x 10^3 MPa)
2. Poisson's ratio lateral I train/longitudinal strain $\mu = \frac{e_{lat}}{e_{long}}$	0.1 -0.3
3 . Unit strengths, based on unconfined uniaxial tests	•
a. Compression f_e	5000-50.000 lb/in.^{2} (34-345 MPa)
b. Tension f_t	400-2500 lb/in.^{2} (2.8-17 MPa)
c. Shear f_x	500-4000 lb/in.^{2} (3.4-28 MPa)
4 Specific weight w =62.4 SG (Eq. 2.1) w = 1000 SG (Eq. 2.1a)	70-450 lb/ft^3 (1120-7200 kg/m^3)

State of Stress About Mine Openings

We concern ourselves first with the state of stress existing in the rock before a mine opening is driven. The *vertical stress* S_y acting on a horizontal plane is equal to the weight of the overlying rock distributed over a unit area and is determined by the depth and specific weight of the rock:

$$S_y = wl = \left(\frac{62.4SG}{144}\right)L = 0{,}433SG\ x\ L\ Lb/in^2 \qquad (10.1)$$

$$S_y = wl == 1000SG\ x\ L\ Pa \qquad (10.1a)$$

For average rock with SG = 2.77, S_y = 1,2 L in lb/in.^{2} (2770L in Pa). The horizontal stress S_x acting on a vertical plane is a function of the vertical stress:

$$S_x = kS_y$$

in which *k* is a constant varying from 0 to > 1.

There are four possible cases for the stress field existing at point within the earth. They are represented diagrammatically in Figure 10.16 and summarized below:

Case 1. No confining pressure or restraint (k=0); occurs at very shallow depths, close lo faults, or adjoining a bench face:

$$S_x = 0 \qquad (10.3)$$

Case 2. No lateral deformation or strain (k=μ/1 - μ); occurs at moderate depths in elastic rock; if μ = 0.25, a typical value of rock,

$$S_x = \left(\frac{\mu}{1-\mu}\right) S_y = \frac{1}{3} S_y \qquad (10.4)$$

Case 3. Hydrostatic pressure (k=1); occurs at great depth or in wet, squeezing, and running ground:

$$S_x = S_y \qquad (10.5)$$

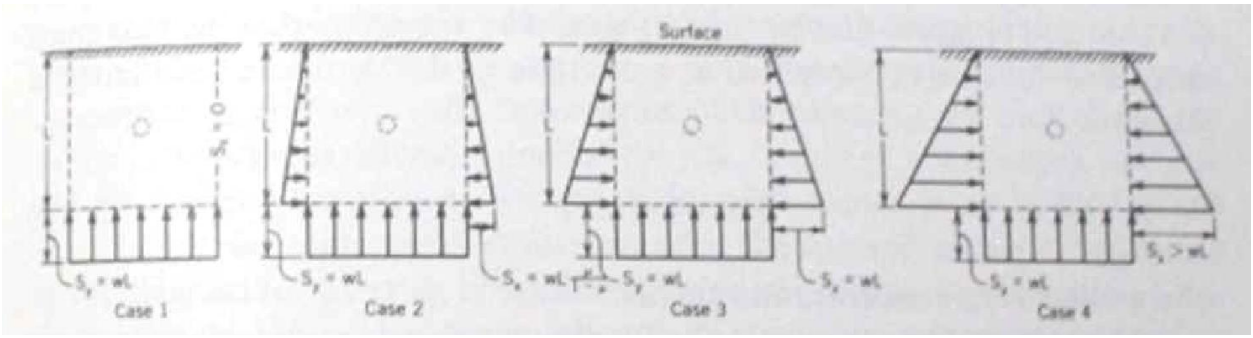


Figure 10.16. Stress fields within the earth. Case 1: No confining pressure Case 2: No lateral strain Case 3: Hydrostatic pressure. Case 4: High lateral pressure (Modif.ed after Panck. 1951 By permission from Louis A. Panek. Denver)

Case 4: High lateral pressure due to tectonic forces (k>1); occurs in regions of recent orogenic or volcanic activity:

$$S_x = kS_y$$

(same as Eq. 10.2) in which k ranges from 2 to 5. Because of its erratic, localized nature, we will disregard case 4.

Assume now that a mine opening penetrates the stress held existing within the earth at a given depth. Knowing the original stale of stress enables us to determine the magnitude of the redistributed stresses. This can be done by exact calculation for a circle (the simplest shape of opening), by numerical techniques and the computer for any opening, by model study and experimental measurements in the laboratory (using photo elasticity or holography! or by field measurements in an actual mine (using toad cells, stress meters strain gages, etc.). The results presented here were obtained i_n model studies and confirmed by other methods.

Since we are concerned only with the magnitude of the maximum stresses about a mine opening, our task is much simplified. From strength of materials and the theory of elasticity, we draw these conclusions (Panck. 1951, Obert 1973b):

1. The maximum stresses occur at the boundary of the opening and act tangentially to it; likewise the radial stress al the boundary is zero, and the shear stress is equal (o one-half the tangential stress.
2. The stresses are independent of the size of opening (but not of the shape or (he side ratio).

27The stresses arc independent of the elastic moduli of the material.

28 The peak values of the maximum (tangential) stress acting on a mine opening occur at the midpoints of the top and sides or at the corners of (he opening (if any),

29The stress redistribution about an opening is negligible at a distance of one diameter from the edge of the hole.

We also adopt two conventions. The first is that tension is represented by the (+) sign and compression by the (-) sign The second is that the maximum tangential stress *s,* is expressed as a multiple of the vertical stress acting in the earth *S/.*

$$s_t = cS_y \qquad (10.7)$$

where *c* is *stress concentration factor.*

Data obtained from the analysis o f a circular opening may be represented as shown in Figure 10.17. Values o f the stress concentration factor for the tangential stress are plotted along vertical and horizontal diameters ex tended into the side walls for cases 1, 2, and 3 (distances are plotted as radii).

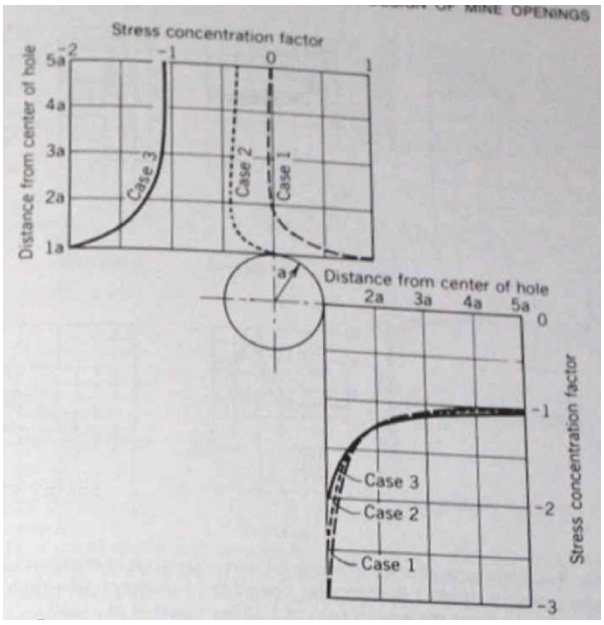


Figure 10.17. Stress concentration factor plotted for tangential stress along vertical and horizontal diameters of a circular opening In three different stress fields (After Caudle and Clark, 1955. By permission from University of Illinois at Urbana-Champaign.)

Alternatively, since the critical stresses occur at the walls of the opening stress concentration factors may be plotted along the boundaries only (Fig 10.18; ***left*** for a circular opening and *right* for rectangles of various side ratios *R/h* and ratio of fillet radius to short dimension *r/h* of). In the symbol convention, ***R*** is width, h is height of opening, and ***r*** is fillet radius of the rounded corner of a rectangle.

Graphically, we may also depict critical stress diagrams for openings as shown in Figure 10.19 ***(left*** for circular, ***right*** for rectangular), in which stress concentration factors at the boundaries of the opening arc plotted for the various stress fields. Stress values at the top, comer (if a rectangle), and side of each opening arc shown. With a circle, peak values occur at the top and side; with a rectangle, peak values occur at the top and corner. Sketching a stress diagram for existing conditions (shape of opening, stress field) is helpful in mine design in visualizing where high stress concentrations and failure may develop.

Theoretical and experimental studies allow us to draw the following conelusions conecming stress about single mine openings (Panek. 1951; Obert, 1973b):

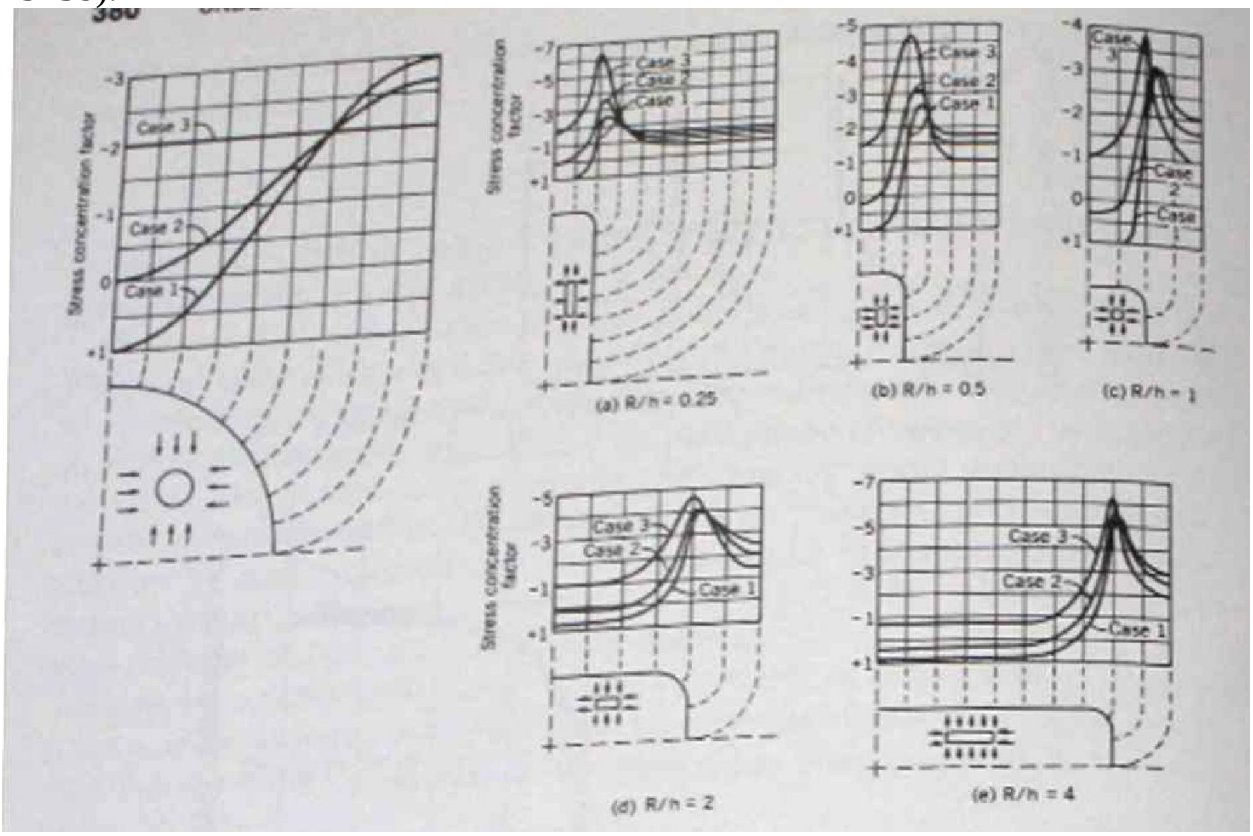


Figure 10.18. *Boundary stress concentrations for mine* openings in different stress fields, *(Left) Circular opening. (Right) Rectangular openings* of different side ratio *(R/h)* anri *ratio of fillet radius to short dimension (r/h) of . (After Obert et* al.. 1960.)

1. Case I stress field is most apt to produce tensile stresses about mine openings, followed by case 2 (case 3 is incapable of producing tensile stresses).
2. Tensile stresses are usually more serious than compressive in causing opening failure: peak tensile values occur at the top o f the opening.
3. Rectangular openings produce higher stress concentrations than circular: the more elongated the rectangle and the sharper the comers, the higher the peak stresses.
4. The preferred shapes of opening (starting with the lowest peak stress concentration factors) for different stress fields are the following:

a. *Case I :* ellipse (vertical orientation), rectangle (vertical), circle, square, rectangle (horizontal).
b. *Case 2:* ellipse (vertical), circle, square, rectangle (vertical), rectangle (horizontal).
c. *Case 3:* circle, square, ellipse (any orientation), rectangle.
5. Stresses are independent of the size o f the opening and properties of the rock.

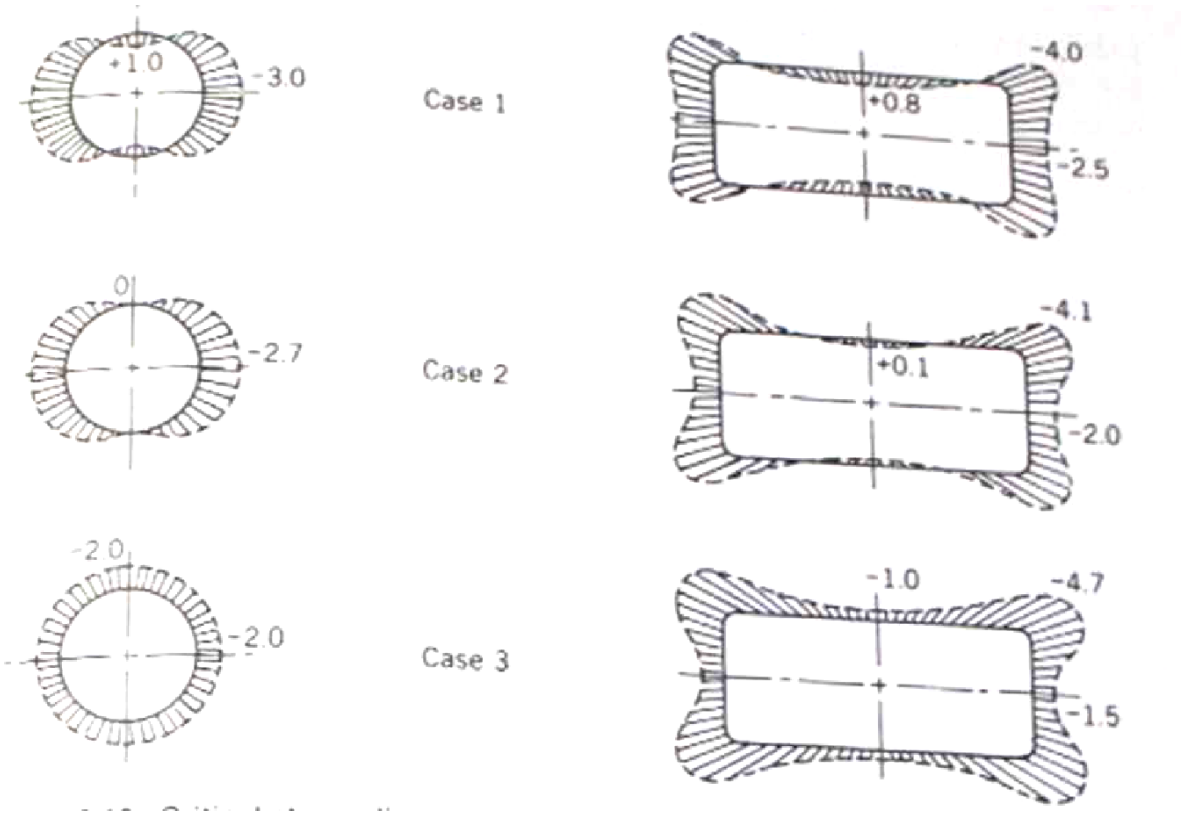


Figure 10.19. Critical stress diagrams for mine openings in different stress fields. Stress values at top, corner (if a rectangle), and side of the opening are indicated. (Left) Circular opening. (Right) Rectangular opening of side ratio R/h = 2 and ratio of filler radius to short dimension $r/h = i$. Plotted for cross section (d, *right), in* Figure 10.18.

While the presence of multiple openings establishes a different redistribution of stresses, the effect is small and may be neglected if the pillar width exceeds the opening width by a factor of 2 (Panek. 1951)

Compilations of critical stresses for the two basic shapes of mine openings, ellipse (including circle) and rectangle (including square), appear in Tables 10.1 and 10.2.

TABLE 10.1 Critical Values of Stress Concentration Factor c on an Elliptical Boundary in Different Stress Fields					
Width - Height Ratio = *R/h*	**Case 1. Top**	**Case 1. Side**	**Case *2.* Top**	**Case *2.* Side**	**Case 3, Top or Side***
025	**+ 1.0**	**—1.5**	**-2.0**	**-1.2**	**-8.0**
0.33	**+ 1.0**	**-1.7**	**-1.3**	**-1.3**	**-6.0**
0.5	**+ 1.0**	**-2.0**	**-0.7**	**-1.7**	**-4.0**
1 (circle)	**+ 1.0**	**-3.0**	**0**	**-2.7**	**-2.0**
2	**+ 1.0**	**-S.0**	**+0.3**	**-4.7**	**-4.0**
3	**+ 1.0**	**-7.0**	**+0.4**	**-6.7**	**-6.0**
4	**+1.0**	**-9.0**	**+ 0.5**	**-8.7**	**-B.0**

Design Procedure

Our objective in the design of mine openings is now clear. Employing the values of stress concentration factor given in Tables 10.1 and 10.2 for the basic shapes of mine openings and various stress field cases, calculate the critical (maximum) values of boundary stress to compare with the measured strength properties of the rock to determine if the opening is safe or will fail. A numerical example demonttrates the procedure.

Example 10.4. A square mine opening is driven horizontally at a depth of 2500 ft (750 m). The fillet ratio и J. Rock properties are SG= 2.77, $\mu=1$, f_c = *20000 lb/in.*^{2}*(10.3 MPa).* Determine the vertical and horizontal stresses in the earth, the critical boundary stresses in uniform stress field for each of the three common cases, and whether the opening will fail. Calculate the factor of safety for one case that is safe.

SOLUTION. Calculate the vertical stress in the earth by Eq. 10.1:

$$S_t = 0{,}433 \text{ SG x L} = 1.2(2500) = 3000 \text{ lb/in.}^2 (20.7 \text{ MPa})$$

Read stress concentration factors from Table 10.2 for R/h =1 (square opening):

Case 1: $S_x = 0$ (Eq. 10.3)

Top c =+1.0, S_1 = + (1.0)(3000) + 3000 lb/in.^{2} >1500— fails

Corner c=3.1, SG x L = 1.2(2500) = 3000 lb/in.^{2}
<=20000 – safe

Case 2: $S_x = \frac{1}{3} S_x = (3000) = 1000\ lb/in^2$ (Eq 10.4)

Top c =-3,1,

Normally, a factor of safety of 1.25 suffices in the design of mine openings, depending on the confidence limits of the data and the permanence of the openings. If the factor of safety is 1 or less, then provision of artificial support is likely necessary. In designing (or induced caving (Chapter 12), the opening shape is chosen to ensure that failure does occur lie (i.e., FS<1). Usually, the *R/h* ratio is the only design parameter that can be varied, since a rectangular cross section is nearly always employed (unless the opening is bored).

If the rock sequence in which the opening is located departs drastically from elastic, homogeneous, and isotropic conditions, then the approach illustrated here is less applicable. In well-bedded rock, for example, as encountered in the mining of coal and most nonmetallies. It is advisable to compute the factor of safety for the corner stress based on the weaker rock strength, either the bed in which the opening is driven or the bed constituting the roof (back). For more accurate results, a different analytical procedure is recommended (Obert et al., 1960).

Extensive literature on rock mechanic* is available, the more noted works being Woodruff (1966). Obert and Duvail (1967), Jaeger and Cook (1976), Goodman(1980), Hock and Brown (1980), Coates (1981), Biron and Arioglu (1983), Budavari (1983), Bieniawski (1984), Brady and Brown (1985), and Hartman (1486).

P R O B LEM S

10.1 A room and pillar coal mine is worked with continuous mining machines under the following conditions.

Room dimensions	6 x 20 ft (1.8 x 6.0 m)
Advance per cut	(10 ft. or 3 m. width) 40 ft (12 m)
Tonnage factor	2 4 f t/ton (0.7 5m/tone$)
Required shift production	1000 tons (910 tonnes)/machine
Net shift working time	6 hr 55 min
Number of shuttle cars pt	2
Capacity of car	6.25 tons (5.67 tonnes)
Delays while loading car	0.6 min
Delays while changer car	0.4 min
Delays while miner placer car	3.0 min/cut

a. How many cuts must be made per shift?
b. Calculate the required capacity of the continuous miner (in tons/min, or tones/min) to achieve the shift production goal.

10,2 A cycle of operations is bong planned for a high-speed drifting project in if metal mine in which one cycle (blasting round) must be realized per shift. To keep up with the cycle, the loading operation must be completed in I hr. The following data are known or assumed.

Size of drift	10 x 15.6 ft (3.0 x 4.8 m)
Round advance	8 ft (2.4 m>
Ore tonnage factor	12.5 ft-Vton (0,39 mVtonne)
Mine car capacity	5 tons (4.5 tonnes)
Preparation time per cycle	30 min
Car change time	1.5 min/car

Determine Ihe correcl rating of front-end loader, compressed-air-opcrated, that should be selected to fit into the cycle, if loaders with the following capacities are available:

1, 2. 4. 5, 6 tons/min
(0.9. 1.8. 3.6, 4.5. 5.4 tonnes/min)

10.3. A coal mine is using the room and pillar method, block version, with rooms and crosscuts 20 ft (6 m) in width, driven on 90-ft (27-m) centers. Entries and entry crosscuts are the same dimensions. If no barrier pillars are left, compute the percentage recovery with and without pillar extraction.

10.4. A раде! of 30 rooms in a room and pillar coal mine is to be driven from a set of three room entries connecting to panel entries. The rooms arc driven on 100-ft (30-m) centers; they are 16 ft (4.8 m> in width and are 300 ft (90 m) in length. Room entries are also 16 ft (4.8 m) in width on 100-ft (30-m) centers. Crosscuts between both rooms and entries are placed on 100-ft (30-m) centers. A barrier pillar 100 ft (30 m) in width will be left on three sides of the panel. A sketch of the mining plan appears in Figure 10.20. Compute the percentage recovery for the mining panel in each of the following cases:

a. No pillar recovery
b. Recovery of chain pillars in rooms and entries
c. Room width increased to 20 ft (6 m) and no pillar recovery

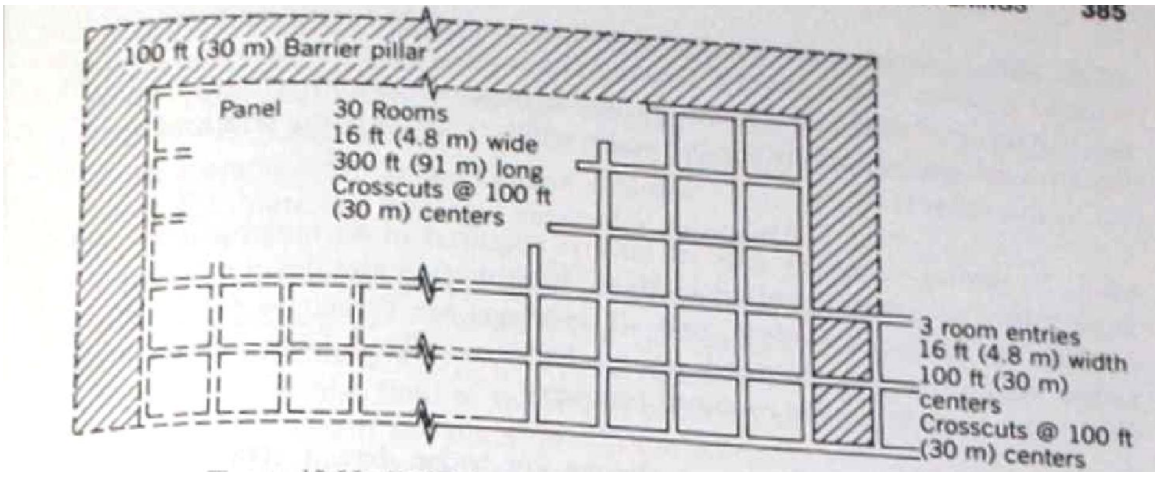


Figure 10.20. Panel in room and pillar mine See Problem 10.4

10.5 A panel of 30 rooms in a room and pillar coal mine is to be drive from a set of three room entries connecting to panel entries. The rooms are driven **20** ft (6 m) wide on 50-ft (15-m) centers and are 300 ft (90 m) in length. Room entries are 15 ft (4.5 m) in width on 50-ft (15-m) centers. Crosscuts between both rooms and entries are placed on 100-ft (30-m) centers. A barrier pillar 50 (15 m) in width will be left on three sides on the panel. A sketch of the mining plan appears in Figure 10.21. Compute the percentage recovery for the mining panel in each of the following cases:

a. No pillar recovery

b. Recovery of room and entry pillars

c. Room width increased to 30 ft (9m), with timbering, and no pillar recovery

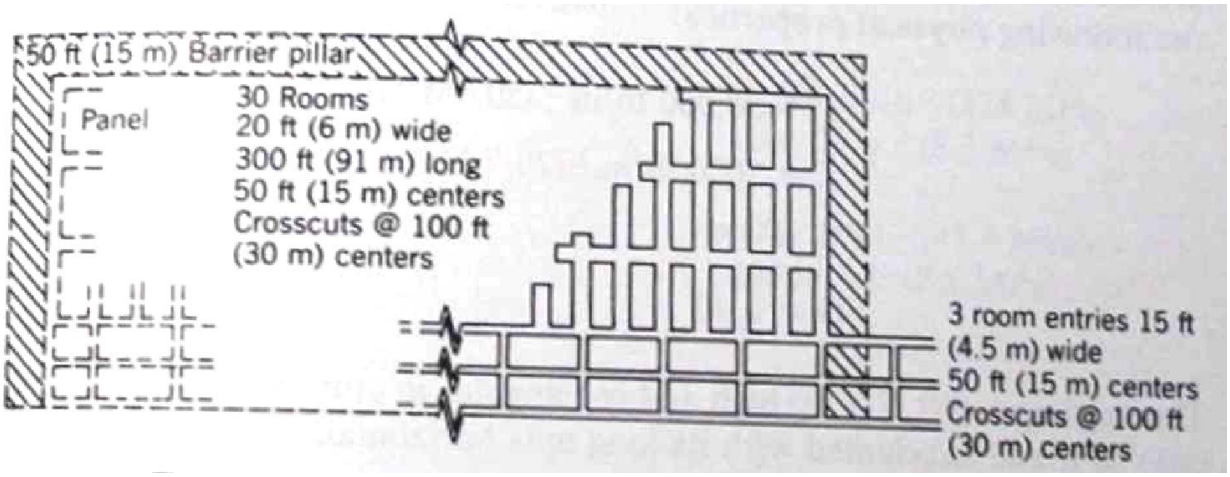


Figure 10.21. Panel in room and pillar mine, See Problem 10.5

10.6. What should be the room and crosscut width in a room and pillar coal mine to achieve 55% recovery, leaving chain pillars but no barriers? Rooms are driven on 110 – ft (33m) centers and crosscuts on 60 – ft (18 m) centers. Both rooms and crosscuts are the same width. Calculate the width of opening to the nearest foot.

10.7. A mining recovery rate of 60 % is required in an underground coal mine using the room and pillar method. Exploitation and development both are land out in block system, and all openings are driven 18 ft (5.4 m) in width. Pillars are not recovered, nor are barriers utilized. Calculate the pillar dimension to provide the required recovery.

10.8. Mine openings of various shapes are to be driven 3000 ft (915 m) underground in a stress field of no lateral strain. The rock overlying and enclosing the openings has the following physical properties:

f_c= 22.500 lb/in.^{2} (155 MPa)

f_t = 1200 lb/in.^{2} (8.3 MPa)

μ=0.25

SG= 2.77

The openings are rectangular in cross section, have a fillet ratio of, and are oriented horizontally with side ratios (R/h) of 1,2,4, and 8.

a. Compute the rock stresses existing in the earth at (hat depth.

b. Sketch and label the stress distribution for each opening.
c. Compute the actual critical stresses.
d. Determine which openings, if any. will fail.
e. Compute factors of safety for those (hat will not fail.
f. Identify the purpose for which each opening might be used.

10.9 A rectangular mine opening is to be driven 5000 ft (1525 m) underground in a hydrostatic stress field. The rock surrounding the opening has the following physical properties:

f_c = 30.000 lb/in.^{2} (207 MPa)
f_t =3000 lb/in.^{2} (20.7 MPa)
μ = 0.5
SG = 2,77

The opening is 8 ft (2.4 m) high and rectangular in cross section, has a fillet ratio of J. and is oriented with its long axis horizontal.

a. What is the widest possible opening (in even multiply of 8 ft, or 2.4 m) which may be driven?
b. Compute the rock stresses existing in the earth at the given depth.
c. Sketch and label the stress distribution for the opening.
d. Compute the actual critical stresses.
e. Compute the factor of safety for the opening (must exceed 1).

10.10. A single rectangular opening 10 ft (3.0 m) in height is driven in rock having strengths of f_c = 18 000 lb/in.^{2} (124 MPa) and f_t = 15 000 lb/in.^{2} (10,3 MPa). Rock specific gravity is 2.3. The opening is located at a depth of 2000 ft (610 m) in a stress field of no lateral pressure and has a fillet ratio of Ł.
a. Determine is the opening will fail when its width is 20 ft (6.1 m).
b. Is there any benefit to reducing the width to 10 ft (3,0 m)? To 5 ft (1.5 m)?
c. What is the maximum safe width of opening?

10.11 Openings 16 ft (4.9 m) wide and 8 ft (2 4 m) high have been driven without failure in a hard-rock mine; the fillet ratio is Ł. The rock massive basalt with a specific gravity of 2.5, compressive strength of 25.000 lb/in.^{2} (172 MPa), and tensile strength of 3500 lb/in^2 (24.1 MPa). In a remote extension of the ore body, at a depth of 4500 ft (1370 m), failure of the mine openings is encountered for the first time. Investigate the failure, determine which stress field prevails in that section of the mine, and specify which part(s) of the opening fail(s).

10.12 An attempt was made to drive rooms 28 ft (8,5 m) in width in a 42 – in seam of coal but failure occurred. Investigating from critical stress standpoint,

what is the maximum safe width of room (in even multiples of side ratio given in the stress concentration table) that can be driven ? The fillet ratio is *I*. The following conditions prevail:

Stress field	No lateral strain
Depth	1000 ft (305 m)
Specific gravity of overburden	2.31
Strength of roof strata	
Compressive	18 000 lb/in.^{2} (124 MPa)
Tensile	1200 lb/in.^{2} (8.3 MPa)
Strength of coal	
Tensile	6000 lb/in.^{2} (41.4 MPa)
Compressive	400 lb/in.^{2} (2.8 MPa)

11 .
UNDERGROUND MINING: SUPPORTED METHODS

11.1 CLASSIFICATION OF METHODS

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

Where they can be utilized, pillars are the ultimate form of ground control because they are in place and because they are capable of providing near-rigid support (Boshkov and Wright, 1973). In certain horizontal mining methods (e.g., room and pillar mining, slope and pillar mining), pillars arc relied upon almost exclusively, supplemented by light artificial supports such as bolts and timber. In the vertical methods (which include practically all sloping), pillars maintain stability between stopes, but they are impractical and seldom employed within stopes because they interfere with mining operations. Heavy artificial supports, therefore, generally have to be resorted to within the openings where use of a supported method is indicated.

Support systems for production workings are chosen to provide varying degrees of controlled wall closure and ground movement. Next to pillars, the most satisfactory form of support is backfill, which approaches 100% in its ability to support the superincumbent load without yielding. In certain instances, some yielding is acceptable and, in fact, preferable because artificial support cannot hold the superincumbent load. Heavy support systems of this type include timber stulls and cribs, timber or steel sets and trusses, and steel jacks, props, arches, chocks, shields, and canopies. Timber is weaker and yields more than steel (sometimes a desirable feature) but is readily available, flexible, workable, easy to install, and economical (Wilhelm, 1982).

In the design of artificial support system for mining methods, an evaluation – preferably quantitative – of the load – carrying capacity of the natural rock structure is prerequisite. Rock mechanics tests are performed evaluate, ate the structural properties of the rock in situ (review Section 10.7) A useful though qualitative term to employ in classifying rock is competency. A *competent rock* is defined as rock which, because of its physical and geologic characteristics, is capable of sustaining openings without any heavy structural supports (Obert et al., 1960). Unfortunately, no field or laboratory tests have been devised to determine

competency in large rock masses, nor can relative terms such as hard and soft or strong and weak satisfactorily express it. Our best approaches are empirical, such as the rock quality designation (RQD) based on drill core evaluation. (Secton 12.4).

The supported class of mining methods is intended for application to rock ranging in competency from moderate to incompetent. There are three methods in this class (Table 3.1):

1. Cut and fill stoping
2. Stull stoping
3 . Square set stoping

11.2 CUT AND FILL STOPING

The only one of the supported class in common use today, cut and fill stopping, is normally an overhand mining method in which horizontal shoes of ore arc excavated in (he slope and replaced with waste as fill. The filling is conducted integrally with the mining cycle and not after completion of the entive mining operations. An estimated 3 % of U.S. underground mineral production is derived from cut and fill stopping, although currently there is renewed interest in the method for mechanized mining of deep ore bodies in heavy ground.

Fill performs unique functions in this method. The most obvious and important is the ground support it provides the weak walls of the deposit. In this regard, compressibility is the most critical fill property, varying from 25 % for mechanically placed dry fill to 5 – 10 % for hydraulically or pneumatically placed fill (Lucas and Haycocks, 1973). A second function served by fill is in providing a working platform on which the next ore slice is drilled and blasted. Because of voids, only about 0.6 ton (or tonne) os fill is required per ton (or tonne) of ore mined.

Cut and fill stopping evolved as a method in which the fill material was coarse-broken muck, derived from development work throughout the mine. The dry fill was dropped in the stope through fill raises from the level above and distributed by slushier (Fig. 11.1). If mine waste rock was not available or was unsuitable, surface waste material (rock, gravel, or sand) was taken underground by skip or through a specially developed system of fill raises. Because of high distribution cost and poor compressibility, mechanically placed fill has been largely replaced by hydraulic fill with is higher productivity, lower cost, and better compressibility (occasionally, pneumatic fill is used instead). This modern version of cut and fill stoping is shown in Figure 11.2. typically, stoping with hydraulically placed fill lends itself to increased mechanization because of the surface is usually hydraulically slurried and piped underground for distribution in the stopes: sometimes, cement is added to produce a weak concrete fill, or natural – setting sulfide tailing are employed for greater fill strength.

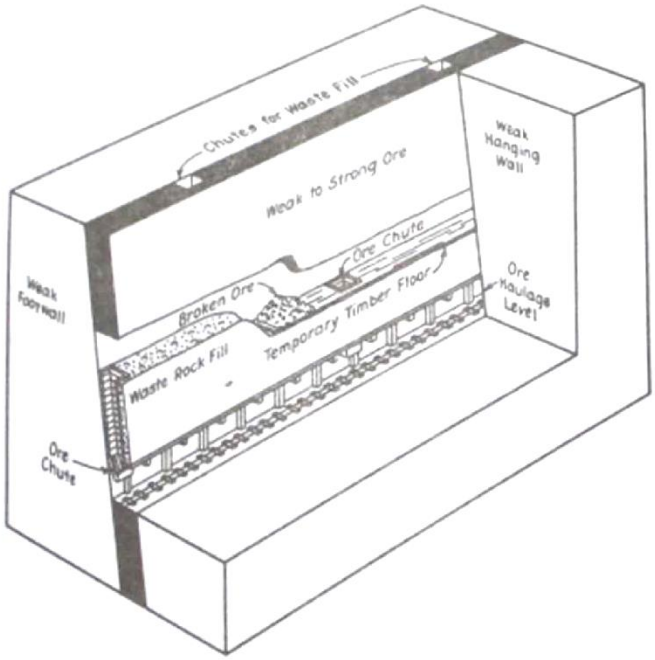


Figure 11.1. Cut and fill stoping using mechanically placed fill and raises for access. (After Lewis and Clark. 1964. Copyright Φ 1964, John Wiley & Sons, New York.)

Hydraulic filling in cut and fill sloping requires special placement and drainage techniques since the slurry contains 30-40% water (Lucas and Haycocks, 1973; Thomas, 1978). All access openings into from below have to be equipped with bulkheads to prevent flooding the haulage drift during filling. Sand barricades or fill fences-earthen dams or burlap – covered lagging on posts – are erected in the stope to control the placement of fill. The tops of man ways and ore passes must be extended above the fill floor to keep them open. To Provide proper drainage of the fill while it sets, percolation drains arc installed along the stope sill, and decantation towers are maintained through the fill; runoff water must be disposed of in the drainage system on the level below. Fresh fill withstands human traffic in a few hours and vehicular traffic after two to four days.

Similar to other vertical exploitation openings, cut and fill stopes are generally bounded by pillars for major ground support. Because the stopes are filled, however, these pillars often can be recovered, in part or totally.

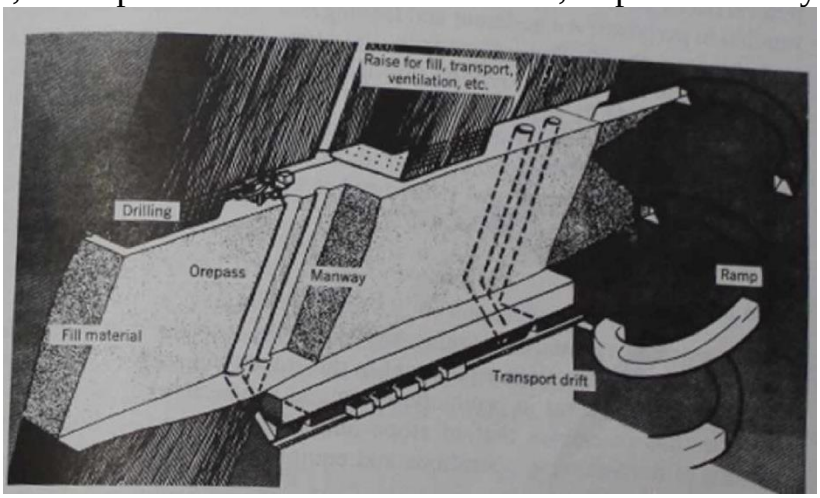


Figure 11.2. Cut and fill stoping using hydraulically placed fill and ramps for access. (After Hamrin, 1982. B y permission from Society of Mining Engineers, Inc.. Littleton, CO.)\

The timing of fill placement in cut and fill stoping is critical to the success of the method, since the fill must be in place in time to assume some or all of the original superincumbent load on the ore in the stope.

Design parameters in cut and fill stoping are determined in much the same way as with any other vertical stoping (not caving) method. In addition to rock mechanics and fill placement considerations, however, stope dimensions are influenced by mechanization factors, such as ease of access, maneuverability of equipment, and production rate requirements. For example, stope height varies from 150 – 300 ft (45 – 90 m) and stope width from 6 – 100 ft (2 – 30 m), limited mainly by rock mechanics and fill placement concerns (Morrison and Russel, 1973). Stope length is determined by mechanization requirements; it varies from 200 – 2000 ft (60 – 600 m) (Lucas and Haycocks, 1973). Typically, the thickness of each horizontal slice removed is 8 – 12 ft (2.4 – 3.6 m) and is mainly a function of the drilling method used (Waterland, 1982). Orepasses must be maintained in the fill; they vary from 6 – 8 ft (1.8 – 2.4 m) square and are spaced up to 200 ft (60 m) apart, depending upon the materials – handling equipment (Pugh and Rasmussen, 1982).

As currently practiced, using hydraulic fill placement, cut and fill stoping is a moderate – scale method.

Sequence of Development

Development of cut and fill stopes follows the same general plan as other vertical stoping methods, but there are significant differences. For example, a haulage drift, crosscuts, and drawpoints or draw drift, together with interlevel raises are driven in the usual way for access, materials-handling, and ventilation purposes. An undercut and loading bells are not necessary, however, because gravity flow of ore from the stope to the drawpoints takes place to u g h orepasses, extended through the fill as the stope advances upward (see Fig. 11.2). Mining above the sill commences by making the first horizontal cut across the stope. If mobile equipment is used in the stope, then ramps must be driven (maximum grade: 15-20°) between levels or else the equipment is captive in the stope until it breaks through to the level above (Thomas, 1978; Waterland, 1982).

Cycle of Operations

Mechanization of cut and fill stoping has led to the introduction of mobile drilling, loading, and haulage equipment in the stopes themselves, with corresponding improvement in production rate and productivity. The production cycle thus resembles that of stope and pillar mining (Section 10.3), consisting of the following operations and equipment (Section 4.1):

Continuous mining: road header in soft to medium rock (e.g., borate) (Sparks, 1980); in hard rock, use conventional cycle

Drilling; pneumatic hand drill (airleg or stoper), drill rig mounting pneumatic or hydraulic machines (percussion or rotary – percussion); hole size 2 – 3 in. (51 – 76 mm)

Blasting: ANFO, slurries; charging by cartridge or bulk by pneumatic loader or pump, firing electrically or by detonating fuse; blasting round consists of horizontal or inclined holes, using overhand small – or moderate – scale method (Fig. 9.9b) (Waterland, 1982; Hamrin, 1982)
Secondary breakage; (in stope) drill and blast, mudcap, impact hammer
Loading: (in stope) LHD, scraper to shute or orepass; gravity flow to drawpoints; (on level) LHD, front – end loader, shovel loader (Lucas and Haycocks 1973)
Haulage: LHD, truck, rail

Incorporation of filling – an auxiliary operation and a form of ground control— into the production cycle during stoping is responsible element of discontinuity in the cycle of operations. Time is consumed in erecting fill barricades and extending raises, moving equipment< placing the fill, and allowing the fill to drain and set (if hydraulically placed). If mechanically placed, a floor or mat is sometimes required to prevent dilution of the ore by the waste. Fill is placed after each horizontal ore slice is completely mined rather than after each blasting cycle; in this respect, cut and fill stoping employs a specialized cycle of operations Minimal roof control (light bolting or timbering) is performed as needed after each blast.

See Sections 10.2 and 10.3 for other auxiliary operations and equipment, of which ventilation is usually first in importance because of limited access and egress from cut and fill stopes.

Conditions

The following is based on Boshkov and Wright (1973), Morrison and Russell (1973), Thomas (1978), and Hamrin (1982).

1. *Ore strength:* moderate to strong, may be less competent than with unsupported methods.
2. *Rock strength;* weak to fairly weak
3. *Deposit shape;* tabular; can be irregular, discontinuous
4. *Deposit dip;* moderate to fairly steep (>45 - 50°); can accommodate flatter deposit if orepasses are steeper than angle of repose
5. *Deposit size:* narrow to moderate width (6 – 100 ft, or 2 – 30 m), fairly large extent
6. *Ore grade:* fairly high
7. *Ore uniformity:* moderate, variable (can sort waste in stope)
8. *Depth:* moderate to deep (<4000 – 8000 ft, or <1.2 – 2.4 km)

Features

The following is based on Lewis and Clark (1964), Lucas and Haycoks (1973), Morrison and Russell (1973), Hamrin (1982), and Waterland (1982):

Advantages

1. Moderate productivity (range 10-20 tons, or 9-18 tonnes, per employee-shift: maximum 30-40 tons, or -7-36 tonnes, per employee shift)
2. Moderate production rate; moderate-scale method
3. Permits good selectivity, sorting; can use waste as fill
4. Low development cost
5. Moderate capital investment; adaptable to mechanization
6 Versatile, flexible, and adaptable
7. Excellent recovery if pillars mined (90-100%), low dilution (5-10%)
8. Surface waste (tailings, etc.) can be disposed o f underground as fill
9. Moderately good safety record

Disadvantages

1. Fairly high mining cost (relative cost. 60%)
2. Costly handling of waste, may be 50% of mining cost
3. Filling complicates cycle, causing discontinuous production
4. Must provide stope access for mechanized equipment
5. Tends to be labor intensive, requiring skilled miners and close supervision
6. Com pressibility o f fill risks some ground settlement and in stability

Applications

With the application o f mechanization in recent years, cut and fill stoping has regained some o f its popularity. Examples include copper (Magma, AZ— sec Florez. 1982), copper-lead (Mt. Isa, Austr.— see Hornsby et al.,1982*)*, gold (Homestake, SD — see Waterland et al., 1982), nickel (Thompson. Man.— see Humphrey and Nicholson. *1979;* Sudbury, Ont.— see Schabas, 1977; Tims, 1982), and silver (Star. ID — see Miner, 1982).

Case Study. Sudbury nickel mines, Falconbridge Nickel Mines Ltd. and Inco Metals Co. Ltd., Sudbury, Ont.

General: major nickel-producing area in the world, including surface and underground mines o f two producing companies; located in Sudbury Basin in central Ontario; mining dates to 1880s
Production: 68 400 tons / day (62 000 tonnes/day) from 12 mines, 3 mills, 2 smelters, and 2 refineries
Mining method: open pit mining, sublevel stoping, cut and fill stoping, and sublevel caving

Variations

Lucas and Haycocks (1973) identify eight varieties of this versatile method, *Underhand cut and fill stoping* (or undercut and fill) is the principal variation of cut and fill stoping, itself an overhand method. Originating in the Sudbury Basin, underhand cut and fill was intended for use in pillar recovery in bad ground (Murray, 1982). Horizontal slices of ore are taken from the top down, and fill is placed against a timber fence along the sides and mat across the floor of the stope. Recovery and selectivity are high, dilution is low, and ore strength can be weaker than in overhand cut and fill stoping. The method is more labor intensive and costly, however. Other older variations are *rill cut and fill stoping*, in which the stope fill is inclined to facilitate the gravity flow of muck to the chutes, and resuing, in which very narrow veins (<6 ft, or <1.8m) are worked by mining and sorting the stope walls separately from the ore.

To improve its productivity, Pugh and Rasmussen (1982) have proposed a variation of the method called *highly mechanized cut and fill stoping.* An inclined timber floor is laid in the stope from the haulage drift below to provide ramp access for rubber-tired drilling and loading – haulage equipment, usually jumbos and LHDs (Fig. 11.3). The ramp is extended as the stope advances, and a wedge-shaped block of ore (1) is minedout. Alternate blocks remain as pillars for subsequent recovery. Following in Figure 11.3, blocks I, II,III, and IV are mined on the retreat. With improved access and mechanization, this variation of cut and fill stopping should prove highly productive and more competitive with the unsupported class of stoping methods.

11.3 STULL STOPING

Stull stopping is infrequently used and relatively unimportant today; the method accounts for less than \% of U.S. underground mineral production. Never widely popular, it is nevertheless useful under certain circumstances (e.g., thin, pitching, tabular deposits with moderately weak wall rock). Overhand stopping in which systematic or random timbering with simple supports is employed for ground control is referred to as *stull stopping.* The timbers are usually single *posts* (in a near-horizontal deposit) or *stulls* (in a pitching deposit) with an optional cap or brace, which are intended to provide temporary support. If the sets are more elaborate and form a listing support structure, the method is called square set stoping (Section 11.4).

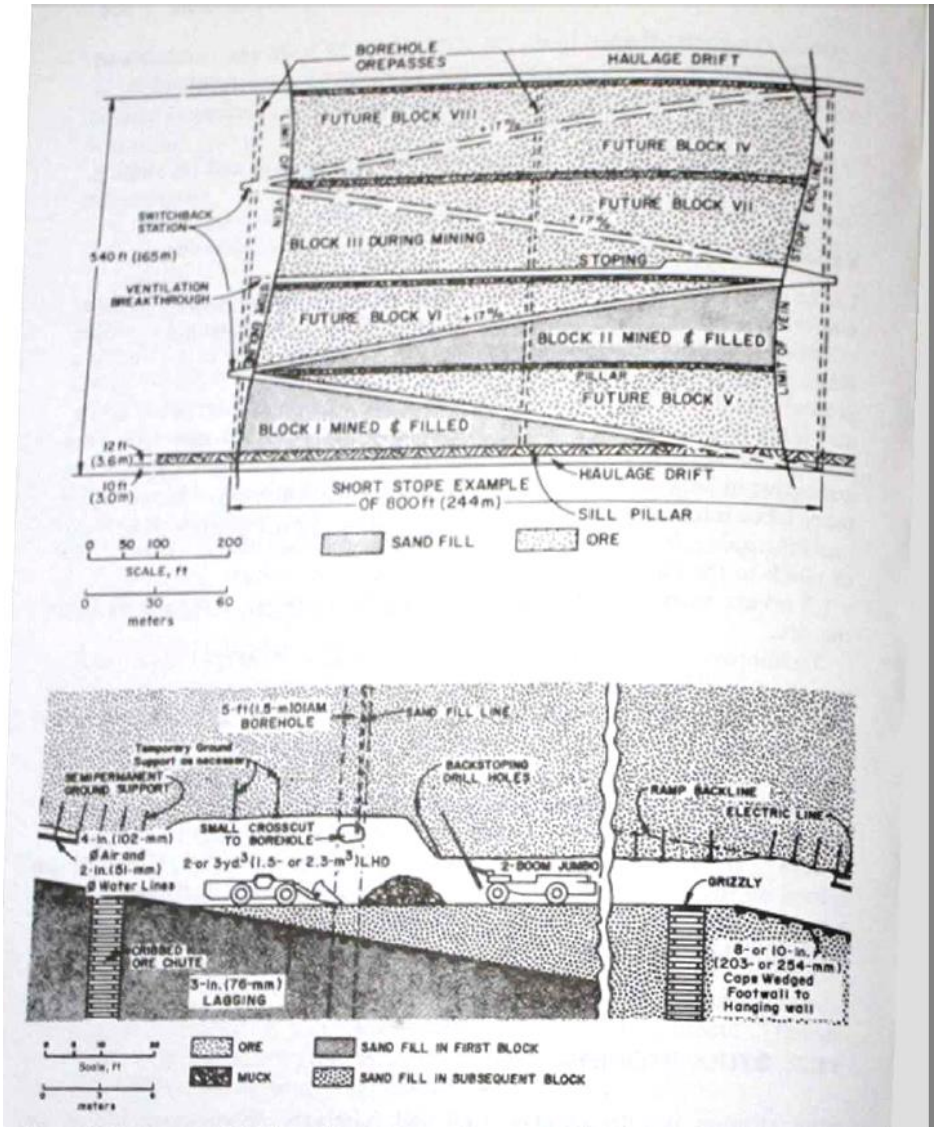


Figure 11.3. Proposed highly mechanized version of cut and fill mining. *(Top)* Location of ramps to permit mining of alternate blocks of ore; stoping in progress in block III. (flo*ftom)* Stopping operations, with drill jumbo and LHD. (After Pugh and Rasmussen. 1982. By permission from Society of Mining Engineers. Inc., Littleton. CO.)

As originally used in underground supported methods, ground control was accomplished by timbering, sometimes supplemented with hackfilling for more permanent support. Eventually, steel sets, jacks, trusses, arches, and patented support designs (chocks, shields, and canopies) were also employed in heavy ground and rock bolts, mesh, cable, and shotcrete in light ground. Where timber supplies are plentiful and not expensive, wood supports still find primary application in stull and square set stoping (and to a lesser extent in room and pillar coal mining). Wood of several species is utilized, depending primarily on local supply. High strength is the most important property sought, although the tendency of wood to split before failure, accompanied by emission of a cracking warning sigh, is also much valued (Lewis and Clark, 1964; Wilhelm 1982; Taylor, 1982). Preferred species are oak, fir, spruce, aspen, hemlock, pine, chestnut, and maple; hickory and willow are too flexible and yield by bending with no warning prior to failure, and brittle woods fail too abruptly. Treatment of all timber to preserve it

and prolong its life is mandatory with application in mines. The advantage of wood over steel for mine support, other than the warning noise before failure, is the ease with which it can be cut or shaped in the working place, although most complicated timber structures (e.g., square sets) are completely framed outside the mine and ready for assembly in the workings. The greater expense of steel supports can only be justified when they are reused.

In stull stoping, as in square set stoping, the means of ground support also provide a working platform for the miners (Fig. 11.4). Slices of ore are mined horizontally or at an angle (but less orderly than in cut and fill stoping), progressing upward from the sill. As supports are installed with advance of the back, flooring consisting of lagging is laid, permitting the miners to reach the back. The lagging doubles in providing a flooring on which the muck may be moved-by gravity if the flooring is inclined, or by slusher if near-flat.

Normally, systematic pillars arc not used in a stull stope. Waste may be left in place as small, occasional pillars. Boundary pillars (sill, crown and rib) are left between slopes. To provide more permanent ground control the stope may be filled periodically (Lucas and Haycocks, 1973; Thomas. 1978). In this modification, stull stoping resembles cut and fill stoping or, if the support becomes more elaborate, square set stoping. It also resembles stope and pillar mining when conducted in more horizontal deposits, although pillars are infrequent and reliance is placed instead on artificial support such as stulls or bolts. Stull stoping is employed, too, for pillar recovery. When the ground in a steeply inclined deposit is bad, then stull stoping can be carried on underhand.

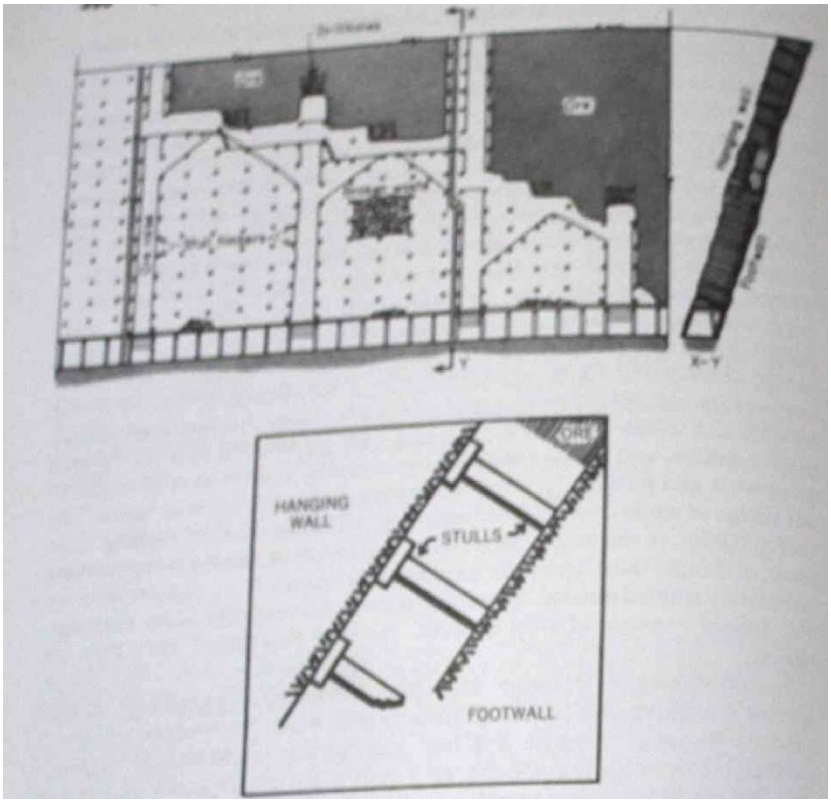


Figure 11.4. Stull stoping in an inclined tabular deposit (top). Two views of stope. (After Jackson and Hedges, 1939). (Bottom) Details of limbering with stulls. (After Paele, 1941, Copyright 1941, John Wiley & Sons New York).

Sequence of Development

Development for Mull storing follows the conventional sequence for verbal stoping methods but is kept minimal and simple. Because it is a *small-scale* methods elaborate development is neither warranted nor necessary. After essential level development is earned out and access to the slopes is prodded (undercutting is not required), raises arc driven between levels. Ore passes are formed as the slope advances, so bells and draw points are unnecessary (generally, the ore passes terminate in chutes to permit total gravity flow of muck through and from the stopes).

Conditions

The following iv based on Morrison and Russell (1973) and Thomas (1978);

1. *Ore strength:* fairly strong to strong, more competent than for cut and fill stoping.
2. *Rock strength:* moderate to fairly weak
3. *Deposit shape:* approximately tabular; can be irregular
4. *Deposit dip:* moderate to fairly steep (>45-50°); can accommodate flatter deposit if ore passes arc steeper than angle of repose
5. *Deposit size:* narrow (<12 ft, or <3.6 m), can be small in areal extent
6. *Ore grade:* fairly high to high
7. *Ore uniformity:* moderate; variable (can sort waste in slope)
8. *Depth:* moderate (<3500 ft, or <1.1 km)

Features

The following is based on *Lewis and* Clark *(1964)* and Thomas (1978);

Advantages

1. *Simple* method, adaptable *to* «mall, irregular ore bodice
2. Requires little *mechanization*; km *capital* cost
3. Low *development* cost
4. *Selective;* versatile *for thin tabular* deposits at any angle
5. Good recovery *if pillar mined* (< 90 %); low dilution (5-10%)

Disadvantages

1.Low productivity
2. Low production *rate;* small-scale method
3. High *mining* cost (relative cost: 7090
4. Labor intensive, slow
5. Heavy timber requirement, high support cost; heavy ground requires fill or pillars
6. Limited applications in current mining

Applications

The future of stull sloping will very likely continue to decline, Its recent, limited applications include silver-lead-zinc (Star. ID — sec Lucas and Hay cocks, 1973) and uranium *(U T* — see Taylor. 1982).
Case Study. Star silver-lead-zinc mine. Hecla Mining Co., Burke. ID
General: *deep* mine. Couer d'Alene district: mining began 1940
Geology: vein deposits; ore galena, marmotite; ore grade 2.5 oz/ton (79 *g/*tonne) Ag. 5 % Pb, 5.6% Zn; reserves 1.5 million tons (1.4 million tonnes)
Production*: 1*100 tons/day (1000 tonnes/day).
Mining methods: cut and fill sloping, stull stoping, and square set stoping.

11.4 SQUARE SET STOPING

The final method in the supported class, square set stoping, is probably the least used o f all mining methods and. in fact, may be approaching obsolescence. In large part, this reflects its unwelcome distinction as the most expensive (along with dimension-stone quarrying) o f all the mining methods.

Its usage has declined to an estimated <1% of U.S. underground mineral production. We include it here not because of its present importance but its historical contribution to world mining. Like room and pillar mining and block caving, square set stoping is distinctively American in origin.

In square set stoping, small blocks of ore are systematically extracted and replaced by a prismatic skeleton of timber sets, framed into an integrated support structure and backfilled floor by floor. So versatile that, it can be operated overhand, underhand, or breast and in any shape of ore body, square set stoping has equal use for first mining and remnant or pillar recovery (Lucas and Haycocks, 1973; Thomas, 1978). It is capable of applications to the weakest ore and rock under the heaviest ground conditions. Only the vertical caving methods (sublevel and block caving) can complete with it in this regard, and then only when subsidence is permitted. If caving is not tolerable, cut and fill stoping may be its only competitor and then only marginally. For the most extreme conditions in mining, square set stoping remains the method of last resort.

During sloping, the square sets arc assembled m place, having been cut to size and framed on the surface; sec Figure 11.5. Note the details of the framing of the timber and the assembly of each joint. Even though the rectangular cross section of the square sets is basically unstable, diagonal bracing is generally not used (Gardner and Vandenburg, 1982). Structural stability is lent to the timber sets by periodic and total backfilling; it is the fill, not the timber, that withstands most of the ground pressure. Square sets are generally about 6 or 8 ft (1.8 or 2.4 m) to the side in cross section and 8 or 10 ft (2.4 or 3.0 m) in height. Usually rectangular, timber cross – sectional dimensions vary from 8 x 8 in. (203 x 203 mm) to 12 x 12 in. (300 x 305 mm).

In standing square sets, it is essential to wedge, brace, or block each timber piece in place to preserve the integrity and rigidity of the entire structure (Lucas

and Haycocks. 1973). Sets are stood so that the axes of the main members are oriented parallel to the three principal stresses in the earth's field (Section 1.7). Much movement or distortion cannot be tolerated, and alignment must be maintained with care (Square set miners are known as skilled carpenters.) In addition to the bare sets, lagging is laid for flooring and fill fences (covered with burlap) are erected. The square sets also form orepasses and man ways, which must be advanced as the stope progresses. To move ore by gravity within the stope, inclined flooring may be laid, sloping to the orepasses. In the very worst of ground conditions – loose or running ground - *forepoling* or *spiling* (lagging driven ahead on the top or sides of the sets) may be necessary to protect the miners from falling rock. Drainage problems are encountered during filling (if hydraulic), and percolation drains and decantation towers may be required as in cut and fill stoping.

As additional support or as a means of leaving waste or low-grade ore behind unmined, pillars m ay readily be utilized in square set stopes.

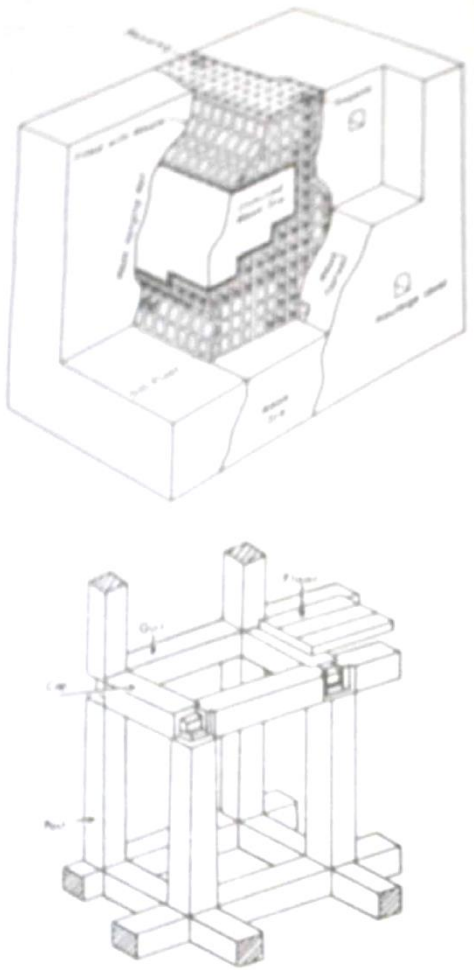


Figure 11.4. Square set stoping in an irregular or massive deposit. (Too) Prismatic arrangement of timber in stope, partially back field. (After Paele, 1941, Copyright 1941, John Wiley & Sons New York)

Conditions

The following is based on Lucas and Haycocks *(1973)* and Morrison *Russell (1973):* and
1. *Ore strength:* weak Jo very weak
2. *Rock strength:* weak lo very weak
3. Deposit shape: any, *regular lo irregular*
4. Deposit dip: any, but *preferably* >45-50° (can accommodate flatter *deposit if orepasses are steeper than angle of repose)*
5. *Deposit size:* any, *generally small*
6. Ore grade: high
7. Ore uniformity: variable (can sort waste in slope or leave pillar)
8. Depth: deep (<8500 ft. or <2.6 *km).*

Features

The following is based on Lewis and Clark (1964), Morrison and Russell *(1973), Gardner* and Vandenburg (1982), and Hamrin (1982):

Advantages

1. Flexible, versatile, adaptive to wide variety o f conditions
2. Suitable *for* worst ground conditions when caving and subsidence not permitted
3. Selective for irregular deposit and variable ore occurrence; waste can be sorted and used as fill or *left* as pillars
4. Excellent recovery (<100%*)* and very low dilution (>0)
5. *Requires little* mechanization; low capital cost
6. Low development cost

Disadvantages

1. Very low productivity (range 1-3 tons, or 0.9-2.7 tonnes, per employee – shift)
2. Very low production rate; small-scale method
3. Highest mining cost, along with quarrying (relative cost: 100%)
4. Most labor intensive; slow; requires highly trained labor
5. Very high limber requirement, cost
6. *Fire* hazard, especially in sulfide ore body, and very poor safety

Applications

No U.S. mines are now known to be using square set stoping exclusively. However, several applications of square set stoping are reported along with other methods, in which square set is used for pillars, remnants, and ore too difficult to mine by any other exploitation method (Thomas, 1978). A few recent or remaining applications include gold (Homestake, SD – see Sisselman, 1976) and lead or lead-

zinc (Bunker Hill, ID – see Songstad, 1982; Burgin, UT — see Rausch et al., 1982; Mt. Isa Austr.).

Case study. Homestake gold mine, Homestake Mining Co., Lead, SD.

General: deep mine, largest U.S. underground gold mine: mining began 1876

Geology: folded ledges in Precambrian Homestake formation of metamorphic, plunging 30-85° to southeast; ore finely dispersed gold quartz-chlorite-cummingtonite schist; ore grade 0.26 oz/ton (8,1 g/tone); reserves 15.4 million tons (14.0 million tones) at 0.21 oz/ton (6.6 g/tonne)

Production: 1.8 million tons/yr (1.6 million tone/yr)

Mining methods: cut and fill stoping, sublevel stoping, and square set stoping.

11.5 SPECIAL TOPIC A: SELECTION AND COST ESTIMATE OF DRILLING EQUIPMENT

The selection and cost estimation of a blasthole drill parallel procedures developed for materials-handling equipment in Sections 7,6, 8.7, and 10.2. We shall use them as general guides, because there are applicable to the selection of equipment for any unit operation integrated into the production cycle of a mine. In the present discussion, we deal with underground drilling equipment to demonstrate the broad application of the methodology.

Drill Selection

While there are a number of performance specifications to be identified in the selection of a production drill, the ones we shall consider of first importance are the following:

1. Type of drill, type of bit, size of bit, power source
2. Drill ability (rate of penetration) in rock specified
3. Blasting factor (area of working face broken per drillhole)
4. Drilling factor (length of drillhole required per unit weight of rock broken)
5. Number of drill units per rig or jumbo (Fig. 9-9c)

Items comprising factor I are selected with the aid of Chapter 4, especially Tables 4.1 and 4.2. Drill ability, the second factor, is based usually on empirical data: Table 11.1 is an example of such data for small – hole percussion drills of the type used underground. (The norm is Barre granite, and rates for other types used of rock are given in relation to the drill ability and abrasion index in Barre granite. The abrasion index is an indicator of bit wear, as drill ability is a measure of penetration rate.) Use of the blasting factor, item 3, is an approximate method of expressing the fragmenting ability of a drill hole and is based on field experience. Similarly, item 4, the drilling factor, expresses fragmenting ability by another measure. Finally, the number of drills required is calculated for the prevailing operating and job conditions as reflected by the four preceding items.

A numerical example demonstrates the selection procedure.

TABLE 11.1 Percussion Drillability and Abrasion Index of Various Rocks Compared to Barre Granite

Rock	Location	Drillability	Abrasion Index
Barre granite	Barre, VT	1.00	1.00
Granite	Dvorshak, ID	1.11	1.14
Granite	California	1.10	0.54
Granite	Newark, NJ	1.05	1.27
Granite	Mt. Blanc, France	0.92	0.86
Granite	Grand Coulee, WA	0.50	2.40
Granite	Bulgaria	0.45	2.29
Granite gneiss	Denver, CO	1.52	1.00
Granite gneiss	Vancouver, BC, Canada	0.89	1.03
Granite gneiss	Hamburg, NJ	0.67	1.46
Quartzite	Capetown, South Africa	1.22	2.70
Quartzite	Corter Dam, GA	1.00	1.40
Quartzite	New Zealand	0.78	1.70
Quartzite	Canada	0.72	3.17
Quartzite	Minnesota	0.56	8.60
Quartzite	Canada	0.33	1.45
Magnetite	Kiruna, Sweden	1.00	1.23
Magnetite	Kirkland Lake, Ont., Canada	0.59	1.41
Taconite	Kirkland Lake, Ont., Canada	0.84	4.13
Hematite (red)	Sarajevo, Yugoslavia	1.50	0.40
Hematite (dark)	Sarajevo, Yugoslavia	2.20	0.70
Siderite	Sarajevo, Yugoslavia	0.90	0.80
Siderite	Suffern, NY	0.89	0.55
Sandstone	Nova Scotia, Canada	2.70	0.14
Sandstone	Ohio	3.10	0.11
Sandstone	New Zealand	2.30	1.20
Shale	Michel, BC, Canada	0.75	2.80
Shale	Scranton, PA	2.00	0.00
Limestone	Davenport, IA	1.79	0.28
Limestone	Portsmouth, NH	1.77	0.65
Limestone	Saratoga, NY	1.22	0.01
Limestone	Tulsa, OK	1.19	0.10
Limestone	Bellefonte, PA	0.94	0.09
Limestone	Buffalo, NY	0.89	0.09

Source: Russell, 1982. (By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Example 11.1. Given the following information regarding a drilling application in a hard - rock mine, select the correct number of drill required for a mobile, trac - mounted drill rig with a capacity of 1 – 4 drill booms:

Mining method	Stope and pillar mining
Stope face	Height 20 ft (6.0 m) Width 24 ft (7.2 m)
Drill power available	pneumatic percussion
Drill bits	2Ł in. (57 mm), carbide, cross
Rock	Denver granite gneiss
Penetration rate, Barre granite	28 in./min (11.9 mm/sec)
Blasting factor (BF)	6.8 ft^2/hole (0.632 m^2/hole)
Depth of round or holes	16 ft (4.8 m)
Delay time in drilling / hole	2.40 min
Drill rounds / shift	3
Allowable drilling time / round	2.0 hr
Tonnage factor (TF)	14.0 ft^3/ton (0.437 m^3/tonne)

Also determine the tons (tonnes) of rock broken and the drilling factor (DF).

Solution. From table 11.1 for Denver granite gneiss, read drillability = 1.52. Therefore, find penetration rate based on rate in Barre granite:

Peneration rate = (1.52)(28) = 42.6 in./min

Required number of holes = $\frac{area\ of\ face}{BF} = \frac{(20)(24)}{6.8} = 70.6$, say 71 holes

Net drilling time = $\frac{depth}{rate} = \frac{(16\ ft)(12\ in./ft)}{42.6\ in./min} = 4.51\ min/hole$

Gross drilling time = net time + delays = 4.51 + 2.40 = 6.91 min/hole

Capacity/drill = $\frac{(2.0\ hr)(60\ min/hr)}{6.91\ min/hole} = 4{,}18\ say\ 17\ holes$

Required number o f drills = $\frac{71\ holes/face}{17\ holes/drill} = 4{,}18\ say\ 4\ drills$

(Go to the next larger whole number if the decimal remainder >0 3)
Select one mobile rig mounting 4 d rills.

Drilling factor is determined by the total length of hole drilled and weight of rock broken:

Total hole *length* = (71 holes)x(16 ft/hole) = 1136 ft/round x 3 rounds/8 hr = 426 ft/hr (avg.)

Volume of rock broken = (20)(24)(16) = 7680 ft^3/round

Weight of rock broken =

$$\frac{volume}{TF} = \frac{7680}{14.0} = 549 tons/round\ x\ 3\ rounds = 1647\ tons/shift\ (1494\ tonnes/shift) +$$

$$8\ hr = 206\ tons/hr(avg).$$

$$DF = \frac{hole\ length}{weight\ of\ rock} = 1136/549 = 2.07 ft/ton(0.68\ m/tonne)$$

In the final selection of rock breakage equipment for the cycle of operation, in a mine, the blasting round is designed along with the drills chosen. In the, way values specified during drill selection for hole size, depth of round blasting factor, and drilling factor can be confirmed.

Drilling Cost Estimate

(For a review of the cost estimation procedure we employed with materials andling equipment in surface mining. see Section 8.7. The same procedure is utilized here for drilling cost estimation. For detailed references on drill selection and cost estimating, see Unger, 1973; Church, 1981; Hemphill, 1981: Russell, 1982; Skodack, 1982).

Recalling that our objective is the separate evaluation of ownership and operating costs in arriving at overall costs, extensive information on drill operation and performance is required if an accurate estimate of hourly and unit costs is to be obtained. Again utilizing the form reproduced in Table 8.4, we first note any differences in the estimating procedure used previously for materials-handling equipment.

A. *Ownership costs*

I. *Depreciation*

a. *Purchase price:* basic cost is for drill rig or jumbo mounting one drill boom, cost of additional drills and booms extra
b. *Freight:* basic weight is for drill rig only; weight of each drill boom is additional
c. *Operating period:* assume 4000 hr/yr unless specified
d. *Economic life:* assume 24,000 hr unless specified **2**. *Interest, taxes, insurance, and storage:* no change

B. *Operating costs*

1. *Consumables cost* (replaces tire cost)

a. *Bits:* base on bit cost, bit life, and hole length drilled per hour

b *Sleet (drill road):* base on steel cost, steel life, and hole length drilled per hour.

2. *Tire and tire repair coin.* disregard if crawler mounted

3. *Repairs, maintenance:* no change

4. *Fuel or power* calculate separately for drilling and tramming (drilling based on equipment electric power consumption, training based on diesel fuel consumption)

5. *Lubrication:* include for hydraulic drill, disregard for pneumatic

6. *Auxiliary fuel:* N/A

7. *Labor:* estimate one or two operators

c. Overall ownership and operating costs, no change

d. *Unit coin:* calculate on unit production ($/ton. or $/tonne) and unit hole length ($/ft. or $/m) b a s e s.

EXAMPLE- 11.2. For the four-boom percussion drill rig selected in Example 11.1, estimate the overall hourly and unit costs. Assume the following conditions:

Capital costs	
Drill rig. I boom	$88.000
Additional boom	$21.000
Rig weight	20.000 lb (9070 kg)
Weight, each boom	2000 lb (907 kg)
Operating costs	
Drilling	5C/kW-hr
Electric power cost	120 kW
Power consumption	
Tramming	
Diesel fuel cost	$1.10/gal (S0.29/L)
Motor rating	100 hp (75 kW)
Fuel consumption	0.75 gal/hp-hr (3.81 L/kW -hr)
Tram time/shift	30 min
Maintenance	75%
Consumables	
Drill bit, cost	$60
Bit life	900 ft (274 m)
Drill steel, couplings, etc., cost	$580
Steel life	6400 ft (1950 m)
Labor wage rate	$15.25/hr

SOLUTION. See Table 11.2. Follow the procedure outlined above. Calculations requiring explanation are summarized below (coded to Tables 8.4 and 11.2).

TABLE 11.2 Cost Estimation Form

4-Boom Drill Mining Equipment Unit

A. Ownership Costs

1. DEPRECIATION
   a. Purchase price = $151,000
   b. −Salvage value (15%) = − 22,650
   c. Freight 28,000 lb @ $4.00/cwt = 1,120
   d. Unloading and moving cost = 110
   e. Delivered price = 129,580, say $129,600
   f. Operating period 4000 hr/yr
   g. Economic life 24,000 hr = 6.0 yr (n)
   h. Depreciation = $129,600 (del. price ~~less tire cost~~) / 24,000 hr = $ 5.40 /hr
2. INTEREST, TAXES, INSURANCE AND STORAGE
   a. Rate = interest 14% + taxes 2% + other 2% = 18%
   b. Average annual investment rate = $\frac{n+1}{2n} = \frac{7}{12}$ = 58.3 %
   c. Average annual investment = $129,600 × 58.3% = $75,557
   d. Annual fixed charge = $75,557 × 18% = $13,600
   e. Fixed charge = $13,600 / 4000 hr/yr = $ 3.40 /hr

TOTAL OWNERSHIP COSTS = $ 8.80 /hr

B. Operating Costs

1. ~~TIRE REPLACEMENT COST~~, CONSUMABLES
   Purchase price 1 set of tires = $_____
   Tire life _____ hr
   Tire cost = $_____ / _____ hr = $ 67.01 /hr
2. TIRE REPAIR COST: ___% × $_____ tire cost = — /hr
3. REPAIRS, MAINTENANCE: 75% × $ 5.40 deprec. = 4.05 /hr
4. FUEL: _____ gal/hr @ $ _____/gal
   OR POWER: _____ kW @ $_____/kW-hr = 9.66 /hr
5. LUBRICATION: _____ gal/hr @ $_____/gal = — /hr
6. AUXILIARY FUEL: = — /hr
7. LABOR: 2 operator @ $ 15.25 /hr = $ 30.50 /hr
   _____ oiler @ _____/hr = _____/hr
   _____ helper @ _____/hr = _____/hr
   Total $ 30.50 /hr
   + 35% benefits $ 10.68 /hr = 41.18 /hr

TOTAL OPERATING COSTS = $121.90 /hr

TOTAL OWNERSHIP AND OPERATING COSTS = $130.70 /hr

UNIT COST $ 130.70 /hr ÷ 206 ton/hr = $ 0.63 /ton
÷ 426 ft/hr = $ 0.31 /ft

A. *Ownership costs*

1a. *Purchase price* = $88,000 + (4 - l)($21.000) - $151,000

b. *Salvage* = 0.15 x $151,000 = S22.650

Weight = 20.000 + (4)(2000) = 28.000 lb

Freight = 28.000 lb x $\frac{\$4.00}{100lb} = \$1120$

d *Unloading and moving cost* = 0.10 x $1120 = $112, say $110

c . *Delivered price* = a - b + c + d = $129.580, say $129,600

f. Use standard operating period = 4000 hr/yr

g. Use standard economic life = 24,000 hr ÷ 4000 hr/yr = 6 yr

2. All calculations self-explanatory,

B. *Operating costs*

1. *Tire cost:* omit (trac-mounted)

Consumables cost:

a. Bits = $\frac{\$60}{900ft} \; x \; 1136\frac{ft}{round} = \$\,75.73/round$

b. Steel = $\frac{\$580}{6400ft} \; x \; 1136\frac{ft}{round} = \; \$102.95/round$

c. Total = $\$178.68/round$ x 3 rounds/ 8 hr = $ 67.01/hr

4. Fuel cost

a. Tramming = 0.75 gal/hp – hr x 100 hp x $\frac{30min}{60/hr} = 37.5 \; gal \, x \; \$1.10/gal - \$41.25 + 8hr = \$5.16/hr$

b. Drilling

$= 120kW \; x \; 0.05/kW - hr =$

$\$6.00/round - hr \; x2hr \; x \; 3 \; rounds \; \div 8 \; hr = \frac{\$4.50/hr}{\$9.66/hr}$

7. *Labor:* Assume 2 operators

C. *Overall: ownership and operating costs* = A + B = $130.70/hr

D. *Unit costs:* hourly production = 206 tons/hr = 426 ft/hr

1. *Cost / ton (tonne)* = $\frac{\$130.70}{206} = \$0.63/ton(\$0.69/tonne)$

2. *Cost/ft (m)* = $\frac{\$130.70}{426} = \$0.31/ton(\$1.02/tonne)$

11.6 S P E C IA L T O PIC B: UNDERGROUND BLASTING

We build now on the theory of explosives and explosive detonation developed in Section 4.3. In Section 6.6, the theory was used in the design of surface blasts; we utilize it now for the design of underground blasting rounds, especially in development openings in which a single free face exists (production blasts underground very much resemble surface blasting in benches with multiple free faces; see Sections 6.6 and 9.4).

Theory of Rock Failure in Blasting

In Section 4.3, we learned when explosives detonate that two effects are primarily responsible for breaking rock: the impact of the shock wave and the expansion of the gas bubble. They are effective in varying proportions, depending upon and rock. In general, powerful explosives (e.g., TNT, nitroglycerin, dynamite) that release more energy fragment by shock wave, and weaker explosives (e.g., ANFO, permissibles) fragment by gas expansion. Correspondingly, shock waves are more effective in hard, brittle rock (e.g., shale, salt, coal).

Shock waves are the initial effect to occur in blasting. As soon as detonation is initiated in the explosive charge, a shock front forms, propagating beyond the charge into the adjoining rock as a stress wave (review Fig. 4.6). The shock front has two components: longitudinal and transverse. The *longitudinal wave* travels at higher velocity with amplitude parallel to its motion, while the *transverse wave* is slower and weaker with amplitude normal to its motion.

The sequence of events, starting with wave action, that occurs in blasting has been described, among others, by Livingston (1956). Hark (1968), and Hemphill (1981). With detonation (Fig.11.6), the longitudinal stress wave propagates into the rock, traveling to the free face(s) as a compression wave, and being reflected back in tension (because of the sharp discontinuity between rock and air. only a small amount of energy escapes into the air as wave energy, vibration, and noise). Since rock is comparatively strong in compression, the initial pulse of the longitudinal wave is only effective in breaking rock close to the charge. Here compressive rock failure is evident by crushing immediately adjacent to the charge and by radial cracks beyond. Major rock fragmentation, however, occurs when the tension wave reflects back from the free face(s); because rock is weaker in tension, it fails in thick tensile slabs as the wave passes, with damage ceasing as the stress level declines. The only effect of the transverse wave, a far weaker wave, is possibly to assist in the formation of release fractures or radial cracks by shear action.

The gas-expansion effect sequentially follows the wave action. It propagates at a slower rate, and while stress waves produce mainly a shattering action, the gas bubble causes a heaving action. The expanding effect of the high-pressure gas radiating through blasting fractures and preexisting joints in the rock is to displace and move the already-broken rock fragments, thus creating further breakage. It

may be concluded that the gas-expansion effect in fragmentation is chiefly one of displacement, reflected in throw and the formation of a crater.

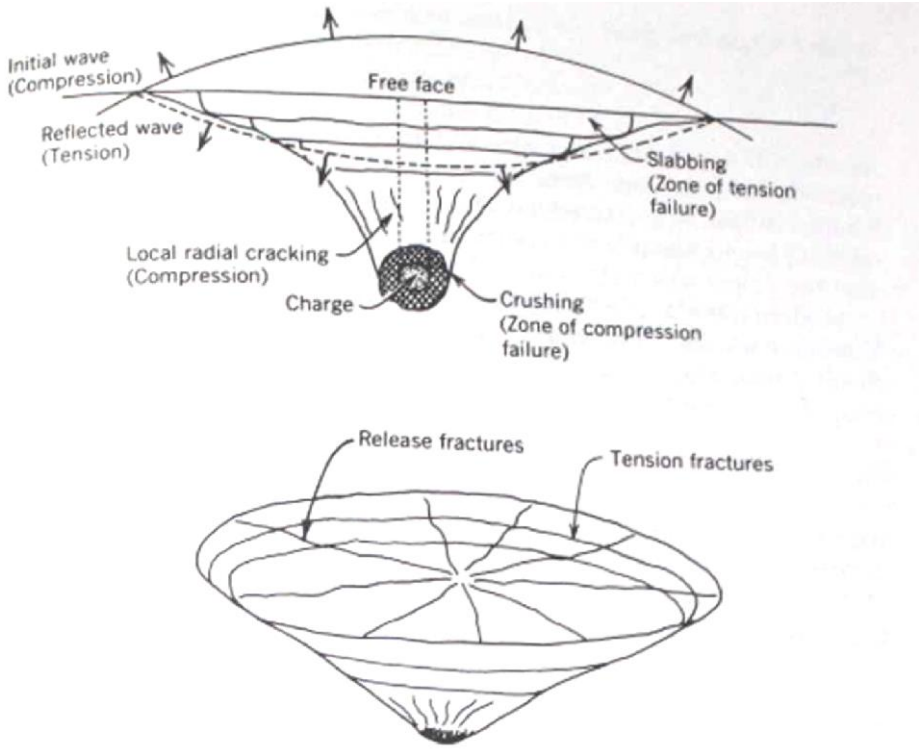


Figure 11.6. Sequence of events in crater formation by rock blasting. Shown are the wave action, steps in rock failure, and the resulting crater.

Figure 11.6 portrays the sequence of events in rock blasting, the nature of rock failure, and the steps in crater formation. Most blasting authorities agree today that, in the majority of cases, it is the combined effects of the stress wave and gas bubble rather than the predominant action of either alone that account for rock fragmentation in the detonation of explosives.

The effect of multiple holes in close proximity and detonated simultaneously is to reinforce the blasting damage produced by a single charge.

Crater Geometry

The dimensional relations of the explosive charge and the resulting crater produced in blasting are unique, affording us the means to analyze or design a blast (Livingston, 1956; Langefors and Kihlstrom, 1978; Gustafson, 1981). The so-called cubical law of blasting derives from eighteenth-century military engineering and applies to a single spherical charge detonated in rock with a single free face:

$$W \propto d^3$$

in which W is weight of explosive and disburden , the depth of charge or the perpendicular distance from the free face to the center of gravity of the charge. While now discredited, the law assumed that the volume of rock broken (proportional to d^2) varied directly with the weight of explosive, and that the crater was a 45° cone with radius and height equal to d.

Modern theories of blasting recognize that (1) the simple cubical – law relationship does not apply, and (2) the shape of the crater departs from a cone. Several relationship have been proposed, probably the most satisfactory being the one by Langefors and Kihistrom (1978).

$$W = ad^2 + bd^3 + cd^4 + \ldots. \qquad (11.2)$$

where a,b, and c are constants. Since a long series is awkward to solve, and decreasing benefits accrue with the addition of each successive term, Eq. 11.2 is often simplified to a empirical, exponential formula, resembling Eq. 11.1 (Livingston. 1956):

$$W = d^2 \qquad (11.3)$$

where *n* is an exponent which in practice varies from 1.5 for elastic rock to 3.4 for plastic.

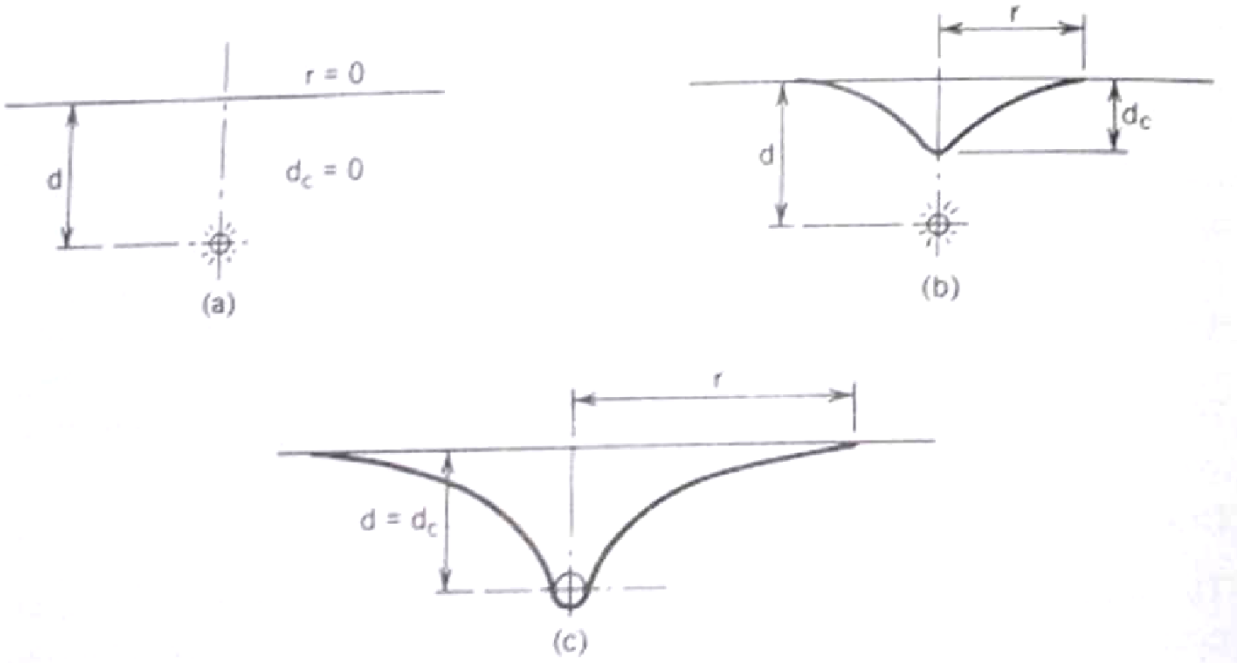


Figure 11.7. Crater geometry relations for a given explosive detonated in a given rock, (a) No crater, (b) Normal crater, (c) Full crater. The charge weight increases for a constant burden between (a), (b), and (c). In (a), $d_e = 0$; in (b). $d_c < d$: and in (c). $d_c = d$.

To determine the value of *n* experimentally for a given explosive and a given rock, a series of blasts is detonated for a fixed burden *d*, increasing the weight of explosive *W* until a full crater is formed (Fig. 11.7). The depth of the crater d_c then becomes equal to the burden d. if the charge weight is insufficient to break any crater (Fig. 11.7a), then $d_c = 0$, and the only effects is to spring the hole around the charge. Results of tests for full craters only with different explosives in the same rock are shown Figure 11.8. Data were obtained for 45 % semigelatin dynamite and ANFO in Idaho Springs, CO. gneiss Equation 11.3 becomes

$$W = d^{1.85}$$

and $r/d = 2.8$ for a full crater in blasting to a single free face.

Development Blasting Rounds

Compared with surface bench blasting or underground slope blasting underground development blasting, difficult, slow, inefficient, and expensive The reasons, of course, that only a single free face exists. Therefore, the objective of each development round is to open up an additional free face as the first holes or *cut*, are detonated. Successive holes, detonated by the appropriate delays, can then break to the cut, improving the effectiveness and efficiency of the blast. Also, it is desirable to break as deep around as possible in development blasting to achieve thc maximum advance thus further reducing costs.

Major factors in the design o f a development blasting round underground are the following:

1. Type of explosive
2. Properties and uniformity o f rock
3. Drillhole diameter
4. Hole pattern, number, and depth
5. Amount of charge per drillhole
6. Order of firing and wiring diagram

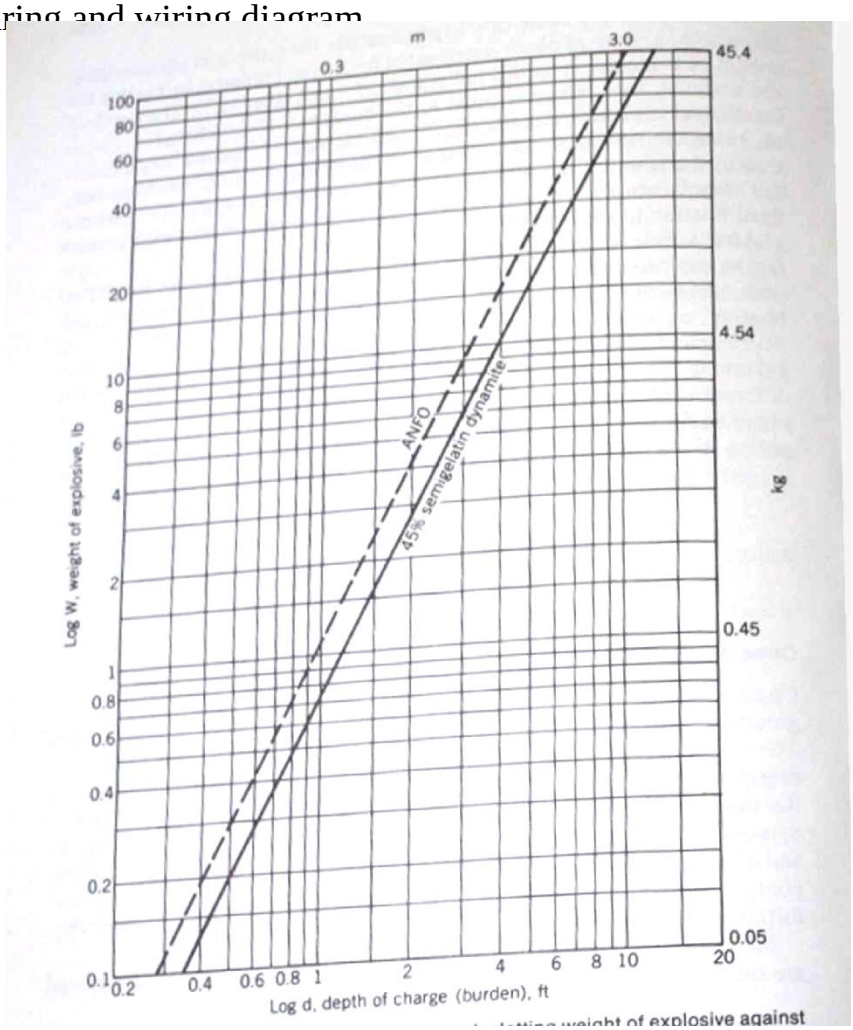


Figure 11.8. Crater data for design of blasting round, plotting weight of explosive against burden. Conditions

Burden for single full crater *(d = dc)*
$W \infty d^{as}$ *r/dc* = 2.8
Single free face
Rock: Idaho Springs, CO gneiss
Explosives: ANFO and 45% semi gelatin dynamite (Modified after Livingston, 1956.)

Hole pattern, or the *blasting round*, is the most important variable to resolve. Rounds are classified by the type of cut employed; they are the following (Obert, 1973c; Anon., 1977; Gregory, 1979);

1. Machine - cut kerf
2. Angled - cut round
3. Burn - cut rounds
4. No - cut rounds

Kerfs are used only in soft to moderately hard rock (coal, salt, potash, etc.) to provide an additional free face for blasting (Section 4.2). A cutting machine of the universal or shortwall type is employed to undercut, top cut, center cut, or shear the face. The use of a kerf substantially reduces the amount of explosive required and lowers the risk of explosion in a flammable atmosphere, such as in a coal mine. Unfortunately, it is not feasible to employ a machine – cut kerf in hard rock.

The other blasting rounds involve only drillholes, unassisted by cutting. Angled – cut rounds were the first developed and still the widest applied. The principal ones used are the V cut, pyramid, and draw, in varying arrangements and combinations of holes (Fig.11.9). Angled cuts are effective in hard, tough rock but cause flyrock and consume large amounts of explosives. The advance with an angled cut is limited to the least dimension of the face, and much skill and care are required to place drillholes properly.

Burn – cut rounds have become popular in recent years because they are easier to locate properly, consume less explosives, and have no limitation on depth of advance. The principle of burn cuts is that certain (usually alternate) holes are left uncharged to provide free faces – thus drilling rather than blasting is utilized to create free faces. A wide variety of hole patterns and even hole sizes is employed in burn cuts (large center holes are the most satisfactory in providing space for shattered rock to move) (Fig. 11.10). Burn cuts are applicable to hard, brittle, homogeneous rock.

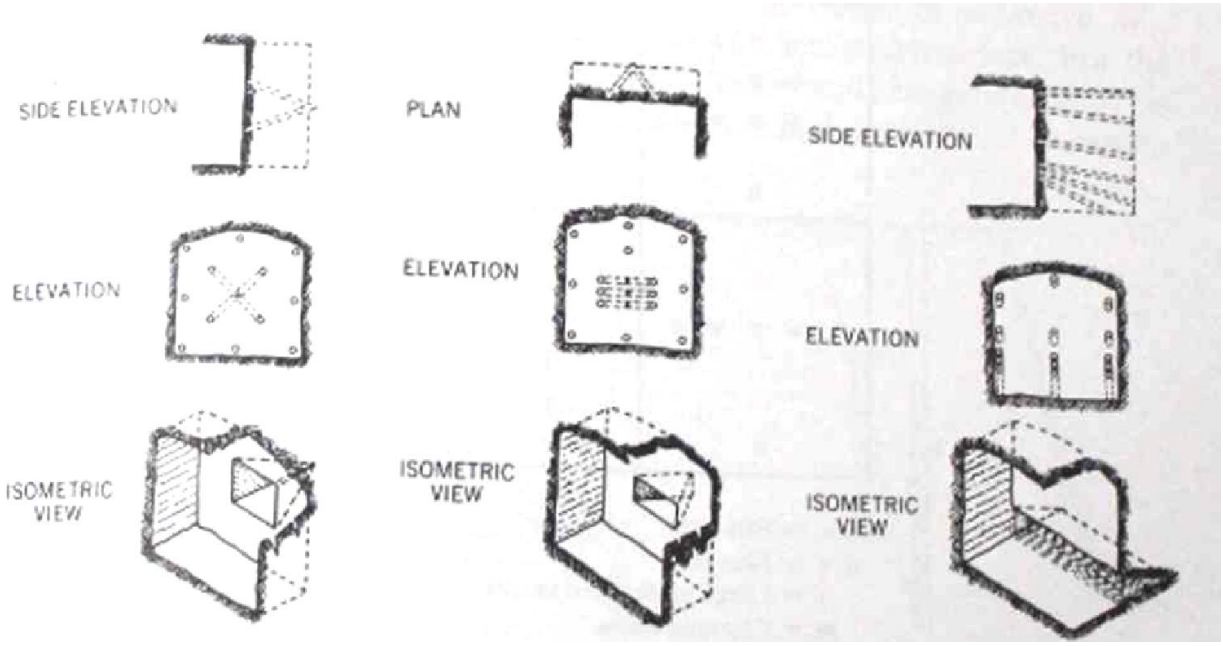


Figure 11.9. Angled-cut development rounds. *(Left)* Pyramid. (*Center*) Triple V. *(Right)* Draw. (After Gregory. 1979. B y permission from Trans Tech Publications. Clausthal-Zellerfeld. West Germany.)

No – cut rounds are of experimental interest. Their principle is that holes drilled in a deepening and expanding helical spiral can produce an initial cut and successively larger and deeper cuts by the proper, careful placement of drillholes and sequence of firing (Livingston, 1956).

As *with* surface blasting, the efficiency of blasting in underground development *is measured by several* parameters, the two most common being *drilling factor and powder* factor. Values for current practice are the following (compare with those for surface mining in Section 6.6):

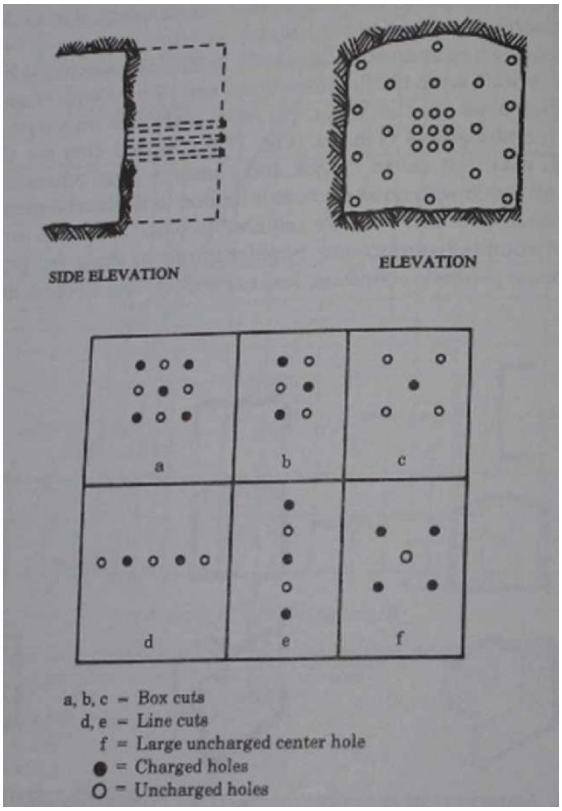


Figure 11.10. Burn-cut development rounds. (After Gregory. 1979. B y permission from Trans Tech Publications, Clausthal-Zellerfeld, West Germ any.)

	drilling factor, ft/ton (m/tonne)	POWDER FACTOR. LB/TON (KG/TON)
Underground development, metallic and nonmetallic ores	3-4 (1.0-1.3)	2-4.5 (1-2)
Underground production. metallic and nonmetallic ores	0.5-2 (0.2-0.7)	0.2-0.4 (0.1-0.2)
Underground mining. coal	1-2 (0.3-0.7)	0.8 (0.4)

Design of Blasting Rounds

Information developed in the preceding sections on the theory of blasting and crater geometry can now be employed in the design of underground development rounds. In this section, we shall deal with the design of only one type of blasting round, the angled cut. Crater data presented in Figure 11.8 will be the basis for our design procedure.

Figure M.U *(left)* illustrates the steps in crater formation and burden relationships for a V-cut round m a development heading (only a portion of the drillholes and the detonation sequence in the face are shown). The first holes detonated are the V cut (1), next the relievers (2), and then the enlargers (3). In designing each sequence of holes, the burden *d* is measured from the nearest free face to the center of gravity of the charge. The amount of explosive required is then read from Figure 11.7. Thus, for the cut holes, the burden is *du* measured to the heading face; the weight of explosive. W, , corresponds to *d*. When the cut is blasted, the first crater forms. For the relievers, burden d_2 is measured to the first crater, and charge weight W, is specified accordingly. Subsequent holes arc handled similarly. A comparison bench blasting in a production round is provided by Figure 11.11 (right).

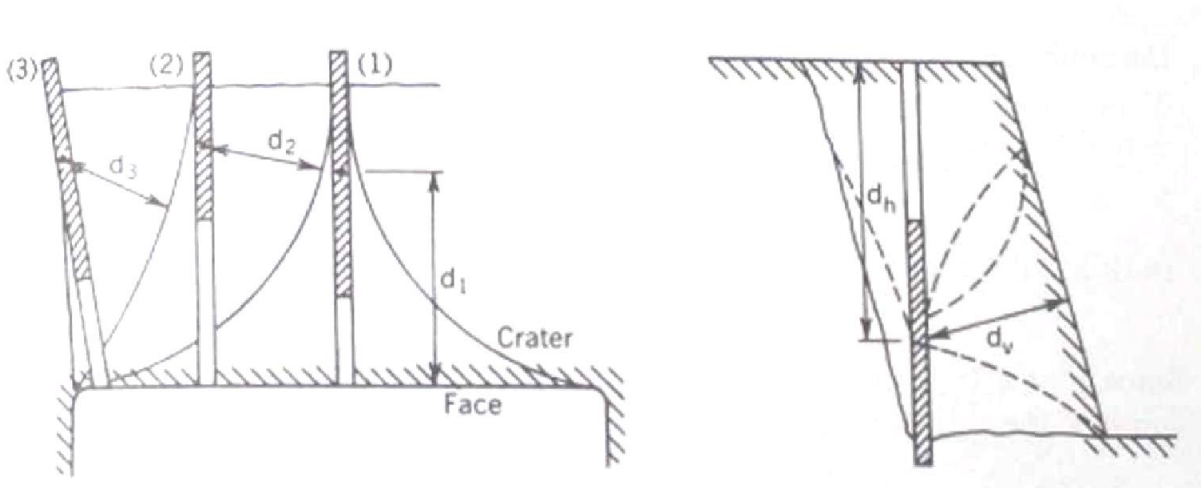


Figure 11.11. Relationship of burden to free face and center of gravity of charge in blasting rounds. *(Left)* Underground development. (1) Cut hole. (2) Reliever hole. (3) Enlarger hole *(Right)* Surface or underground production.

While the resulting design is approximate and must be adjusted for variations in actual practice, it can be used successfully to obtain a trial blasting

round. Since the configuration is only approximate, a further simplification will be made in the numerical example and problems by locating the center of gravity of all holes at the depth of round or anticipated advance.

Example 11.3. (a) Design the eight – hole V cut of a blasting round intended to pull 6 ft (1.8 m) in an 8 x 10 ft (2.4 x 3.0 m) drift in Idaho Springs gneiss (TF = 12.1 ft^3/ton, or 0.38 m^3/tonne). The explosive is ANFO of specific gravity 0.8, fired electrically with millisecond delays. Drillholes diameter is 3 in.(76 mm).

(b) The following data apply to the entire blasting round:

Total explosives consumption	150 lb (68 kg)
Total length of blasthole drilled	165 ft (50 m)

Calculate the powder factor and drilling factor for the round:

SOLUTION. (a) From Figure 11.8, the ANFO curve, read w_1 = 34.5 lb for d_1 = 6 ft. Find loading density:

Explosive specific weight w_1 =(62.4)(0.8) = 49.9 lb/ft^3

Hole are $A = \frac{\pi}{4}(\frac{3}{12})^2 = 0.0491 ft^2$

For unit length *l*,

Loading density = (0.049) (49.9) = 2.45 lb/ft (3.65 kg/m)

Determine explosives, charge per hole in cut:

Charge/ hole, W_1 =34.5/8 = 4.31 lb

Length of charge/ hole, L= 4.31/2.45 = 1.76 ft (0.53 m)

Since *L* = 1.76 ft. and the center of gravity of the charge is located at the depth of the round *d* = 6 ft. the length of each cut drillhole *L h* must be

$$L_k = 6 + \frac{1.76}{2} = 6.88\, ft\ (2.10\, m)$$

(b) Calculate powder factor:

Weight of rock broken, W = $\frac{(6)(8)(10)}{12.1} = 39.7\ tons/round$

Powder factor PF = $\frac{150}{39.7} = 3.78\ lb/ton (1.89\ kg/tonne)$

Calculate drilling factor:

Drilling factor Df = $\frac{165}{39.7} = 4.16\ ft/ton (1.40\ m/tonne)$

In practice, drillholes are generally overdrilled (about 6 in., or 152 mm) to ensure that the full depth of rounds is pulled. This requires that the center of gravity of the charge be shifted closer to the collar of the hole. Also, when cartridges rather than bulk explosives are used, loading densities are calculated to the nearest one – half cartridge.

After the cut is designed, the burden on succeeding holes is read from either a plan or elevation view of the round, selected so that a true distance, without foreshortening, is read. Further, it is advisable, to check the desired radius of crater against the design *r/d,* ratio to ensure that the crater formed by the cut holes breaks to the full dimensions of the face.

PROBLEMS

11.1 The cycle in driving a raise conventionally by drilling and blasting must be completed in a 7-hr shift. Allowing 0.8 hr for scaling and bolting 1,6 hr for blasting 0.2 hr for chute pulling, and 20% of shift time for delays calculate the rating of two stoper percussion drills (in in./min. or mm/sec' penetration rate) required with two operators to achieve a 6-ft (1.8-m) advance per shift. The raise is 6 x 8 ft (1.8 x 2.4 m) in cross section, and 30 6 $\frac{1}{2}$-ft (2.0-m) holes are required. Setup time for drilling is 20 min, collaring time is $\frac{1}{2}$ mm per hole, and steel change time is 1 min per hole.

11.23. A percussion drill jumbo is operating in a drift in a hardrock mine under the following conditions:

Rock	Minnesota quartzite
Drilling rate (in Barre granite)	52 in./min (22 mm/sec)
Position, collar, retract time	3.2 min/hole
Drillhole depth	9 ft (2.7 m)
Shift drilling time	7.2 hr

What is the output of one drill per shift, measured in total feet (meters) of hole?

b. If three rounds are required per shift, and each round consists of 68 holes, how many drills must be mounted on the jumbo?

c. If the working face is 18 x 26 ft (5.5 x 7.9 m) and the tonnage factor is 11 ft^3/ton (0.34 m^3/tonne), calculate the tons broken per shift and the drilling factor.

11.3 A percussion drill rig demonstrates the following performance in a mine:

Rock	Bellefonte, PA limestone
Drilling rate (in Barre granite)	28 in./min (11.9 mm/sec)
Position, collar, retect time	2.52 min/hole

Drillhole depth	12 ft (3.6 m)
Number of holes/round	58
Drilling time/round	2 hr
Number of rounds / shift	3
Shift time	8 hr

Pertinent cost data are as follows:

Delivered price	$140 000
Operating period	6000 hr/yr
Life	33 000 hr
Operating cost	$ 32 000 hr

Determine the following:
a. Number of drills required for the conditions
b. Ownership cost. $/hr
c. Unit drilling cost. $/ft (S/m)

11.4 Given the following information, calculate and compare the unit costs ($/hr or S/ft or m of drillhole, and per ton or tonne of rock broken) of drilling with pneumatic and hydraulic equipment in underground development work, employing multiple headings in hard-rock mining. Select the more economical drill rig for this application. Assume one rig with 1 to 4 drills each.

	PNEUMATIC	HYDRAULIC
Design and performance data		
Drill rounds/shift (8-hr)	3	3
Drill face	20 x 16 ft	20 X 16 ft
	(6.1 x 4.9 m)	(6.1 X 4.9 m)
Drillhole depth	12 ft (3.7 m)	12 ft (3.7 m)
lid diameter Drilling lime/round	2 m. (51 mm) 2 hr	hydraulic 2 in, (5| mm)
Dialing factor	6 4 П1	2 hr 6 4 ft
Hitch dritlability (based on	<0.59 m:)/hole	(0.54 nr t-hole
Barre granite)	0.8	0.8 66 in./min

Drilling rule (in Barre granite)	36 in,/min	
positioning, collaring, retract	(15 mm/sec)	<28 mm/min)
time	2.0 min/hole 175	2.0 min/hote 175 lb/ft’
Rock specific weight	lb/ft*	
Capital costs	<2X00 kg/m')	(2800 kg/m1)
Drilling rig. 1 boom	$84,000	4125 non
each additional boom	Si 8.(НИ) 20.000	
Rig weight	lb	524.000
Weight, each boom	(9070 kg) 2000 lb	(9070 kg) 2000 lb
Operating period	(907 kg)	(907 kg)
	4000 hr/yr	4000 hr/yr
Life	20.000 hr	20.000 hr
Operating costs		
Electric power (drilling)	6tf/kW-hr	6tf/kW-hr
Power consumed	110 kW	90 kW
Diesel fuel (tramming)	% I -05/gal	SI.05/gal
Tram time/shift	(50.26/L)	(S0.26/L)
	30 min	30 min
Motor rating	100 hp	100 hp
	(75 kW)	(75 kW)
Fuel consumed	0.75 gal/hp-hr	0.75 gal/hp-hr
Hydraulic fuel @ $1.50/gal	(3.81 L/kW-hr)	(3.81 L/kW-hi
(S0.40/L)	0	1.5 gal/round (5.7 L/round)
Maintenance	60%	75%
Consumables		
Drill bit. cost	$80	$55
Bit life	500 ft	500 ft
	(152 m)	(152 m)
Drill steel, couplings, hose,		
etc.. cost	$840	$560
Steel life	4000 ft	4000 ft
	(1220 m)	(1220 m)
Labor wage rate (use 1 operator/		
1-4 booms)	$16.00/hr	516.00/hr

11.5 Given the following information, calculate and compare the unit costs ($/hr or $/ft or m of drillhole, and per ton or tonne of rock broken) of drilling with pneumatic and hydraulic equipment in underground development work employing multiple headings in hard-rock mining. Select the more economical drill rig for this application. Assume one rig w.th 1 - 4 drills each.

	PNEUMATIC	HYDRAULIC
Design and performance data	3	**1**
Drill rounds/shift (8 hr)	18 x 24 ft	18 x 24 ft
Drill face	(5.5 x 7.3 m)	(5.5 x 7.3 m)
	14 ft (4.3 m)	14 ft (4.3 m)
Drillhole depth	**2**i in.	л l ■
Bit diameter	(64 mm)	24 in.
	2.5 hr	(64 mm)
Drilling time/round	7.3 ft^3	2.5 hr
Blasting factor	(0.68 m^2)/hole	7.3 ft^2
		(0.68 m^2)/hole
Rock	New Jersey granite gneiss	
Drilling rate (in Barre granite)	32 in./min	56 in./min
	(14 mm/sec)	(24 mm/sec)
Positioning, collaring, retract time	2.25 min/hole	**2.0** min/hole
Rock tonnage factor	13.5 ft/ton	13.5 ft^J/ton
	(0.42 m/tonne)	(0.42 m/ tonne)
Capita! costs		
Drill rig, I boom	592,000	$135,000
each additional boom	$22,000	$28,000
Rig weight	**20,000** lb	**22.000** lb
	(9070 kg)	(9070 kg)
Weight, each boom	**2000** lb	**2000** lb
	(907 kg)	(907 kg)
Operating period	4000 hr/yr	4000 hr/yr
Life		
Operating costs		
Electric power (drilling)	40/kW-hr	4^/kW-hr
Power consumed	**110** kW	90 kW
Diesel fuel (tramming)	$1.10/gal	$1.10/gat
	(S0.29/L)	(S0.29/L)
Tram time/shift	**20** min	**20** min
Motor rating	100 hp (75 kW)	100 hp (75 kW)
Fuel consumed	0.75 gal/hp-hr	0.75 gal/hp-hr
	(3.8! L/kW-hr)	(3.81 L/kW-hr)
Hydraulic fuel @ $1.SO/gal	pneumatic	hydraulic
(S0.40/L)	0	1-5 gal/round
Maintenance	75%	(5.7 L/round)
		№%
Consumables		
Drill bit, cost	S90	560

Bit life	450 ft	450 ft
	(137 m)	(137 m)
Drill steel, couplings, hose. etc., cost	$860	$540
Steel life	3600 ft	3600 ft
Labor wage rate (use 1 operator/	(1097 tn)	(1097 m)
1-4 booms)	516,40/hr	$l6.40/hr

11.6. Design the cut holes only of a blasting round for a 4 x 6 ft (1.2 mm) crosscut employing an angled cut of your choice (arrangement and number of holes) and based on the charge-burden chart provided for Idaho Springs gneiss and dynamite or ANFO (Fig. *И*.8). Pull the *maximum* advance possible (what is it?). Name the cut you have selected, and sketch it and the crater in two views. Loading density in the holes is 1.5 lb/ft (2.23 kg/m) for either explosive. Calculate the weight of explosive and length of drillhole required.

11.7. A V-cut blasting round is to be employed in driving a 5 x 7 ft (1.5 x 2.1 m) raise, (a) What is the maximum depth of round that can be pulled? (b) Using two holes in the V, design the cut portion of the round *only* to permit an advance of 4,5ft (1.4 m), Make use of the charge-burden chart for Idaho Springs gneiss and dynamite, assuming a loading density of 1.4 lb/ft (2.08 kg/m) of drillhole. Sketch the holes and craters in two views, and calculate the footage of hole, charge, and weight of explosive.

11.8a. A 7 x 10 ft (2.1 x 3.0 m) drift is being driven in Idaho Springs gneiss using 45% semigelatin dynamite. Using a V-cut blasting round, determine the *minimum* number of holes in the cut only required to produce a *maximum* advance per round. Specify the advance, hole depth, and number of holes. Holes arc overdrilled 0.5 ft (0.15 m) and sealed with 0.5 ft (0.15 m) of stemming. Use the blast design (W vs. *d)* chart, assuming that the crater breaks to within 0.5 ft (0.15 m) of the end of the hole rather than to the center of gravity of the charge. (In other words, this is a theoretical design, modified for practical application.) Take loading density as 0.9 lb powder/ft drillhole (1.35 kg/m). Sketch the cut design in two views.

b. Design the round to the same specifications using ANFO.

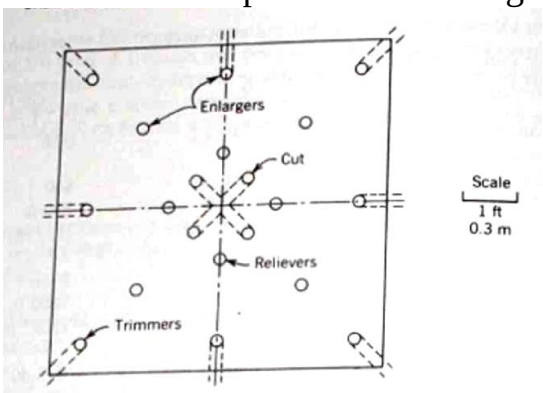


Figure 11.12. Pyramid-cut blasting round. See Problem 11.9.

11.9. Using an angled cut of the four-hole, pyramid type, design a complete blasting round for development work in a 6 x 6 ft (1.8 x 1.8 m) drift, employing millisecond delay firing (Fig. 11.12):

Rock	Hard gneiss (Idaho Springs formation): tonnage factor12 ftVton (0.375 mVtonne)
Explosive	Semigelatin dynamite. 45% strength; density 122 sticks/box (50 lb. or 22.7 kg); standard li **x 8** in. (32 **x** 203 mm) can ridge
Depth of round	5 ft (1.5 m)
Hole diameter	$1\frac{3}{8}$ in. (35 mm)

Determine the following.

a. Sketch of blasting round (to scale), showing placement of drillholes, order of firing, and shape of craters formed
b. Explosives requirement for each hole, calculated to nearest halfstick
c. Total number of holes required
d. Total length of drillhole
e. Drilling factor, in ft/ton (m/tonne)
f. Total explosives consumption, in lb (kg) g. Powder factor, in lb/ton (kg/tonne)

11.10 Using an angled cut of the six-hole V type, design a complete blasting round for development work in a 6.5 x 6.5 ft (2.0 x 2.0 m) drift employing millisecond delay firing (Fig. 11.13):

11.11

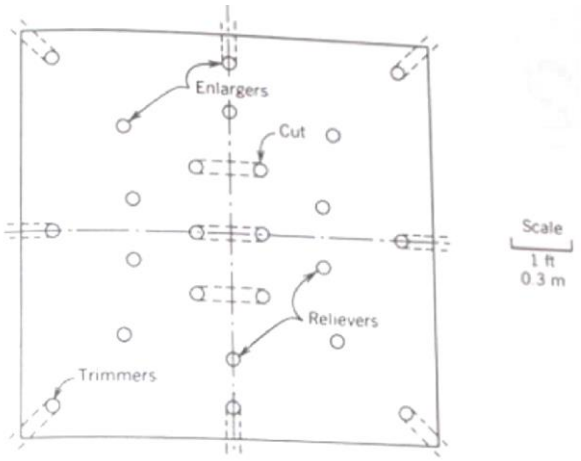


Figure 11.13. V-cut blasting round. See Problem 11.10.

Rock — Hard gneiss (Idaho Springs formation); tonnage factor 12 ft^J/ton (0.375 m^3/tonne)

Explosive — Semigelatin dynamite, 45% strength; density U2 sticks/box (50 lb. or 22.7 kg); standard Ц x К i_n. (32 x 203 mm) cartridge

Depth of round — **6** ft (1.8 m)

Hole diameter — li in. (38 mm)

Determine Ihe following:

- **a.** Sketch of blasting round (to scale), showing placement of drillholes, order of firing, and shape of craters formed
- **b.** Explosives requirement for each hole, calculated to nearest halfstick
- **c.** Total number of holes required
- **d.** Total length of drillhole
- **e.** Drilling factor, in ft/ton (m/tonne)
- **f.** Total explosives consumption, in lb (kg)
- **g.** Powder factor, in lb/ton (kg/tonne)

12
UNDERGROUND MINING: CAVING METHODS

12.1 CLASSIFICATION OF METHODS

We have concentrated our efforts so far on classes of mining methods **that** require exploitation workings to be held open, essentially intact, for **the** duration of mining. If the ore and rock are sufficiently competent, unsupported methods are adequate; if ore and rock arc incompetent to moderately competent, then supported methods must he used. We now encounter a class of methods in which the exploitation workings arc designed to collapse: (hat is, caving of the ore or rock or both is intentional and the very essence of the method.

We define ***caving methods*** as those associated with induced, controlled, massive caving of the ore body, the overlying rock, or both, concurrent with and essential to the conduct of mining. There are three major caving methods (Table 3.1):

1. Longwall mining
2. Sublevel caving
3. Block caving

Long wall mining is used in horizontal, tabular deposits, mainly coal; the others have application in inclined or vertical, massive deposits, almost exclusively metallic or nonmetallic. The caving class accounts for about 15% of U.S. mineral production, a sharply increasing amount. In cost, this class includes a moderately priced method as well as the two cheapest of all underground methods.

Because the exploitation openings are deliberately destroyed in the progress of mining, the caving class is truly unique. Rock mechanics principles are applied to ensure that caving, in fact, does occur – rather than to prevent the occurrence of caving. In effect, the cross-sectional shape of the undercut area (i.e.. the width-to-height, or R/h ratio) is sufficiently elongated to cause failure of the roof or back (see Section 10,7). Further, development openings have to be designed and located withstand shifting and caving ground, as well as subsidence that usually extends to the surface. Production must be maintained at a steady, continues level to avoid

disruptions or hangups in the caving action. Good mine engineering and supervision are indispensable to a successful caving operation.

12.2 LONGWALL MINING

Longwall mining is an exploitation method used in fairly flat-lying thin tabular deposits in which a long face is established across a panel between sets of entries and retreated or advanced by narrow cuts, aided by the complete caving of the roof or hanging wall (Fig. 12.1). While the length of face, or wall, is measured in hundreds of feet (hundreds of meters), the width of the working place is narrow, measured in feet (meters). The longwall is kept open by a system of heavy-duty, powered, yielding supports that form a cantilever or umbrella of protection over (he face. As a cut or slice is taken along the length of the wall, the supports retract, advance, and reengage, allowing the roof to cave behind. The caved area is called the *gob*.

A very old method, longwall mining originated in the coal mines of Europe in the seventeenth century and is used almost exclusively in most coat- producing countries outside the United Stales (Lucas and Haycocks. 1973). Strangely enough, it has enjoyed widespread success in the United States only since the 1960s, when self-advancing, hydraulic support systems were perfected (Trent and Harrison, 1982). Other innovations that have led to its rapid adoption here arc development of mobile, flexible, armored conveyors. high-speed continuous mining machines, and roof control and caving practices grounded in sound rock mechanics principles.

In 25 years, longwall mining has grown significantly in popularity in U.S. coal mining and now accounts for over 15% of underground production. There are currently over 100 longwalls in operation (Merritt, 1984). Longwall mining has been much less widely used in metal and nonmetal mining and overall is responsible for about I \% of all U.S. mineral production. It is considered a *large-scale* method of exploitation and is one of the t*o cheapest underground methods.

Mistakenly classified by some as a supported method of mining because of the elaborate roof support system employed, longwall mining more properly belongs to the caving class of methods. The reason has to do with the role of caving in the mining process, which is to aid in the breakage of the in-

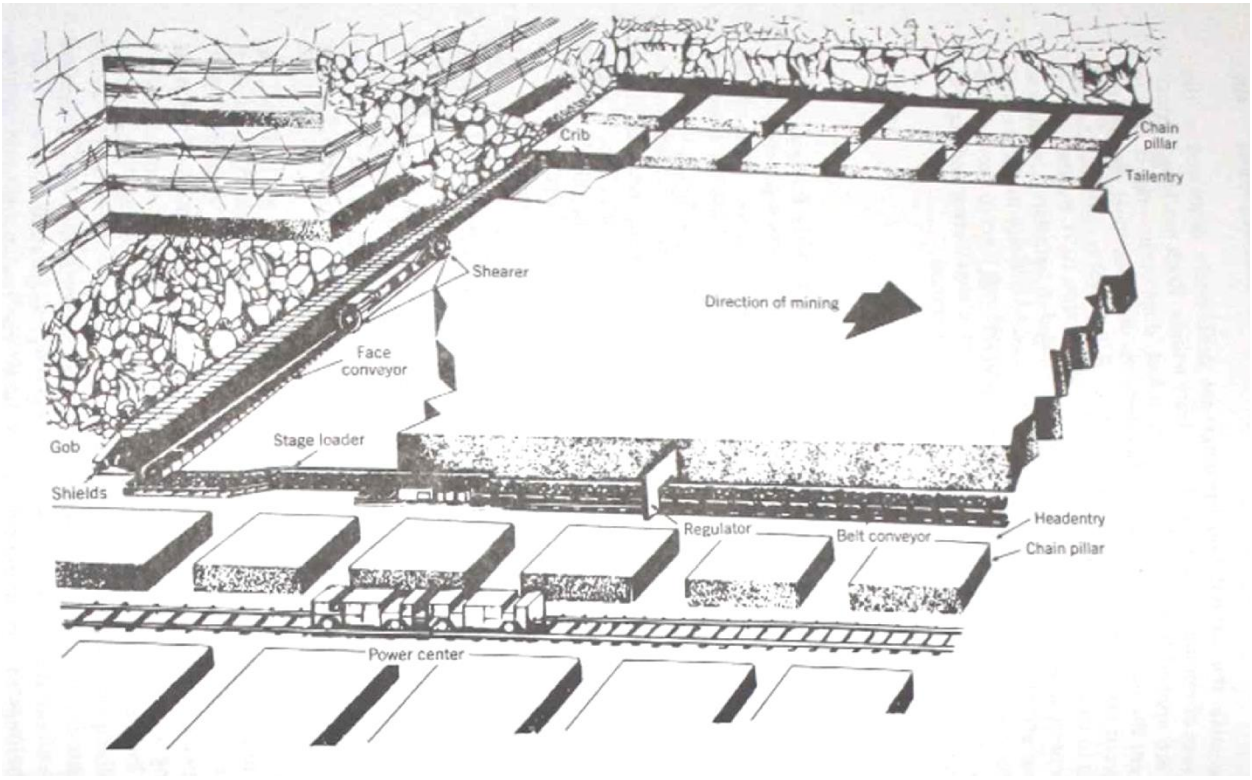


Figure 12.1. Longwall mining of coat. (After Peng and Chlang. 1984. By permission from Eickhoff Corp , Pitlsburgn.)

situ mineral while permitting the immediate roof to cave safely, venting excessive superincumbent loads from damaging the supports (Trent and Harrison, 1982). Ideally, the immediate roof should be thin bedded and moderately weak so it will break and collapse; stronger, thick – bedded formation do not preclude the use of a caving method, so long as they occur higher in the geologic column. The floor or footwall should be competent to provide a firm foundation for the roof supports.

Subsidence is an environmental hazard of longwall mining. Depending on the depth of cover and area mined, caving and collapse of the overlying strata will eventually extend from the mining horizon to the surface. Although some damage to the surface must be expected, it can be controlled by maintaining a uniform rate of advance of the longwall and the caving, if the surface subsides evenly, then destruction of structures mi water courses need not occur (Peng and Chiang, 1984). At one time resist ant support systems comprised of pack walls, cribs, and backfill were utilized in an attempt to prevent or minimize subsidence, but they are rarely utilized today, having given way to modern yielding supports which control caving and subsidence but do not prevent it.

Rock bursts are another potential hazard in longwall mining, especially at great depths (>2500 ft, or >750 m). Bursts are accompanied by the very rapid release of large amounts of strain energy and may have violent, often explosive-like repercussions (Trent and Harrison, 1982). Damage to personnel and property is frequent. In coal mines, rock bursts (coal bumps) arc often triggered by the explosive venting of strata gas (methane); in metal mines, tough brittle rock such as quartzite may fracture and fail violently, bike earthquakes, rock bursts are extremely difficult to predict. They can, however, be prevented or relieved by good

mining practices (e.g.. elimination of sharp comers in workings, of remnant pillars of inadequate size, of erratic rates of extraction).

Because it is difficult theoretically or experimentally to determine the suitability of ore, rock, or coal for caving, an empirical scale of *cavability* is often employed (Peng and Chiang. 1984). Its use is not limited to longwall mining but applies to the entire caving class. More regarding cavability will be presented under block caving, the ultimate of caving methods. If a strong, massive coal-minc roof is judged unsuited for longwall mining, artificial breakage means (e.g., explosives, water jet, mechanical roof breaker) may render it cavable (Tandanand, 1980).

Design parameters to be considered in longwall mining center on direction of mining, dimensions of openings, and types of equipment (Thomas, 1978; Stefanko and Bise, 1983). Mining may be carried out on the advance or retreat; in the United States, federal regulations currently favor retreat longwall mining. Figure 12.2 depicts a longwall lace being mined on the retreat, that is, toward the main entries and primary access. Notice that multiple openings (three, or two with a variance from MSHA) are required as in room anti pillar mining, if the deposit is coal (Anon., 1984a).

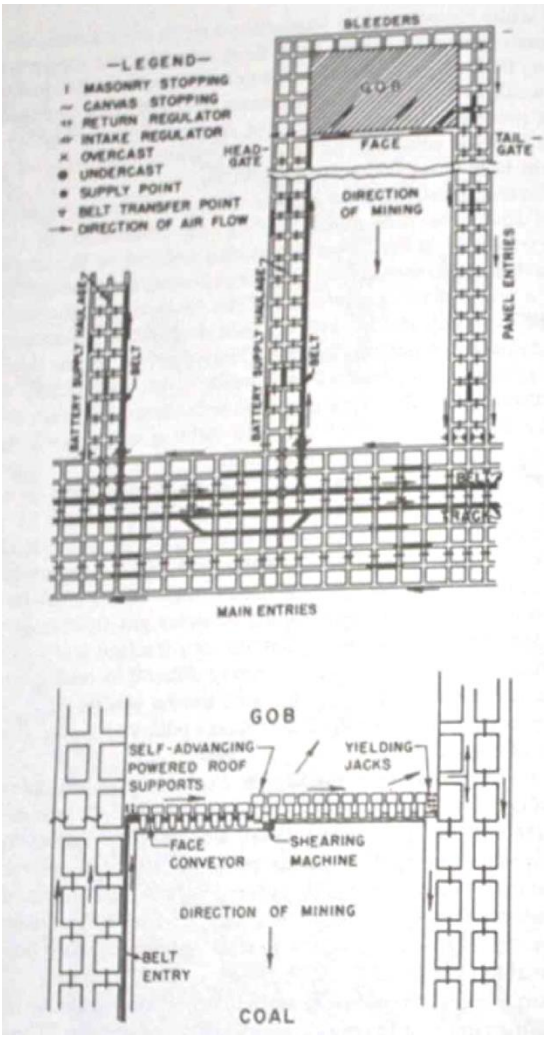


Figure 12.2. Retreat longwall mining of coal; plan views. *[Lett)* Development and exploitation stages, showing main entries, panel entries, bleeders, and longwall face. Haulage and ventilation systems Indicated *(Right)* Enlargement of face area showing equipment. (After Schroder. 1973 By permission from Society of Mining Engineers. Inc.. Littleton. CO.)

The relative dimension of the workings (face length, face width, depth of cut, and panel length) are shown approximately to scale, although the face length is foreshortened in the detailed diagram. The kinds of equipment required and their location are also shown.

Typical longwall dimensions are as follows (Deems, 1984):

Panel length	3000–9000 ft (900–2700 m)
Face length (panel width)	500–1000 ft (150–300 m)
Face width	15–25 ft (2.4–3.6 m)
Face height (seam height)	3–15 ft (0.9–4.5 m)
Depth of cut (plow, shearer)	3-30 in. (76–762 mm)

The average face length of U.S. longwall s has increased from 465 ft (142 m) in 1976 to 563 ft (172 m) in 1983. Average lengths m West Germany currently are 769 ft (234 m) and in the United Kingdom 662 ft (202 m)

Equipment choices involve mainly two unit operations: breakage and roof support (face haulage is also important, but armored, flexible, chain-and- flight conveyors are usually the choice) (Trent and Harrison. 1982; Stefanko and Bise, 1983). l or breakage, two continuous types of mining machines are employed, plows and shearers. A *plow* takes a shallow cut and is limited to thin or moderately thick scams, but is low in capital and maintenance costs a *shearer* mines a deep cut and is applicable to moderate to high scams, but is high in both costs (sec Fig. 9.9f). Because it is more productive, however. a shearer is usually checaper in overall cost. In this country, shearers arc in the preponderance (probably 90%). For support, cither chocks, shields, or combination units arc used (all are self-advancing and also serve to advance the face conveyor) (Fig, 9,9i). They are similar in design, *chotks* having four joints and vertical legs and *shields* three joints, angled legs, and a protective canopy on the caving side. The trend in the United States is strongly to shields (usage is now about 80%). Detailed views of a longwall face are shown in Figure 12.3.

Because of its uniquely continuous nature (except for infrequent moves to a new panel), longwall mining is unlike any other mining method and yields phenomena! production rates and productivities. Using the hypothetical cases of Stefanko and Bise (1983) for coal mining by the four common methods and variations (see Section 10.2), favorable production rates and productivities in a 63-in. (l.6-m) seam are estimated to be the following:

	ROOM AND PILLAR	SHORTWALL	LONGWALL	
	CONVENTIONAL	CONTINUOUS	CONVENTIONAL	CONTINUOUS
Production, tons (tonnes)/ shift	504 (457)	446 (405)	586 (532)	1208 (10%)
Face crew	10	6	6	7

Sequence of Development

Longwall development is strikingly similar to development in room and pillar mining By referring to Figure 9.6. we can see how a coal mine is ideally laid out for either method of mining. Main entries are driven across the property, from which orthogonal panel entries divide the coal into large blocks, minablc by either thc room and pillar or longwall method. Because entry driving is slower and more costly than exploitation and must be completed before a retreat method can be used, continuous miners rather than conventional loading equipment are increasingly employed for development (sec Section 10.2 and Fig. 10.4). One difference in longwall mining is that bleeder entnes at the end of the panels are required and not optional (Fig.12.2)

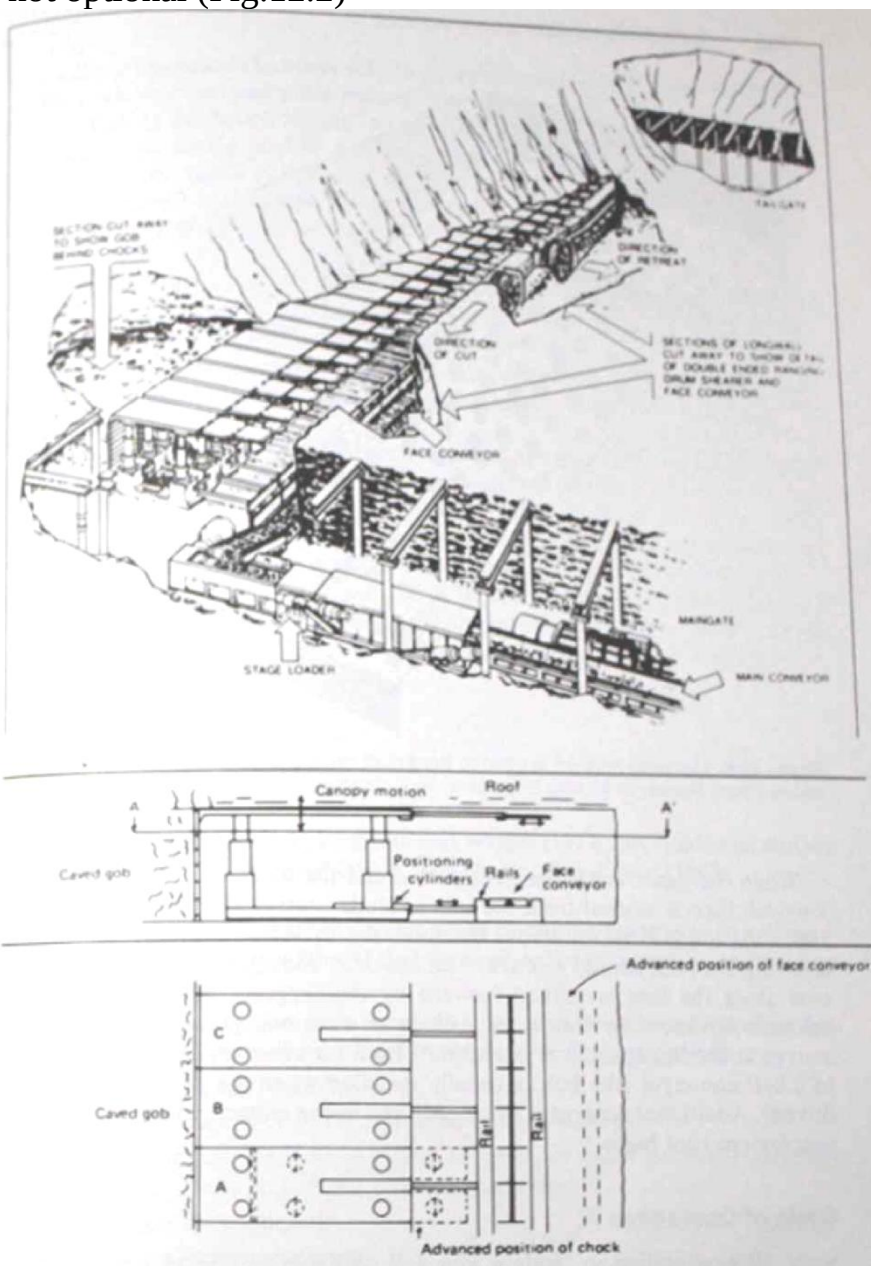


Figure 12.3. Detailed views of a longwall face and its equipment, (fop) Cutaway through face and headgate. (*Center*) Cross section through chocks. (*Bottom*) Plan view of face showing chocks and conveyor. (After Stout. 1980 By permission from McGraw-Hill, Inc.. New York)

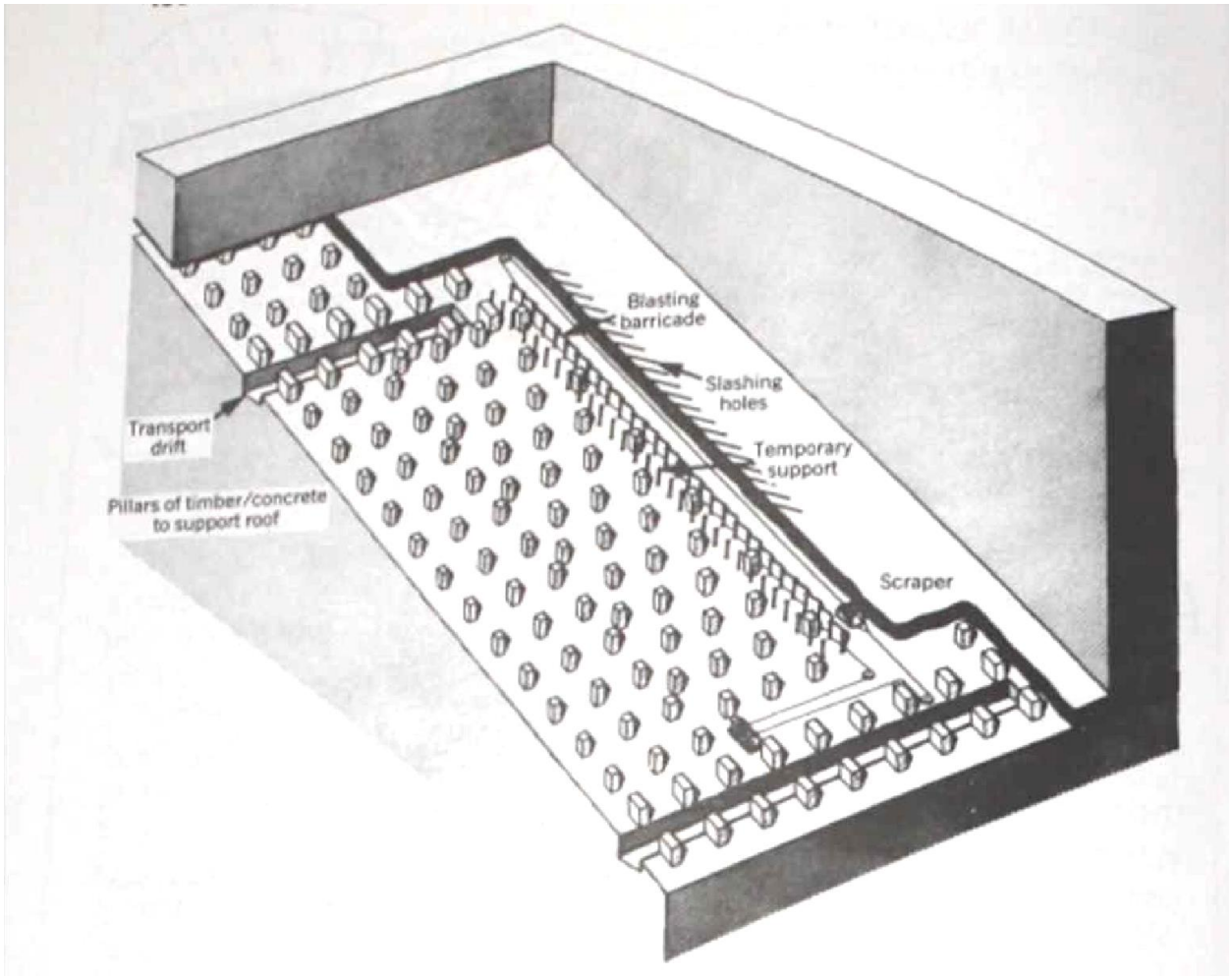


Figure 12.4. Longwall method applied to hard-rock mining. (After Hamrin. 1982. By permission from Society of Mining Engineers, Inc.. Littleton. CO.)

When the panel entries are completed and thc bleeders established, the longwall face is started from the inner bleeder entry (Trent and Harrison, 1982). As a cut is taken across the face, the hydraulic supports advance, allowing the roof behind to cave. The armored conveyor used to transport coal along the face is snaked forward by the supports, while the mining machine advances by means of a chain or rack and pinion. As the coal arrives at the headgate, it is transferred from the chain-and-flight conveyor to a belt conveyor (the belt is usually installed when the panel entries are driven). Additional support is often required in the entries, where cribs may supplement roof bolts.

Cycle of Operations

Since all exploitation in modern longwall coal mining utilizes continuous type mining equipment, the production cycle of operations is a modified one, consisting only of mining + haulage, with ground control integrated as an auxiliary operation (Sections 4.1, 10.2). In this regard, it closely resembles the cycle in room and pilar mining. the development cycle, of course, is identical. In hard – rock mining, a conventional cycle of operations must be utilized, consisting of drill + blast + load

+ haul, together with ground control; it is typical of that used in stope and pillar mining (Section 10.3.).

In longwall coal mining, the production cycle consist of the following operations and equipment (most are illustrated in Fig. 9.9.):

Mining (breaking and loading): shearer (single drum, double drum), plow, water jet (experimental) (Lucas and Haycocks, 1973; Kuti, 1979; Trent and Harrison, 1982; Campbell 1984)
Haulage: armored chain – and – flight face conveyor, belt entry conveyor (Lucas and Adler, 1973; Trent and Harrison, 1982)

Auxiliary operations for longwall arc headed by ground control, in which hydraulically actuated, self-advancing supports (shields, chocks) provide a continuous canopy of protection over the face (Stefanko and Bise, 1983, Peng and Chiang. 1984). Other operations of importance arc health and safety (gas control, dust control, ventilation), power supply, maintenance, and materiel supply, similar to those in room and pillar mining (Section 10.2). Of these, dust control and ventilation are most essential because of the excessive dust concentrations inherent in longwall mining. For other auxiliary operations, refer to Table 4.1.

Conditions

The following is based on Boshkov and Wright (1973), Morrison and Russell (1973), Trent and Harrison (1982), and Stefanko and Bise (1983):

1. ***Ore strength:*** any, but should crush rather than yield under roof pressure; preferably material that is weak and can be cut by continuous- type miner

2. *Rock* ***strength:*** weak to moderate, must break and cave; ideally, thin- bedded in intermediate roof; floor must be firm, nonplastic

3. ***Deposit shape:*** tabular

4. *Deposit dip:* low (<12°), preferably flat, uniform

5. ***Deposit size:*** large areal extent (>! mi^J, or >260 hectares); thin-bed- dcd (3-15 ft, or 1-5 m); uniform thickness

6. ***Ore grade:*** moderate

7. ***Ore uniformity:*** uniform

8. ***Depth:*** moderate (500-3000 ft, or 150-900 m. or deeper for coal) to very deep (<12.000 ft, or <3.5 km for noncoal)

Features

The following is based on Morrison and Russell (1973), Thomas (1978), Trent *and Harris on (1982), Stefanko and Bise (1983),* and Peng and Chiang (1984);

Advantages

1. Highest underground productivity: outstanding continuity of operations and low labor intensity resulting in high output (U.S. average face productivity 107 Ions, or 97 tonnes, per employee-shift. Raw coal)
2. Fairly low mining cost* least of the underground methods (relative ’ cost 20%, same as block caving)
3. High production rate; large-scale method
4. Approaches continuity of production, permitting nearly simultaneous cycle of operations to be conducted
5. Suitable for total mechanization, remote control, and automation
6. Low labor requirement
7. Fairly high recovery (70-90%) and low dilution (10-20%); theoretically. nearly 100% extraction is possible if entry chain pillars are recovered
8. Concentrated operations, facilitating transport, supply, and ventilation
9. Applicable to deep scams under bad roof conditions
10. Very good health and safety, especially with regard to roof-fall accidents (although dust may pose a serious environmental problem)

Disadvantages

1. Caving and subsidence occur over wide areas, approaching 10-80% of mined height; controllable
2. Method very inflexible and rigid in layout and execution; no selectivity except in varying height of opening somewhat
3. Mining rate should be uniform to avoid roof-support and subsidence problems (interruptions in production caused by strikes or lengthy delays should be minimal)
4. High capital cost, totaling $18.000-525,000'ft ($59,000-$80.000/m) of face
5. Reliance on single production unit means costly delays, interruptions in production
6. High moving costs
7. Heating in gob may create temperature-humidity problem and spontaneous combustion

Applications

Use of the longwall method in U.S. bituminous coal but in hard-rock mining is growing steadily, but in hard – rock mining it seems likely to remain small. In coal applications in all parts of the country are becoming numerous (most now are in the Appalachian field); examples include Consolidation Coal No, 95 (Shinnston, WV – see Shaver, 1982), Jim Walter Resources No.3. (Adger, Al – see Brass, 1980), and Sunnyside No.1 (Price, UT-see Huntsman, 1982). In metallic and nonmetallic mining, applications include copper (White Pine, MI), iron (Europe), gold (Witwatersrand, So Africa), potash (UT), trona (Green River, WY – see Wilson and Rao, 1982), and uranium (Radon, UT).

Case Study. North River No. 1 bituminous coal mine, Pittsburg and Mid way Coal Co., subsidiary of Chevron Corp., Berry. AL (Brezovec, 1982; Matthews, 1986).

General: located at moderate depth (600 ft, or 180 m) in Warrior Basin in Appalachian coal field

Coal: steam grade (moisture 7 %, ash 13%, sulfur 2.3%, calorific value 12.300 Btu/lb or 28,600 kJ/kg); Pratt seam 52 in. (1.3 m) thick; roof friable, laminated sandstone and shale; reserve 300 million tons (270 million tonnes)
Production: 1 million tons/yr (0.9 million tonnes/yr)

Equipment: 50-in. (1.3-m) double-drum shearer, two-leg shields, 101-in (260-mm) chain-and-flight conveyor, 42-in. (I.l-m) panel belt conveyor. 10 continuous miners for development and exploitation

Variations

Shortwall Mining. The most important modification of the longwall method is shortwall mining, actually an evolutionary or transitional variation and. in some classifications, a method in its own right. We elect to subordinate it in our classification because of its similarity to other coal mining methods and its limited and declining use—there are now fewer than a dozen operating faces in the United States (Stefanko and Bise, 1983). Widely considered a compromise between room and pillar and longwall, shortwall mining actually originated from the room and pillar method, although it resembles a shortened version of longwall mining. It is transitional, because a company considering converting from room and pillar to longwall can try shortwall for only a modest equipment investment. Shortwall mining may also be favored under these conditions: shallow depth, massive roof, small reserve, and varying geologic conditions (Thomas, 1978; Stout, 1980).

In layout, shortwall mining is very similar to longwall mining (Fig. 12.5).

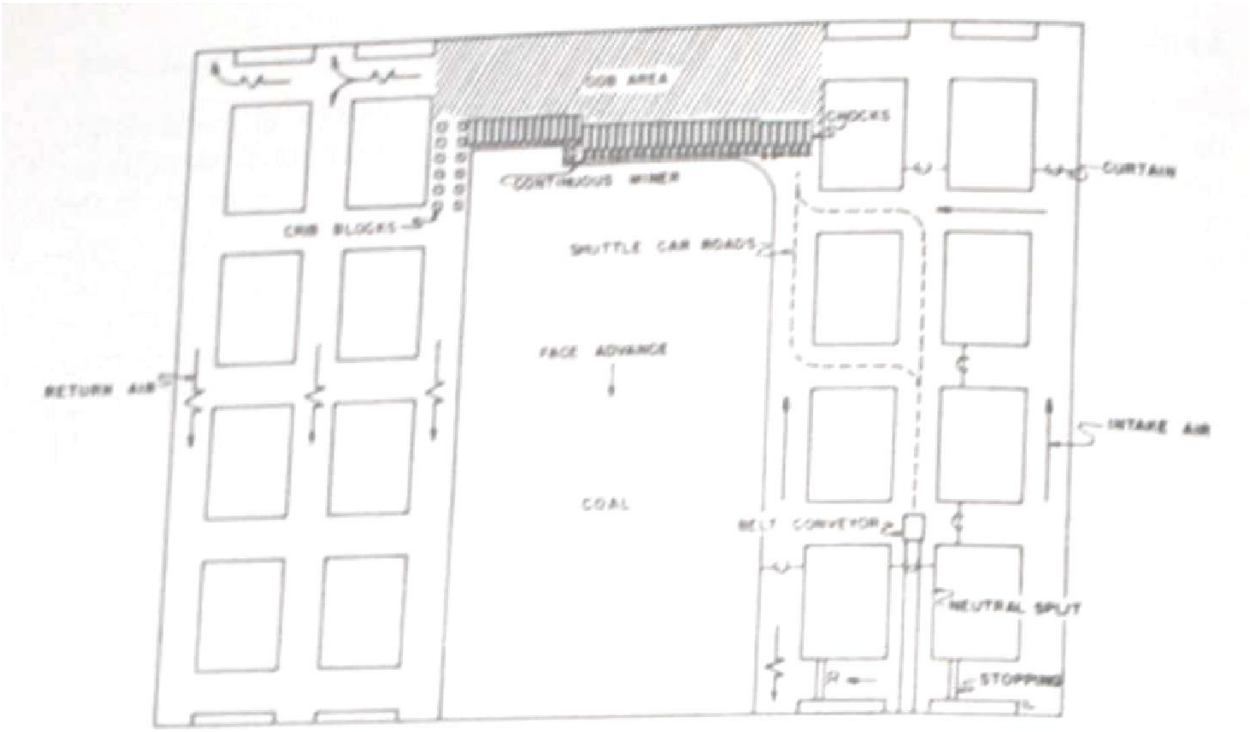


Figure 12.5. *Shortwall* mining of cost (After Schroder,1973. By permission from Society of Mining Engineers, In c . Littleton. CO)

The principal geometric difference is face length, which is only 150-200 ft (45-60 m) in short wail (Trent and Katen, 1982). The major equipment difference is the use of room-and-pillar-type continuous miners and shuttle cars, resulting in a width of cut of 10-12 ft (3-5 m). much greater than that in longwall. Roof support is of thc chock type, extensible and longer and sturdier than that used on longwalls. In application, shortwal! is limited to scam heights of 3.5-12 ft (l.l-3.6m), depths of 200-1700 ft (60-500mj, and panel lengths of 2000-4000 ft (600-1200 m).

Performance-wise. thc shortwall crew requirement is somewhat less than for longwall (6-7). but recovery, production rate, and productivity are also less— see the comparison earlier in this section. Capital investment is about S8000-S12.000/ft ($26,000- $39.000/m) of face. however, significantly lower than for longwall (Trent and Katen, 1982). Its relative cost is 40%, higher than that for related underground methods In other features, shortwall mining is also something of a compromise (it is considered a moderate-scale method). It displays greater flexibility, tolerates more geologic variation, and requires less moving time than longwall; on thc other hand, it is inferior to room and pillar in these regards. For U.S. case studies, sec the references by Palowitch and Zachar (1976) and Merlo (1982). The only country today in which shortwall mining is holding its own is Australia.

Other Variations. A variation of longwall mining that conceivably could have as great an application in the United States as it does in Europe and elsewhere *is pitch mining.* At least one such operation is in existence today, in the western U.S. coal fields where steeply dipping seams are commons (R e y n o ld s . 1983). At

Snowman Coal Co, in Colorado, a longwall is operated cross-pitch on a 30° dip using standard equipment; with some modifications, the longwall method should be practical here as elsewhere at dips <60°.

An important and revolutionary modification of longwall mining is remote *control and automation(* also see Chapter 14 on novel methods). Research has demonstrated that longwall is ideally suited for both features, which are now being used experimentally in the United States and elsewhere (Green,1985). Advancing both roof supports and face conveyor remotely and automatically is a prime application; automatic control of the mining machine is more difficult and requires the perfection of reliable, rugged sensors and (Tregelles and Barham, 1979; Broussard and Palowitch, 1979). Microprocessor technology affords not only greater safety but superior, faster control and savings in manpower.

12.3 SUBLEVEL CAVING

The two remaining caving methods are applicable to near-vertical deposits of metallic or nonmetallic minerals. In *sublevel caving,* overall mining progresses downward while the ore between sublevels is broken overhand the overlying waste rock (hanging wall or capping) caves into the void created as the ore is drawn off. Mining is conducted on sublevels from development drifts and crosscuts, connected to the main haulage level below by ramps, ore passes, and raises. Since only the waste is caved, the ore must be drilled and blasted in the customary way; generally, fan hole rounds arc utilized.

Figure I2.6 illustrates sublevel caving in a steeply dipping ore body. Because the hanging wall eventually caves to the surface, all main and secondary development is located in the footwall. In vertical crow section, the sublevel drifts and crosscuts arc staggered so that those on adjacent sublevels are not directly above one another (Fig. 12.7). Thus fan holes driven from one sublevel penetrate vertically to the second sublevel above. Development and exploitation operations are of necessity carefully planned so (hat adjacent sublevels are engaged in sequential unit operations, as shown in Figure 12.6.

In actuality, modem sublevel caving bears little resemblance to sublevel caving of yesteryear (Lewis and Clark, 1964). Formerly a small-scale, labor intensive method requiring heavy timbering, sublevel caving evolved through research and development (R&D) into a highly mechanized, *large- scale* method of mining using only nominal support. Much of the progress look place in Swedish iron-ore mines, whence the technology was exported around the world. In the United States, sublevel caving is not widely used, probably accounting for < 2% of all underground mineral production, mainly in metals. Where conditions arc appropriate (thick, steeply pitching seams), sublevel caving is also beginning to find use in coal mining, especially anthracite. In design and operating practices, sublevel caving is among the most advanced of all mining methods. Sound engineering is indispensable to its conduct. I he reason is related to the complexity of the caving action and the necessity of controlling it.

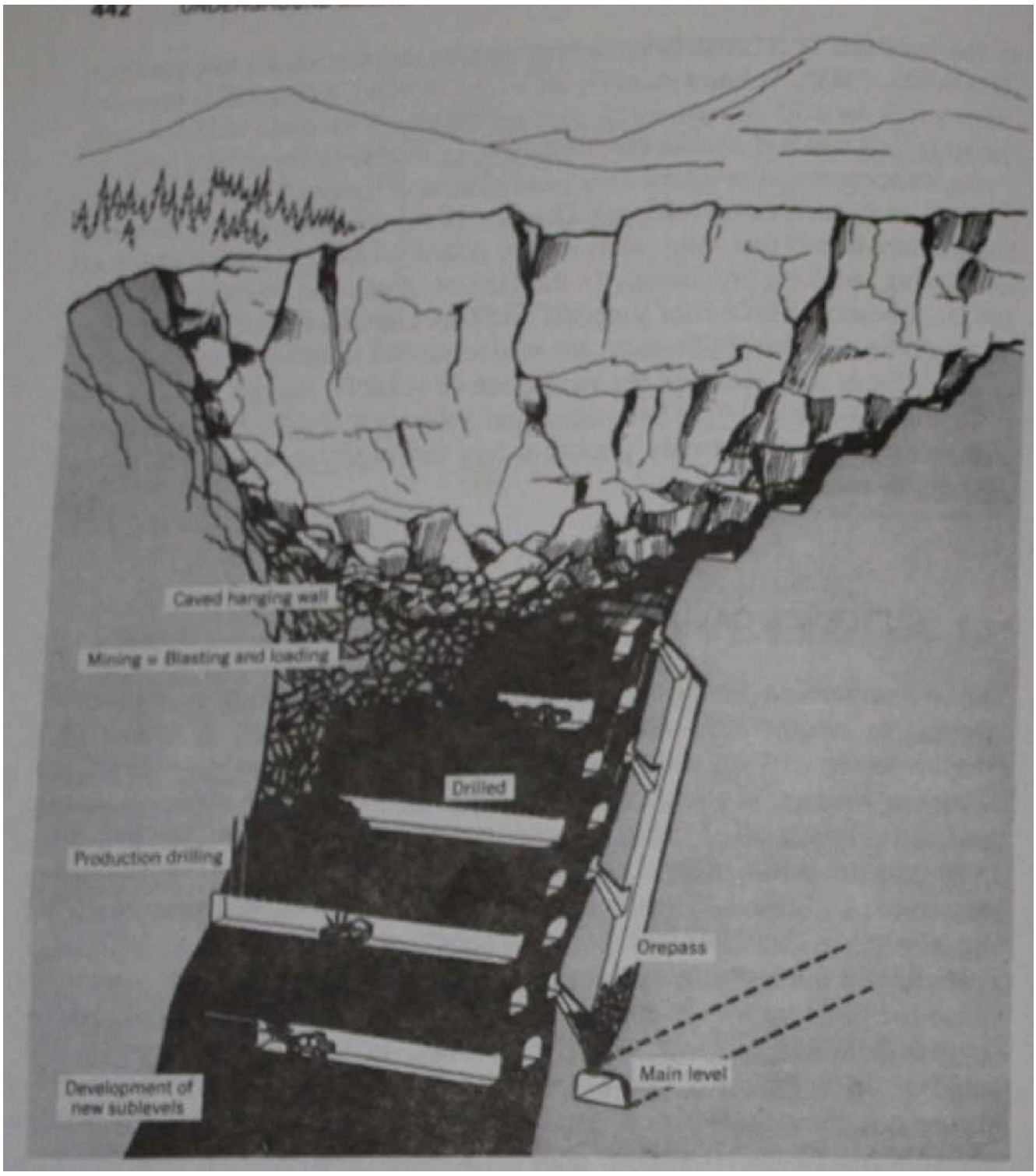


Figure 12.в. Sublevel caving in a large, steeply dipping ore body. Different unit operations are conducted on adjacent sublevels (After Hamrin 1982 By permission from Society of Mining Engineers, Inc . Littleton. CO)

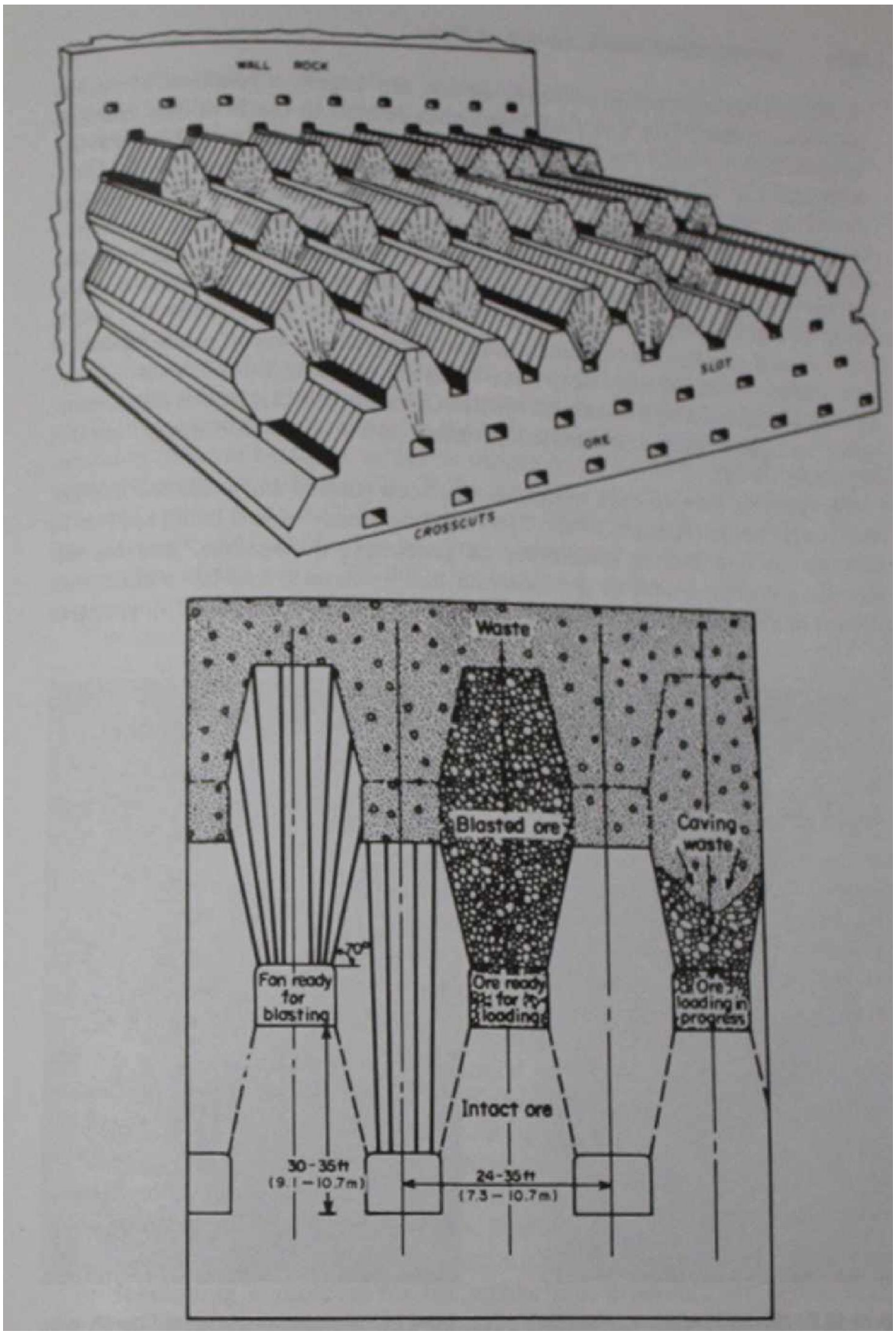


Figure 12.7. Layout for sublevel caving. *(Top)* View showing caving slot, sublevel crosscuts, and fanhole drilling round. (After Baase et al.. 1982.) (floffom) Details of sublevel layout and unit operations. (After Cokayne. 1982. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Design parameters in sublevel caving are largely a function of ***caving*** *mechanics* ***the*** branch of rock mechanics related to the breakage and collapse *of* consolidated material in place and their flow downward by gravity. Although the ore has to be drilled and blasted in the sublevel method, £ overlying rock comprising the capping or hanging wall is undercut and caves Extremely careful controls must be exercised m drawing the ore to avoid excessive dilution. ***Draw control*** is the practice of regulating the withdrawal of ore in the sublevel crosscuts so as to optimize the economics of the draw (Cokayne, 1982). Premature cutoff results in poor recovery, while tardy cutoff produces excessive dilution. Generally, a cutoff grade based on economics is employed to determine when the mucking should cease and the next fan (ring) of holes should be blasted ('Nilsson. 1982a). In one large mine, rapid sampling and a computer program are utilized for grade control (Thomas, 1978).

The gravity flow of bulk materials has been studied and analyzed in bins, silos, and chutes (Kvapil, 1982). The resulting theory is now being applied to sublevel caving and to specifying its geometry, dimensions, and layout. Models are very useful to demonstrate flow principles and have been successful in simulating gravity draw in various caving methods. The simplest are two dimensional,

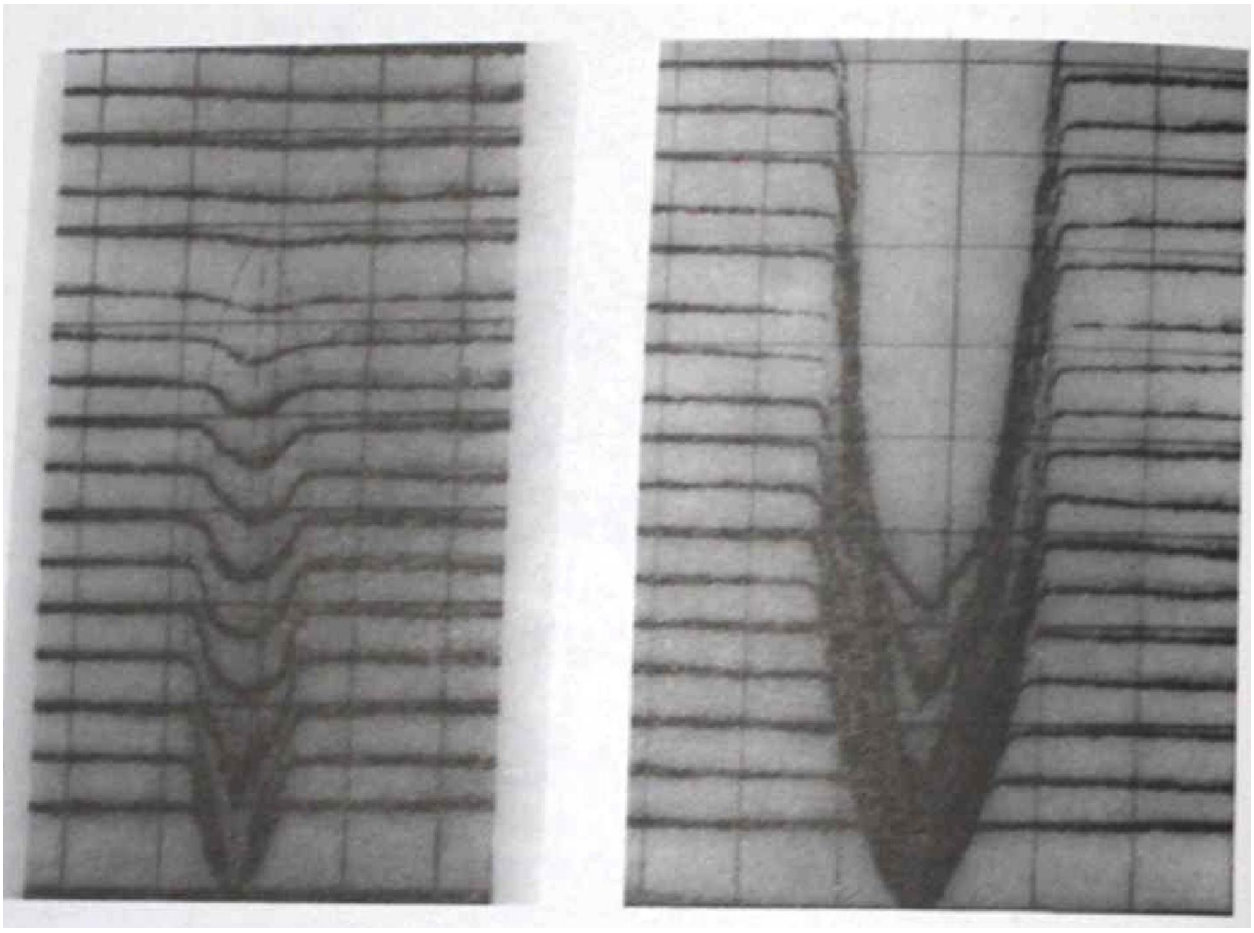


Figure 12.8. Model studies representing successive phases of bulk-material flow in sublevel caving. *(Left)* Vertical ellipse formed at the boundary of gravity motion, delineating the active zone above each crosscut. *(Right) Advanced* phase of gravity flow in drawing ore. (After Kvapil, 1982 By permission from Society of Mining Engineers, Inc., Littleton, CO.)

are two dimensional, consisting of two vertical, parallel glass plates filled with horizontal layers of colored sand , as the sand flows by gravity through an open in g at the bottom the layers distend and depict the flow pattern (Fig l2.8,. At the boundary of gravity motion, an ellipse forms (in three-dimensional space, it becomes an ellipsoid); we refer to it as a *gravity-flow ellipse* (Fig.12-8, left). As more sand is withdrawn, the ellipse distorts and forms a funnel open to the surface (Fig. 12.8, *right).* In actual caving, it is this funneling action into the overlying waste which dilutes the ore during drawing. Theoretically, no dilution would occur if only an ellipsoid of ore were withdrawn.

Caving mechanics as demonstrated by model studies and the derived theory' determines many design parameters in sublevel caving (Janelid, 1975). For example, varying with the bulk properties of ore and waste, the optimum interval between sublevels ranges from 30-45 ft (9 1-13 7 m) the center spacing of crosscuts 25-35 ft (7.6.-10.7 m), and the dimensions of crosscuts about 9-15 ft (2.7-1.8 m) square (Thomas, 1978; Cokayne, 1982). The geometry of fan holes for blasting is governed similarly; the maximum inclination of the fan is 70-80°, the burden 4-6 ft (1.2-1.8 m), and the inclination of the outer drillholes 60-65', based on the gravity-flow ellipse.

The overall applicability and suitability of sublevel caving, as with all methods in this class, are determined by (1) the cavability of the overlying rock and *(2)* the acceptability of surface subsidence. Both factors are inherent in the method and ultimately govern the decision to use it.

Sequence of Development

Another characteristic of caving methods is that they require extensive development prior to and during mining, in sublevel caving, as much as 1520% of the ore production takes place during development (Cokayne. 1982; Hamrin, 1982). The major portion of the development is horizontal, both on the haulage level and sublevels (drifts, crosscuts), although some is inclined (access ramps) or near-vertical (orepasses).

Assuming a steep pitch, and depending on the width of the ore body, sublevel caving operations are arranged longitudinally in narrow deposits and transversely in thick deposits. If the deposit has a flat dip but is sufficiently thick, sublevel caving may still be applicable (Cokayne, 1982).

Haulage level, and optionally sublevels, are usually laid out in a grid of drifts and crosscuts analogous to room and pillar or longwall development. Multiple, interconnected openings facilitate transportation and. more importantly, substantially improve ventilation.

In developing a sublevel for ore extraction, crosscuts are driven across the deposit to the hanging wall or cave boundary (see Fig, 12.6; note the deepest sublevel). At the end of the crosscuts, slot raises are driven 10 the cave above, then slabbed off to form slots the shape of the fanhole round. The first blast breaks into the slot; subsequent rounds pull against the broken muck. Several rounds may be blasted simultaneously to initiate the first cave on the uppermost sublevel. Subsequently, multiple blasts in adjacent crosscuts or even an entire sublevel arc detonated as needed to meet production requirements.

Cycle of Operations

Modern sublevel caving became possible with the widespread adoption in noncoal mining of mobile mechanized equipment, generally diesel powered. The advent of mechanization was especially beneficial to sublevel caving because it requires so much development work. All exploitation operations
are now mechanized also.
Because sublevel caving has application almost exclusively in hard-rock mines, continuous mining or mechanical-boring equipment for exploitation arc not feasible: hence a conventional cycle of operations is followed. Drilling and blasting are the most critical to perform well, since success in caving waste as well as mining ore depends on efficient, effective rock breakage. Drill holes are carefully aligned (by surveying! and controlled in the fan pattern; charging with explosives is done entirely mechanically. Materials handling is carried out conventionally, the ore being transported to dump points and then dropped through ore passes to the haulage level.
Notice in Figures 12.6 and 12.7 that unit operations are sequenced by crosscuts or even sublevels. This permits greater efficiency in exploitation but also is essential to proper caving action.
The following operations and equipment comprise the production cycle (Section 4.1; Fig. 9.9); development operations closely parallel those in other hard-rock mining.
Drilling: fandrill jumbo, two- or three-boom, pneumatic or hydraulic machines (percussion or rotary-percussion); air- or diesel-powered; hole size 2-5 in. (51-76 mm); drilling factor 0.7 ft/ton (0.2 m/tonne) (Hamrin 1982)

Blasting: ANFO. slurries; bulk charging by pneumatic loader or pump, firing electrically or by detonating fuse; blasting round for caving method (Fig. 9.9b) (Cokayne, 19И2)

Secondary blasting (on sublevel): drill and blast, mudcap, impact hammer

Loading: LHD. overhead loader, front-end loader, gathering-arm loader
Haulage: LHD. truck, shuttle car on sublevel; gravity flow through ore- pass; rail conveyor on haulage level

The important auxiliary operations in sublevel caving are ground control and ventilation. Neither is complex in this method of mining. Support of development openings is the main requirement, and normal light bolting, timbering, or shotcreting is sufficient (excessive support is not appropriate for the method). Ventilation .s easily carried out if the multiple sublevel crosscuts arc interconnected, as in room and pillar mining; control devices and auxiliary ventilation afford good air quality and distribution at the face further necessary auxiliary operations arc health and safety (other than ventilation), maintenance,

power supply, drainage, and materiel supply. Sec Table 4.1 and Sections 10.2 and 10.3 for additional particulars.

Conditions

The following is based on Boshkov and Wright (1973), Morrison and Russell (l473). Thomas (1978), Cokayne, (1982). and Hamrin (1982):

1. ***Ore strength:*** moderate lo fairly strong, requiring blasting; may have occasional heavy or sticky /ones, but require enough competence to stand without excessive support; less strength than for unsupported methods, but more than for block caving
2. ***Rock strength:*** weak to fairly strong; may be blocky, but should be fractured or jointed and cavable; preferably moderate to large fragments. no fines to dilute ore
3. ***Deposit shape:*** tabular or massive (if elongated along one axis, preferably vertical); may be moderately irregular
4. ***Deposit dip:*** fairly steep 060°) or vertical; can be fairly flat if deposit thick
5. ***Deposit size:*** large, extensive vertical or areal extent; thickness >20 ft (>6 m)
6. ***Ore grade:*** moderate
7. ***Ore uniformity:*** moderate, no sorting possible (some dilution acceptable)
8. ***Depth:*** moderate (<4000 ft, or <1.2 km)

Features

The following is based on Boshkov and Wright (1973), Thomas (1978). Cokayne (1982), Hamrin (1982), and Kvapil (1982):

Advantages

1. Fairly high productivity (range 20-40 tons, or 18-36 tonnes, per employee-shift; maximum 40-50 tons, or 36-45 tonnes, per employee- shift)
2. High production rate: large-scale method
3. Fairly high recovery (80-90%, attaining 125% if dilution excessive)
4. Suitable for full mechanization
5. Somewhat adaptable, flexible, and selective: no pillars required
6. Good health and safety

Disadvantages

1. Moderate to fairly high mining cost (relative cost: 50%)
2. Moderate to high dilution (10-35%): highly sensitive with regard to dilution
3. Caving and subsidence occur, destroying surface
4. Draw control critical to success of method

5. High development cost
6. Must provide stope access for mechanized equipment

Applications

Sublevel caving in its present form has had little application to date in the United States; interestingly, the original method was widely used in the iron ranges of the Lake Superior district, and its main use today is still in iron ore mining. In addition to Sweden, Canada has many applications. Examples worldwide include anthracite coal (Valley View, PA—see Green, 1985), bituminous coal (Ajka, Hungary—sec Ravasz, 1984), copper (Craigmont, BC—see Haase ct al., 1982), copper-lead (Ml. ba, Australia—see Hornsby el al.. 1982), iron (Pea Ridge, MO—sec Tucker, 1981; Kiirunavaara, Sweden—see Janelid, 1975; Guslafsson, 1981; Heden et al., 1982), and nickel (Sudbury, Ont.—see Johnston, 1982).

Case Study. Kiirunavaara iron ore mine. LKAB, Kiruna, Sweden.

General: world's largest underground mine, located in major iron-orc- mining district of Scandinavia, within Arctic Circic; operating company is 96% state-owned; mining dates to 1904
Geology: large deposit, shaped like elongated disc (0.1 x 2.5 mi, or 160 m x 4 km), extends to I mi (1.5 km), dip 50-70[3]; sedimentary in origin, tilted, and metamorphosed; ore fine-grained magnetite with apalite; grade 60-67% iron, O.I-0.5% phosphorus
Production: N0,000 tons/day (100,000 tonnes/day) of ore and waste, or 24 million tons/yr (22 million tonnes/yr) of ore; N hoisting shafts, side by side
Mining method: sublevel caving, with variations

Variations

Top Slicing. This is an example of the variation predating the method itself.
Top slicing is an underhand caving method in which horizontal slices of ore are extracted by driving contiguous sublevel drifts and crosscuts and then collapsing them to cave the overlying waste. Essentially obsolete today, the method gave rise to sublevel caving in its original form. Formerly the standard method for mining the Lake Superior iron ores, top slicing today finds limited use for pillar recovery and as a hybrid method with longwall mining or square set stoping.

In top slicing, a raise is driven to the level above; a sublevel is established immediately below the level (Fig 12.9). Since the method is used only in bad ground, heavy timbering is required in the drifts and crosscuts, with sets nearly touching, and forepoling or spiling advanced overhead for protection. As a crosscut reaches the ore body boundary, it is caved on the retreat by removing or blasting the timber sets. A lagging floor is laid before caving so that the overlying waste

does not contaminate the ore below. After a complete sublevel is caved, the next sublevel below is mined. The accumulation of caved waste rock and crushed, broken timber is called mat.

Although the method is applicable to the next – to - worst ground conditions (weak but not running) and to irregular deposits it highly labor intensive, has heavy timber requirements, and is unproductive (small – scale) and expensive (relative cost: 90%). For the latter reasons, top slicing has fallen into disuse, replaced by other caving methods. References are contained in Lewis and Clark (1964), Thomas (1978), and Jackson (1982).

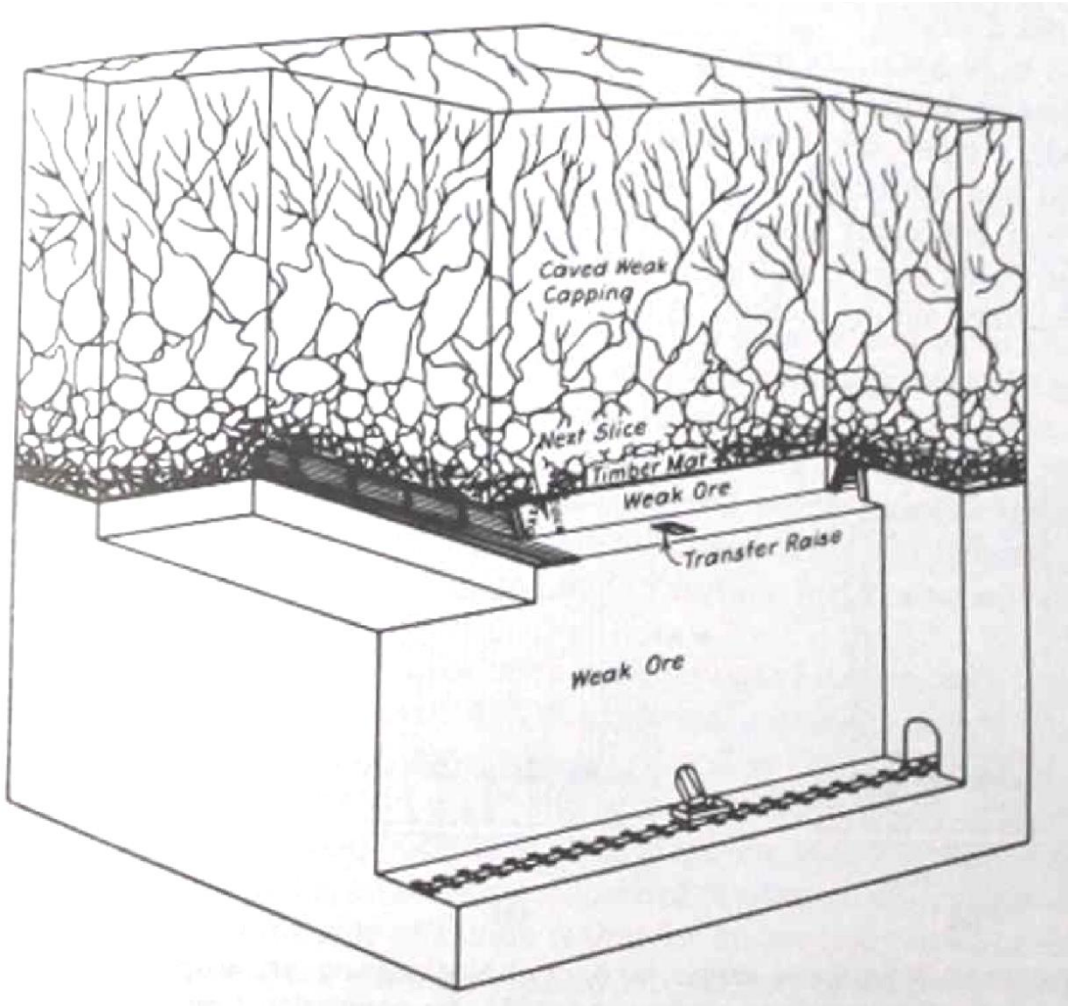


Figure 12.9. Top slicing in massive deposit. (After Lewis and Clark, 1964. Copyright 1964, Joh n Wiley & Sons, New York.)

12.4 BLOCK CAVING

The mass – production exploitation technique in underground mining that has the potential to rival surface mining in output and cost is block caving. Among underground hard – rock miners, the old, widely known adage is, “If you can’t block – cave it, forget it” – a somewhat apt if not entirely accurate assessment.

Block caving is the mining method in which masses, panels, or blocks of ore are undercut to induce caving, permitting the broken ore to be drawn off below. If the deposit is overlain by capping or bounded by a hanging wall, it caves ,too, breaking into the void created by drawing the ore (Fig. 12 10). Unlike sublevel caving, both ore and waste are caved in block caving. Like sublevel the caving proceeds in columnar fashion to the surface, often belling out to a stable angle of repose (50-90" from the horizontal: see Section 9.2) (Richardson, 1981). The result is massive subsidence, exacerbated by the exceptionally high production rate and great areal extent characteristic of block caving.

A truly American mining method, block caving was invented in the years following World War I to cope with the exploitation of the massive, lowgrade copper-porphyry deposits of the southwestern United States. Developed and perfected in the United States, the technology rapidly spread abroad, especially where American companies operated, and now is applied in probably more countries and to more diverse commodities than any other subsurface mining method. A *large-scale* method, block caving is utilized to produce about 10 % of U.S. underground metals and nonmetals and 3 % of all underground minerals.

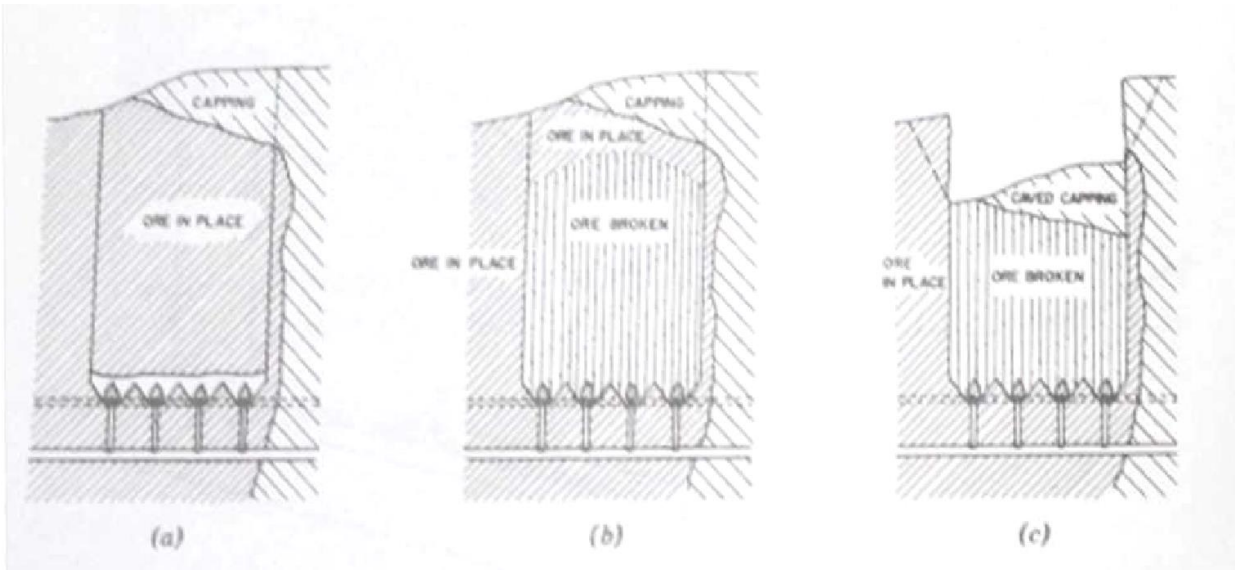


Figure 12.10. Progressive stages (a b. c) of block caving, show ing caving o f ore and waste (After Trepanier and Underwood. 1981 By permission from Society o f Mining Engineers. Inc.. Littleton. CO.)

Caving mechanics provides the basis for understanding and controlling the operating factors in block caving as it does in sublevel caving. Determining the cavability of an ore body is the first major task. Safe and efficient block caving is not possible if the ore (or rock) forms stable arches or break into fragments of excessive size, defined as being too large to pass through the system of drawpoints, bells, finger raises, and so forth (McMahon and Kendrick, 1969) Reliable methods to predict the cavability of an ore body are essential and they must be applied in the initial planning stage. Results of the cavability study influence not only thc

mining plan and costs for extracting (drawing) the ore body but subsequent costs for secondary blasting, loading, haulage-hoisting, and even mineral processing (Kendorski,1978). The basic question to be resolved is not only will it cave, but can it be caved, drawn, handled, and delivered to the surface for processing—economically?

Several rock mechanics indices have been proposed as measures of cavability (McMahon and Kendrick, 1969). Empirical *cavability numbers* have been devised at various block caving mines, based on operating experience; unfortunately, they are valid only at the properties where they originated. *Secondary blasting* has also been used as an empirical indicator; the amount of explosives consumed is an inverse measure of the degree of fragmentation achieved in caving. One rule of thumb is that for an ore body to be cavable, 50% of the ore fragments should break to 5 ft (1.5 m) or less in maximum dimension (Tobie and Julin, 1982). Finally, the *rock quality designation RQD,* which is the percentage of intact (≤4 in, or ≥102 mm, in length) core recovered from a drillhole, affords a means of predicting cavability (Obert, 1973a, 1973b). MvMahon and Kendrick report some success in correlating these indeces.

Cavabtlily is not just a matter of achieving acceptable fragmentation and optimum operating costs. From a safety standpoint, the ore or capping must not arch over long distances for long periods of time I he formation of stable arches not only disrupts the caving operation but very likely will cause air blast and concussion in the mine when they suddenly collapse. The classic - etamplc occurred at the Urad molybdenum mine in Colorado: an arch 300 x 400 ft (90 x 135 m) in are a formed over the entire mine and violently collapsed after nine months, causing extensive property damage by air blast (Lucas and Haycocks. IWI. Only the prior evacuation of all employees from the mine prevented wholesale loss of life.

After cavabiliiy determination, the second application of caving mechanics is in draw control and drawpoint spacing (Richardson. 1981). Involved is the gravity-flow ellipse or ellipsoid referred to earlier in “Sublevel Caving" (Section 12.3), modified by the inflow of waste us the caving funnel progresses upward into the capping In block caving, this is termed the *zone of draw.* Figure 12.11 depicts caving action as a function of drawpoint spacing. With theoretically ideal spacing (Fig. 12.11b), the ellipses arc contiguous. Excessive spacing (Fit: 12.1 lb) or deficient spacing (Fig. 12.1lc) produces zones of draw which may yield unsatisfactory grade control and create weight problems on sill pillars. In plan, draw points may be arranged in hexagonal (d) or square pattern (c). Practically, to insure that the /.ones of draw completely blanket the ore body, drawpoint spacing is reduced somewhat, permitting minor overlap of the zones, as seen in (d) and (c).

As block caving is initiated—whether in a block, panel, or mass—the caving action commences as undercutting of a critical-sized area is completed and progresses upward: it also progresses in a prescribed pattern across the ore body. Once caving begins, the only means of regulating it as well as the production of ore is through draw control (DeWolfe. 1981). Because of irregularities in the contact between ore and waste but. more important, because of tunneling in the

ore, some dilution is inevitable if a high recovery is achieved. Effective draw control optimizes grade control, recovery, and dilution. How this is accomplished is idealized in Figure 12.12, where panel caving is progressing across a deposit, regulated by the draw - control rate. Caving is being initiated by undercutting on the right while full production (100%) is being realized on the left (DeWolfe, 1981), Draw-control rates range from zero and 15% at the draw points on the right lo 100% of maximum tonnage on the left, with intermediate rates (30-75%) in between. Depending on cavabiliiy of the ore, the actual rate ("velocity") of draw varies from 6 in. (152 mm) to 4 ft (1.2 m) per day. with mine production rates varying from 6000-65.000 tons/day (5400-59,000 tonnes/ day) (Ward. 1981; Tobic and Julin. 1982). In practice, drawpoint spacing ranges from 15-40 ft (4.5-12 m).
There are other parameters involved in mine design when using block caving, and a discussion of two of the most important follows.

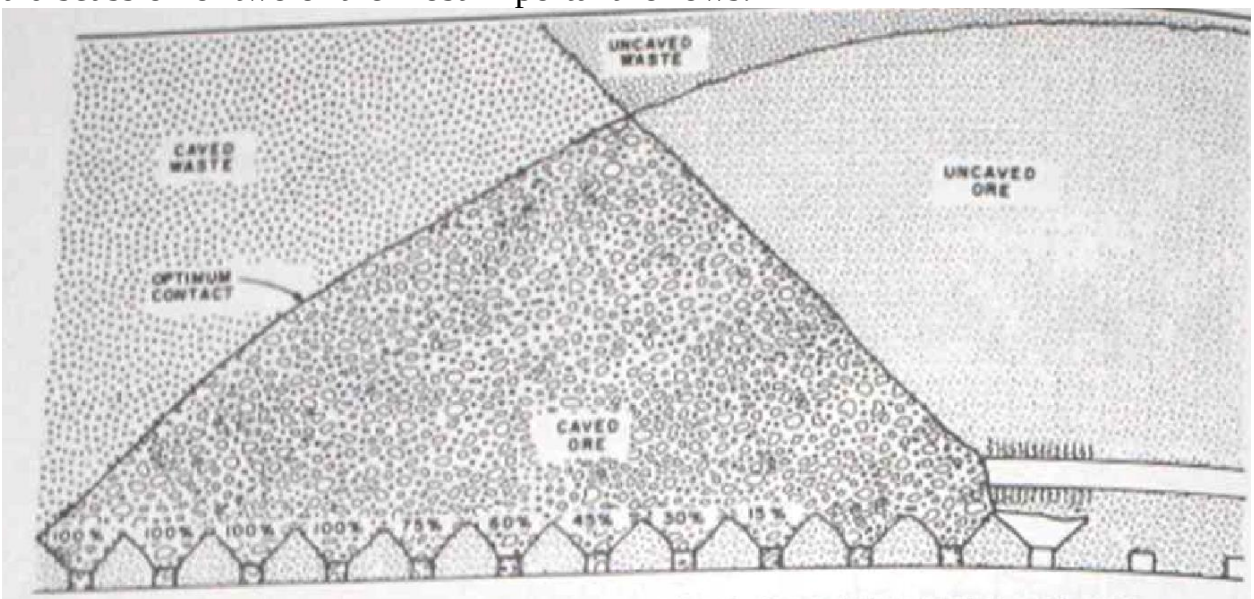


Figure 12.12. Progression of cave across an ore body (from left to right), with differential draw-control rates indicated. The plane of contact between caved ore and waste is inclined, typical of panel and mass block caving. (After DeWolfe, 1981. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

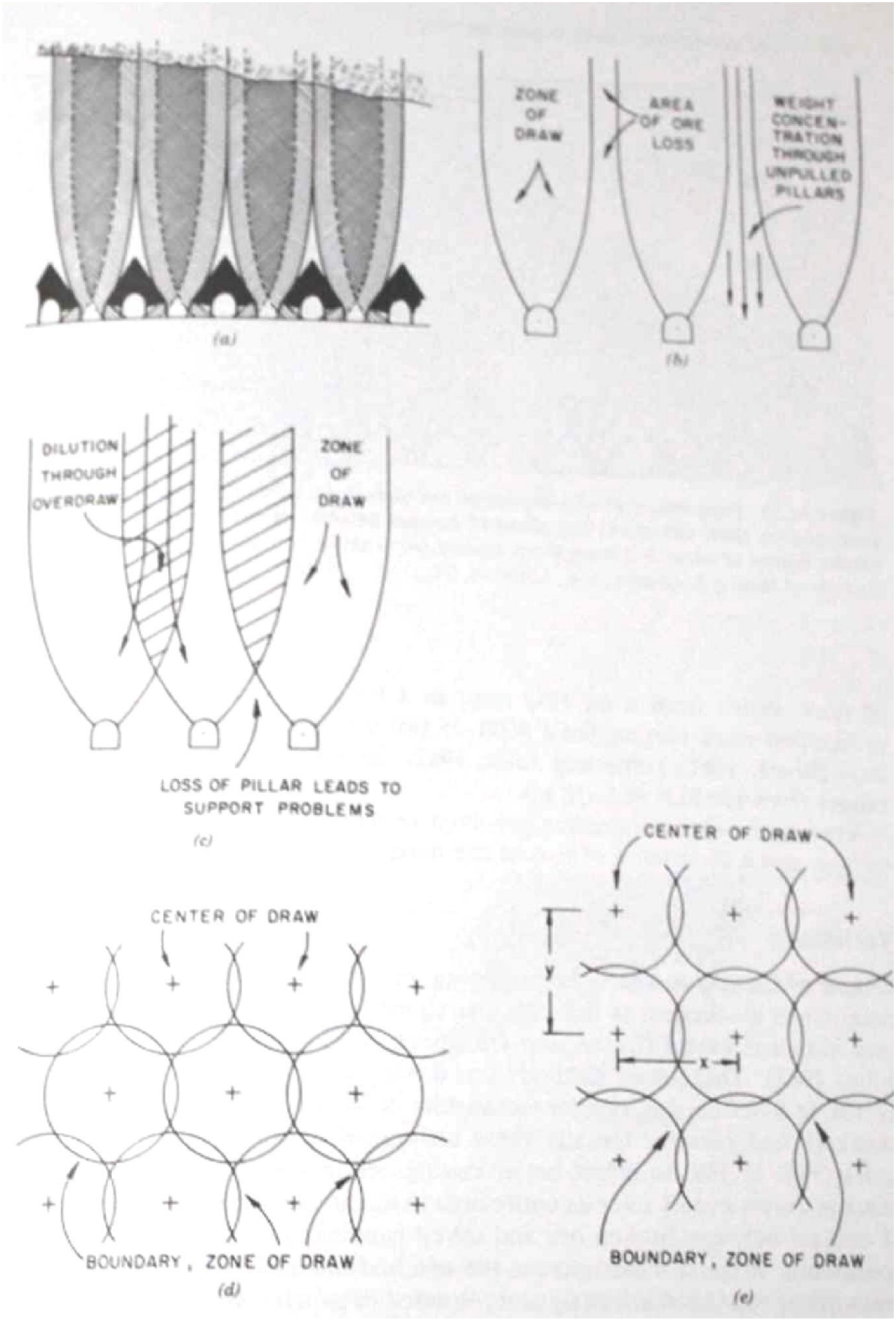


Figure 12.11. Effect of drawpoint spacing on zone of draw in block caving. *(Top)* Vertical sections (al Theoretically ideal spacing. (After Weiss et al.. 1981.) (b) Excessive spacing (c) Insufficient spacing. *(Bottom)* Plan views, (d) Suitable hexagonal spacing (•) Suitable square spacing. (After Richardson. 1981. By permission from Society of Mining Engineers. Inc.. Littleton. CO.)

Variations

Shape of Caving Area. In block caving, one of (he critical parameters that determines its success is the size and shape, or pattern, of the horizontal area which is caved (Lucas and Haycocks, 1973; Ward, 1981; Tobie and Julin, 1982). Three area methods arc used; block, panel, and mass (Fig. 12.13). In *block caving,* regular rectangular or square areas are undercut in a checkerboard pattern. Usually these are mined in alternating or diagonal order (Fig. 12.13a) to effect better caving action and draw control. Each block is drawn evenly over its entire area to maintain a near-horizontal plane of contact between broken ore and caved capping. In *panel caving,* orc in continuous strips is mined across the ore body (Fig, 12.13b). Manageable areas are caved simultaneously and retreated in panels. An inclined plane of contact between orc and capping maintained during caving is shown in Fig. 12.12. In *mass caving,* there is no area division into block or panels; irregularly sized prisms are mined as large as consistent with caving properties of the ore and stresses on the openings below (Fig. 12.13c). The plane of contact between ore and waste is generally inclined as undercutting proceeds on the refreat across the ore body.

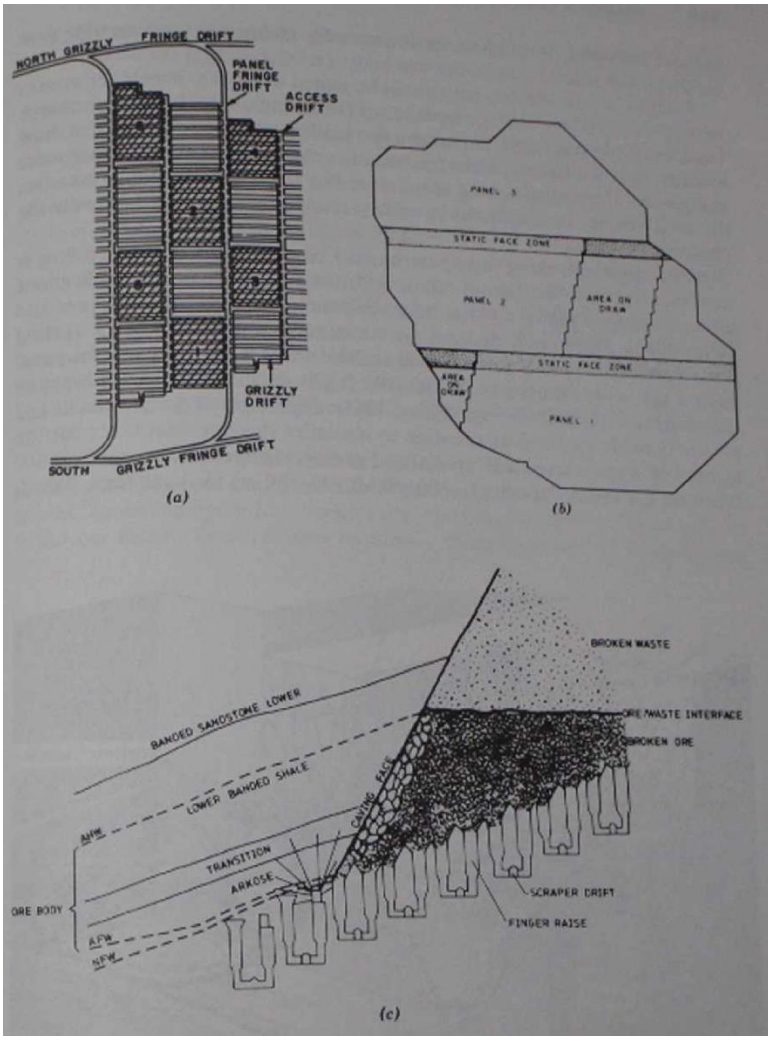


Figure 12.13. Three shape variations of block caving, (a) Block version of block caving. (After Tobie et al.. 1982.) (b) Panel caving. (After DeWolfe, 1981). (c) Mass caving. (After Pearson, 1981 By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Although dimension in popularity, the block version of block caving is still widely employed (about 50 %), with the remainder of the applications approximately equally divided between panel and mass caving (Tobie and Julin, 1982). Horizontal dimension, commonly employed in block version vary from, 100 x 100 ft (30 x 30 m) to 300 x 300 ft (90 x 90 m), for panel caving the width range, from 100 - 100 ft (30-90 m), with pillar, between the panel, of 10- 30 ft (3-9 m) (Pillar, 1981). The height o f the ore block, and panel, и made as large as possible to minimize development costs per ton (tonne) of ore; if varies with spatial end geologic factor, and range, from 100 ft (36 m) for small deposits to 300-400 ft (90 - 180 m) for large one, (Ward, 1981; Tobie and Julin, 1982). The overall height of ore and copping may action 1500 – 4000 ft (450 – 1200 m).

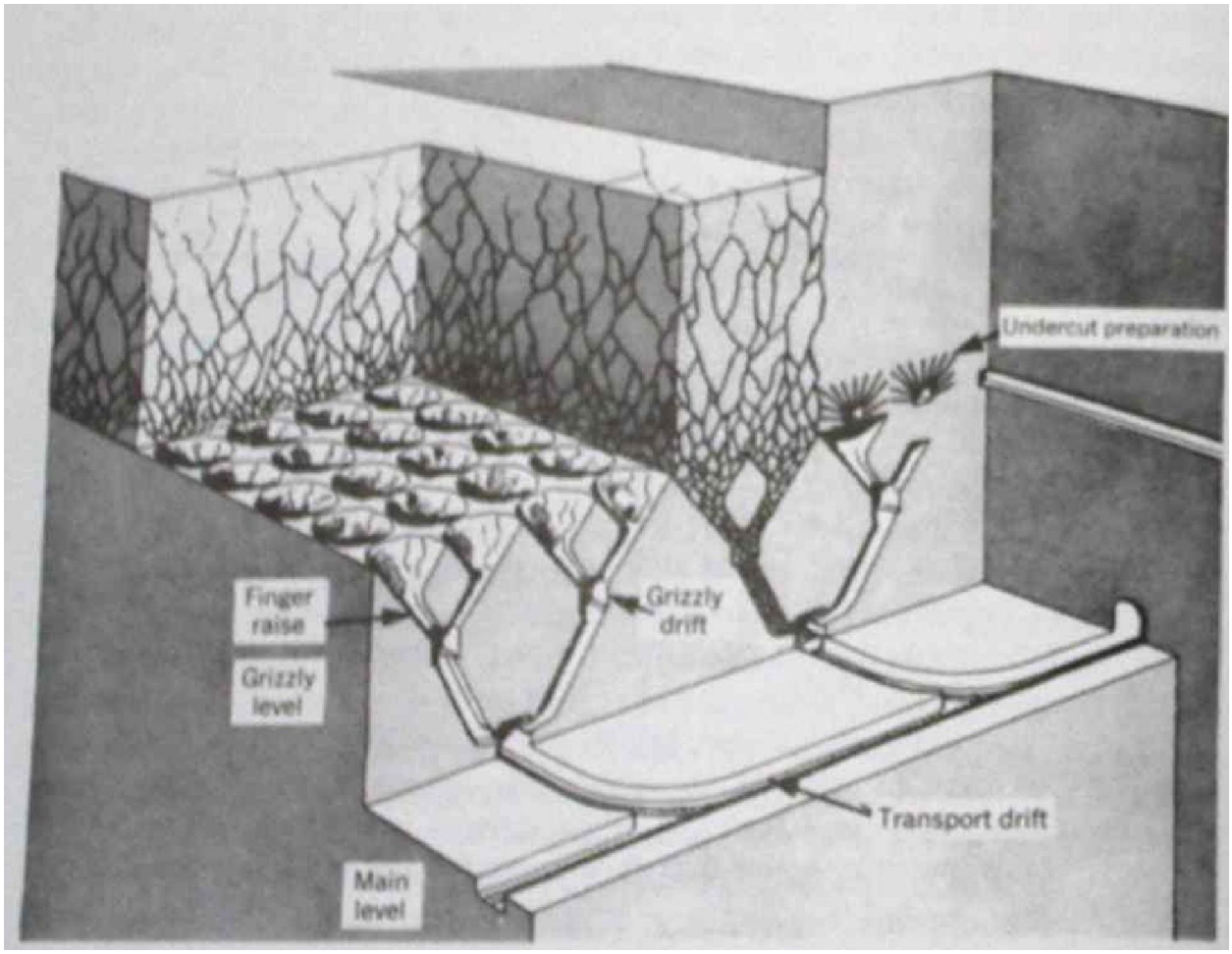


Figure 12.14. Block caving, with gravity draw and grizzly d rifts (After Hamrin, 1982 By permission from Society of Mining Engineers. Inc.. Littleton. CO.)

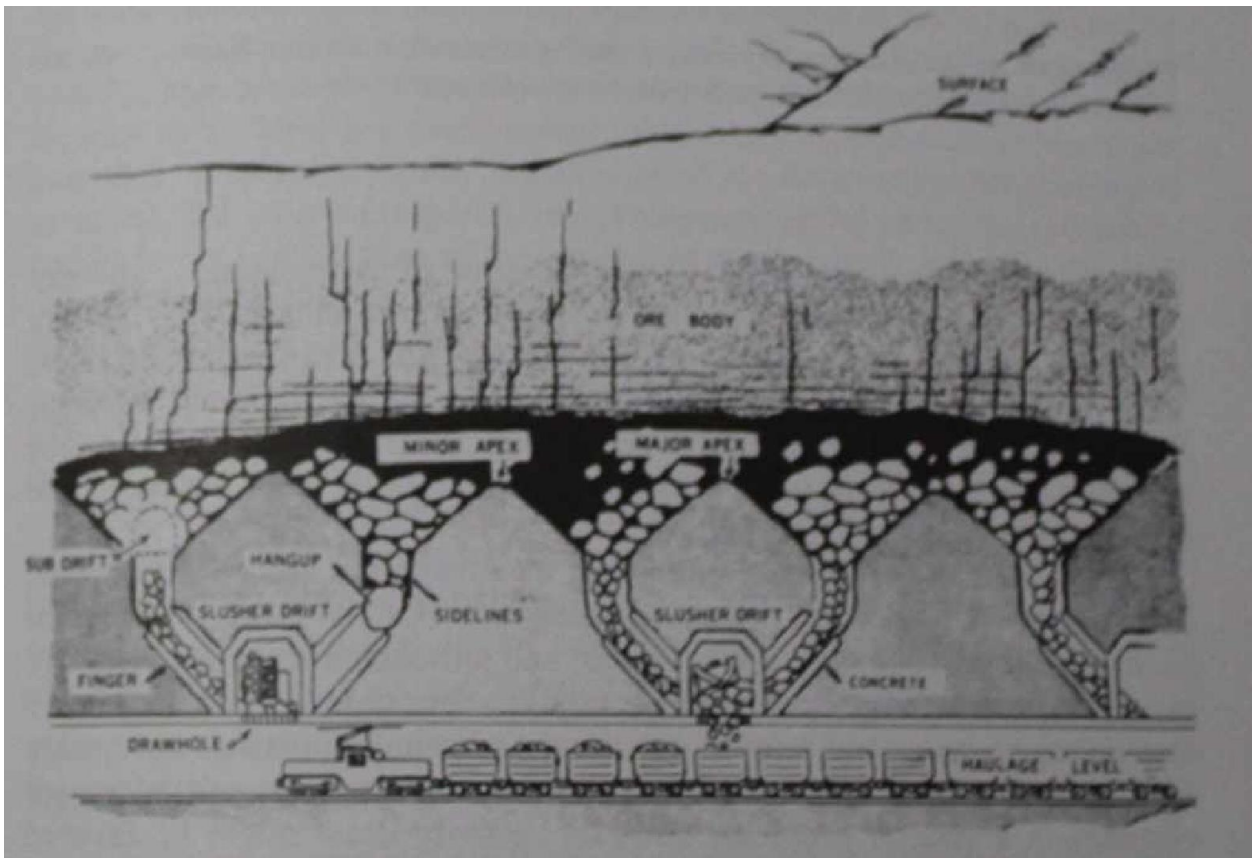


Figure 12.15. B lo ck caving, with scraper loading in slusher drifts (After Johns. 1961 By permission from Society of Mining Engineers. Inc., Littleton. CO.)

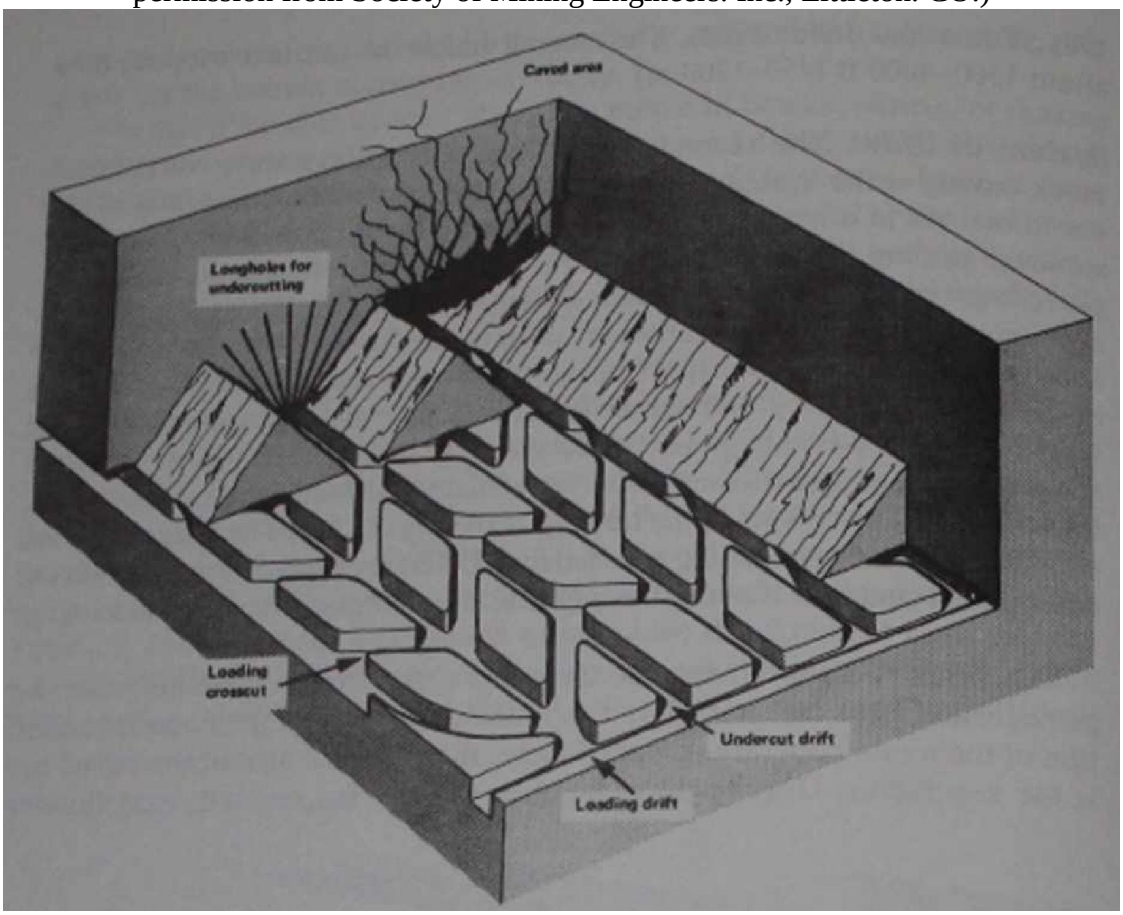


Figure 12.16. Block caving, with machine loading in draw drifts. (After Hamrin, 1982. By permission from Society of Mining Engineers. Inc.. Littleton, CO.)

System of Draw. The second design basis for distinguishing varieties of block caving is the system of draw or method of materials handling of the caved ore. As in other vertical exploitation methods, such as shrinkage and sublevel stoping (Section 10.4 and 10.5), three versions are currently employed: gravity, slushier, and loader.

The primary determinants of the means of materials handling are geologic and physical characteristics of the ore body and the financial condition of the mining operation Specifically, the fragmented size of the caved ore is the key factor. Gravity draw requires a finely fragmented easy – flowing ore: slushers arc suitable for a moderate degree of fragmentation; and mechanical loaders accommodate very coarse ore. Similarly, the gravity method is best applied to large, regular deposits and high production rates, while the slusher and loader method are more suited to small, irregular deposits and less production. Larger capital costs arc entailed with the slusher and loader methods, but the gravity method is labor intensive and requires extensive development.

Each system of materials handling has its advantages and disadvantages; several are evident in the preceding paragraph. Of the three, mechanical loading is the most flexible and versatile; and although it is capital intensive and provides poorer grade control, it requires less development and fewer draw points, is faster and more productive, and usually provides acceptable draw control at the lowest overall cost. If ground control of the development openings is a serious problem, however, gravity draw might be favored because its openings are smaller in area although more numerous. In the final analysis, the choice of materials-handling system is not determined by costs alone but must take into account compatibility with the natural conditions and ore recovery - dilution results (Ward 1981).

Of the three means of draw, slusher is most widely used today (about 45%), loader is next (30%), and gravity is least and declining day (25 %) (Pillar).

Sequence of Development

Development for block caving typically is extensive and expensive, but generally less so than for sublevel caving on unit – cost basis. Main – level development commences from the shaft station in the usually way, providing for high-speed, high-capacity haulage and ample ventilating – airflow capacity. Main haulageways are often paralleled by laterals, interconnected by crosscuts, to ensure good ventilation and to provide adequate lanes or stub crosscuts for loading. One or more sublevels *are required* for grizzly or slusher operations in the gravity and slusher variations of block caving.

To provide ore-drawing facilities, chutes drawn points, or trenches are prepared in the ore body under the block to be caved (Hamrin, 1982). They lie adjacent to the haulage drifts and crosscuts on the main level, as may be seen in Figures 12.4-16. Finger raises to serve as orepasses are then driven to the sublevels (slusher or grizzly) above, if any, and to the undercut sublevel, where they are belled out; other finger raises serve manways. The most critical development operation is undercutting, which is carried out in the usual manner (Section 10.4)

but with the intention of initiating caving by the carefully controlled removal of pillars. Because of the large expanse of opening and tonnage of ore involved, the dangers of premature collapse, hangup, or air blast are ever-present. Since the ground becomes heavy and may shift, substantial reinforcement support for all development openings is essential. The interval required for development of the sublevels, orepasses, and undercut is a function of the method of draw, gravity requiring the most and mechanical loading the least (a large amount of ore is tied up by this development and can only be recovered if the ore body below is mined).

Boundary weakening other than by undercutting, is rarely performed today in block caving (Ward, 1981). Occasionally, comer raises are driven on one side of the undercut block and slabbed off to create a narrow vertical slot. Shrinkage sloping bas also been utilized to form a thin slot. Having been freed on two sides, even the most competent rock and resistant block usually begins to break and cave. In a new mine, boundary weakening may provide a guarantee that caving will, in fact, result, and it can usually be eliminated as more rock mechanics knowledge is gained.

Cycle of Operations

As in most other vertical sloping and caving methods for hard-rock mining, the development and exploitation cycles of operations are separate and distinct. Each is highly mechanized but uses equipment especially designed for its function. Continuous mining machines are generally not applicable in hard rock (except raise borers and occasionally TBMs), so the cycle of operations for development is a conventional one of drill + blast + load + haul, similar to that in stope and pillar mining (Section 4.1 and 10.3). The equipment utilized is pictured in Figure 9.9. For exploitation, a truncated cycle of materials – handling operations is utilized, since only loading and haulage is necessary (rock breakage is performed by the caving action). The cycle of operations in block caving consists of the following.

Drilling (undercut): pneumatic – or hydraulic – powered, percussion or rotary – percussion drill rig or longhole drill on column; hole size 2 – 3 in. (51 -76 mm)
Blasting (undercut): ANFO, slurries; bulk charging by pneumatic loader or pump, firing electrically or by detonating fuse; slabbing round
Secondary blasting (on sublevel or in -haulage drift): impact hammer, dynamite bomb, drill and blast, mudcap
Loading (through bells and orepasses): gravity flow and chute; (at drawpoints): LHD, front-end loader, overhead loader, slusher
Haulage (on main level): LH D , rail, truck, belt conveyor, slushier

Like some other caving methods (longwall and top slicing) block caving requires elaborate ground-control measures (Hamrin, 1982). Reinforcement *as* well as conventional support is often called for in development openings (e.g., raises, orepasses, slusher drifts, haulage drifts) that serve a production function. The usual forms of ground support are bolts shotcrete steel sets or arches cable, or mesh.

Reinforcement consisting of reinforced concrete or steel liner is often installed to prevent excessive wear or damage to important, openings, especially slusher and haulage drifts. The next most important auxiliary operation is ventilation; large airflow quantities are essential for dust, fume, and heal control and are obtainable through the multiplicity of development openings (Tobie and Julin, 1982). Other key auxiliary functions arc provided by drainage, materiel supply, maintenance, and power supply (Table 4.1; Sections 10.2, 10.3).

Conditions

The following is based on Boshkov and Wright (1973), Lucas and Haycocks (1973), Morrison and Russell (1973), Thomas (1978), Weiss et al. (1981), Hamrin (1982), and Tobie and Julin (1982):

1. *Ore strength:* weak to moderate or even fairly strong, preferably soft or friable, fractured or jointed, not blocky; caves freely under own weight when undercut; free –running, not readily oxidized.
2. ***Rock strength:*** weak to moderate, similar to ore in characteristics; distinct ore-rock boundary
3. ***Deposit shape:*** massive or thick tabular, fairly regular
4. ***Deposit dip:*** fairly steep (>60°) or vertical; can be fairly flat if very thick
5. ***Deposit size: very*** large areal extent; thickness >100 ft (>30 m>
6. ***Ore grade:*** low, ideal for disseminated ore, most suitable of underground methods for low-grade deposits
7. ***Ore uniformity:*** fairly uniform and homogeneous; sorting not possible
8. ***Depth:*** moderate (>2000 and <4000 ft, or >600 and <1200 m); depth must be sufficient to develop overburden stress which exceeds rock strength

Features

The following is based on Lewis and Clark (1964), Lucas and Haycocks (1973). Morrison and Russell (1973), Thomas (1978), Ward (1981), Hamrin (1982), Tobie and Julin (1982):

Advantages

1. Fairly high productivity (range 15-40 tons, or 14-36 tonnes, per employee-shift; maximum 40-50 tons, or 36-45 tonnes, per employee- shift)
2. Fairly low mining cost, least of the underground methods (relative costs 20%. same as longwall mining)
3. Highest production rate of underground methods; large-scale method
4. High recovery (90-125%), hut dilution may be high (10-20%)
5. Hock breakage in production occurs entirely by caving, induced by undercutting; no drilling and blasting cost (except for secondary breakage).

6. Suitable for gravity-draw or fully mechanized materials handling; repetitive. standardized operations
7. Ventilation usually very satisfactory ; good health and safety (except in undercutting and some drawing)

Disadvantages

1. Caving and subsidence occur on large scale; wholesale damage to surface
2. Draw control critical to success of method
3. Slow, extensive, costly development
4. Maintenance of openings in production areas is substantial and costly if pillars load excessively
5. Rigid, inflexible method
6. Hazardous work due to hang-ups in grizzly and slushier sublevels and at drawpoints, with some risk of air blast throughout mine.
7. Possible spontaneous combustion in ore or rock during caving if drawing slow or delayed (risk high if sulfide content is <45%)

Applications

If mineral deposits that are otherwise prime candidates for recovery by open pit mining occur at uneconomic depths, and underground methods must be considered, then block caving is the logical first choice. It is remarkable how similar in application are open pit mining and block caving; the main point of difference is the depth of occurrence of the ore body to which they are applied. Examples of block caving mines are asbestos (Thetford Mines – Que. – see Trepanier and Underwood, 1981), copper (Lakeshore, AZ; San Manuel, AZ – see Tobie et al., 1982; Nehanga, Zambia – see Pearson. 1981; Erfsberg East, Indonesia – see Ward, 1981), diamonds (Premier, S. Africa – see Owen, 1981), iron (Mather, MI – see bluekamp, 1981), and molybdenum (Urad, Co, Climax, Co – see Vera, 1981; NM – see Shoemaker, 1981).

Case Study. Henderson molybdenum mine. AM AX. Inc.. Empire. CO.
General- major molybdenum producer atop the Continental Divide at an elevation of 10.4(H) ft (3.2 km); required $500 million capital investment to open: mining commenced in 1976
Geology- two partially overlapping quartz-molybdenite ore bodies, occurring within rhyolite porphyry intrusive in granite; elliptically shaped in plan with dimensions of 2200 x 3000 ft (670 x 910 m) and average thickness of 6(H) ft (185 m); deposit maximum depth 3600 ft (1080 m); ore molybdenite; grade 0.42% MoS,: reserves 242 million tons (220 million tonnes)
Production: 30,000 tons/day (27.200 tonnes/day) ore. or 50 million Ib/yr (23 million kg/yr) Mo; 3 production levels; mill located 14.4 mi (23.0 km) distant, through 9.6 mi (15.5 km) tunnel; development period 10 yr

Mining method: panel block caving with LH D loading equipment

12.5 SPECIAL TOPIC: MINE VENTILATION

The most vital of the auxiliary operations in underground mining is ventilation. It largely maintains the quality and quantity of the atmospheric environment and is the mainstay of the miner's life – support system and the health and safety program.

Mine ventilation is the process of total air conditioning responsible for the quantity control of air, its movement, and its distribution. Other processes specifically help to accomplish quality control (e.g., gas and dust control)or temperature-humidity - control (e.g., air cooling and dehumidification, heating). Because of its versatility, only ventilation carries out aspects of all three control function. When the control of the atmospheric environment is complete -that ... when there is simultaneous control of the quality, quantity, and temperature – humidity of the air in a designated space – then we are employing *total air conditioning.*

In recent years, environmental standards in mines have been raised substantially (Hartman. 1982a). Although threshold limits are based on human endurance and safety, we are increasingly concerned for standards human comfort as well. The reasons have to do with cost effectiveness as well as humanitarianism. Worker productivity, job satisfaction, and accident prevention correlate closely with environmental quality.

Conditioning functions and processes commonly used in mines consist of the following:

1. Quality control
a. Gas control
b. Dust control
2. Quantity control
a. Ventilation
b. Auxiliary or face ventilation
c. Local exhaust
3. Temperature-humidity control
a. Cooling and dehumidification
b. Heating

Processes may he applied individually or jointly. We concentrate here on mine ventilation as the most important and universally used process

Quality Control

Chemical contaminants— principally gases and dusts—constitute a variety of hazards in the mine atmosphere. Gases may be suffocating, toxic, radioactive, or explosive. Dusts may be nuisance, pulmonary, toxic, carcinogenic, or explosive.

Contaminants occur naturally (e.g., strata gases such as methane in coal beds or radon gas from radioactive ores) or are introduced by mining activity (e.g., diesel or blasting fumes, smoke from fires, all dusts). Even human breathing liberates a contaminant (carbon dioxide) while consuming oxygen, admittedly in small amounts.

The engineering principles *of mine air quality control,* for both gases and *dusts,* arc *the* same: *they* are

1. *Prevention or* avoidance
2. Removal or elimination
3. *Suppression or* absorption
4. Containment *or* isolation
5. *Dilution or reduction*

Different practices are employed to implement these control principles. For example, in the room and pillar mining of coal with continuous equipment, the two major contaminants are methane gas and coal dust. In this case, corresponding to the five control principles above, the following control practices would be considered for application:

	GAS	DUST	BOTH
1	Nonsparking bits Airway sealant	Sharp bits	Good mining practices Explosion-proof atmosphere
2.	Methane drainage	Dust collector Good housekeeping	Bleeders
3.	Sprayfans	Water sprays Rock dusting	Water infusion
4.	Sealing old workings	Hood enclosure	Separate split of air
5.	Main ventilation Auxiliary ventilation	Main ventilation Auxiliary ventilation	Main ventilation Auxiliary ventilation

While not all these principles and practices would be utilized in every mine, enough would t>e employed to cope with existing hazards and contain them within threshold limit values promulgated by MSHA and other health and safety authorities (Anon., 1984a. I984d). Ideally, they should be practiced in the order in which they are given; this results in the best quality control and cost effectiveness. Note that ventilation is listed last, not because it is least effective but because it is universally applicable and most effective when coupled with other measures.

The quantity of ventilation, *Q,* required to dilute an airborne hazard is determined by the following relation:

$$Q = \frac{Q_g\,(1 - \mathrm{TLV})}{\mathrm{TLV} - B_g} \qquad (12.1)$$

where Q_g is contaminant inflow' rate, is concentration of contaminant in normal intake air, and TLV is threshold limit value of contaminant.

Example 12.1. Calculate the air quantity necessary to dilute methane gas in a coal mine to i'ts TLV when the inflow rate is 250 ft^3/min (0.118 m^3/see) and its concentration in the intake air is 0.1%.

Solution. Consider the limit tor methane to be based on its explosibility (is also suffocating in high concentrations but nontoxic); MSHA prescribes the TLV = 1 %, at which work must cease during the exploitation stage (Anon., 1984a), Employing Eq. 12.1.

$$Q = \frac{(250)(1 - 0.01)}{(0.01 - 0.001)} = 25{,}000 \text{ ft}^3/\text{min } (11.8 \text{ m}^3/\text{sec})$$

Example 12.2. Calculate the air quantity necessary to dilute bituminous coal dust during continuous mining to its T LV when the dust generation rate is 2.5 g/min (0,0055 Ib/min) and the concentration in the intake air is 0.5 mg/m^3 (38.9 x 10^{-6})

SOLUTION. Consider the TLV for coal dust, set by MSHA, to be its respirable limit (2 mg/m^3) (Anon., 1984a). Expressed on a percentage basis the TLV becomes s o small that it may be omitted from the numerator in Eq.12.1. For consistency of units, f t is defined as a weight flow rate, and TLV and *B*, are weight concentrations (the calculation is carried out more easily in S.I. units):

$$Q = \frac{2.5 \times 10^3}{2 - 0.5} = 1667 \text{ m}^3/\text{min} = 27.8 \text{ m}^3/\text{sec } (58{,}900 \text{ ft}^3/\text{min})$$

Quantity Control

Quantity control in mine ventilation is concerned with supplying air of the desired quality and m the desired amount to all working places throughout the mine. Air is necessary not only for breathing-a remarkably low -20 ft^3/min (0.01 m /sec) per person usually suffices-but to disperse chemical and physical contaminants (gases, dusts, heat, and humidity) as well. Because breathing requirements are easily met, federal and/or state laws provide for a higher minimum quantity of air per person (100-200 ft^3/min or 0.05-0.09 m^3/sec), a minimum quantity at the face (3000 ft^3/min, or 1,4m^3/sec) or in the last crosscut of a coal mine (9000 ft^3/min, or 4.3 m^3/sec) or a minimum velocity at the coal face (60 ft/min, or 0.3 m/sec) (Anon., 1984a). Mine ventilation practice is heavily regulated in the United States as well as in the rest of the world, especially in coal and gassy (noncoal) mines, and other statutes relate to air quantities required to dilute diesel emissions, blasting fumes, radiation, dusts, battery emissions, and many other contaminants.

Rather surprisingly, legal requirements for minimum airflow seldom govern in determining the design quantity for the mine. Most mines operate with ventilation quantities far in excess of those legislated ; the *best* ventilated mines circulate million of ft^3/min (thousands of m^3/sec) of attaining ratios o f *the* weights of airflow to mineral produced of 10-20 tons/ton (tones /tonne)

The desi*gn* basis for specifying the amount of airflow in working places is usually the critical velocity or quantity required to disperse or dilute contaminants (airflow must he well into the turbulent range). If the quantity needed for dilution is inadequate for effective dispersion or cooling, then the velocity and hence the quantity is increased to an adequate level. (Economics plays a rofe also since excessive airflows squander fan horsepower.) Critical velocities at the working face range from 100-400 ft/m in (0.5 - 2.0 m/sec), unless cooling is a consideration, in which case velocities may attain 400-600 ft/min (2.0 - 3.0 m/sec). Knowing the area *A* and velocity *V,* the quantity can then be determined:

$$\mathbf{Q = VA} \qquad \mathbf{(12.2)}$$

EXAMPLE 12.3. Calculate the quantity of air required to dilute 80 ft^3/min (0,038 m^3 /sec) of carbon dioxide, a strata gas, in the stopes of a metal mine and to maintain a critical face velocity of 150 ft/min (0.76 m/sec). The cross sectional area of each stope is 15 x 20 ft (4.5 x 6.0 m) and the CO, content of the intake air is 0.03%. The TLV established by MSHA for C 0 2 is 0.5%.

SOLUTION .

1. Calculate the required dilution by Eq. 12.1:

$$Q = \frac{(80)(1 - 0.005)}{0.005 - 0.0003} = 16{,}900 \text{ ft}^3\text{/min } (7.98 \text{ m}^3\text{/sec})$$

2. Calculate thc required quantity for effective dispersion by Eq. 12.2:

$$Q = (150)(15)(20) = 45.000 \text{ ft}^3\text{/min } (21.24 \text{ m}^3\text{/sec})$$

3. Specify the critical quantity as thc larger value (item 2 governs):

$$Q - 45.000 \text{ ft}^3\text{/min } (21.24 \text{ m}^3\text{/sec})$$

Theoretically, contamination by the C 0 2 in human exhalation ought to be considered, too, but because it is so small *(Q g* = 0.1 ft^3/min, or 47 x 10^{-6} m^3/sec, per person), it can be neglected in this example.

Once quantity requirements in all the working places have been specified, it is then necessary to create a *pressure difference* in the mine to provide the desired flows. Either *natural ventilation,* in which the pressure difference results from thermal energy (like the chimney effect), or *mechanical ventilation,* in which the rotational energy of a fan is converted to fluid-flow energy, may serve as energy sources. In modern mines, only fans are relied upon to provide a pressure difference for ventilation (natural ventilation is too variable, unreliable, and insufficient in magnitude to be utilized). Both centrifugal and axial-flow fans are in common use, and they may be installed in either the blower, booster, or exhaust

position (except that underground booster fans are prohibited in coal mines). Axial – flow fans installed as – exhausters appear in Figure 12.17.

Calculating the pressure difference required over all working places and airways is prerequisite to the selection of a mine fan. Resembling the well – known Darcy equation from fluid mechanics, the Atkinson equation is generally utilized in mine ventilation:

$$H = \frac{KPLQ^2}{5.2A^3} \qquad (12.3)$$

where II is pressure difference, *K* is a friction factor, *P* is perimeter, and is length (actual length plus equivalent length due to shock loss).



Figure 12.17. Axial-flow fans installed in the exhaust position at the surface of a coal mine (By permission from Jeffrey Div.. Dresser Industries. Columbus. OH.)

Friction factors are selected from a table, experience, or actual measurement. So times, Eq. 12.3 is written in the form

$$H = RQ^2 \qquad (12.4)$$

where R *is airway resistance* = $KPL/5.2A^3$

EXAMPLE12.4. Given: a rectangular mine opening of cross-sectional dimensions 6 x 20 ft (1.8 x 6 m), length 7500 ft (2250 m), and friction factor 45×10^{-10} lb-min^2/ft4 (8.35×10^{-3} kg/m^3). Calculate the pressure difference for an airflow of 140.000 ft^3/min (66.1 m^3/sec).

Solution

$$P = 2(6+20) = 52 \text{ ft} \qquad A=(6)(20) = 120 \text{ ft}^2$$

Using Eq. 12.3,

$$H = \frac{(45)(10^{-10})(52)(7500)(14000)^2}{5.2(120)^3} = 3.83 \text{ in. water} \quad (953 \text{ Pa})$$

(Note: The inch of water is a unit of pressure measurement; 1 in. water = 5.2 lb/ft^3 , or 249 Pa.)

Mine ventilation circuits, like electrical circuits, arc arranged with airways in series or parallel or as combination series-parallel circuits called networks. Simple circuits can be solved mathematically using principles derived from electrical theory, but because of their complexity networks are best solved by computer and are beyond the scope of this discussion. In series circuits, H and *R* values are cumulative, and *Q* is constant. In parallel circuits, *Q* values arc cumulative and inversely proportional to $\sqrt{R}$ for a given airway, and *H* is constant. Referring to Eq. 12.4, for series circuits,

$$H = (R_1 + R_2 + \ldots + R_n)\, Q^2 = H_1 + H_2 + \ldots + H_n \qquad (12.5)$$

and for parallel circuits,

$$R_{eq} = \left(\frac{1}{\sqrt{1/R_1} + \sqrt{1/R_2} + \cdots + \sqrt{1/R_n}} \right)^2 \qquad (12.6)$$

$$Q_1 = Q \sqrt{\frac{R_{eq}}{R_1}} \quad \text{etc.} \qquad (12.7)$$

where R_{eq} is the equivalent circuit resistance.

EXAMPLE 12.5. Four airways arc arranged as a series circuit. Their resistances, in units of 10^{-10}in.-min^2/ft^6 (N – sec^2/m^8), are

23.50 (2.627)
1.35 (0.151)
3.12 (0.349)
3.55 (0.397)

If the quantity is 100 000 ft^3/min (47.2 m^3/sec), find the pressure difference across the circuit.

Solution. Use Eq. 12.5:

$$H = (23.50 + 1.35 + 3.12 + 3.55)(10^{-10})(100.000)^2 = 31.52 \text{ in. water } (7.84 \text{ kPa})$$

Example 12.6. If the airways of Example 12.5 are rearranged as a parallel circuit, calculate the pressure difference across the circuit and the quantity of airflow in airway I.

SOLUTION. Use Eqs. 12.6, 12.4, and 12.7:

$$R_{eq} = \left(\frac{1}{\sqrt{1/23.5} + \sqrt{1/1.35} + \sqrt{1/3.12} + \sqrt{1/3.55}}\right)^2 (10^{-10})$$

$$= 0.214 \times 10^{-10} \text{ in.-min}^2/\text{ft}^6$$

$$H = (0.214)(10^{-10})(100{,}000)^2 = 0.214 \text{ in. water } (53.25 \text{ Pa})$$

$$Q_1 = 100{,}000 \sqrt{\frac{0.214 \times 10^{-10}}{23.5 \times 10^{-10}}} = 9540 \text{ ft}^3/\text{min } (4.50 \text{ m}^3/\text{sec})$$

The *H-Q* relation (Eq. 12.4) for an airway is termed its *characteristic;* for the entire mine, it is called the *mine characteristic.* A graphical plot of pressure difference vs. quantity results in a parabolic-shaped curve; we refer to it as the *mine characteristic curve.*

Example 12.7. Calculate and plot the characteristic curve of the mine of Example 12.4.

SOLUTION. Given H = 3,83 in. water for Q= 140,000 ft^3/min, calculate several points by Eq. 12.4:

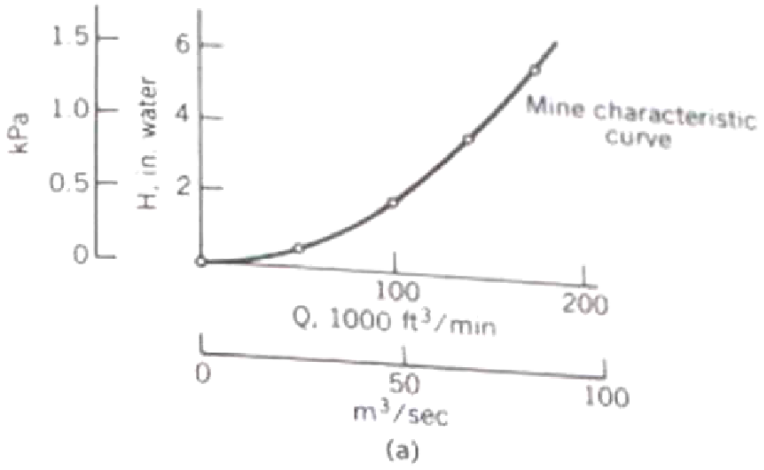
Q, ft^3/min (m^3/sec)	H, in. water (Pa)
0 (0)	6.0 (1493)
50,000 (23.6)	6.6 (1642)
100,000 (47.2)	6.3 (1568)
150,000 (70.8)	5.5 (1369)
200,000 (94.4)	4.0 (995)
250,000 (118.0)	2.1 (523)
300,000 (141.6)	0 (0)

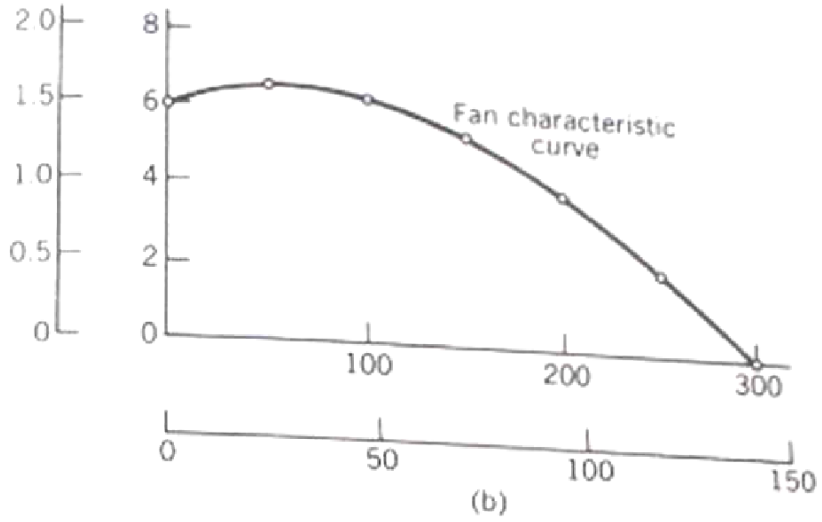
Plot the fan characteristic curve.

SOLUTION. See the fan curve in Figure 12.18b.

Characteristic curves are used mainly to determine the system operating point when a fan is connected to a mine. The *operating point* is the combination *of* pressure difference and quantity at which the fan and the mine are in equilibrium. *If the* fan and mine characteristic curves are plotted to scale on the same *H-Q*

graph, the operating point lies at their intersection. It is a useful procedure to employ for the selection of a fan.





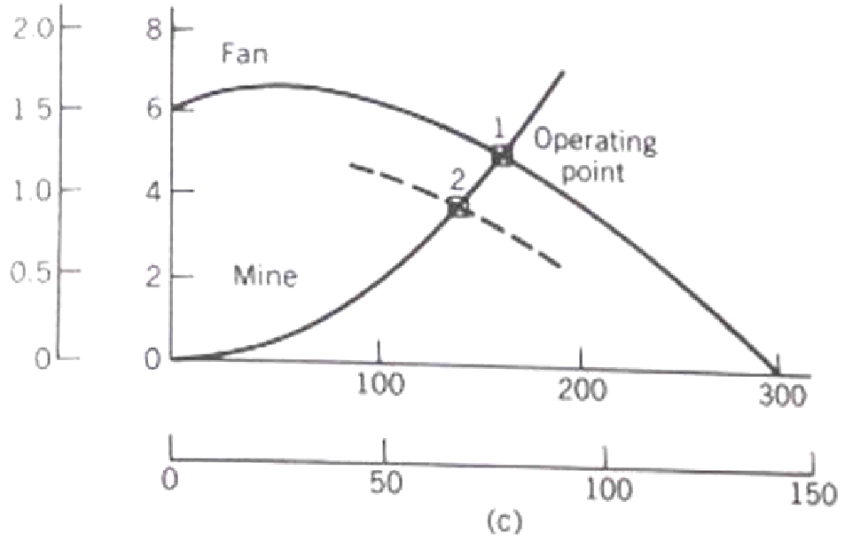


Figure 12.18. Mine and fan characteristic curves. See Examples 12.7, 12.8, 12.9 and 12.10.

Example 12.9. If the mine and fan whose characteristic curves arc depicted in *Figure* 12.18a and b comprise a ventilation system, determine the operating point graphically.

SOLUTION . See the curves in Figure 12.18c. Read at operating point 1*:* Q=160.000 ft^3/min (75.5 m^3/sec) and H= 5.0 in. water (1244 Pa).

In selecting a fan for a given mine, it may be the ease that no fan curve intersects the mine curve at the required pressure difference and quantity. The fan's performance may be modified, within limits, by changing its rotational speed, blade pitch (if axial-flow), or its inlet vanes (if centrifugal) – or additional fans may be installed in the system. Performance curves or data are necessary to predict the effects on fan performance of changing the latter two parameters, but fan laws enable us to determine the effect o f a change in speed *n:*

$$\frac{Q_2}{Q_1} = \frac{n_2}{n_1} \quad \text{and} \quad \frac{H_2}{H_1} = \left(\frac{n_2}{n_1}\right)^2 \qquad (12.8)$$

Thus if the performance (operating point or characteristic curve) of a fan is desired at another speed, the *H-Q* relation is readily determined by Eq. 12.8. Likewise, the rotational speed o f a fan required to deliver a specified pressure difference and quantity o f airflow can also be found by the same equation.

EXAMPLE 12.10. If the speed o f the fan whose characteristic curve is plotted in Figure 12.18c is 1750 rpm, determine the speed at which it must operate to deliver Q = 140.000 ft^3/min (66.1 m^3/sec) and H= 3.83 in. water (953 Pa) when connected to the mine whose characteristic curve is also plotted in Figure 12.18c.

SOLUTION. Plot operating point 2 for the *H-Q* relationship specified (Fig .12.18c). Calculate the required speed n_2 by *Eq.* 12.8:

$$n_2 = n_1\left(\frac{Q_2}{Q_1}\right) = 1750\left(\frac{140{,}000}{160{,}000}\right) = 1531 \text{ rpm}$$

The entire fan curve can be transposed to the new speed if desired by applying Eq.12.8 (a portion is shown).

If many fan and mine curves are involved, as when selecting the most economical fan over the life of a mine, then the graphical simulation or algebraic solution o f intersecting curves is best carried out by computer.

PROBLEMS

12.1 Calculate the minimum width of undercut required in a block caving mine to insure a successful cave, given the following:

Stress field	No lateral strain
Rock strength, compressive	9700 lb/in.^{1} (66.9 MPa)
tensile	810 lb/in.^{2} (5.6 MPa)
Depth of cover	1500 ft (460 m)
Undercut height	12 ft (3.7 m)
Fillet ratio	$\frac{1}{6}$

12.2 Nitrogen gas occurs as a normal constituent (78,09%), *of* the atmosphere we breath and may be found as a strata gas in mines. If nitrogen is liberated of the atmosphere we breathe and may be found as a strala gas in mines. If nitrogen is liberated at the rate of 8.5 ft^3/min (0.00401 m^3/sec) in a mine and the TLV is 80 %, calculate the *quantity of* drillhole required to maintain air quality standards.

12.3a A circular airshaft 8ft (2.4 m) in diameter transmits an airflow of 250 000 ft^3/min, (118 m^3/sec) into a mine when a fan is connected to it. *If a si*milar new shaft 10ft (3 m) in diameter having the same length, friction factor, and prepare difference it sunk to replace the original shaft, what airflow will result? Disregard the rest *of* the mine.

b. The pressure difference for a mine is 4.0 in water (995 Pa) with an air quantity of *200.000* ft^3 /min *(94.4* m^3/sec).What is the pressure difference if the flow is halved?

12.4a. A circular ventilation shaft is to be sunk with the following characteristics.

Diameter	16 ft (4.9 m)
Length	1400 ft (427 m)
Friction factor	25 x 10^{-29} lb-min^2/ft^4 (4.6 x $10^{-3}$$kg/m^3$)

Calculate the pressure differential with an airflow of 215,000 ft' min (101 m'/sec).

b. If the remainder of the mine consists of a main shaft and haulageway having a pressure difference of 0.41 in. water (102 Pa), determine the overall pressure difference if the ventilation shaft is connected in series to the rest of the mine.

12,5a. Develop an expression to calculate the pressure difference across multiple, identical, parallel airways (number *N)* in a coal mine, neglecting the effect of crosscuts, seam irregularities, and so on. Start with Eq. I2.3 for the pressure difference in a single airway.

b. Using your expression from part a. calculate the pressure difference in the water in the following multiple airways:

N = 5 airways
Cross section = 5 x 18 ft (1,5 x 5.5 m)
K = 30 x 10^{10} lb - min^2/ft^4 (5.5 x 10 kg/m^3)
L = 3900 ft (1190 m)
Q = 175.000 ft^3 /min (82.6 m^3/sec), total flow

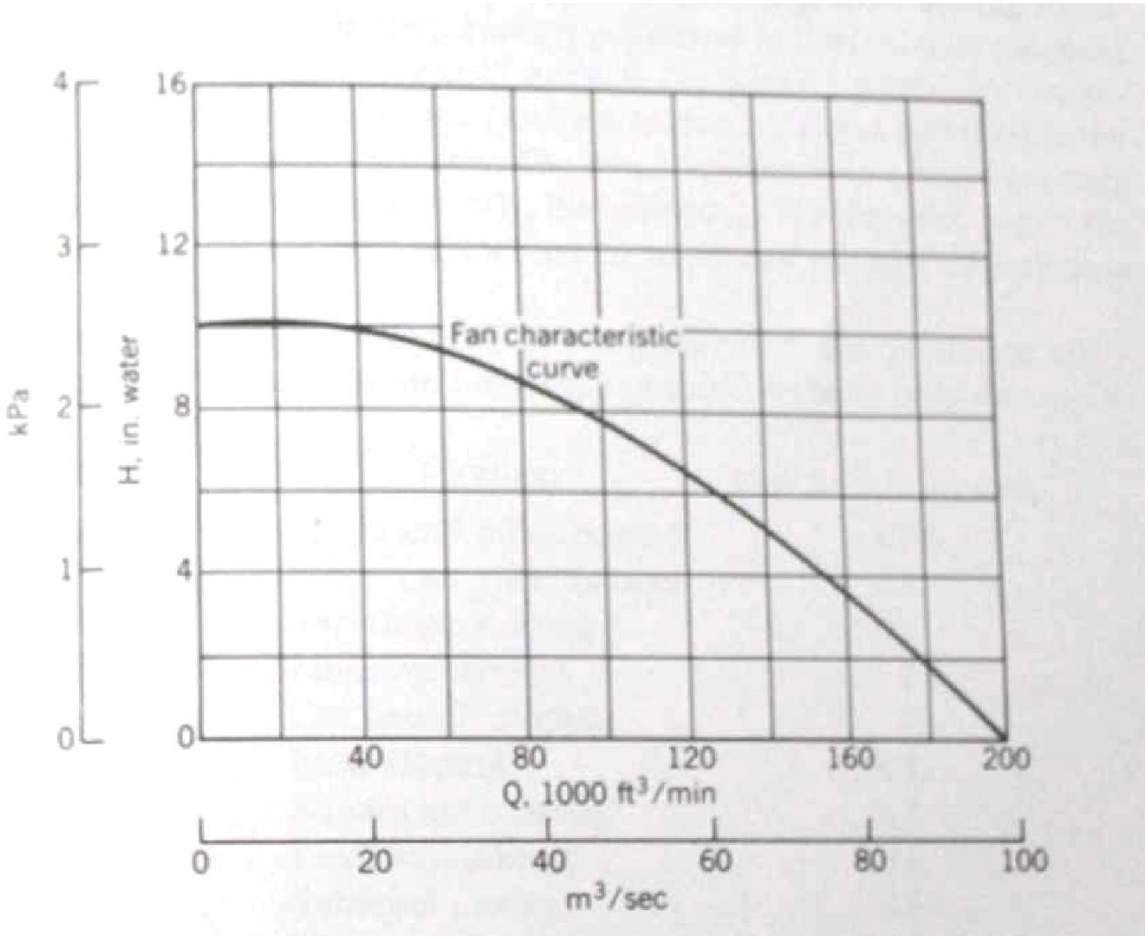


Figure 12.19. Fan characteristic curve. See Problems 12 6 and 12 7.

c. A fan whose characteristic curve is also plotted in Figure 12.19 is available to supply air in the mine and overcome its pressure difference. Determine the operating point of the system (mine + fan) graphically, and read the actual quantity of air and pressure difference in the system.

d. If the desired H-Q does not result, what speed change will be necessary for the fan if the original speed is 2800 rpm?

13 .
UNDERGROUND MINING: METHOD COMPARISON AND SUMMARY

13.1 METHOD RECAPITULATION

In outline and format, this chapter summarizing underground mining parallels Chapter 8 on surface mining. Having examined in the last four chapters the development and exploitation of mineral deposits by traditional underground, methods, we are now in a position to compare and summarize the principal features of these methods. The result will provide a basic not only for comparison of all mining methods, but ultimately for the selection of the most suitable and cost – effective method to mine any mineral deposit (see Chapter 15).

Reiterating the portion of Table 3.1.a classification of exploitation methods, concerned with underground mining, ten methods have been discussed:

CLASS	METHOD	percent U.S. USE
Unsupported	Room and pillar mining	62%
	Stope and pillar mining	15
Supported	Shrinkage stoping	<1
	Sublevel stoping	7
	Cut and fill stoping	3
	Stull stoping	<1
Caving	Square set stoping	<1
	Longwall mining	11
	Sublevel caving	<2
	Block caving	3_ -100%

The approximate usage percentages refer to current U .S. mineral production and the extent to which each method finds application underground.

In this recapitulation of underground methods, we compare their sequence of development, cycle o f operations, conditions o f use, characteristics, and importance.

13.2 *SEQUENCE OF DEVELOPMEN T*

As explained in Section 9.1, the critical factors in underground development are natural and geologic: ore and rock strength, the presence of groundwater and the rock-temperature gradient. Also important are certain social economic - political-environmental considerations such as availability of a trained work force, financial constraints, antipollution legislation, subsidence restrictions, and underground atmospheric standards. Referring again to the 11 general steps in mine development (Section 3.2), those aspects which find unique expression in underground mining are step 5 (environment) and step 5 (access openings).

Environmental Restrictions

Surface subsidence and a life-threatening atmosphere are the peculiar environmental hazards encountered in underground mining. Restrictions against subsidence may limit recovery with unsupported methods, dictate more extensive ground-control measures with supported methods, and rule out caving methods altogether: controlled subsidence where acceptable usually requires modifications in the mining method. Elaborate m.ne ventilation systems are often necessary to provide adequate control of the quality, quantity and temperature-humidity of the underground atmosphere (Section 12.5).

Access Development

Much of Chapter 9 was devoted to a discussion of the special provisions for access openings required in underground mining. The types, purposes, siting, and construction of development openings for primary, secondary, and tertiary access were described. Their specialized nature and inordinately high construction cost argue for taking special pains with underground mine layout.

Pattern of Development Because underground exploitation methods spatially tend to fall in either of two planes, horizontal or vertical, mine development similarly follow's a horizontal or vertical pattern (inclined openings are less commonplace). The two most prevalent underground methods— room and pillar mining and stope and pillar mining— along with longwall mining are essentially horizontal methods, and hence most of the related development conforms to a horizontal pattern. All other underground stoping and caving methods and many of their associated developmet openings are vertical in nature. In all cases, however, primary or main openings (shafts, slopes, declines, etc.) must be sunk nearly vertical to the mineral deposit and certain secondary openings (entries, drafts, etc.) driven nearly horizontal in the deposit to provide necessary access. Thus the pattern of development in an underground mine is commonly a combination of vertical and horizontal openings, standardized and limited in number to restrict costs.

13.3 CYCLE OF OPERATIONS

Unit operations in underground mining are grouped into (1) production and (2) auxiliary cycles; they are further distinguished as (1) development and (2) exploitation relative to the stage of mining.

Production Operatio n s

In Table 9 .1 and Figure 9.9a-j, a classification of underground equipment and unit operations was presented for the production cycle, distinguishing applications for cyclic and continuous mining and for various types of development and exploitation openings. That information arranged according to a different format in Table 13.1, permitting common cycle of operation for the principal underground methods to be summarized and compared as follows (symbols relate to Table 13.1):

Unsupported methods													
Room and pillar mining	**Bla**	+	Ala	+	Alb		A2a	+	A2h	+	AV		
Stope and pillar mining	B2a	+	A2a	+	A2b	+	**4V**						
	Ala	+	Alb	+	Ate	4.	A2a	+	A2b	+	A2c		
Shrinkage stoping Sublevel	B2c	+	A2a	+	A2b	+	A2c						
stoping Supported methods	Ala	+	Alb	+	Ale	+	A2a	+	A2b	+	A2c		
	Ala	+	Alb	+	Ale		A2a	+	A2b	+	A2c		
Cut and fill stoping	Ala	+	Alb	+	Ale	4.	A2a	+	A2b	+	A2c		
Stull stoping	Ala	+	Alb	+	Ale	+	A2a	+	A2b		A2c		
Square set stoping	Ala	**j.**	Alb	+	Ale	+	A2a	+	A2b	+	A2c		
Caving													
Long wall mining	B2b	+	A2a	+	A2b	+	A2c						
Sublevel caving	Ala	+	Alb	+	Ale	+	A2a	+	A2b	+	A2c		
Block caving	Ale	+	A2a	+	A2b	+	A2c						

(Note: Development is standardized for most methods in coal and hard rock; see Table 9.1. In block caving, operations are not listed for undercutting, more properly a development activity. Continuous mining is uncommon in hard rock and appears under the stope and pillar method only as an example; on the other hand, it is used exclusively in longwall mining of coal and other soft minerals.)

TABLE 13.1 Summary of Unit Operations Comprising Different Production Cycles in Underground Mining

Unit Operation	Condition	Equipment
A. Standard Cycle		
1. Rock breakage		
a. Drilling	Horizontal	Jumbo, airleg, drifter, auger rotary
	Vertical up	Stoper, fandrill
	Vertical down	Jumbo, down-hole, roller-bit rotary
b. Blasting	Dry	ANFO, permissible, airdox, cardox
	Wet	Slurry, dynamite
c. Secondary blasting	Stoping	Drill and blast, impact hammer, drop ball, shaped charge, bomb, mudcap
2. Materials handling		
a. Excavation	Stoping	Gravity-flow (chutes, drawpoints)
	Transfer	LHD, front-end loader, shovel, overhead loader, gathering-arm loader, slusher
b. Haulage	Horizontal	Truck, shuttle car, LHD, slusher, rail, belt conveyor, chain conveyor
c. Hoisting	Vertical, inclined	Skip, cage, conveyor, pipeline
B. Modified Cycle		
1. Breakage (kerf)		
a. Cutting	Coal, soft mineral	Shortwall, universal cutting machine
2. Continuous mining		
a. Room and pillar	Coal, soft mineral	Ripper, boring, milling, auger, drum machine
b. Longwall	Coal, soft mineral	Plow, shearer, water jet
c. Stoping	Soft, medium rock	Tunnel-boring machine, roadheader, raise borer, shaft borer

Auxiliary Operations

Auxiliary tasks performed in underground mining arc listed in their entirety in *Section 10.2 for* coal mining and modified in Section 10.3 for hard-rock mining. Those peculiar to the underground regime are the following:

1. Ventilation and air quality control
2. Subsidence control
3. Roof and ground control, backfilling
4. Water and flood control
5 .Waste disposal
6. Lighting
7. Power distribution
8. Communications
9. Personnel transport

13.4 GEOLOGIC AND NATURAL CONDITIONS

Geologic, natural and spatial conditions are crucial to the optimal selection and utilization of underground mining methods. They are summarized in Table 13.2, based on requirements for the individual methods set forth in Chapters 10.11, and 12. As with the similar list for surface methods in Table 8.2, it is not presented in detail but is useful for comparison of the compatibility of particular methods and conditions.

With regard to the applicability of underground methods, certain general observations may be offered (compare with the list for surface methods in Section 8,4):

1. Underground mining is versatile with regard to depth (shallow to deep) and ore and rock strengths (weak to strong); but it is somewhat restrictive with regard to deposit shape, dip, and size and ore grade and uniformity.
2. Within economic limits, some undcrground method can be applied to the mining of almost any deposit, regardless of natural conditions, and at any depth.
3. Undcrground mining, with one exception (block caving), is not well suited to the exploitation of large, low-grade disseminated orc bodies.
4. Unsupported methods are thc most widely used, adaptable to modified surface mining equipment, and amenable to mechanized, mass-production techniques.
5. Supported methods are most appropriate for difficult natural conditions but are less productive and have fallen into disuse.
6. Caving methods are highly productive and low-cost, but restrictive in application because of size requirements and subsidence.

Selection of Methods

Although, in general, underground mining cannot economically compete with surface mining at very shallow depths (a few hundred feet or meters), underground methods lend to compete only with one another at moderate to great depth (thousands of feel or meters). While all the geologic, natural, and spatial factors identified in Table H.2 arc operative, those that tend to he most critical in the decision-making process are (I) ore and rock strengths and (2) deposit shape, dip. and size. Pending a complete feasibility analysis of all the technological, economic, and safety considerations, preliminary selection of candidate underground methods often originates with these factors. Brown (1985) and others have devised a quantitative approach to mine design that utilizes rock mechanics principles.

Before proceeding, it is useful to quantify thc relative terms we have used thus far for ore and rock strength. Typical values with examples are given in Table 1.1.3. Using these mineral and rock types and/or their compressive strengths, applicable candidate methods may be identified in Tables 8.2 or 13.2 or in the tables that follow.

To facilitate the tentative selection process, it is helpful to display factors and methods in decision matrices, based on Table 13.2 but rearranged. Several such charts have been devised, and two of the more useful for underground methods are reproduced here.

TABLE 13.2 Comparison of Deposit Conditions Favorable to Underground Methods

	Unsupported			
Factor	Room and Pillar	Stope and Pillar	Shrinkage	Sublevel
1. Ore strength	Weak to moderate	Moderate to strong	Strong (should not pack)	Moderate to strong
2. Rock strength	Moderate to strong	Moderate to strong	Strong to fairly strong	Fairly strong to strong
3. Deposit shape	Tabular	Tabular, lenticular	Tabular, lenticular	Tabular, lenticular
4. Deposit dip	Low, preferably flat	Low to moderate	Fairly steep	Fairly steep
5. Deposit size	Large, thin	Any, preferably large, moderately thick	Thin to moderate	Fairly thick to moderate
6. Ore grade	Moderate	Low to moderate	Fairly high	Moderate
7. Ore uniformity	Fairly uniform	Variable	Uniform	Fairly uniform
8. Depth	Shallow to moderate	Shallow to moderate	Shallow to moderate	Moderate

TABLE 13.2 (continued)

	Supported			Caving	
Cut and Fill	Stull	Square Set	Longwall	Sublevel Caving	Block Caving
Moderate to strong	Fairly strong to strong	Weak to fairly weak	Any (should crush, not yield)	Moderate to fairly strong	Weak to moderate, cavable
Weak to fairly weak	Moderate	Weak to very weak	Weak to moderate, cavable	Weak to fairly strong, cavable	Weak to moderate, cavable
Tabular to irregular	Tabular to irregular	Any	Tabular	Tabular or massive	Massive or thick tabular
Moderate to fairly steep	Moderate to fairly steep	Any, preferably steep	Low, preferably flat	Fairly steep	Fairly steep
Thin to moderate	Thin	Any, usually small	Thin, large areal extent	Large, thick	Very large, thick
Fairly high	Fairly high to high	High	Moderate	Moderate	Low
Moderate, variable	Moderate, variable	Variable	Uniform	Moderate	Fairly uniform
Moderate to deep	Moderate	Deep	Moderate to deep	Moderate	Moderate

TABLE 13.3 Classification of Ore and Rock Strength

Mineral or Rock	Relative Strength	Compressive Strength, lb/in.² (kPa)
Coal, decomposed and badly altered rock	Very weak	<6,000 (<40,000)
Friable sandstone, mudstone, weathered rock, soft shale	Weak	6,000–14,500 (40,000–100,000)
Shale, limestone, sandstone, schistose rock	Moderate	14,500–20,000 (100,000–140,000)
Most igneous rock, strong metamorphic rock, hard limestone and dolomite	Strong	20,000–32,000 (140,000–200,000)
Quartzite, basalt, diabase	Very strong	>32,000 (>220,000)

Source: Modified after Hamrin, 1982.

TABLE 13.4 Selection Chart for Noncoal Underground Mining Methods

	Deposit Properties						Deposit Geometry							
	Ore Strength			Rock Strength			Beds		Veins			Dip or Inclination		
Method	Weak	Moderate	Strong	Weak	Moderate	Strong	Thin	Thick	Narrow	Wide	Masses	Flat	Moderate	Steep
Stope and pillar		X	X		X	X	X	X				X	X	
Shrinkage stoping		X	X		X	X			X	X			X	X
Sublevel stoping		X	X		X	X			X	X	X			X
Cut and fill stoping		X	X	X	X				X	X	X		X	X
Stull stoping		X	X		X				X				X	X
Square set stoping	X			X	X				X	X	X		X	X
Sublevel caving		X	X	X	X					X	X			X
Block caving	X	X		X	X					X	X			X

Sources: Modified after Anon., 1974; Halls, 1982.

Noncoal Deposits. Table 13.4 displays information from Table 13.2 in a different format. Applicable methods are limited to hard – rock mining.

Coal and Noncoal Deposits. The most widely applications of the various selection methods is also the oldest, originated by Peele in 1941 and modified by Lucas and Haycocks (1973) and Thomas (1978). It is shown in Table 13.5 and most closely resembles a decision matrix. To use the table, one proceeds from left to right, considering successively factors of (1) deposit geometry and (2) deposit strength. Candidate methods are identified in the right – hand column. Because comparative terms employed in Tables 13.2, 13.4, and 13.5 differ, there is not complete agreement among the various approaches.

More elaborate and sophisticated selection procedures – applicable to both surface and underground methods – are reserved for Chapter 15.

TABLE 13.5 **Selection Chart for All Underground Mining Methods**

Deposit Shape	Deposit Orientation	Deposit Thickness	Ore Strength	Rock Strength	Applicable Method(s)
Tabular	Horizontal, flat	Thin	Strong	Strong	Room and pillar mining, stope and pillar mining
			Weak, strong	Weak	Longwall mining
		Thick	Strong	Strong	Stope and pillar mining
			Weak, strong	Weak	Sublevel caving
	Vertical, steep	Thin	Strong	Strong	Shrinkage stoping, sublevel stoping
			Strong	Weak	Cut and fill stoping, square set stoping, stull stoping
			Weak	Strong	Square set stoping
			Weak	Weak	Square set stoping
		Thick	Strong	Strong	Shrinkage stoping, sublevel stoping
			Strong	Weak	Cut and fill stoping, sublevel caving, square set stoping
			Weak	Strong	Sublevel caving, block caving, square set stoping
			Weak	Weak	Sublevel caving, block caving, square set stoping
Massive	—	—	Strong	Strong	Shrinkage stoping, sublevel stoping
			Weak	Weak, strong	Sublevel caving, block caving, square set stoping

Sources: Modified after Peele, 1941; Lucas and Haycocks, 1973; Thomas, 1978.

TABLE 13.6 Comparison of Advantages and Disadvantages of Underground Methods

	Unsupported				Supported			Caving		
Characteristic	Room and Pillar	Stope and Pillar	Shrinkage	Sublevel	Cut and Fill	Stull	Square Set	Longwall	Sublevel Caving	Block Caving
1. Mining cost	30%	30%	50%	40%	60%	70%	100%	20%	50%	20%
2. Production rate	Large	Large	Moderate	Large	Moderate	Small	Small	Large	Large	Large
3. Productivity	High	High	Low	High	Moderate	Low	Low	High	Moderate	High
4. Capital investment	High	Moderate	Low	Moderate	Moderate	Low	Low	High	Moderate	High
5. Development rate	Rapid	Rapid	Rapid	Moderate	Moderate	Rapid	Slow	Moderate	Moderate	Slow
6. Depth capacity	Limited	Limited	Limited	Low	Moderate	Limited	Unlimited	Moderate	Moderate	Moderate
7. Selectivity	Low	High	Moderate	Moderate	High	High	High	Low	Low	Low
8. Recovery	Moderate	Moderate	High	Moderate	High	High	Highest	High	High	High
9. Dilution	Moderate	Low	Low	Moderate	Low	Low	Lowest	Low	High	High
10. Flexibility	Moderate	High	Moderate	Low	Moderate	High	High	Low	Moderate	Low
11. Stability of openings	Moderate	High	High	High	High	Moderate	High	High	Moderate	Moderate
12. Subsidence	Moderate	Low	Low	Low	Low	Moderate	Low	High	High	High
13. Health and safety	Good	Good	Good	Good	Moderate	Moderate	Poor	Good	Good	Good
14. Other	Highly mechanized, caves with pillar recovery, good ventilation	Mechanized, fair ventilation	Gravity flow in stope, labor intensive, simple method	Gravity flow in stope, mechanized, large blasts, good ventilation	Gravity flow in stope, mechanized, requires backfill	Gravity flow in stope, labor intensive, simple method	Gravity flow in stope, labor intensive, high timber cost	Highly mechanized, continuous, rigid, expensive moves	Mechanized, draw control critical	Gravity flow in undercut, low breakage cost, good ventilation, draw control critical

13.5. Advantages and disadvantages

The same kind of comparison of features can be drawn for underground as for surface methods, with a few minor differences. Table 13.6 summarizes the advantages and disadvantages of the various underground methods; it is similar to Table 8.3 for surface methods. Definitions developed in Section 8.5 are applicable, with the following modifications (see characteristics as numbered in Table 13.6):

Item11. *Stability of openings:* refers to relative ease or difficulty of maintaining mine openings during exploitation: more broadly termed ground control.

Item12. *Subsidence (replaces environmental risk):* caving to the surface the most serious environmental hazard associated with underground mining: varies with the class of methods.

*(Note; It*em 13 in Table 8.3. waste disposal, is omitted in Table 13.6 because disposal of mine waste with underground methods is a relatively, problem, nearly independent of the method.)

Because most comparisons arc expressed qualitatively only, the student is referred to Chapters It). II. and 12 for more detailed, quantitative information.

13.6 IMPORTANCE AND SUMMARY

In our introduction to the topic of underground methods (Section 9.1), we conceded that il one is seeking maximum production at minimum cost, surface mining would nearly always win hands down any contest with underground mining. In summarizing the importance of surface mining (Section 8,6). we further made the claim that, viewed against the most decisive criteria— safely, technology, and economics—surface mining is almost always the easy winner.

While true, these statements are not conclusive or final in that they overlook (1) limitations in the suitability of a mineral deposit for surface mining and (2) the variety and versatility of underground mining methods. And while surface mining is over five limes as popular as underground mining in the United States, Section 9.1 identifies 10 important minerals that are produced in major or significant amounts by underground methods. Factors (Section S.6) that may favor underground mining for a given mineral deposit are excessive depth, high capital investment, poor selectivity, high environmental risk or land-reclamation costs, or prohibitive waste-disposal requirements. Other factors that tend to support underground mining are modest reserves, limits on dilution, adverse climate, or lack of availability of water for surface aqueous methods.

Under these special conditions, then, we conclude that underground mining may *offer the* better— if not the only— exploitation alternative in method *selection.* Surprisingly, however, even in the arena *of* mine production rate, *underground* bulk methods at their best compete respectably with surface mining.

For the most productive of underground mining methods, peak annual *production* rates have been recorded as follows:

METHOD	MINE	MINERAL	PRODUCTION/YR, 10* TONS (10^s)	
Sublevel caving Mock caving	Kiininuvaara. Swed.	Iron	24.0	(21.8)
	San Manuel, AZ	Copper	15.6	(14.2)
Slope and pillar	While Pine. Ml	Copper	6.9	(6.3)
Room and pillar	Pittsion Moss No. 3. VA	Coal	4.3	(3.9)

Coupled with waste output in addition to these figures, underground mining compares quite favorably with maximum annual tonnages of 32 million tons (29 million tonnes) for open pit mining and 20 million tons (18 million tonnes) for open cast.

Granted, the crucial criterion of comparison is cost, and it favors underground mining only under extraordinary circumstances. In certain instances today for example, in the coal industry—longwall can compete costwise with open cast mining. Likewise, in the copper industry, block caving occasionally rivals open pit mining.

For the variety of reasons cited in Section 9,1, there is cautious optimism about rising future trends in the conversion of surface to underground mining. Two substantive studies (Pfleider. 1973c; Anon., 1974) support that contention. Pfleider points out. for example, that while U.S. underground mineral production is only 15% of the total, world underground production is 32%. Whether underground mining regains (he prominence it once held remains to be seen.

After discussion of novel mining methods, we will be in position in the final chapter to examine more general method-selection procedures and to compare all current mining methods.

13.7 SPECIAL TOPIC: DETERMINATION OF MINING COSTS AND PROFITS

Several preceding chapters have contained sections (2.8, 3.5, 3.8, 5.4, 6.2, 8.7, 11.5) dealing with cost estimation for various unit operations in mining. Now that the nomenclature and procedures are well established, it remains to employ the entire process to determine overall production costs and profits of a complete mining operation. Reiteration of terms introduced in Section 3.8 and reference to a financial source like Gentry and O'Neil (1984) are helpful.

Unit vs. Total Coat Bases

We continue to base all calculations on unit (S/ton. or $/tonne) costs These can readily be converted to a total basis if needed.

Direct vs. Indirect Costs

When equipment is involved in a mining activity, the direct cost is calculated $_s$ the sum ot the ownership and operating costs. Indirect or overhead charges are usually computed as a percentage of the direct cost.

Overall Mining Cost

The overall mining cost equals the sum of the direct and indirect costs for all unit operations or activities in the cycle and for all stages of mining. (As an approximation or check on ihe exploitation cost, the method outlined in Section 6.2 can be utilized.)

PRODUCTION COST

If lo the mining cost are added all other costs of producing mineral (beneficiation. smelting, refining, transportation, etc.), the overall production cost results* Miscellaneous costs—plant depreciation, royalty charges, and taxes (other than income)—should be included, too, if applicable.

Overall production costs = mining cost + other production costs + miscellaneous costs

Value

The exact value of the product from a mineral operation can only be determined if a smelter schedule or sales contract is available. Lacking those, value estimates can be based on published market prices for the specified mineral commodity in the area under consideration. For metallic and non- metallic ores, market information is published monthly in *Engineering and Mining Journal* and weekly in *Metals Week*. For coal, prices are listed in *Coal Week.* (All three journals are published by McGraw-Hill Publishing Co New York.) Usually, a unit value calculation is based on grade or coal quality and the estimated overall recovery.

Profit

Knowing the value of a mineral product and the costs to produce it, the gross profit can be calculated on a unit basis. If the federal depletion and income tax rates arc then applied, the net profit (or loss) results.

Gross profit = value - cost
Net profit - gross profit - taxes + depletion allowance

EXAMPLE 13.1. The following cost data for the unit operations and stages of mining and other production activities and charges have been determined at an underground copper mine; for simplicity's sake in these calculations, the unit operations costs have already been estimated.

Rock breakage	$1.10/ton	($1.21/tonne)
Materials handling	1.45	(1.60)
Auxiliary operations	0.30	(0.33)
Indirect charges	10 %	
Prospecting and exploration	1-50	(1.65)
Development	2.65	(2.92)
Plant depreciation	1-85	(2.04)
Processing (beneficiation. smelting, refining)	8.50	(9.37)
Transportation	6.30	(6.94)
Royalty	2.25	(2.48)
Federal income tax rate	34%	

The ore grade is *2.2%* copper, estimated recovery is 85%, and the prevailing market price is $0.80/lb ($1.76/kg). Determine all costs, the value of the ore, and the gross and net profits.

SOLUTION. Calculate mining and production costs:

Direct exploitation cost	= S1.10 + 1.45 + 0.30 = S 2.85/ton
Indirect exploitation cost	= (0.10)(2.85) = 0.29
Exploitation cost	= $ 3.14/ton
Prospecting and exploration cost	=1.50
Development cost	= 2.65
Overall mining cost	= $ 7.29/ton
Other production costs	= 58.50 + 6.30 = 14.80
Miscellaneous costs	= $1.85 + 2.25 = 4.10
Overall production costs	= $26.19/ton ($28.81 / tonne)

Calculate ore value :
Unit value = ($0,80/1b) (0.022) (2000lb/ton) (0.85)= $29.95/ton ($32.98/tonne)

Calculate profile

Gross profit = $29.92 - 26.19	= $ 3.73/ton
Income tax = (0.34)(3.73)	=1.27
Net profit	= $ 2.46 / ton ($2.71/tonne)

Similar computations may involve the determination of cutoff grade and break-even market price (e.g., see Examples 2.1 and 2.2). Also, depletion

allowance has been neglected in Example 131 (see Section 3.5). Its effect, if applicable, would be to increase the net profits of the mining company by reducing the taxable income. For calculation procedure, see Halls (1982).

Example 13.1 dealt with a metal commodity and yielded a modest profit, But profits in the coal industry are similarly low. Currently, the overall production cost in coal mining is estimated to be $20-30/ton ($22 – 33 /tonne), with sales prices in the $20-35 ($22-39) range. Both cases demonstrate the mining adage that – "cost follows price" – unfortunately, not vice versa.

PROBLEMS

13.1. The following data have been estimated in the feasibility study of a zinc deposit:

Grade	3.4%
Recovery	80%
Overall production cost, tax, royalty	522.87/ton ($25.21/tonne)
Market price	44.19/lb (97.42/kg)

a. Calculate the break-even market price (e/lb, or e/kg) of zinc metal.
b. Calculate (he cutoff grade *(%)* of zinc ore.

13.2. A uranium deposit is available for purchase. The following financial data are provided:

Mining cost	$ 8.00/ton	($8.82/tonne)
Milling cost	6.00	(6.61)
Transportation cost	7.50	(8.27)
Taxes	3.50	(3.86)
Royalty	5.00	(5.51)
Government support price (U_3O_8)	$30.00/lb	(S66.14/kg)
Recovery	90%	

What is the cutoff grade, in lb/ton (kg/tonne) and percent?

13.3. Given the following operating anti cost data, estimate all the unit mining and production costs and unit profit (in $/ton. or S/tonne) for an underground zinc mine using the stope and pillar method. Specifically, determine the following (ail except item I arc expressed on a unit basis in $/ton, or $/tonne):

a. Productivity, in tons (or tonnes) per employce-shift
b. Unit operation costs for breakage, handling, and auxiliary purposes
c. Direct exploitation cost
d. Indirect (overhead) cost
e. Overall exploitation cost
f. Overall production cost
g. Gross and net profit (or loss)
h. Check of exploitation cost by approximate method in Section 6.2 (use same constants)

Production data:

Daily output	28,800 tons/day (26,130 tonnes/day)
Work time	8 hr/shift, 3 shifts/day, 5 days/wk, 50 wk/yr
Work force	270 employees
Average wage rate, including fringes	$15.50/hr

Mining data:

Face	20 x 25 ft (6.1 x 7.6 m)
Depth of round	15 ft (4.6 m)
Tonnage factor	12.5 fl^3/ton (0.39 m^3/tonne)
Rounds/shift/face	2

Drilling (rubber-tired, hydraulic. 2 boom jumbos—no spares):

Drilling factor	2.0 ft/ton (0.67 m/tonne)
Average drilling rate	5.25 ft/min (1.60 m/min)
Ownership and operating cost/drill rig	$ 105/hr

Blasting (ANFO, pneumatic loader):

Powder factor	1.1 lb/ton (0.55 kg/tonne)
Explosives cost	12</lb (26.5?/kg)
Additional blasting costs	20% of explosives cost

Loading (rubber-tired, front-end loaders—no spares):

Load cycle/face	2 hr
Ownership and operating cost/loader	$125/hr

Haulage (rubber-tired trucks—2 spares)*

Trucks/loader	3
Ownership cost/truck	$21/hr
Operating cost/truck	$49/hr
Hoisting (balanced, drum, skip); Cost	8e/ton (8.8 e/tonne)

Auxiliary operations (roof control, ventilation, drainage, etc.):

Cost	130/ton (14.30/tonne)
Indirect cost (overhead):	

Rale. % of direct cost	10%	

Other production costs:

Exploration	$l.30/ton	($1,43/tone)
Development	2.15	(2 37)
Plant	1.60	(1.76)
Processing	7.35	(8.10)
Transportation	5.10	(552)
Royalty	1.75	(1.93)

Income tax rate, % of gross profit:	34%

Market price, U.S., Zinc metal—see latest quote in ***Engineering and Mining***

Ore grade, as mined	3.4%
Overall recovery	80%

13.4 Given the following operating and cost data, estimate all the unit mining and production costs and unit profit (in $/ton, or $/tonne) for an underground lead mine using the stope and pillar method. Specifically, determine the following (all except item 1 are expressed on a unit basis in $/ton, or $/tonne):

a. Productivity. in ions (or tonnes) per employee shift
b. Unit operation costs, grouped as breakage, handling, and auxiliary purposes
c. Direct exploitation cost
d. Indirect (overhead) cost
e. Overall exploitation and mining costs f. Overall production cost
f. Gross and net profit (or loss)
g. Check of exploitation cost by approximate method in Section 6.2 (use same constants)

Production data:

Daily output	14.400 tons/day (13,060 tonnes/day)
Work time	8 hr/shift, 3 shifis/day, 5 days/wk, 50 wk/yr
Work force	135 employees
Avenge wage rale, including fringes	$ 15,50/hr

Mining data:

Pace	18 x 26 ft (5.5 X 7.9 m)
Depth of round	16 ft (4.9 m)

Tonnage factor	12.48 ft^3/ton (0.39 mVtonne)
Rounds/shift/face	2

Drilling (rubber-tired, hydraulic, *2* biwm jumbos—no spares):

Drilling factor	2.0 ft/ton (0.67 m/tonne)
Average drilling rate	5.1 fl/min (1.55 m/min)
Ownership and operating cost/drill rig	$118/hr

Blasting (ANFO, pneumatic loader):

Powder factor	1.3 lb/ton (0.65 kg/tonne)
Explosives cost	12e/lb (26.5^/kg)
Additional blasting costs	20% of explosives cost

Loading (rubber-tired, front-end loaders—no spares):

Load cycle/face	2 hr
Ownership and operating cost/loader	$134/hr

Haulage (rubber-tired trucks—2 spares):

Trucks/loader	3
Ownership cost/truck	$23/hr
Operating cost/truck	$56/hr

Hoisting (balanced, drum, skip);

cost	11e/ton (12.1 e/tone)

Auxiliary operations (roof control ventilation, drainage, etc):

cost	27e/ton (29.8e/tonne)

Indirect cost (overhead);

Rate, % of direct cost	10 %

Other production costs;

Exploration	$1.10/ton	($1.21/tone)
Development	1.95	(2.15)
Plant	1.40	(1, 5 4 **)**
Processing	7.15	(7.88)
Transportation	4,90	(5.40)

Royalty	1.25	(1.38)
Income tax rate, % of gross profit:	34%	

Market price, U.S., zinc metal—see latest quote in Engineering and Mining Journal

Ore grade, as mined	4.8%
Overall recovery	85%

14

NOVEL METHODS OF MINING

14.1 CLASSIFICATION OF METHODS

There are several unique mining methods (hat are not included among the traditional surface and underground methods already described. We shall term them ***novel methods,*** defined as methods that employ nontraditional principles or technologies, or exploit uncommon resources, and that are not yet widely accepted in practice.

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

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

TABLE 14.1 Classification of Novel Mining Methods

Method	Probable Commodities	Locale		Likelihood of Use		
		Surface	Underground	Existing	Promising	Questionable
Rapid excavation	Noncoal (hard rock)		×	×		
Automation, robotics	All	×	×	×		
Hydraulic mining	Coal, soft rock		×	×		
Methane drainage	Coalbed methane	×	×	×		
Underground gasification	Coal		×		×	
Underground retorting	Hydrocarbons		×		×	
Ocean mining	Metal, nonmetal	×			×	
Nuclear mining	All	×	×			×
Extraterrestrial mining	Metal, nonmetal	×	×			×

The novel methods which display enough promise or have attracted enough notoriety to discuss at this time are indentified in Table 14.1. (expanded from Tables 3.1). Observe that the first two are not associated with specific methods of exploitation; they relate particularly to one or more unit operations in the mining cycle and may be incorporated into existing methods, also evident that most are designed for other surface or under- ™«.ind miniriK often with specific commodities in mind, and these are Soted Finally the likelihood of their commercial application « estimated a_s *(,)* limited existing use. *(2)* promising but noi yet in use. and (3) questionable

°r Because oHheir limited application or unknown potential, none of the novel methods of Table 14.1 warrants as extensive coverage as thc tradi- lional surface and underground methods. Accordingly, wc will confine our discussion to a brief explanation of each novel method, an identification of its likely uses, a listing of its advantages and disadvantages, and some comments on its current status and future promise. A summary concludes the
chapter.

14.2 RAPID EXCAVATION

Rapid excavation is the concept of replacing the intermittent operations or rock breakage and materials handling in mining with a system of continuous extraction. Notice that we say *concept* rather than *practice* because truly continuous rapid excavation is not yet attainable. Nonetheless, progress is being made, particularly in breakage and less notably in handling. One of the related deterrents lo the elimination of the cyclic nature of mining, incidently. is the difficulty of incorporating the major auxiliary operations— ground control, ventilation, gas and dust control, drainage, and materiel supply—into the production operations without interrupting or delaying advance of the working face. Figure 14.1 illustrates

schematically the principle and components of rapid excavation with a TBM (tunnel-boring machine, also called a *mole),* in which breakage, handling, support, and ventilation arc attempted simultaneously.

The term *rapid excavation* was coined in the 1960s to convey the concept of continuous, high-speed mining or tunneling (Howard, 1967). It received wide acceptance and remains the ultimate goal of those who seek to maximize the performance of the entire extraction system (Anon., 1968). Equally applicable as a concept to both surface and underground mining, rapid excavation in usage is applied mainly to the subsurface and, as a goal, to hard rock.

While the entire operating cycle is involved, the element of rock breakage lies at the heart of rapid excavation. It is nearly always accomplished by mechanical means (Section 4.2), but any of the forms of rock attack noted in Table 4 .1 could be candidates (to date, hydraulic fragmentation is the only other mechanism to be used commercially, mainly to supplement mechanical) (Souder and Evans, 1983).

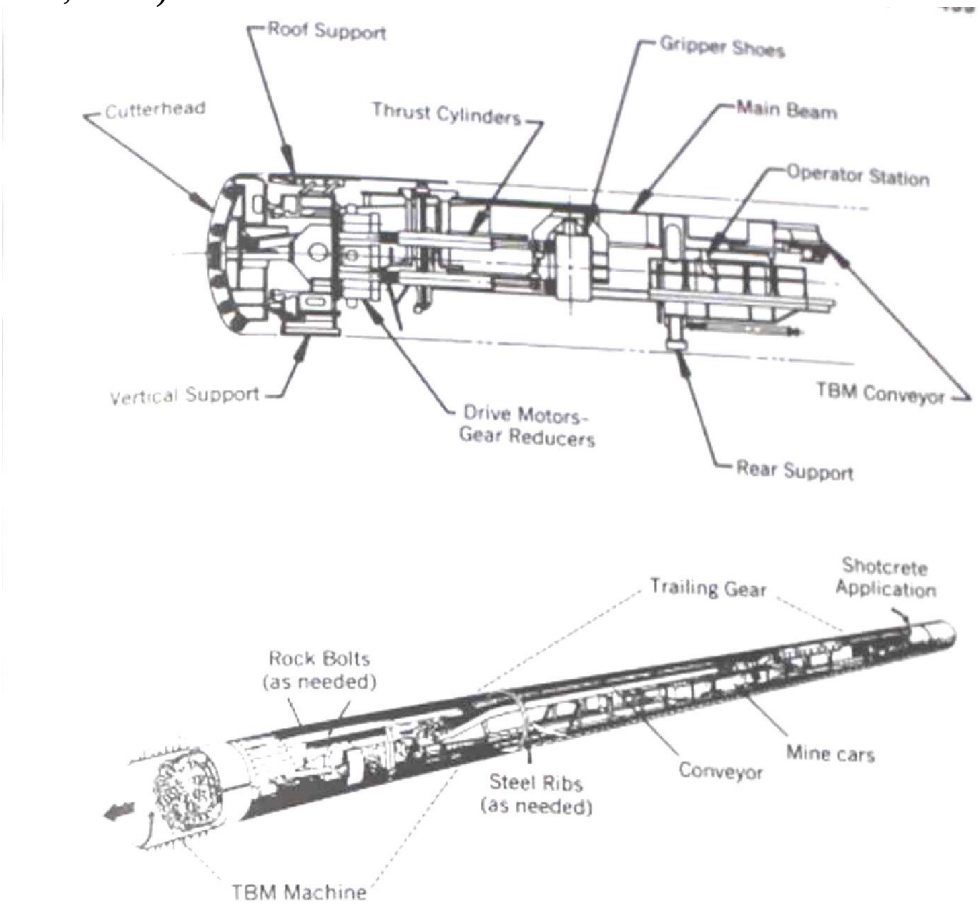


Figure 14.1. Diagrams of tunnel*boring machines (TBMs), (TOP) Boring machine unit and constituent parts. (After Brockway, 1983. By permission from Society of Mining Engineers, inc.. Littleton. CO.) (Bottom) TBM with trailing gear, (After Sager et al 1984. By permission from National Research Council, Washington, DC.)

In terms of application, two types have become popular, the continuous miner for coal and soft rock and the boring machine for soft to medium-hard rock (the latter

is more closely identified with rapid excavation). Continuous mining machines arc widely used for both development and exploitation in room and pillar and longwall mining. Boring machines in soft- and hard-rock mining find application mainly for development, in tunnels and drifts (T B M s), raises (raise-boring machines— RBM s), and shafts (shaft-boring machines— S B M s) (Obert, 1973c; Lucas and Haycocks, 1973). In terms of machine design, either miners or borers may be full-face or partial-face, meaning that a single pass suffices and a round or elliptical hole results (full-face), or multiple passes utilizing a pilot hole, reamer, or an oscillating head are employed (partial-face). The various types of machines are identified in Table 9.1 and pictured in Figures 9.8 and g.

Historically continuous excavating machines have been in existence for a good many years, preceding the invention of dynamite (in 1867). The first such machine, a TBM, appears to have originated in Italy in 1846; it util.zed percussive-blow energy to produce a slot around the face (Robbins 1984). TBMs were used in the earliest attempts to tunnel under the English Channel starting in 1865; most of them used drag-bit or disk cutters. Modern rock-boring machines were developed in the 1950s, although successful coal continuous miners were in operation a decade earlier.

Progress in rapid excavation can be measured by several performance parameters: (I) hardness of the rock penetrated, (2) time percentage of machine availability, (3) diversity of application. (4) rate of advance, and (5) cost of advance. In all respects, rapid excavation has made gains. Because it is most easily determined by laboratory test, compressive strength is widely employed as a measure of rock hardness or resistance to penetration. Table 14.2 presents a system of rock classification and boring application based on compressive strength. The range of applicability has been steadily rising; boring is now used for rock of medium hardness and occasionally for hard rock. Note the similarity of Table 14.2 to Table 13.3; both employ compressive strength as a readily quantified measure of rock strength or hardness.

Rock compressive strength is also a useful criterion to employ in deter-mining or estimating boring rate and cutter (bit) cost of continuous excavation machines (Fig. 14.2). In these graphs, the data are for full-face TBMs, although the trends are the same for raise and shaft borers and partial-face machines and continuous miners. The drastic and limiting effects of rock strength are p la in ly evident from Table 14.2 and Figure 14.2.

TABLE 14.2 Rock Classification for Boring Machine Application

Rock Type	Relative Hardness	Compressive Strength, lb/in.²	(10^3 kPa)	Type Cutters, If Applicable
Shale, clay, hard-pan	Soft	<6,000	$<41)	Milled tooth, kerf, disk, drag
Dolomite, sand-stone, lime-stone, marble	Medium	6,000–12,000	(41–83)	Milled tooth, kerf, disk
Hard limestone, gneiss, soft granite	Medium hard	12,000–25,000	(83–172)	Milled tooth, tungsten car-bide
Hard granite, diorite basalt	Hard	25,000–35,000	(172–241)	Tungsten carbide
Quartzite, taconite	Very hard	>35,000	(>241)	N/A

Source: Modified after Obert, 1973c.

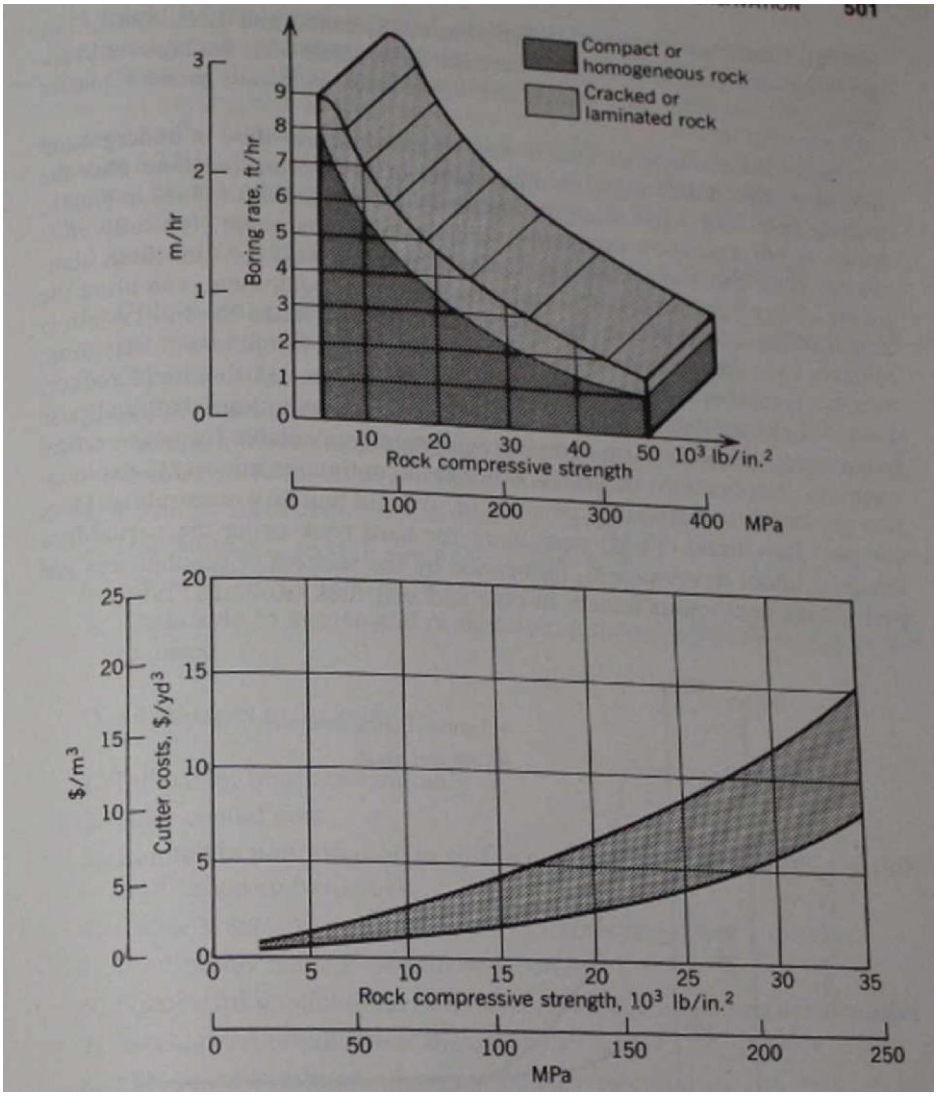


Figure 14.2. Effects of rock compressive strength on (fop) boring rate and (bottom) cutter cost, in 1973 dollars, of boring machines. *(Top,* after DesLauriers and Broennimann, 1979. *Bottom,* after Obert, 1973c. By permission from Society of Mining Engineers, Inc.. Littleton, CO.)

Machine availability is not high but improving. Performance studies of TBM s indicate that it typically ranges from 35 to 50%, which is the percentage of time that the boring machine is actually penetrating rock (Obert, 1973c). Continuous miners do little better (stefanko and Bise, 1983). The principal delays are caused by mechanical breakdowns, haulage interruptions, and lost time due to auxiliary operations, especially ground control and ventilation.

One of the major deterrents to applying rapid excavation in underground mining is the rather massive dimensions of the boring machine plus the trailing gear. The TBM alone in Figure 14.1 about 50 ft (15 m) in length, which not excessive compared with a continuous miner, typically 35 ft (10 m) long. But the addition of the usual (railing gear for tunneling, comprised of haulage, ventilation, and ground – control equipment, can bring the length of the entire system to over 450 ft (135 m). When the full system is utilized, then applications in mining are limited to development of very long, near - horizontal openings (entries, tunnels, slopes, etc.) (Uthus and Crocker, 1976; Brockway, 1983). Similar problems exist in continuous shaft and raise excavation, although more compact equipment is available for near-vertical openings, Applications of TBMs, other than continuous miners, to exploitation are largely experimental because of size and mobility constraints. More compact horizontal (TBM) equipment for hard rock using the partial-face design is under development, influenced by the success of roadheaders and partial-face continuous miners in coal and soft rock (Robbins. 1984).

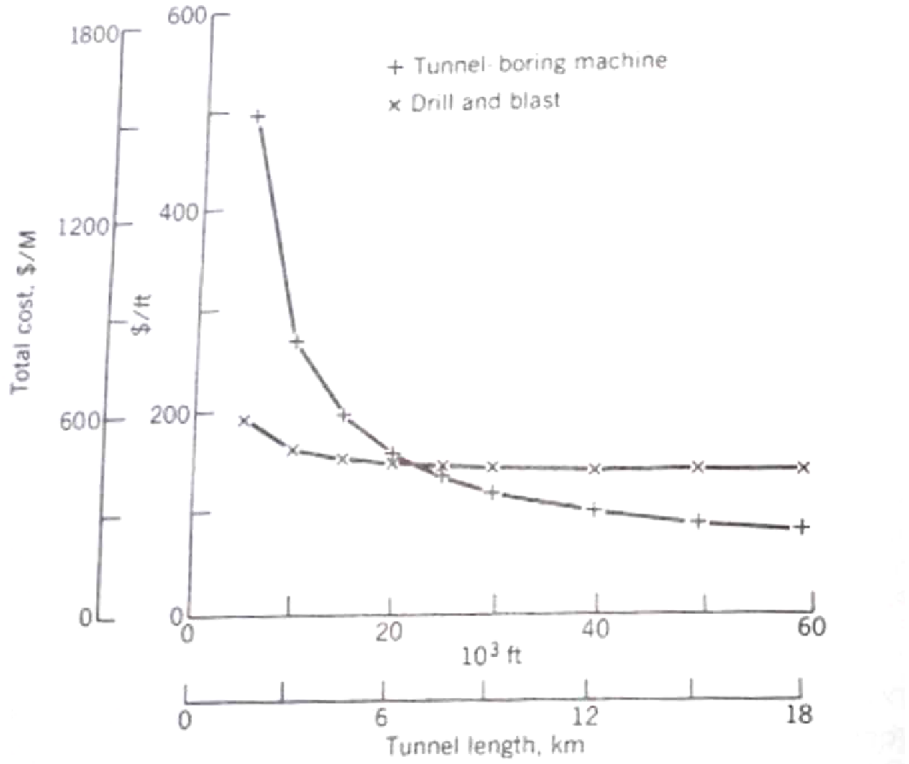


Figure 14.3. Cost comparison of conventional cyclic mining and a T B M a s a function of tunnel length. (After Sharp et al., 1983. By permission from Society of M in in g Engineers. Inc., Littleton. CO.)

Ultimately, the success of rapid, continuous excavation in displacing conventional, cyclic operations must be demonstrated by its cost competitiveness (Sharp et al., 1983). Ideal though it may be as a concept, rapid excavation will be adopted widely in mining only if it is more economical. Not unexpectedly, tunnel length is a key factor in determining the cost effectiveness of a TBM (Fig. 14.3). Note that the crossover point for the hypothetical systems illustrated is about 22.000 ft (6.7 km), which is excessive for most mining situations.

Advantages o f rapid excavation are the following (Obert. 1973c):

1. Utilizes principle of continuous extraction of material in place, including breaking and handling
2. Capable of high rates of advance under optimal conditions
3. Eliminates blasting damage to and irregularities in walk , creating a more stable opening, reducing support needed ventilation resistance; eliminates over break reducing
4. Reduced labor requirement and high productivity
5. Operating and overall costs possibly lowered
6. Reduced safety hazards to personnel
7. Applicable to any spatial orientation, although specialized equipment

Disadvantages include these:

1. Inflexible, restricted mobility
2. High capital cost
3. Limited to uniform rock in soft to medium-hard range, with occasional application to hard rock
4. Cutter wear and cost excessive in unfavorable rock conditions
5. Continuous haulage seriously deficient
6. Successful applications in hard-rock mining limited to development
7. Mechanical breakdowns disastrous to output
8. Close and continual alignment usually essential

Needed R&D in rapid excavation have been exhaustively studied and reported (Anon., 1968; Hartman. 1970b). Among others, two areas are currently attracting attention. One is a large full-face SBM that requires no pilot hole or reaming; several are under development (Grieves 1981; Robbins 1984; Anon., 1985b), The second is continuous excavation in hard-rock exploitation, comparable to continuous coal mining (Sparks, 1980; Anon,, 1982b; Robbins, 1984), System concepts suggested for two stoping methods, shrinkage and cut and fill, are shown in Figure 14.4. Materials handling as well as rock breakage pose problems.

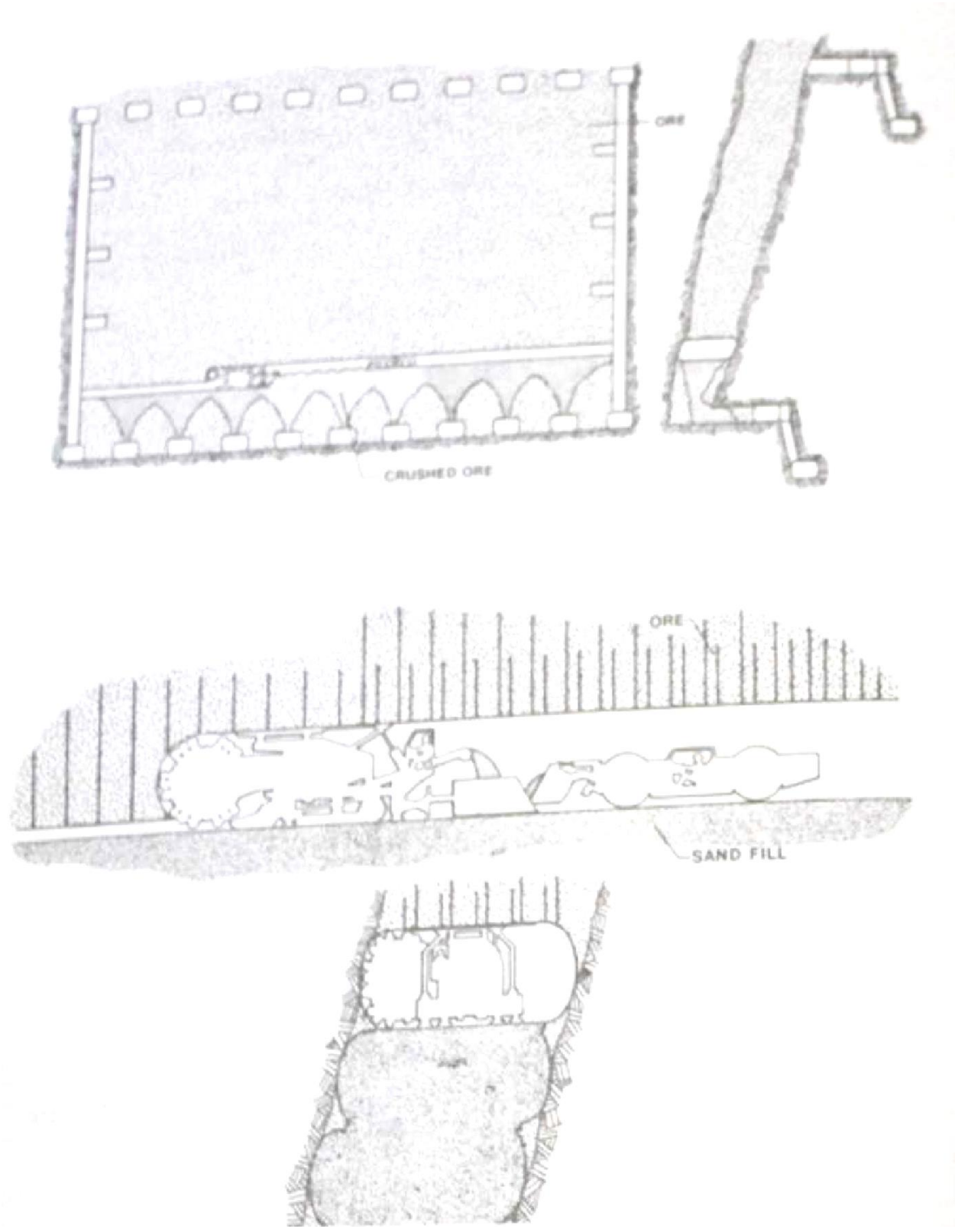


Figure 14.4. Conceptual application of TBMs to hard – rock exploitation. (Top) Shrinkage stoping (Bottom) Cut and fill stoping (After Hobbins, 1984. By permission from Society of Mining Engioners Inc, Litliton CO)

14.3 AUTOMATION AND ROBOTICS

The mining industry has only recently begin to experiment with high technology. To date, applications have been isolated and successes few. While it is too early to say with certainly, there are cautions signs that the situation may be about to change. The reason is due largely to rapid computerization in the industry.

The progression of high technology within the industry may be measured along the following scale:

Mechanization → remote control → automation → robotics

Based on past progress, current innovations, and future indications, wc arc well advanced through mechanization, experimenting with remote control and automation, but still far removed from robotics applications If this assessment is correct, mining badly lags the rest of industry.

The reasons are hard to fathom. Enhancement of productivity and cost savings arc universally regarded as the overriding incentives for the adoption of mechanization and automation. Bui an even more compelling argument tor mining should be the opportunity for a drastic improvement of safety, an urgent need in an industry with the poorest safety record.

We shall examine briefly the nature of remote control, automation, and robotics, their features, and the status of their development within the mining industry. Like rapid excavation, they do not constitute a distinctive method of mining but an adaptation that can improve—even revolutionize— unit operations.

Remote Control

Remote control permits machine operations to be controlled at a distance. It may or may not involve automation as well, a separate function. The level of control exercised in mining is almost always manual: hence with remote control, the operator can be stationed at some distance from the machine Presumably, productivity and cost savings arc enhanced because machine operation is more effective and continuous. Savings also accrue from improvements in safety lo the operator: less likelihood of falls of ground, reduced dust and gas concentrations, less noise, and reduced risk of injury from machinery (Murphy. 1985).

Experience lo dale with remote-equipped continuous miners, the oldest and most common remote-control application underground, has not been universally successful. Murphy estimates that 700 remote-controlled miners have been installed in the United States, but fewer than 200 are still being used on a regular basis. He gives (he following as reasons for their lack of acceptance:

1. Keduclion in the sensory cues that the continuous miner operator has with a remote-control console
2. Problems with maintainability and/or reliability of a remote-control unit
3. Regulatory agency resistance to remote control without proper safeguards

Interestingly, (he first may be the most serious drawback. In surveys, continuous-miner operators rank the following in order of sensory importance to machine operation: (I) touch. (2) sound, and (3) sight. Hence operators rely more on vibratory and auditory cues received from (he machine than on visual ones. In a remote-control situation, the miner operator loses all of the vibratory cues and most of the auditory cues. Special training of the operator will help, but further refinements in control devices (e.g., additional instrumentation) may be required to replace the lost sensory cues.

Maintenance and reliability problems, the second disadvantage, continue to plague underground applications of remote control. A lack of uniformity in performance of (he various commercial units now available is a common complaint. Quite clearly, remote-control units must perform with the same reliability as the machines to which they are attached.

The third reason for lack of acceptance, strangely, is safety-related. Minor controversy has arisen between mining companies and regulatory agencies in some instances over the extent of canopy protection to provide the operator, frequency of gas checks at the face, brattice extension, depth of cut, and so forth. In the main, these disagreements have been resolved by negotiation.

Remote-control applications in mining, both surface and underground, have been reported as follows (Pfleidcr, 1973b; Murphy, 1985; Anon., 1985c; Scott, 1985b):

OPERATION	COMMODITY	LOCALE
Rock breakage		
Drill	Metal, non metal	Surface, underground
Raise borer	Metal	Underground
Materials handling		
Loader Auger	Coal, metal Coal	Underground
Continuous miner	Coal, nonmetal Coal	Surface
LHD unit	Met a 1	Underground
Train	Metal, coal Coat	Underground Surface,
Ground control		Underground
Mobile roof support	Coal	Underground
		Underground

Next to continuous miners, the next most common applications of remote control in mining are longwall shearers, LHDs, and trains. Especially where cost savings in labor are realized as well the safety advantages of remote control can become a compelling argument favoring its adoption for a variety of unit operations.

Automation

Problems of worker acceptance, more intense than those encountered with remote control, plague automation in industry in general. Rouse and Morris (1985) have postulated, however, that it is really the level of automation which determines

whether or not automation is accepted by the individual. They recommend instituting an educational program to gain employee support and to ascertain acceptable levels of automation prior to making any changes.

From its inception, technology has affected human beings, sometimes favorably and sometimes adversely. At some point, machines were invented that could perform simple tasks autonomously. They progressed to total replacement of humans in certain jobs. Automation, regarded initially as a technology to assist people, has emerged increasingly as one to replace people. Labor hostility toward automation has been the natural result.

And yet we accept sophisticated levels of automation in everyday life. We heat and cool our homes by thermostats, telephone long distance with the aid of automatic switching, travel in self-service elevators, drive automobiles with automatic transmissions and cruisematic speed control, and fly in airliners equipped with autopilot and auto land systems. It is obvious that these levels of automation are quite acceptable to the majority of the populace.

Automation is generally considered to perform five functions (Rouse and Morris. 1985). They are, with definitions and examples, the following:

FUNCTION	DEFINITION	EXAMPLES
Synthesis	Generation of alternatives	Medical diagnosis, chemical analysis, airborne geophysical prospecting
Analysis	Evaluation of alternatives	
Decision	Selection among alternatives	Fail-safe nuclear and military systems
Control	Implementation of alternatives	Thermostats, automatic transmissions, autopilots
Monitoring	Observation of results	Smoke detectors, fire alarms, burglar alarms

Probably the least advanced of these functions is decision making because the computer must be programmed (by human,) to choose among alternative action. Likewise, automated synthesis and analysis require interpretation by knowledgeable, highly trained professionals.

The mining industry, disregarding numerous applications for mineral processing, is in the very early stages o f automation. Improved performance safety and economics, more flexible use of equipment and space and increased organizational control are some of the attractive features o f automation Drawbacks are human alienation, system malfunctions, complexity and high capital cost Mining applications of automation include the following (Pfleidcr. 1973b; Wood. I980; King, 1982: Arm strong and McNider, 1985; Scott, 1985a):

OPERATION	COMMODITY	LOCALE
Hock breakage	Metal, nonmetal	Surface
Drill		
Raise borer	Metal	Underground
Materials handling		Surface
Auger	Coal	
Continuous miner	Coal, nonmetal	Underground
Roadheader	Coal	Underground
Longwall shearer	Coal	Undcrground
Train	Metal	Surface, underground
Bell conveyor	Coal	Underground
Hoisting	All	Underground
Ground control		
Roof bolter	Coal	Underground
Longwall chock, shield	Coal	Underground
Ventilation and gas control		
Mine monitoring	Coal	Underground
Fan operation	Metal	Underground

Various mining units equipped with remote control and/or automation are shown in Figure 14.5 and 14.6. The most prolific application in U.S. mines has been air – quality monitoring, where reportedly 42 computerized installations are currently in use (Welsh, 1985).

Robotics

Robotics, an advanced form of automation, involves the programmed, remote control of certain machine tasks by a mechanical actuator or robot. Applied in industry on a very limited basis at present, robotics has been installed on factory assembly lines at present, robotics has been installed on factory assembly lines to perform repetitive tasks or in highly dangerous or ultraclean environments for a variety of simple jobs. Its potential for mining is just now being investigated (Wood 1980; Bartel, 1983). There are reports that a Soviet mining machine has been operated by a robot, and studies in the United States have analyzed the suitability of repetitive operations (e.g., roof bolting) for the application of robotics (Oppenhum and Rehak. 1985). The coupling of low – cost microprocessors to mechanical actuators offers attractive possibilities for increased safety and improved productivity in industries such mining.

Features

In summary, remote control, automation, and robotics offer these advantages and disadvantages in mining (Murphy, 1985. Rouse and Morris, 1985; Oppenheim and Rehak, 1985):

Advantages

1. Increased productivity
2. Reduced operating cost
3. Improved operating safely
4. Improved operating performance
5. More flexible use of equipment and space
6. Increased organizational control and continuity
7. Better machine response

Disadvantages

1. Increased capital cost
2. Low system reliability and high breakdown rate
3. Susceptibility of electronics failure in hostile mine environment
4. Worker alienation or lack of acceptance; union opposition
5. Reduction of operator sensory control, if remote
6. Insistence of regulatory agencies on additional safeguards
7. Variability of natural conditions and mobility of mine operations
8. Limited successful use in mining to date

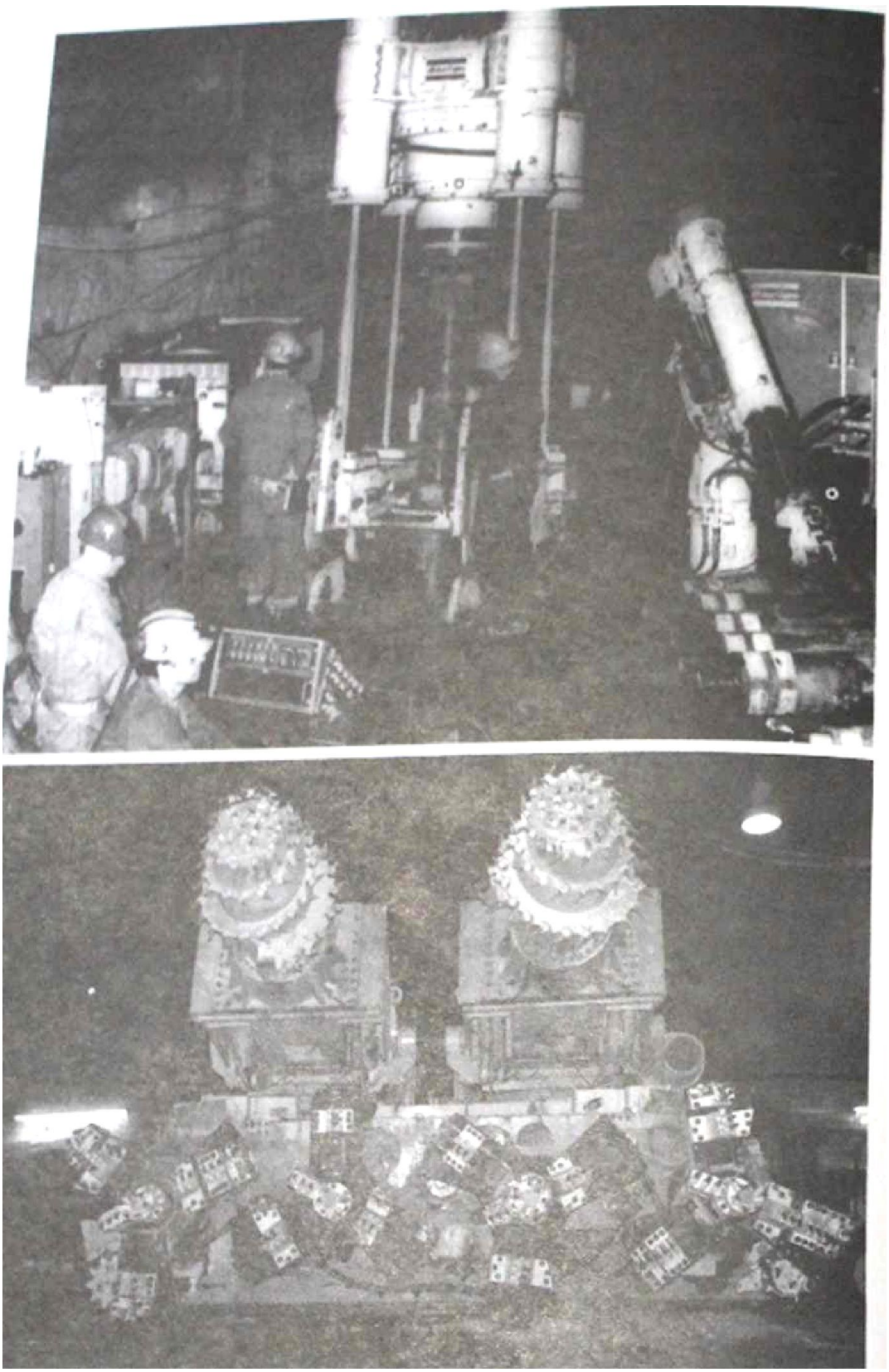


Figure 14.5. *(Top)* Hard-rock raise-boring machine (R8M) equipped with remote control and automation. This unit is computer-monitored and controlled for semiautomatic operation. (After Anon., 1983e By permission from American Mining Congress, Washington, DC.) (*Bottom* J Automated roadheader for potash and coal. The twin cutter booms are equipped with electronic sensors to distinguish ore and waste and are laser-guided and computer-controlled. (By permission from AEC, Inc.. State College, PA.)

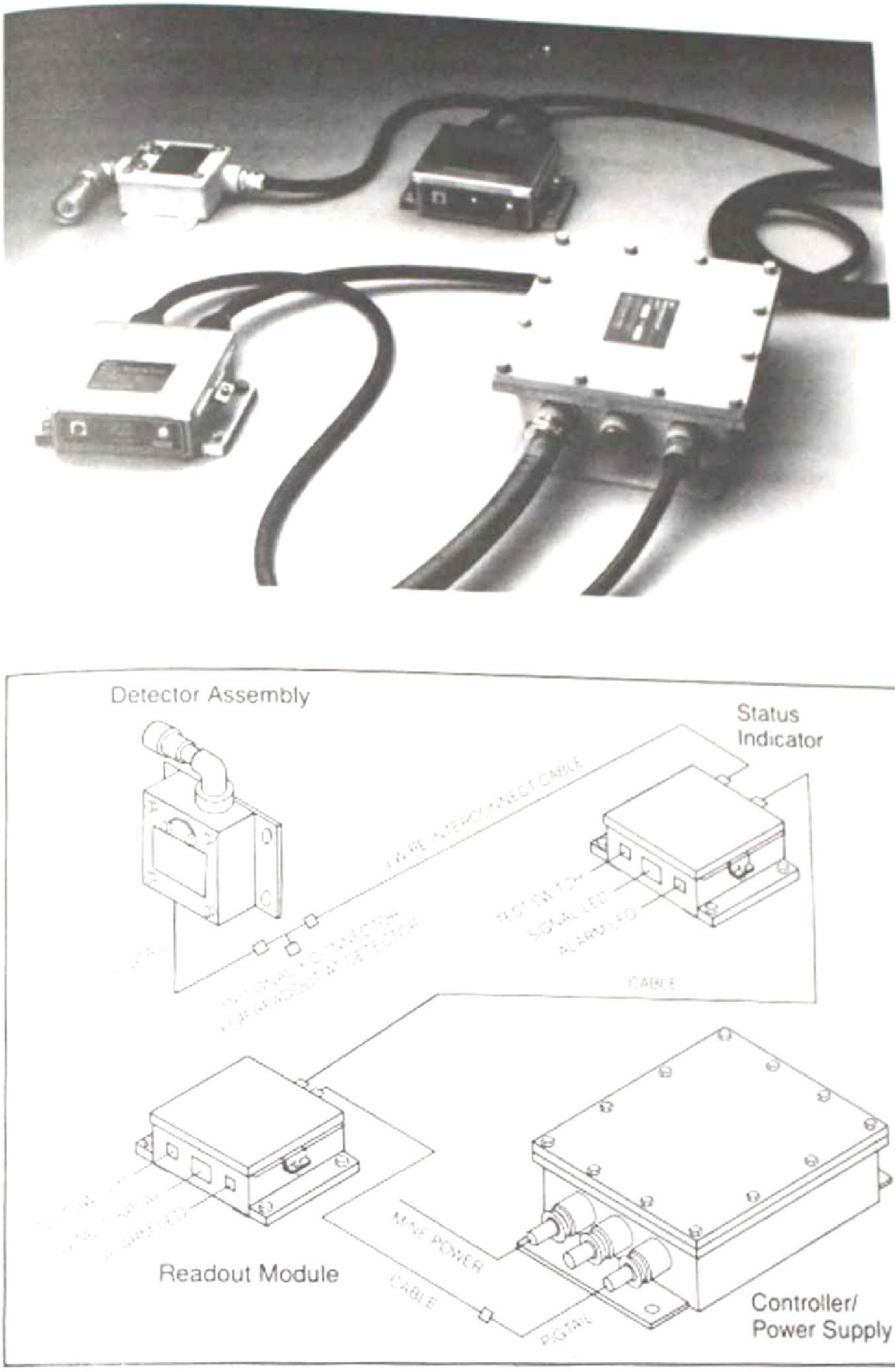


Figure 14.6. Automated methane monitoring system, used to detect gas, energize warning systems, and shut down power to a continuous miner at the face of a coal seam. (Top) Components. (Boffom) Schematic. (By permission from Mine Safety Appliances Co., Pittsburgh.)

Future

Those k n o w le d g e a b le in the *field* are confident that further technological progress can *bring* automation and robotics to mining (Bartel. 1983; Murphy, 1985). *They* are *less* sanguine that the industry and its personnel are ready *for high technology. Rouse* and *Morris* (1985) suggest that automation must he *regarded as* a means rather than an end and urge that efforts to implement it *must be* objectives-driven instead of technology-driven.

14.4 HYDRAULIC MINING

In hydraulic mining, a very high pressure jet of water, steady or pulsed, fragments consolidated mineral or rock in place. Thus it has application as a primary *extraction or* mining mechanism, although it is limited at present *mainly to softer* materials. When coupled with mechanical action (for mining or *for* cutting, drilling, or boring as well), then hydraulic attack becomes a secondary or supplemental technique.
The range of applications of hydraulic energy to mining is broad, as demonstrated by the following:

FUNCTION / APPLICATION	DEPOSIT OR MATERIAL
Hydraulic penetration (drilling, etc.)	Rock
Hydraulicking (extraction)	Placers (sand, gravel)
Hydraulic mining (extraction)	Coal, soft rock
Hydraulic transport (haulage, hoisting)	All bulk material

Previous discussion has been directed to hydraulic penetration (Chapter 3) and hydraulicking (Chapter 7). In this section, we will consider hydraulic mining and, to a lesser extent, hydraulic transport.
Hydraulic mining utilizes the kinetic energy of a fluid jet to break and excavate material from the solid (in a completely hydraulic system, bulk transport occurs as well). Consequently, its effectiveness or cutting rate is primarily a function of nozzle (jet) size, flow rate, pressure, force, and power (Fowkes and Wallace, 1968). Other important operating factors are standoff distance (range), the attack angle at which the jet stream impacts the face, and jet traverse rate (Jerem ic, 1979). The effects o f several nozzle parameters are shown in Figure 14.7. In most hydraulic systems, because o f pump characteristics, pressure and flow rate are not independent variables (Fig. 14.7a); in general, high flows are associated with low pressures, a combination suitable for the breaking or mining of soft materials (W o o d , 1980).
Results indicate that cutting rate is directly proportional to both pressure and flow rate (Fig. 14.7b) as well as to force (Fig. 14.7c) and kinetic energy(Fig. 14.7d).

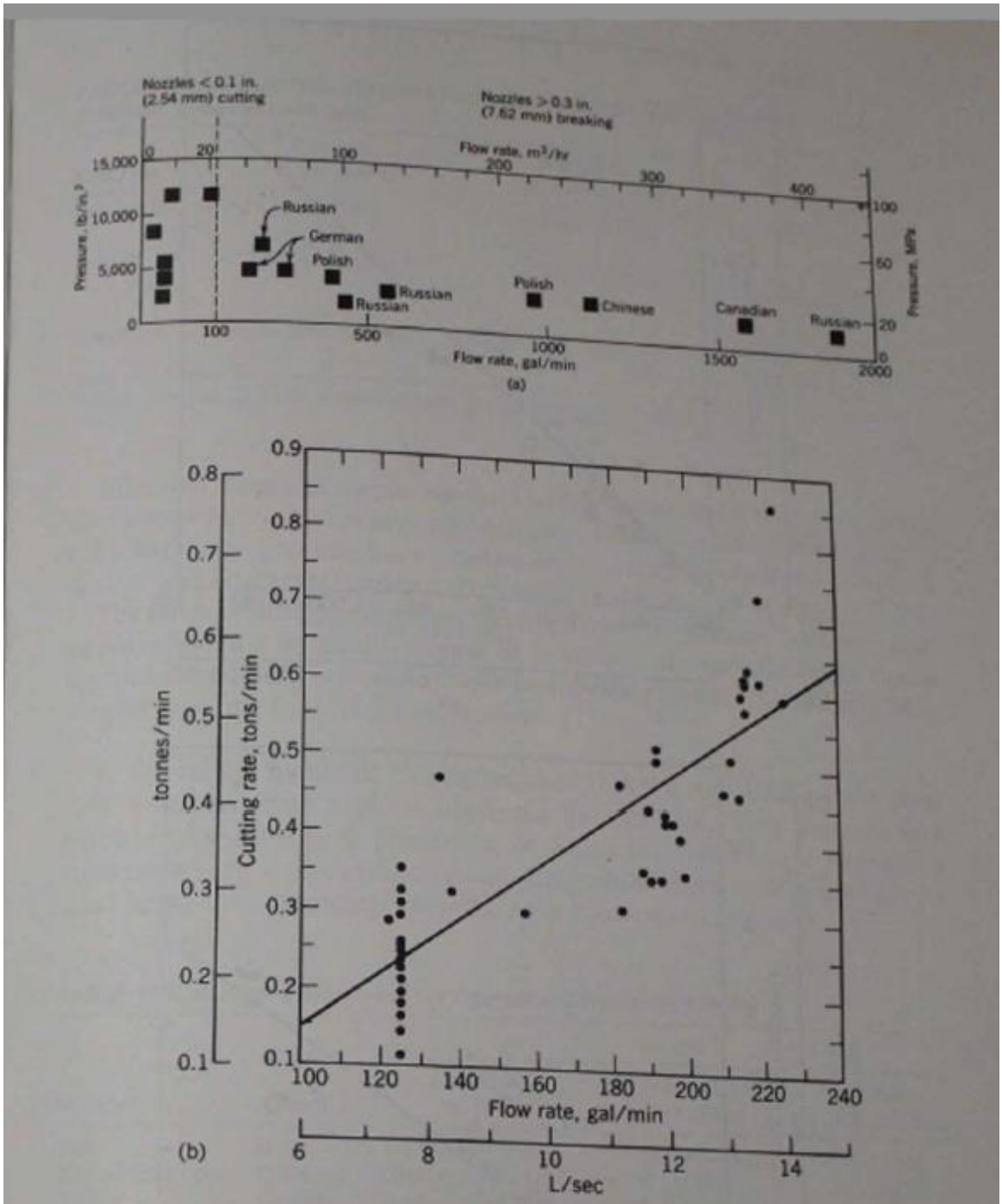


Figure 14.7. Cutting rate parameters in hydraulic mining. (a) Correlation between nozzle pressure and flow rate. Scales not linear near zero; cutting and breaking ranges shown. (After Jeremic, 1979. By permission from *Mining Magazine*, London.) (b) Effect of flow rate on cutting rate. (c) Effect of force on cutting rate. (d) Effect of kinetic energy on cutting rate. (After Fowkes and Wallace, 1968.)

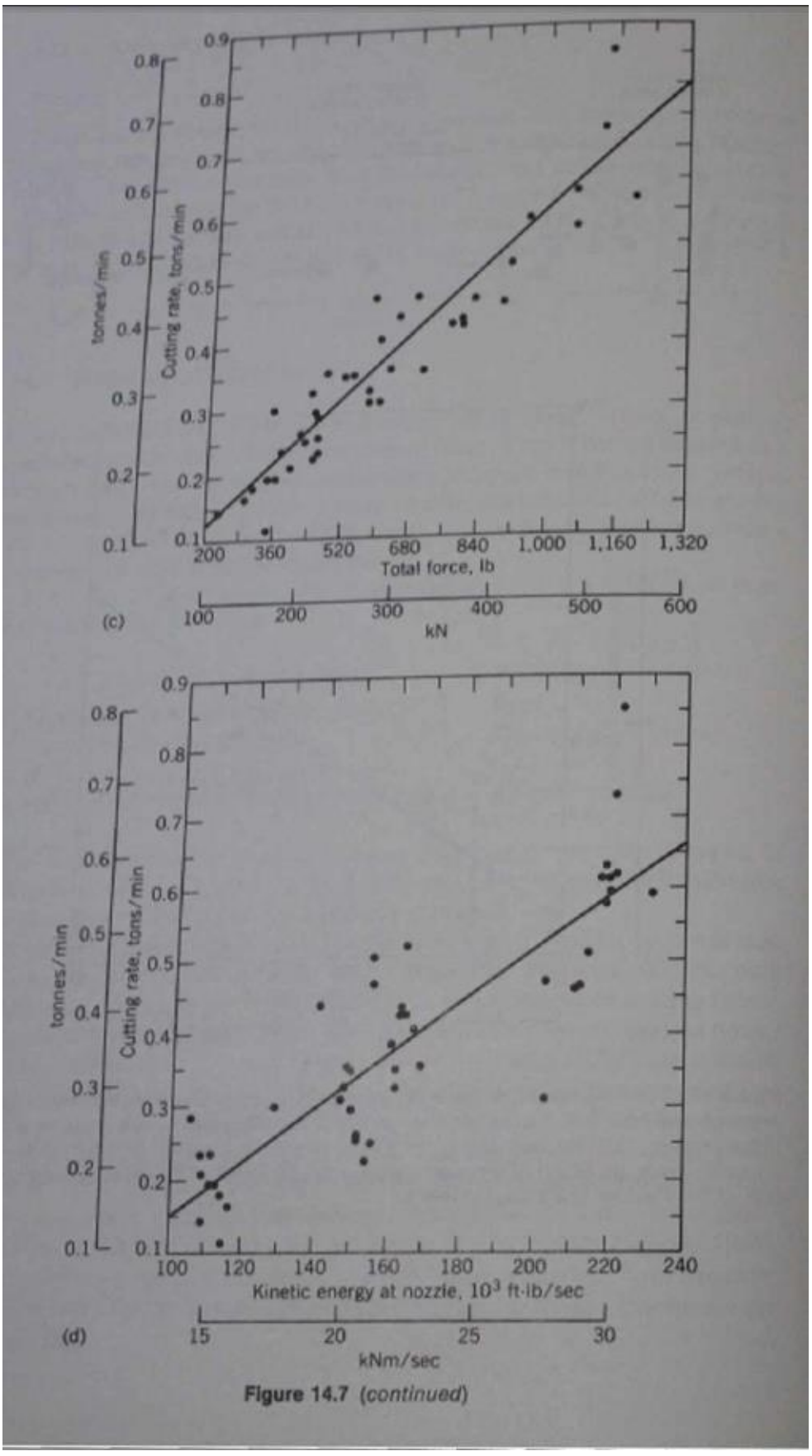


Figure 14.7 (*continued*)

TABLE 14.3 Comparison of Nozzle Pressures and Flow Rates for Different Forms of Hydraulic Excavation in Coal

Function	Pressure, lb/in.² (MPa)	Flow Rate, gal/min (m³/sec)	Application
Penetration	>14,500 (>100)	<110 (<0.007)	Drilling, extraction of thin seams
Cutting	2180–14,500 (15–100)	<220 (<0.014)	Cutting kerfs, combined hydromechanical action
Breakage	1450–5080 (10–35)	220–2000 (0.014–0.125)	Extraction, continuous mining

Source: Jeremic, 1979. (By permission from *Mining Magazine*, London.)

Effective ranges of nozzle pressure and flow rate have been established for various hydraulic excavation functions (Table 14.3). It is interesting to note that, in progressing from penetration to cutting to breakage, pressures decline and flow rates increase. This trend is also evident in Figure 14.7 a.

The range of design parameters, (nozzle diameter, pressure, and flow rate) employed in the hydraulic mining of a variety of minerals under actual operating conditions is given in Table 14.4. From Figure 14.7 and Table 14.4, we can draw the following conclusions:

I . Successful hydraulic mining require, that the threshold nozzle pressure associated with a given substance be exceeded. That pressure is a function of various rock properties, of which the compressive strength it most important. As an approximation, the threshold pressure of a steady let must be equal to or greater than the rock compressive strength.

TABLE 14.4 Design Parameters for Operational Hydraulic Mining

		Nozzle		
Material	Country	Diameter, in. (mm)	Pressure, lb/in.² (MPa)	Flow Rate, gal/min (m³/sec)
Soil	U.S.[a]	1.5–6 (38–152)	100 (0.7)	2500 (0.16)
Bituminous coal	U.S.[b]	0.25–0.56 (6–14)	4000 (27.6)	300 (0.02)
	Canada	0.60–1.2 (15–30)	1700 (11.7)	1300 (0.08)
	Germany	0.67 (17)	1300 (9.0)	480 (0.03)
	USSR	0.75–0.87 (19–22)	1500 (10.3)	1980 (0.13)
Anthracite coal	U.S.[b]	0.40–0.46 (10–12)	5000 (34.5)	300 (0.02)
Sandstone, soft	U.S.	0.62 (16)	1000 (6.9)	400 (0.03)
Gilsonite	U.S.	—	2000 (13.8)	—
Hard rock	U.S.[b]	0.02–0.10 (0.5–2.5)	>25,000 (>170)	150 (0.01)

[a] Hydraulicking.
[b] Experimental only.
Sources: Anon., 1962; Malenka, 1968; Lucas and Haycocks, 1973; Jackson, 1980; Wood, 1980; Anon., 1985d.

TABLE 14.5 Production Output and Productivity in Hydraulic Coal Mines

	Canada	USSR		
	Michel	Novokutznetz	Jubilee	Red Army
Mining method	Sublevel	Sublevel	Room and pillar	Room and pillar
Seam thickness, ft (m)	33–49 (10–15)	2.6–66 (0.8–20)	20–13 (3–4)	3.3 (1)
Seam pitch, degrees	25–55	66	10	8–10
Monitor output, tons/min (tonnes/min)	6.1 (5.5)	3.3 (3.0)	2.2 (2.0)	1.1–2.2 (1.0–2.0)
Mine production, tons/day (tonnes/day)	8800 (8000)	13,200 (12,000)	4400 (4000)	1300 (1200)
Productivity, tons/ employees-shift (tonnes/ employee-shift)	28 (25)	17 (15)	44 (40)	22 (20)

Source: Jeremic, 1979. (By permission from *Mining Magazine*, London.)

Disadvantages . .

1. High water requirements, extensive piping needed
2. Inefficiency o f energy transfer in breaking material; high power and energy requirements
3. Difficulty o f control o f monitor; irregular shape of opening
4. Potential safety hazard with high jet pressure
5. Mineral must be insoluble in water
6. High atmospheric humidity a problem in hot mines
7. Applications limited to soft to medium-hard rock

Hydraulic Transport

Hydraulic transport or conveying, a much-publicized version o f which is slurry pipelining, includes both haulage and hoisting o f bulk material in suspension. In some respects, the development of hydraulic transport parallels that of hydraulic mining, but it is also fueled by intense competition with other surface coal-haulage systems (e.g.. rail, belt conveyor, truck, barge). Hydraulic transport is not new (Pfleidcr, 1973b). The concept goes back to 1891, although the first commercial pipelines were not built until 1957 (a 72-mi, or I ! 6-km. line for gilsonite in Utah and Colorado and a 108-mi, or I74-km. line for coal in Ohio). M any have been built worldwide, the majority on the surface for coal and for ores of copper, gold, iron, limestone, and phosphate (Link, 1982). The longest to date is the 273-mi (439-km) Black Mesa coal pipeline in Arizona. Major advantages are the low operating cost,

reliability, efficiency, and low labor requirements, disadvantages are the high capital cost, inflexibility, and high water cost (which the water is recycled). can For us, the attractive feature of hydraulic transport is to couple it with hydraulic mining in a unified system. Element, of the system exist if nut he entirety. Hydraulic haulage is commonplace on the surface as slurry nine mining but is less common underground. Hydraulic hoisting has been tried experimentally for a variety of minerals, for lifts up to I200 ft (360 m) and in recent years has met with some success for coal (Singhal, 1970- Petry 1982) A mine in West Virginia now uses hydraulic haulage and hoisting for traditionally mined coal, produced by longwall and continuous mining (Fig 14.10). The potential exists, of course, for an all-hydraulic mine, combining breakage with transport.

In a sense, the future o f hydraulic transport hinges on political developments rather than technological. Proposals exist to establish a surface network o f high-capacity slurry pipelines to transport coal across the United States (Fig . 14.11). Legislation to extend a federal right of eminent domain to the builders, vitally needed if the pipelines are to acquire rights-of-way at reasonable cost, has not yet successfully passed the Congress.

Slurry pipelines can also operate with liquids other than water. While only experimental, bulk transport o f coal by methanol, crude oil, or liquefied Natural gas, ammonia, or carbon dioxide is under consideration (Link 1982; MCKeever, 1983).

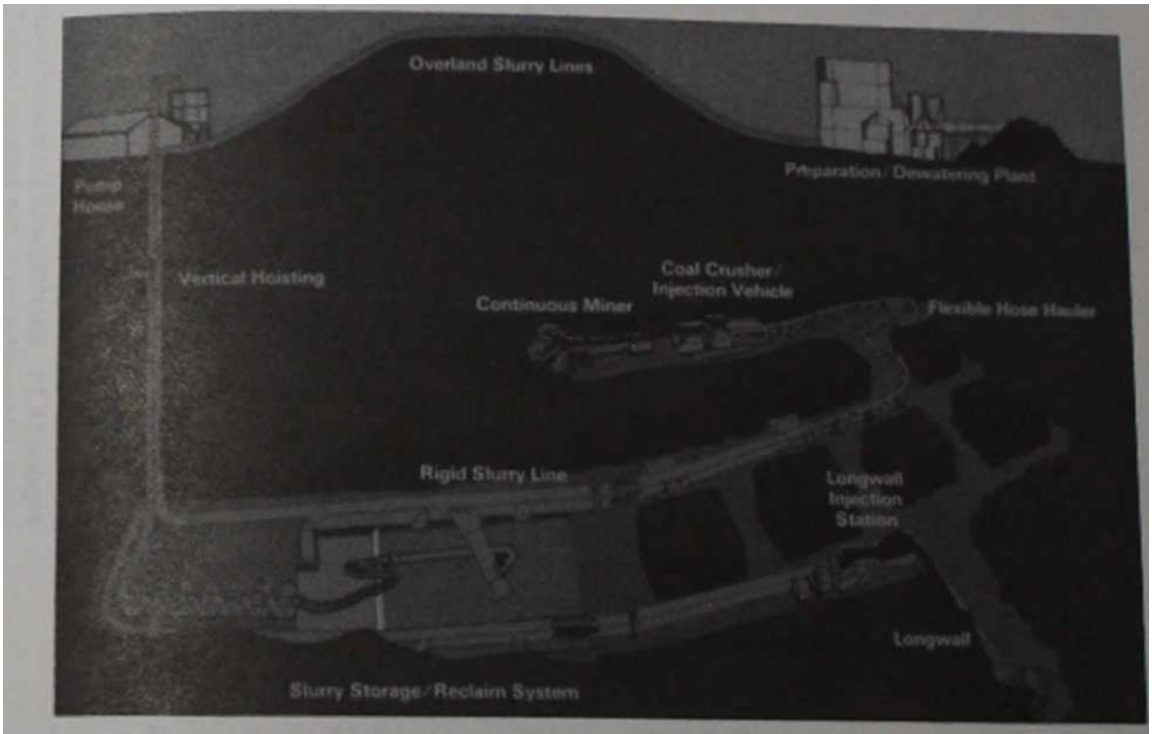


Figure 14.10. Hydraulic transport system for traditional longwall mine. Coarse coal produced by two continuous miners and a longwall shearer is hydraulically hauled and hoisted by slurry pipeline to the surface. (After Petry, 1982. By permission from American Mining Congress, Washington, DC.)

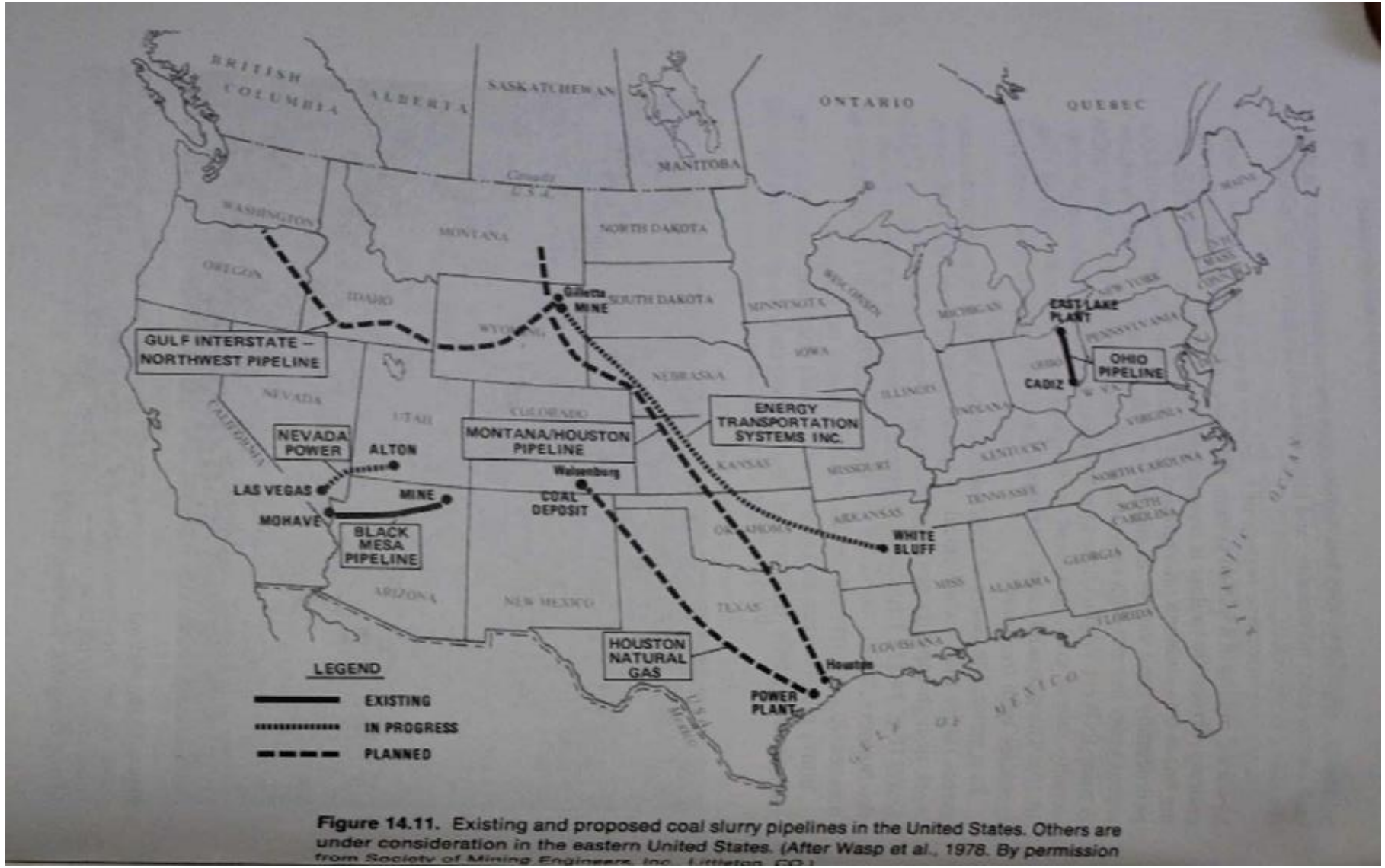


Figure 14.11. Existing and proposed coal slurry pipelines in the United States. Others are under consideration in the eastern United States. *(After Wasp et al., 1978. By permission from Society of Mining Engineers, Inc., Littleton, CO.)*

14.5 METHANE DRAINAGE

A novel mining method, methane drainage is not a new method. Attempts to drain firedamp (methane) from coal seams date to 1730 in England, with the First successful, controlled system installed in 1943 in Germany (Buntain, 1983). Today, over 60 % of the coal mines in the United Kingdom practice drainage, as do many other European mines, but it is still relatively uncommon (although taking hold rapidly) in the United States.

Methane drainage , also called coal degasification is the practice of removing the gas contained in a coal seam and adjoining strata through wellbores, drill holes, and pipelines. In some respects, it resembles borehole mining, although operations may be conducted from either surface or underground. Drainage is also closely akin to the well production of natural gas, the principal constituent of with traditional mining.

In their ventilating airstreams, U .S . coal mines purge some 250 million ft^3(7 . 1 million m^3) of methane daily (Dunn 1982) It is an inefficient, costly, unsafe, and archaic practice. The successful and widespread use of methane drainage abroad demonstrates that (1) air dilution requirements can be drastically but safely reduced and that (2) the methane captured by drainage can be utilized commercially. Mines employing methane drainage report the capture of 50-60% of the face emissions (Thakur and el)avis 1977-Dunn,1982; Perry et al.,1982).

Independent of mining and the safety motivation of drainage coal bed methane is classified by the U .S . Department of Energy as one of four unconventional gas resources. While not definitely established total U. S. coal methane resources are estimated to be 300 trillion ft' ($8{,}5 \times 10^{12}$ m^3), with about 200 trillion ft^3 ($5.7*10^{12}$ m^3) believed to be recoverable (Anon.' 1981b). Candidate coal fields with large resources that seem to bear the most promise are Green River (WV and CO), Arkoma (AR and OK) Piceance (C O), and Northern Appalachia (P A , O H , W V , and K Y). Currently the largest commercial production comes from the Warrior Basin (A L). Individual mines now being degasified or suitable for degasification have methane emission levels o f 5-12 million ft^3/day (0.14 to 0.34 million m^3 /day) (Irani et al.. 1972).

While an unconventional gas resource, coal bed methane in the United States is an attractive one, because coal seams still remain that arc accessible and un mined. Unlike natural gas reservoirs, however, coal scams are thin and impermeable except for their natural fracture system *(butt* and *face cleats).* M ethane occurs occluded and adsorbed as layers of gas in the cleats.

The principal difference in conventional and unconventional coal bed occurrences of methane lies in the way in which the gas is transported When a *methane drainage well or a mine opening it driven into the team* , a low- pressure *space is created: the gas is liberated from* **its** *adsorbed* **state and** *migrates* to the *well or opening* (Ertekin, 1984)

Methane occurrence in a coal seam is expressed as the volume of gas *contained per unit weight of coal It m dicatet the potential of a seam to* **produce**

methane *Methane* emission *it measured at the* volume flow raleper **unitlength** of *drainage hole* **Characteristics** of U.S. *coal team t* noted for *their high methane content and liberation rate are* given in Tabic 14.6. A *variety of faction* control the output of methane drainage. They include *phyttcal properties o f the c*oal seam (difTusiviiy. reservoir pressure, permeability, *and gat content),* mining method (if in progress), and drainage method (Thakur and *Dahl.* I*ЧН2}.* One of the best predictors of emission rate
κ *the product o f daily coal production* and depth o f mining; that is. the greater the production or depth or both, the greater the gas liberation rate (Irani el al.. 1972).
there are four techniques in use in the United States to drain methane from solid *coal* (Thakur and Davis. 1977):
1. Vertical wells from the surface, with hydraulic stimulation
2. Inclined holes from the surface
3. Horizontal holes from shaft bottoms
4. Horizontal holes from mine entries
Vertical holes arc least productive and. if they arc hydraulically fractured, may damage the roof or floor of the scam and interfere with mining. Inclined *h o le s are* promising, hut drilling and completion costs are high. Either of the Iasi two methods. using horizontal holes, is productive, with entry drilling more popular and cheaper. Holes are usually 3-5 in. (76-127 mm) in diameter and up to 2000 ft (600 m) in length. A typical underground methane drainage plan for solid coal using horizontal holes appears in Figure 14.12. Figure 14.13 *(top)* pictures the surface installation for a vertical drainage hole, When the mining method in use causes concurrent caving and subsidence of the overburden, methane from the entire gas-emission space flows into the mine, which acts as a natural low-pressure sink (Thakur and Dahl, 1982). Drainage of methane from the gob becomes even more essential if mining is to continue safely. Two methods are in use in the United States (Thakur et al.. 1985): (I) Vertical holes are drilled from the surface over the proposed mining area, almost to the top of the coal scam: and (2) cross-measure holes are drilled at an angle of 30-40° from the vertical over the intended mining area. Vertical holes are more widely used and generally regarded as more productive. Methane drainage from a gob using vertical holes is shown in Figure 14.13 *(bottom).*

TABLE 14.6 Methane Characteristics and Emission Rates in U.S. Coal Seams

Seam	Gas Content, ft^3/ton (m^3/tonne)	Composition, $\%CH_4$	Calorific Value, Btu/ft^3 (MJ/m^3)	Depth, ft (m)	Average Emission Rates, million ft^3/day per 100 ft (million m^3/day per m)
Beckley (WV)	262 (8.2)	99.2	1002 (37.4)	900 (274)	15 (13)
Mary Lee (AL)	430 (13.4)	96.1	970 (36.1)	1500 (457)	10 (9)
Pittsburgh (PA, WV)	220 (6.9)	90.8	922 (34.3)	700 (213)	15 (13)
Pocahontas (VA)	450 (14.0)	96.9	1003 (37.4)	1700 (518)	10 (9)

Source: Thakur and Dahl, 1982. (Copyright © 1982, John Wiley & Sons, New York.)

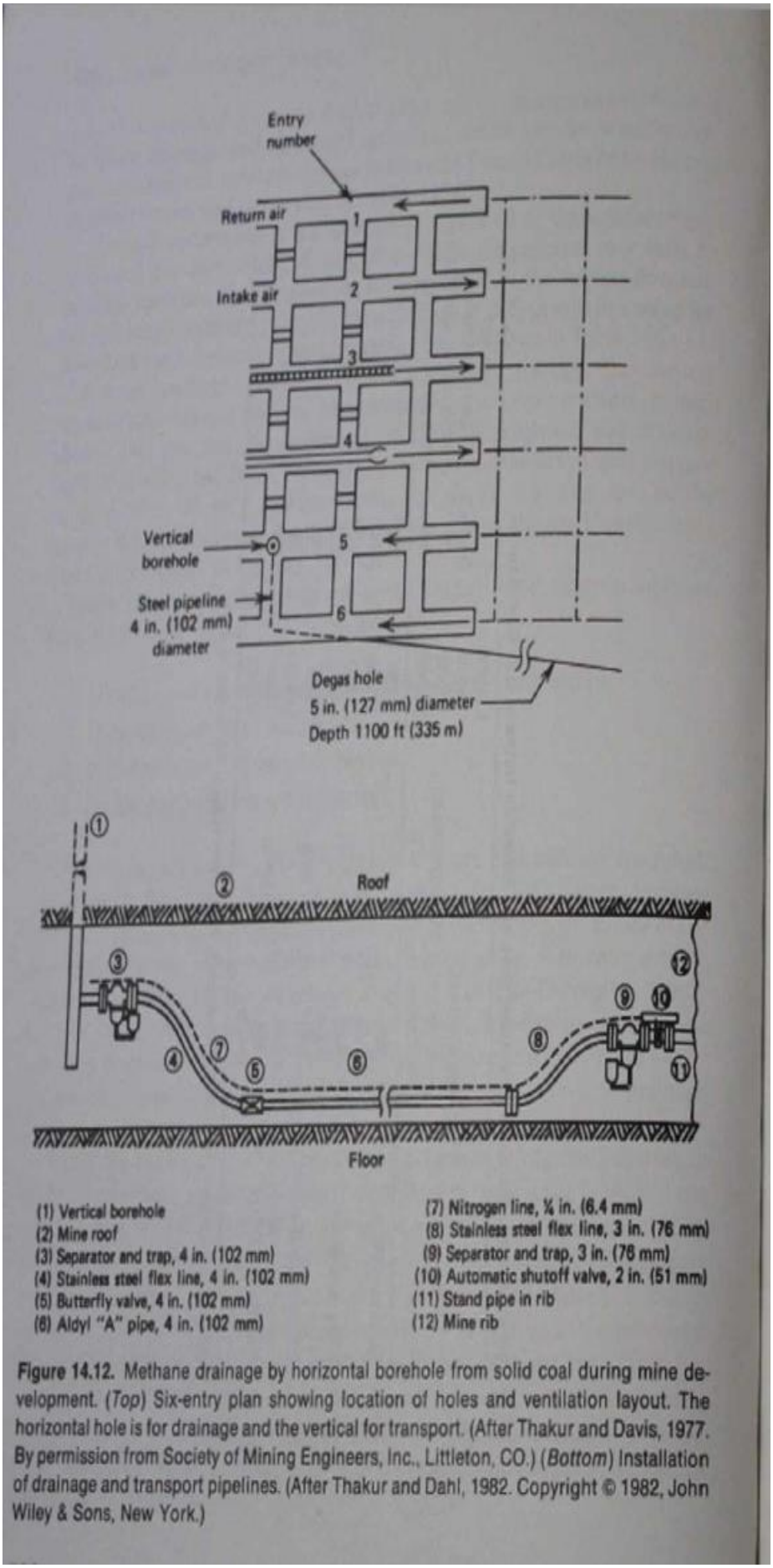
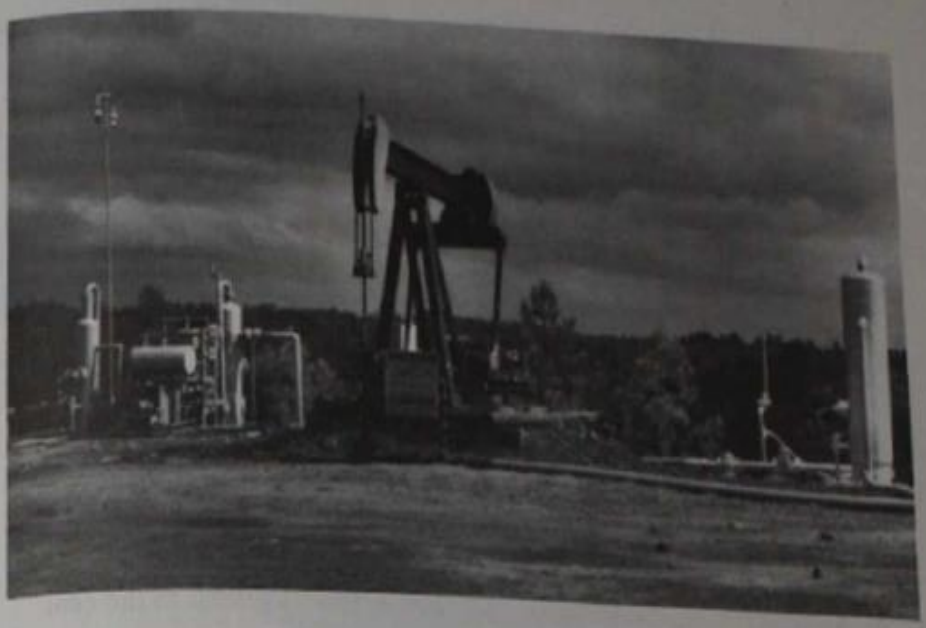


Figure 14.12. Methane drainage by horizontal borehole from solid coal during mine development. (*Top*) Six-entry plan showing location of holes and ventilation layout. The horizontal hole is for drainage and the vertical for transport. (After Thakur and Davis, 1977. By permission from Society of Mining Engineers, Inc., Littleton, CO.) (*Bottom*) Installation of drainage and transport pipelines. (After Thakur and Dahl, 1982. Copyright © 1982, John Wiley & Sons, New York.)



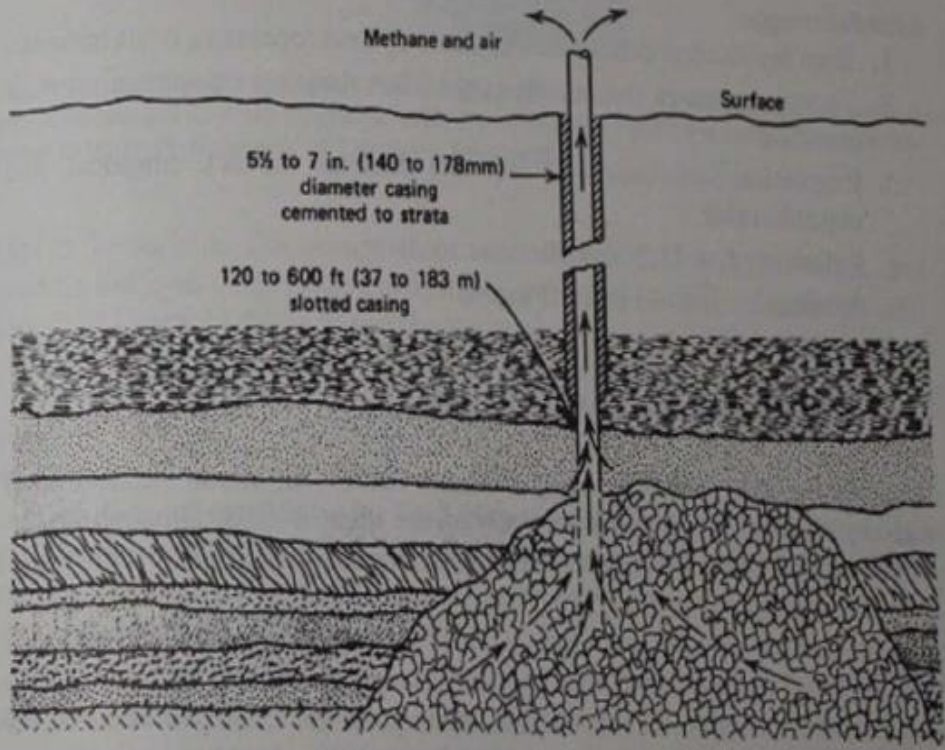


Figure 14.13. (*Top*) Methane drainage by vertical well. Shown are the pump, separator, dehydrator, and compressor at the surface. (After Dunn, 1982. By permission from Kaneb Services, Inc., Houston.) (*Bottom*) Vertical-well drainage of gob area in longwall mining or room and pillar mining. Well is usually cased to prevent loss during caving. (After Thakur and Dahl, 1982. Copyright © 1982, John Wiley & Sons, New York.)

Drainage and transport require the installation of suitable suction pumps in the system. Those operating underground must meet permissibility standards for an explosive atmosphere and arc specially designed (Buntain. 1983)

Features of methane drainage are as follows (Thakur and Dahl, 1982; Buntain 1981; Ertekin. 1984); for a current information source, sec the *Quarterly Review of Methane from Coal Seams Technology,* published by the Gas Research Institute. *Chicago.*

Advantages

1.Mine safety enhanced by degasifying the coal seam in advance of, during, and *after mining*

2. Ventilation costs reduced in proportion to the amount o f methane removed from the mining area

3 . *Methane* produced can be used locally as a fuel or marketed commercially

4. An unconventional gas resource recovered in addition to the coal

Disadvantages

1. High installation cost *of* an elaborate system (operating costs minim al)

2. Delays caused in the mining cycle when drainage equipment must be *installed*

3. Regulation and control o f methane drainage inexact, empirical, and unpredictable

4. Relatively few *U.S.* installations to date

5. Application limited *to coal* seams

14.6 UNDERGROUND GASIFICATION

Like several other novel methods we have discussed, in-situ gasification of coal is not new. W . Siemens conceived the idea in 1868, although underground experiments (by others) were not conducted until 1914, with extensive tests beginning in the 1920s (Marsden and Lucas, 1973). M an y European countries as well as Canada, Japan, and the United States have experimented with underground gasification, but the Soviet Union probably has the only commercial installation, producing gas from about 1 .000.000 tons/yr (900.000 tonnes/yr) of coal (Stephens, 1980). Interest in the United States has intensified since oil prices rose drastically in the 1970s, and extensive field experiments are currently being conducted by the U .S . Department of Energy and the private sector in Texas, Washington, West Virginia, and Wyoming (Stephens, 1980; H ill et al., 1985).

Underground gasification involves the partial combustion of coal in place, generally through boreholes, with the collection o f gaseous by-prod

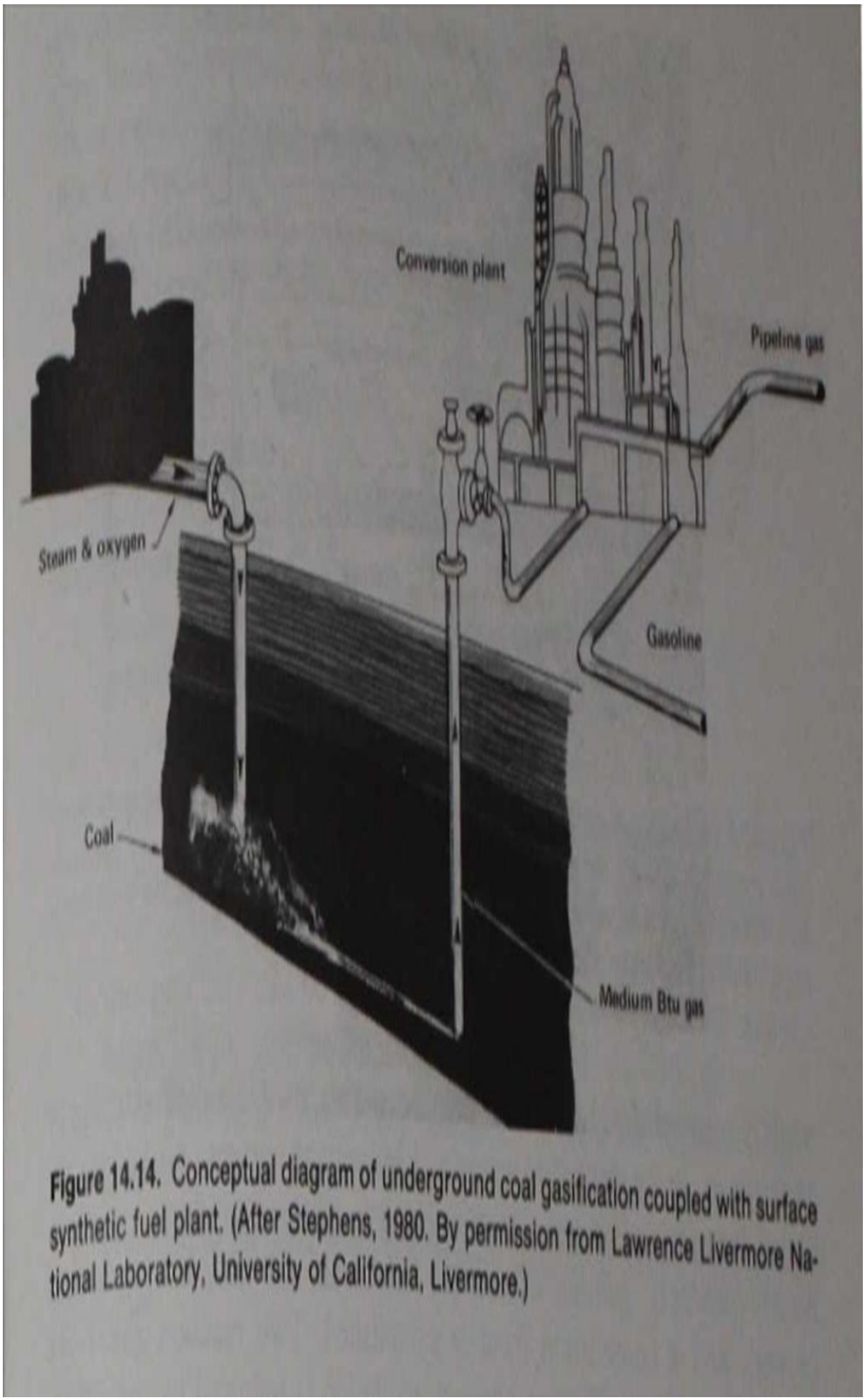


Figure 14.14. Conceptual diagram of underground coal gasification coupled with surface synthetic fuel plant. (After Stephens, 1980. By permission from Lawrence Livermore National Laboratory, University of California, Livermore.)

ucts at the surface The objective is to extract thermal energy from the coal scam m the form of uel gases to avoid conventional mining A n alternative shown m Figu re 14.14 is to use the gases as feedstock for the production of petrochemicals or sy nlhctic fuels, such as gasoline (Stephens 1980) U nderground co a l gasification in vo lves three basic stages (Zvyaghim sev, 1982): 1. D rillin g o f v e rtica l o r inclined access holes from the surface through the coal seam , in p airs, one hole serving as the inlet for air and the other as the outlet lo r the gaseous products. Occasionally, horizontal boreholes from old mine working shave been used. 2 . Formation of reaction channels in the coal seam (linkages) between the injection and production holes, permitting the coal to interact with the air in a moving com bustion front after ignition.

3. Gasification o f the coal by supplying an air blast through the inlet hole and removing the gaseous products through the outlet hole. The two boreholes and the interconnecting channel constitute an underground gasifier. Figure 14.15 illustrates underground gasification using percolation, one of five methods o f preparing a channel in a coal seam. Path linkage between

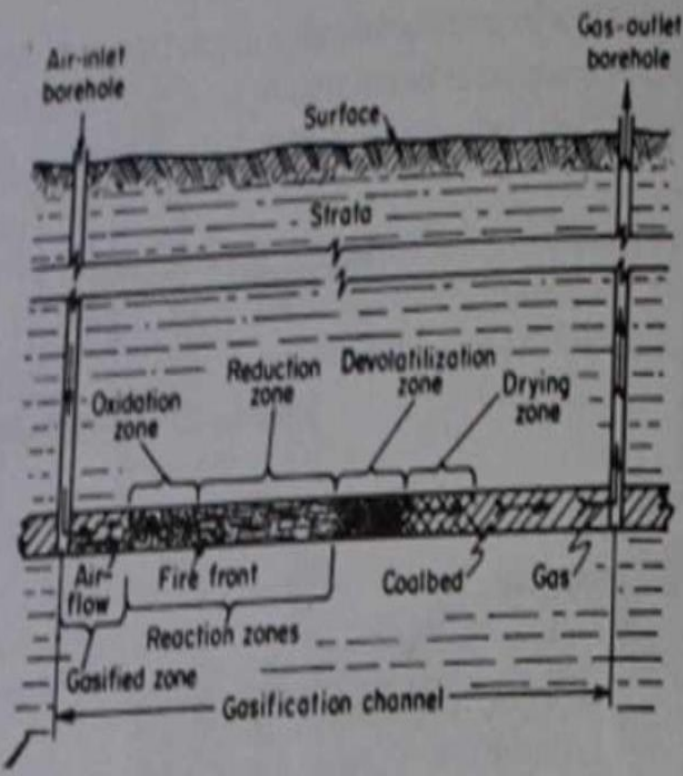


Figure 14.15. Borehole method of underground coal gasification, using backward-burning percolation. Shown are the seam, boreholes, linkage or gasification channel, combustion front, and reaction zones. (After Marsden and Lucas, 1973. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

wells is accomplished by injecting compressed air, by hydraulic fracturing or penetration, or by electro linking using high-voltage current. Air is then injected (oxygen and/or steam may be used in addition to obtain a higherquality product), ignition occurs (by electrode, thermite bomb, or gas burner), and a combustion front is established. Two reaction zones are formed along the gasification channel; oxidation is followed by reduction.

Combustible products are carbon monoxide, hydrogen, and some hydrocarbons, while noncombustible products are carbon dioxide and nitrogen. The system shown in Figure 14.15 is backward-burning; that is, the coal is ignited at the bottom of the outlet borehole, and the combustion front retreats toward the inlet.

Design parameters range as follows (Marsden and Lucas. 1973; Stephens. 1980; Zvyaghintsev. 1982);

1. Minimum seam height 3-6 ft (0.9-1.8 m)
2. Minimum seam depth 200-400 ft (60-120 m)
3. Well spacing 80-100 ft (24-30 m)
4. Intake gas (air, oxygen, etc.) pressure 100-125 lb/in.^{2} (690-860 kPa)
5. Coal consumption per well pair 100 tons/day (90 tonncs/day)
6. Gas yield 30 ft^3/Ib (1.9 m^{3}kg) coal
7. Gas production per well pair 5 million ft^3/day (0.14 million m^3/day)
8. Gas quality 75-125 Btu/ft^3(2.79-4.66 MJ/m^3)

A product with an enhanced heating value of about 450 Btu/ft^3 (16.8 M J/m ^{3}) results from injecting oxygen and steam instead of air. Coal rank is not a critical parameter; in fact, lignite and subbituminous are preferred to bituminous, because they have a lower swelling index and are therefore easier to link and gasify (anthracite is considered unsuitable because of its minimal chemical reactivity). Coal permeability, always low, is a limiting factor in achieving linkage and is enhanced by the presence of a prominent cleat system. Other design factors and considerations are discussed by Marsden and Lucas (1973).

The same sources (plus Murray, 1982) describe the features of underground coal gasification:

Advantages

1.Replaces traditional mining at a cost competitive with that of underground methods (but not surface methods)
2. Less environmental impact
3. Applicable to low-grade coal deposits (thin. deep, pitching, low-rank. Previously worked, adverse geology,etc) that are uneconomic to
4.Gas products can be used locally without enhancement but require upgrading for pipeline shipment (>15-18 mi, or >25-30 km)
5. Very good health and safety conditions (miners not exposed under-
6. Potential to increase recoverable U .S. coal resources by three to four Times

Disadvantages

1.Gas has low heating value without upgrading
2 Leakage during degasification may be high (5-15%)
3 High coal losses during combustion (20-40%)
4 Low thermal efficiency (15-40%)
5. Surface subsidence follows degasification
6 Possible contamination o f ground water by toxic by-products
7 Difficult to regulate and control
8. U .S . technology commercially unproven
9. Application confined to coal

Complete early references on underground coal gasification are contained in Capp et al. (1963) and Gibb (1964)

14.7 UNDERGROUND RETORTING

Certain natural hydrocarbon deposits— oil shale and tar sand— are unique in that the hydrocarbon minerals they contain occur in the solid state. Kerogen is found in *oil* shale and bitumen in far sand. Their occurrence throughout *the* world is widespread albeit *low-grade,* but because o f economics, com*mercial exploitation* is limited and is expected to rely largely on novel methods. *The* Athabasca tar sands o f Alberta, Canada, in part occur near the surface, and *production* by traditional, shallow methods began in the 1960s (Love. 1985); however, mining deeper sands awaits the development of non traditional methods. Likewise, some of the most attractive oil shale reserves are *found in the Green River* formation ol Colorado. Utah, and Wyoming, with marginal deposits located in the East, but all except the near-surface *deposits will* probably have to be exploited by novel methods, many still only conceptual.

The unique nature of these solid hydrocarbon deposits suggests that a novel method applying heat, not unlike underground gasification of coal, would be successful on fluidizing the kerogen or bitumen. A promising method underground retorting, in which the oil-bearing formation in-sutu is subjected to combustion or is heated by electricity or by forcing through high-temperature, high-pressure gases or liquids. Traditional exploitation methods, either surface or underground, may be combined with surface retorting, but the mining is not novel. The discussion here is limited underground retorting and, because of their prevalence in the United States, to oil shales.

Mineralogically and petrographically, oil shale is a fine-grained marlstone containing a solid, insoluble organic substance called kerogen (Marsden and Lucas, 1973), When heated to 900-950°F (480-510° C) kerogen decomposes to oil, gas, and a carbonaceous residue of spent shale (char). The chemical process of pyrolysis or cracking of kerogen is termed *retorting* . The main Object of in-situ (underground) retorting is to extract the kerogen in «he form of liquid fuels with gaseous by-products. While simply described, he process involves complex thermodynamic problems of heal transfer, pyrolysis kinetics and process control.

In addition, two natural characteristics of the oil shale act as deterrents: (1) Its very low thermal conductivity impedes efficient heat transfer, and (2) its extremely low porosity and permeability hamper fluid passage. . . , . ,

The inadequate permeability of oil shale is the most critical problem to overcome. It generally requires the creation of artificial fractures and channels in the formation in order to retort it in place. Broadly, the different techniques to accomplish fracturing plus retorting fall in two generic categories: (I) true in-situ retorting and (2) modified in-situ retorting. *True in-situ operations* are carried out entirely from the surface, usually through boreholes, while *modified in-situ methods* require some underground mining.

When necessary with true in-situ retorting, formation permeability is increased by massive fracturing prior to retorting (A non., 1980b). This is achieved in three ways, similar to methods used in conventional petroleum production or underground gasification of coal: (1) hydraulic or pneumatic fracturing by high-pressure fluid injection; (2) application of high-voltage electricity or electro linking; and (3) explosives fracturing by large-scale detonation of liquefied blasting agents or slurries.

Various processes for true in-situ retorting are envisioned: one concept appears in Figure 14.16. Six steps are involved (Anon.,1980b; Russell, 1981; Kalia and Gresham, 1985)

1 . Drilling a predetermined pattern of wells, probably three to five in number, in which an injection hole is ringed by production holes

2. Dewatering the zone to be retorted if the deposit occurs in the water table and the method requires a dry environment.

3 . Fracturing the deposit to create or increase permeability

4. Igniting the bed by burner or injecting a hot fluid to provide heat for combustion and pyrolysis.

5. Pumping compressed air and/or oxygen into the formation through an injection well to support combustion and force hot gases through the fractured rock to aid retorting.

6. Recovering the retorted oil through the producing wells

Step 3 is omitted if the oil shale is sufficiently permeable for pyrolysis to be carried out by a superheated fluid (natural gas, air, or steam) in step 4. ”

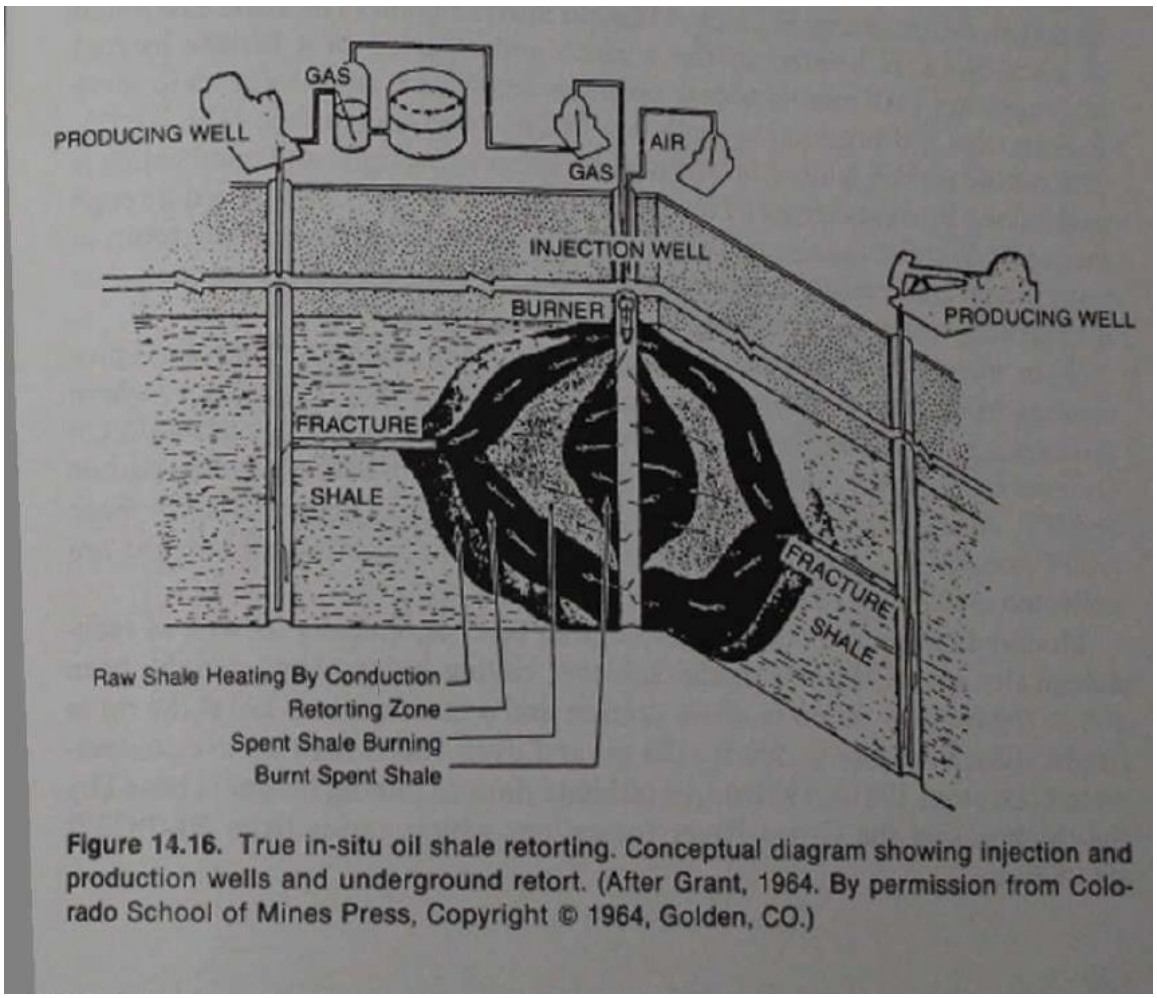


Figure 14.16. True in-situ oil shale retorting. Conceptual diagram showing injection and production wells and underground retort. (After Grant, 1964. By permission from Colorado School of Mines Press, Copyright © 1964, Golden, CO.)

Pyrolysis in underground shale retorting is similar to combustion in underground coal gasification.

Limited field tests have been conducted of true in-situ retorting, but many uncertainties remain, and commercial application is remote. Use appears restricted to relatively thin and permeable deposits under shallow cover (anon., 1980 b)

In contrast, modified in situ retorting has undergroune full-scale field tests and has advanced to a moderate level of technological readiness. While no commercial operations exist at the time of this writing, they appear feasible, economics permitting (Anon., 1980b)

In the modified process, the permeability of oil shale deposits is increased substantially by mining 20-40 % of the shale to creative a void (or voids) in the deposit and then blasting the remainder of the shale into the void to rubblize it (Russell, 1981). The block and chamber formed in the shale constitutes an underground retort. Various mining methods are employed to create the chambers; most excavate a single void near the bottom of the retort, but sublevel caving can be utilized to form multiple voids. A sequence of block and retort development by sublevel caving is illustrated in Figure 14.17

The steps in-situ retorting are as follows .(A n o n .. 1980b; Kalia a n d Gresham, 1985). After main and secondary development opening, are dm en to the block to be retorted, single or multiple voids are excavated, and the oil shale remaining in

place is blasted into the voids. (The shale excavated in each block is hoisted to the surface and retorted in a facility located aboveground.) All mining access openings are scaled, and boreholes to serve as injection and production wells arc drilled from the surface (Fig . 14.17).

The rubble is then ignited (at the top) by burning fuel gas, and combustion is maintained by injecting air. The burning /one progresses downward through the retort pyrolyzing the oil, which is collected at the bottom o f the retort in a sump and pumped to the surface. Shutting of The air supply extinguishes the burning when the entire block has been pyrolyzed. Four major zones arc encountered in a burning underground retort, proceeding from the top down (Dayton. 1981a, 1981b): (I) preheat zone, where the inlet gas is heated by thc spent shale (from 300-1500°F, or 150-815°C); (2) combustion zone, where oxygen is consumed by burning residual carbon (I500°F, or 8I5°C): (3) retorting zone, in which kerogen begins to decompose (900°F, or 480°C); and (4) cooling zone, where the retorted oil and gas are collected (I6 0 °F, or 70°C).

Modified in-situ retorts must be large to be economically as w ell as technologically sound. Experimental sublevel caving blocks have usually been 160 x 160 ft (50 x 50 m) in cross section and a maximum o f 270 ft (80 m) in height. Block heights to 500 ft (150 m) and even 700 ft (210 m) are contemplated (Dayton. 1981a, 1981b). An ultimate limit to mining height is posed by the thickness o f the Green River formation, which varies from 50-1870 ft

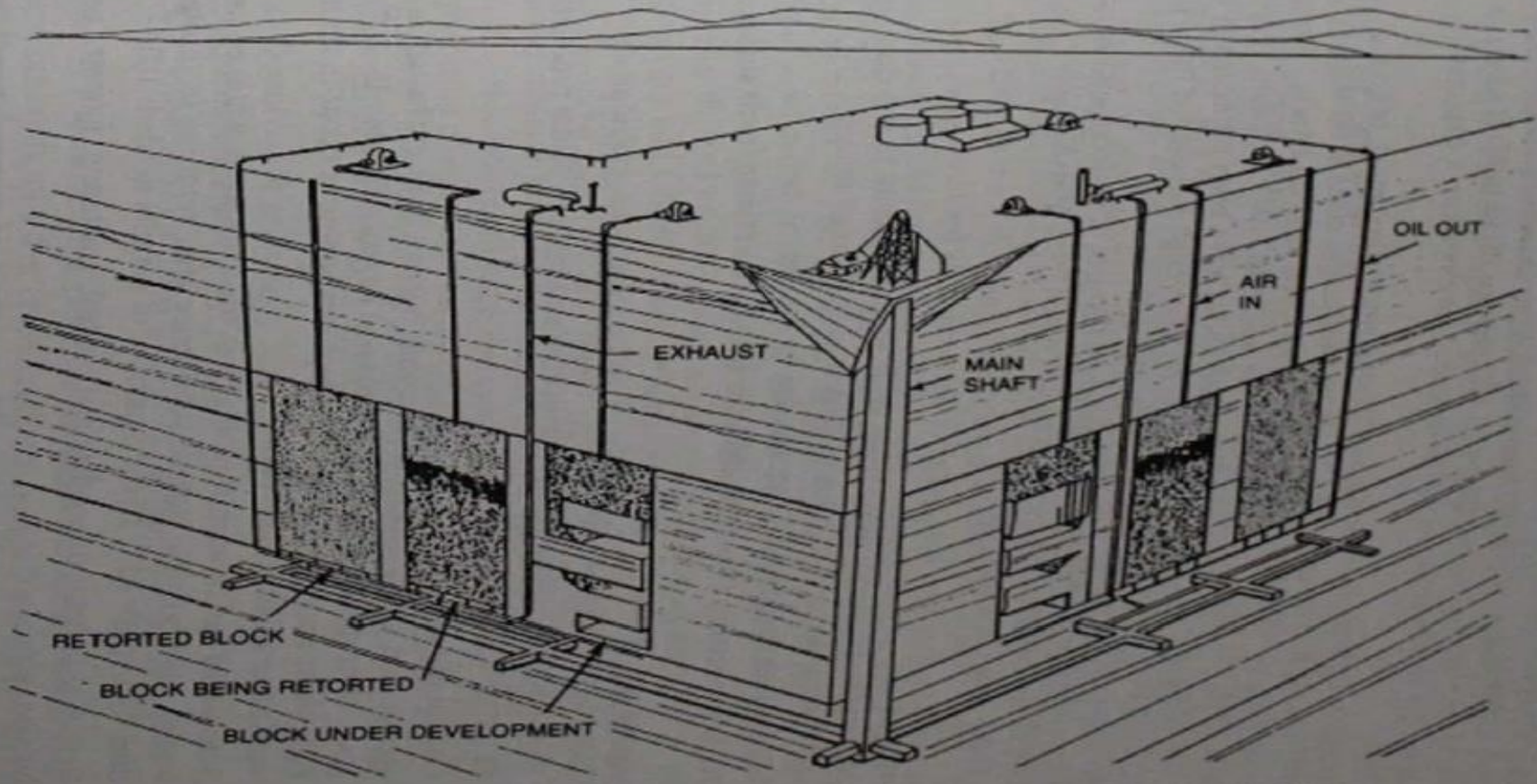


Figure 14.17. Modified in-situ oil shale retorting. Conceptual diagram showing injection and production wells and underground blocks, rubblized by sublevel caving, being retorted. (After Jee, 1977.)

The production scale must be similary large to be economic. It has been projected that a modified in-situ operation would have to mine and retort on surface 20,000 tons/day (18,000 tonnes/day) and blast and retort underground 80,000 tons/day (72,000 tonnes/day). Which shale containing 30 gal/ton (125 L/tonne), the yield would be 55,000 nnl/day (8750 m^3/day) of oil. An estimated 20 blocks (retorts) would be under development or exploitation at one time.

Mining-retorting costs for the modified in-situ method must, of course, be competitive with conventional petroleum production costs if oil shale is to compete with oil. To date, that goal has been attained (Anon., 1980 b; Russel , 1981;Dayton, 1981 a; DeGabriele and Aho, 1982; Rajaram , 1985). Capital costs alone for the mine described above have been estimated at $3 billion. Unit costs for the most economic oil shale mine still exceed those for conventional oil by at least $5/bbl ($30/m^3)

We can identify the following as following as important features of underground retorting of oil shale, referencing the sources just noted (plus Marsden and Lucas, 1973; Kalia and Gresham, 1985):

Advantages

1.Makes the recovery of oil from shale technologically (if not economically) feasible, and at a lower cost than traditional mining

2. Minimal environmental damage due to mining

3. Higher recovery (45-50%) than for traditional mining in oil shale

Disadvantages

1. Restrictive method, applicable only to solid hydrocarbons

2. Economically uncompetitive with conventional petroleum production at today's prices; high risk

3. Capital-intensive

4. Health and safety risks considerable but not well evaluated (e.g., gassy atmosphere constitutes explosion hazard and adds 15—20% to equipment cost, combustion and high temperatures occur in retorts, ventilation requirements high)

5. Substantial environmental problems in waste disposal, water contamination, and possible surface subsidence after retorting (estimate of time to prepare E1S: 2 years; number of permits needed: 100)

6. Retorting difficult to regulate and control (fracturing, ignition, combustion, etc.)

7. Private investment discouraged by government ownership and regulations

8. Waste disposal required for 80% of mined material (e.g., 16,000 tons/ day, or 14,400 tonnes/day. in the previous example)

9. High energy requirements for retorting (estimated 730 MJ/tonne 630.000 Btu/ton. Or 730 MJ/tonne)

10. High water requirements for surface retorting (1.4-3.6 bbl/ton, or
0 .3-0.6 m'/tonnc), potentially a serious problem in an arid region

11. Socioeconomic impacts great on a sparse population.

14.8 OCEAN MINING

Ocean mining is probably the only mining method-traditional or novel-more bedeviled by legal and political constraints than by technological or economic. Not that all technical and economic problem have been solved; many remain , as do environmental and social issues. But the current unsettled state of international offshore boundaries and the unresolved jurisdiction over deep-sea mineral deposits have paralyzed all major progress in ocean mining, coastal petroleum activity excluded.

The Law of the Sea Treaty, negotiated in 1980 and signed in 1982 by 130 nations, was intended to resolve all jurisdictional and usage disputes over the oceans, including mining rights. The United States (and three other nations), however, has refused to sign or to abide by the treaty. Instead, the United States in 1983 declared its jurisdiction over all mineral deposits within a so-called Exclusive Economic Zone (also ark , posits treaty), extending 200 nautical miles (370 km) from its coasts, mainland and possessions, and encompassing an area of 625 million mi^2 ($1.6*10^9$ km^2) (McGregor and Offield, 1983). What impact this action and other unilateral U.S. declarations may have on the development of an ocean mini remains to be seen (Blitgen, 1982). An earlier step, the issuance in 1981 of interim mineral leasing regulations as an incentive, has had no apparent effect. Aside from petroleum and natural gas, there has been relatively li.Ue
exploitation of even shallow coastal deposits to date (Baram et al 1978
Mineral resources in the marine environment fall into three categories
(Cruickshank and Marsden, 1973):

1. Dissolved in seawater
2. Unconsolidatcd on the seabed
3. Consolidated on the seabed or in the underlying bedrock.

Table 14.7 identifies the principal minerals and elements (including fresh water) presently mined or considered recoverable from the oceans. They fall in all three economic mineral groups (metal, nonmetal, and fuel) but. Excluding petroleum, constitute only about 0.1% of today’s world mineral production, with a value of about $2 billion (Cruickshank et al., 1968). Distribution among the three groups is about 25%, 25%, and 50%, respectively

TABLE 14.7 Classification of Marine Mineral Deposits

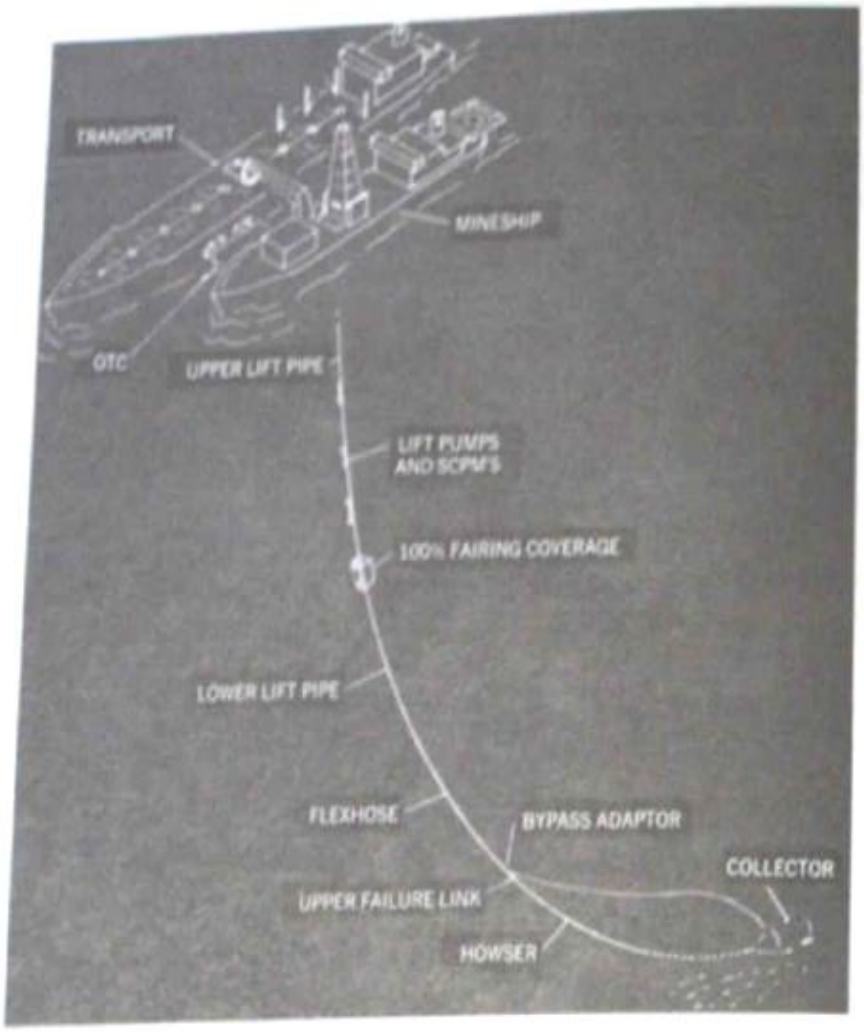
Dissolved	Unconsolidated			Consolidated
	Continental Shelf, 0–700 ft (0–200 m)	Continental Slope, 700–12,000 ft (0.2–3.5 km)	Deep Sea, 12,000–20,000 ft (3.5–6.0 km)	
Seawater	*Nonmetallics*	*Authigenics*	*Authigenics*	*Disseminated, massive, vein, tabular, or stratified deposits of:*
Fresh water	Sand and gravel	Phosphorite	Ferromanganese nodules associated with:	Coal
metals and salts of:	Lime sands and shells	Ferromanganese oxides and associated minerals	Cobalt	Ironstone
Magnesium	Silica sand	Metalliferous mud with:	Nickel	Limestone
Sodium	Semiprecious stones	Zinc	Copper	Sulfur
Calcium	Industrial sands	Copper	*Sediments*	Tin
Bromine	Phosphorite	Lead	Red clays	Gold
Potassium	Aragonite	Silver	Calcareous ooze	Metallic sulfides
Sulfur	Glauconite		Siliceous ooze	Metallic salts
Strontium	*Heavy Minerals*			Hydrocarbons
Boron	Magnetite			
Uranium	Ilmenite			
Other elements	Rutile			
Metalliferous Brines	Monazite			
Concentrations of:	Chromite			
Zinc	Zircon			
Copper	Cassiterite			
Lead	*Rare and Precious Minerals*			
Silver	Diamonds			
	Platinum			
	Gold			
	Native copper			

Source: Cruickshank and Marsden, 1973. (By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Our discussion of novel mining methods for ocean minerals will be limited to those in the second category category above, unconsolidaded deposits on the seabed. The technology for recovery of the first group, dissolved minerals, is well advanced, comprising a field of knowledge more akin to chemical or metallurgical engineering than to mining engineering. Not all processes are economic at present, of course, but probably all natural elements exist in seawater, and many in time may be recoverable. Those in the third group, consolidated deposits, will (1) probably be minable with technology borrowed from offshore petroleum production, mainly platforms of various types and artificial islands, if distant from shore; or (2) continue to be exploited from shore if within 8-10 mi (13-16 km), the limit imposed by ventilation practice (Wood. 1980). Some offshore technology applicable to consolidated deposits is also discussed in the following.

In Table 14.7. unconsolidated mineral deposits are sub classified on the basis of increasing depth of occurrence as continent shelf (<700 ft, or <200m), continental slope (700-12.000 ft or 0.2-3.5 km), and deep sea (> 12.000 ft, or >3.5 km
Of the approximately 70 active mining operations in unconsolidated deposits, all are located in shallow beach-type deposits on the continental shelf (Cruickshank and Marsden, 1973). The discovery in the 1950 s of vast deposits, of ferromanganese nodules, also rich in cobalt, nickel, and copper, as well as massive sulfide plumes formed along rift zones in deep basins of several of the pecans has lent a power full incentive to mining at great depth (Mero, 1965; MeGregor and Offield, 1983)
Geologically, unconsolidated marine deposits, regardless of depth, resemble placer deposits found on dry land, because they are mechanical concentrations of (generally) heavy minerals. As such they are amenable to the principles of placer mining and, conceivably to modifications of the practices, especially dredging (Sections 7-2 and 7-3) It is very likely that novel methods being developed to mine the deeper unconsolidated deposits will utilize first-generation devices that resemble present-day mechanical and hydraulic dredges, manned remotely by sophisticated automation.

An ocean-mining system conceptually consists of three components; (1) platform. (-) coupling and hoist, and (3) excavating unit (Fig. 14.18. *top)* Originally, in the transition from fresh to salt water and from shallow to deep bodies of water, the platform was floating and moored at the surface the coupling was rigid, and the excavator was integrally attached (Fig. 14. 8 *bo,lorn).* That is changing, as the figure suggests, in the evolution of ocean mining. In the future, the excavator may be bottom-mounted as a slave unit the coupling will necessarily be flexible, and the platform may be located at the surface or bottom. An entirely submersible system-except for the surface transport facility— may ultimately be possible.
Excavation units to date have been modified from dredge designs (Fig. 14.19, *top).* Both mechanical and hydraulic units have been utilized. While the digging action of hydraulic dredges is less positive and efficient, they permit the use of flexible and continuous materials-handling (hoisting) sys



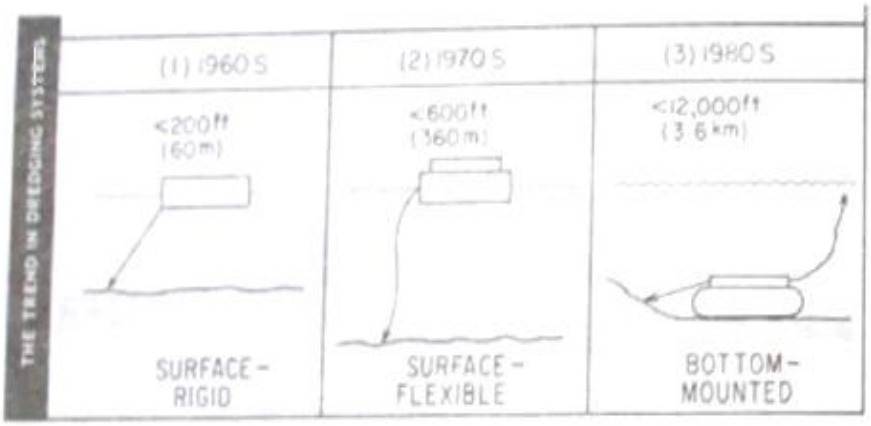


Figure 14.18. (*Top*) Conceptual configuration for deep-sea mining system. (After Halkyard, 1979. By permission from American Mining Congress, Washington, DC.) (*Bottom*) Transition in nature of coupling between excavating unit on ocean bottom and platform at surface: (1) surface-rigid, (2) surface-flexible, and (3) bottom-mounted. (After Cruickshank et al., 1968. Copyright © 1967, *Engineering and Mining Journal*, New York.)

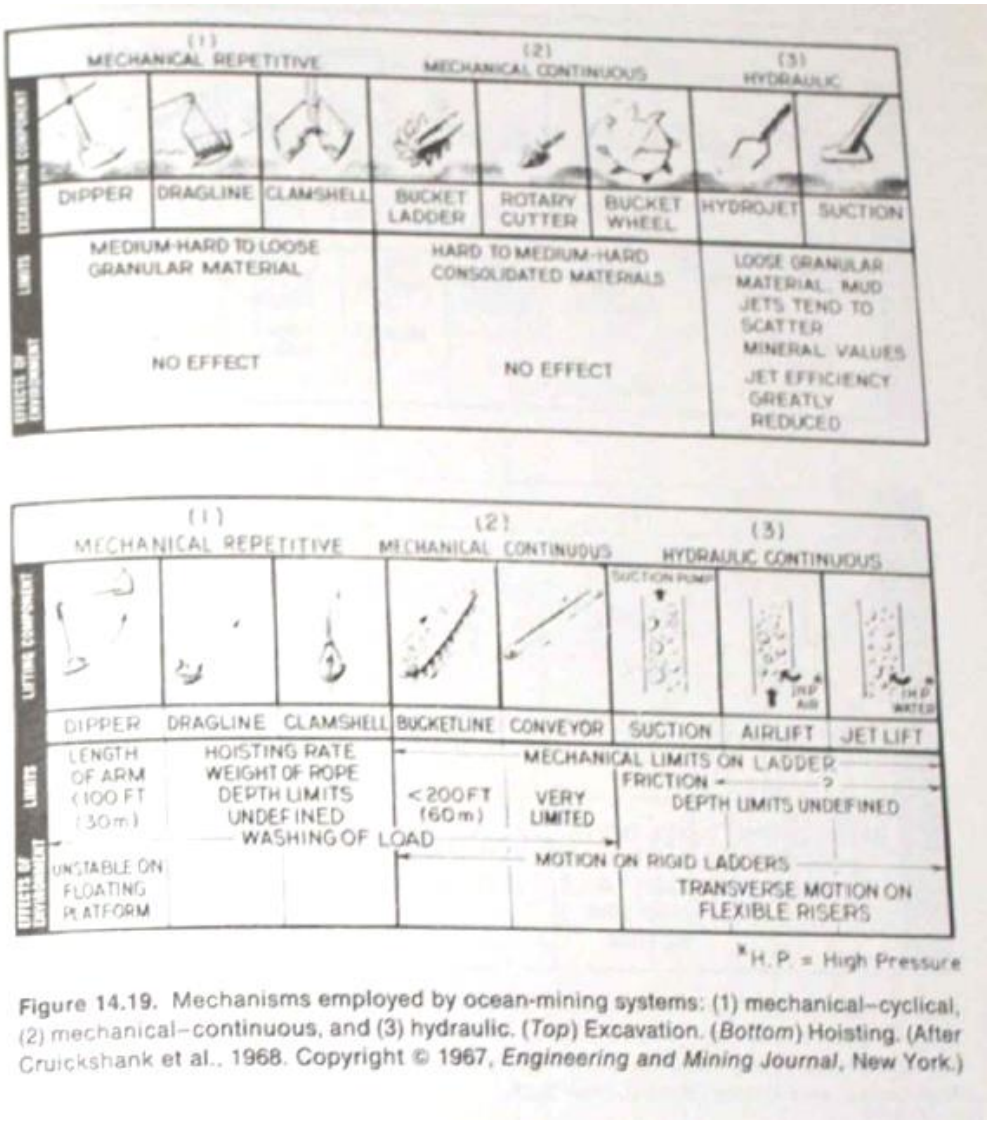


Figure 14.19. Mechanisms employed by ocean-mining systems: (1) mechanical–cyclical, (2) mechanical–continuous, and (3) hydraulic. (*Top*) Excavation. (*Bottom*) Hoisting. (After Cruickshank et al., 1968. Copyright © 1967, *Engineering and Mining Journal*, New York.)

Tems , either hydraulic or airlift (fig 14.19, bottom). Cruickshank and Marsden (1973) and Li (1975) offer detailed comparisons of various underwater excavators. The third component of an ocean-mining system is the platform (Fig. 14.20, *top).* Floating surface units still prevail, but for stability, submerged platforms (borrowed from offshore oil well technology) are contemplated. Eventually, seafloor platforms with bottom-mounted excavator units arc also envisioned. Positioning and control devices (Fig. 14.20, *bottom)* must be elaborate for surface platforms, especially if rigid coupling is required;

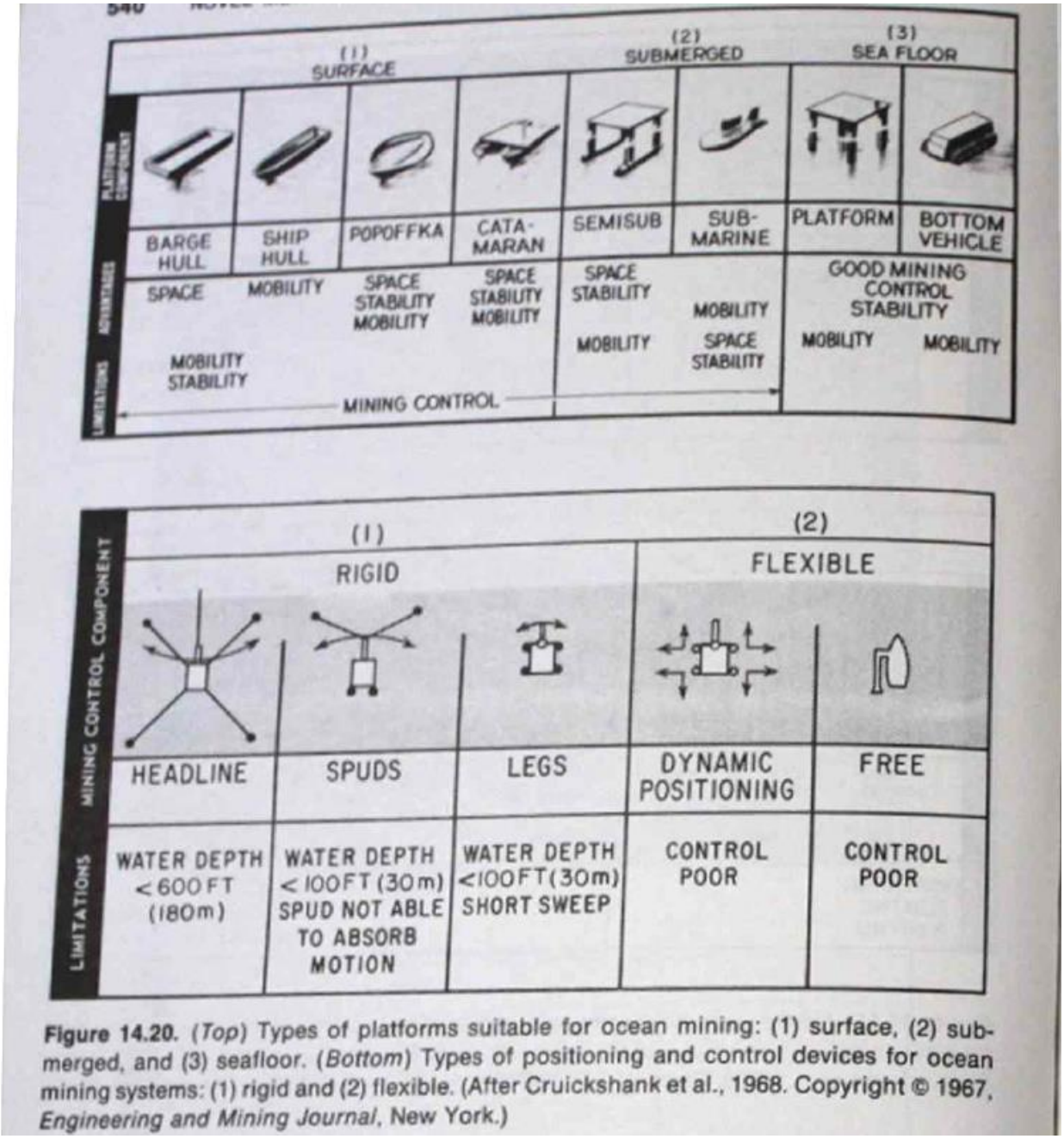


Figure 14.20. (*Top*) Types of platforms suitable for ocean mining: (1) surface, (2) submerged, and (3) seafloor. (*Bottom*) Types of positioning and control devices for ocean mining systems: (1) rigid and (2) flexible. (After Cruickshank et al., 1968. Copyright © 1967, *Engineering and Mining Journal*, New York.)

they are much simpler if flexible coupling is permitted but may provide poorer system control.

Engineering design of ocean-mining systems has advanced from the study and drawing-board stage to model tests in deep water (Donkers and de Groot, I974; Tinsley, I975; Welling, I976; Halkyard, 1979). A comprehensive array of the system options under consideration, including transport and processing, appears in Figure 14.21. Although many competing designs are available, probably the major component feature to be resolved is the nature of the excavating unit. The competition among existing designs is basically between mechanical and hydraulic dredges (Li, 1975; Petters and Brockett 1984).

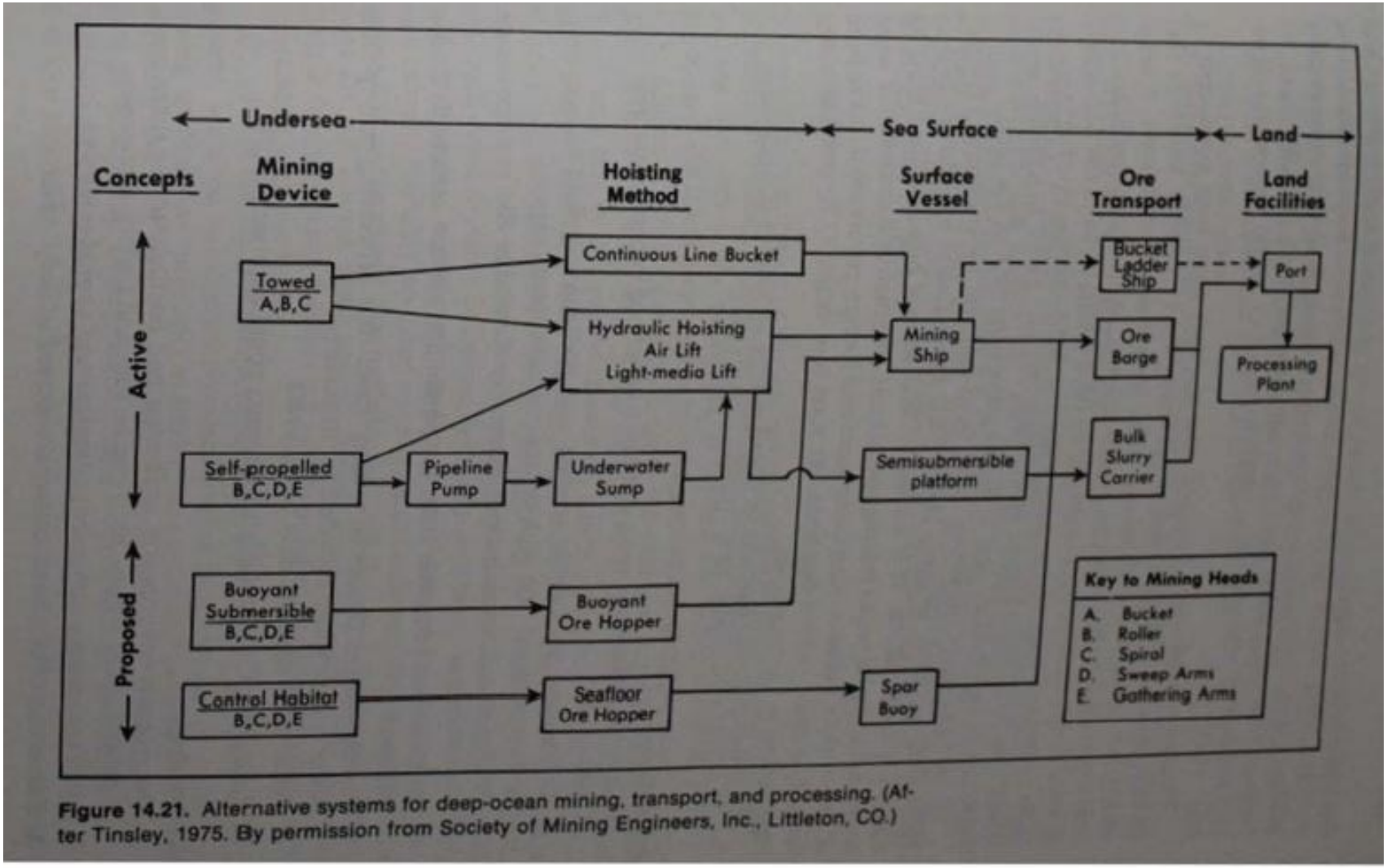


Figure 14.21. Alternative systems for deep-ocean mining, transport, and processing. (After Tinsley, 1975. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Extensive production rate and cost analyses have been conducted by private sources and government agencies. They generally conclude that within the *bounds of* technology under development or forseen, a deepocean mining industry is economically feasible. Mining costs, probably 25% *of total production* costs, arc projected (mid-1980s) to be in the S25-50/ton (528-Wfonne) range for nodules valued at $ 150-200/ton ($ 167-222/tonne) (Hartman *1962:* Cruickshank et *al.*, 1968; Tinsley, 1975). In addition, transportation is expected to cost 25% and processing *50% of* the total.

Investment costs, however, have been and will be enormous (Tinsley,1975: Welling, 1976; Blitgen. 1982). K&D expenditures by industry to date approach $500 million. It is estimated that several billion dollars must be expended to make ocean mining operational. International consortia of some *of the* world's major mining companies and foreign governments have been *formed to meet capital* demands. At least four major U.S. companies are *represented in* the consortia.

We summarize the features *of* ocean mining as follows (Cruickshank and Marsden. 1973; Welling, 1976; Baram et al., 1978):

Advantages

1. Deposits extensive, comprised of a variety of minerals, and relatively high-grade
2. Technology thoroughly studied and preliminarily field-tested; some systems untried, but with no serious conceptual gaps
3. Economics of exploitation apparently favorable for systems under consideration
4. Existence of consortia from the public and private sectors to exploit deposits
5. Environmental impact of mining small (but processing may pollute water unless waste recycled for disposal)
6. Placer-type deposits easy to excavate
7. Recovery of unconsolidated deposits could approach 100%

Disadvantages

1. Deep-ocean environment unfamiliar and hostile; storms (hurricanes, typhoons, tidal waves, etc.) can be life- and facility-threatening
2. Most technology unproven, complex, and capital intensive
3. Remote control and automation untested in deep water
4. Servicing and salvage at great depth difficult, expensive time consuming, and
5. Questions of ownership and mineral rights unresolved, matters of international law at stake; military action and piracy possible threats
6. Great distance from land exacerbate supply and shipping problems
7. Sociopsychological problems due to shipboard isolation likely for crew

14.9 OTHER NOVEL METHODS

Only two novel mining methods are considered to be questionable with regard to practical application. Both are new, and neither has yet been utilized. It is unlikely that they will be applied in the near future, but because of their potential value and the possibility that changes may be made in constraints on their use, they are described here briefly under this miscellaneous category of methods.

Nuclear Mining

The peaceful application of nuclear energy for fragmentation and excavation purposes in mining, petroleum, and construction has been the subject of endless speculation and proposals. It formed the major thrust of Project Plowshare, th R&D program on peaceful uses of the atom sponsored by the U.S. Atomic Energy Commission (now the Nuclear Regulatory Commission) in the 1950s and 1960s. Since ratification of the Limited Nuclear Test Ban Treaty late in that period, however, which prohibited surface detonations and underground tests venting to the surface, applications of nuclear blasting and mining are no longer seriously being advocated. In this respect, nuclear mining falls into the same category as ocean mining, in which political-legal considerations comprise the major deterrents to commercial use.

Unfortunately, nuclear mining faces formidable safety-assurance obstacles as well. Probably no other environmental risk troubles the public consciousness to the extent that radiation hazards do—and rightly so. Distinctions mare not always made among nuclear reactors, nuclear blasting, and nuclear weapons when hazards arc assessed, however, and blasting is often condemned with the bomb. Because of the current unfavorable image projected by nuclear energy in general, we need not discuss proposed mining uses exhaustively. Surface mining applications, primarily crate ring for excavation purposes, are ruled out under the Test Ban Treaty and for safety reasons and can be dropped from consideration. Underground-mining uses of nuclear blasting are more promising; they produce a cavity, crushed rock in a rubble chamber, and a fractured zone (Fig. 14.22). Some likely applications are for block caving, in situ leaching, underground coal gasification, and underground oil shale retorting.

Assessment of the legitimate merit of these potential mining applications is not easy. Much general and speculative information is contained in the publications of Lawrence Livermore National Laboratory operated by the University of California, especially the proceedings of the five Plowshare Symposia (the last: Anon., 1970). From a mining perspective, however, the literature is less prolific. Some helpful references are Hartman (1966), Anon. (1967b), Hardwick (1967), Williams et al. (1969), and Marsden and Lucas (1973).

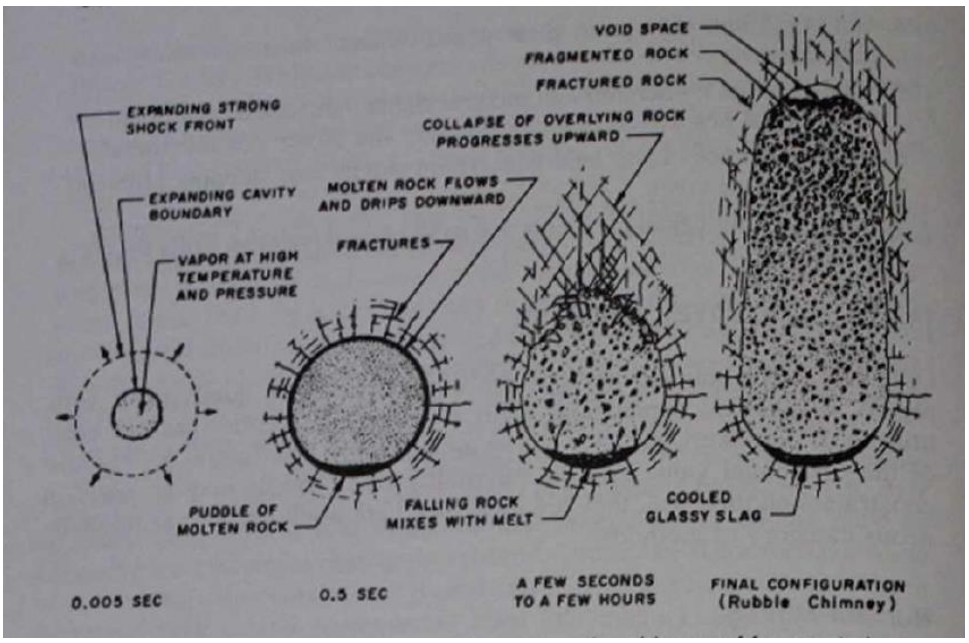


***Figure* 14.22** Sequence of events in formation of a cavity, chimney of fragmented rock, and fractured zone during detonation of an underground nuclear explosive. Time intervals are indicated. (After Anon.. 1967b. Copyright © 1968. *Engineering end Mining Journal*, New York.)

Based on these sources, we can deduce the following with regard to the technological, economic, and safety feasibility of nuclear mining, bearing in mind that field tests have been limited to noncommercial civil and military experiments at the Nevada Test Site:

1. The general engineering feasibility of nuclear blasting for mining purposes has not been demonstrated. Certain uses appear attractive, but specific tests designed to explore mining applications are needed.
2. Technological feasibility is lacking because there is insufficient knowledge about scaling relations, effects in different media, results with large yields, and effects of multiple shots; further, much technical information is classifield.
3. Economic feasibility cannot be assessed conclusively from published data but appears most attractive with large yields (i.e. > 10 kilotons, or >9 kilotonnes, of TNT equivalent)
4. Safety feasibility is not reasonably—let alone conclusively – established, even for contained underground shots. Aside from serious doubts about radiation, there are environmental concerns related to groundwater contamination and seismic damage.

It is not likely that the international political scene will soon change and permit the resumption of evil underground tests of nuclear explosives. Until it does, nuclear mining will remain unproven and unexploited.

Extraterrestrial Mining

Easily the most remote of all the novel mining possibilities, *extraterrestrial mining* raters to the exploitation of mineral resources on the moon and elsewhere in outer

space Questions of a political-legal-economic nature aside, technological feasibility remains the abiding imponderable. Speculation about lunar mining has been rampant since NASA's Ranger and Apollo landings on the moon (Ruzic, 1964; Delinois, 1966; Penn, 1966) The nature and extent of lunar mineral resources, of course, is not well known even after Soviet and American exploratory landings. Analyses of surface samples collected by probes, drills, and human explorers indicated a porous, lava-like mantle on the moon. Based on similar geologic occurrences on earth, there has been speculation that " commercial" deposits of iron ore, some of the base metals, precious metals, diamonds, and rare earths may exist on the moon. If mining is someday conducted on the moon, native supplies of oxygen, water, and mineral fuels would be desirable if not requisite for sustained production. Without corroborative evidence, affirmative answers cannot be given, but oxygen (combined with other elements) and water (as ice) may exist. Conventional fuels and energy (other than a remote chance of geothermal) are unlikely, however, which means that external power supplies— perhaps solar, nuclear, or fuel-cell— would have to be transported from earth to the mine site. Extraterrestrial mining beyond the moon (planets, comets, asteroids, etc.) is even further out in the realm of conjecture. One thing seems certain in either lunar or outer-space mining: Exploitative systems will probably have to be automated or robotic, because the environment is too hostile for *long term* human endurance. Lunar drills and vehicles used in the Apollo *landings should lend thems*elves well to automation (Peckham and Dallas. |«/M). *Other mining* operations (e.g.. blasting, loading) pose more formidable *problems (Delinois. I'M).* The technological feasibility *of* unmanned as well as *manned* space *exploration* has been demonstrated by the U.S. and Soviet lunar *programs,* but sustained rather than short-term missions arc beyond *present* capabilities. We have said nothing *of economic* feasibility. Why should anyone want to mine on the moon when ample minerals remain on earths Establishment of a *hoped-for* permanent *camp or colony on* the moon could provide the rationale, since self-sufficiency in minerals would be imperative; but such an outpost has been *indefinitely postponed. If there* is justification—and it may involve noncommercial space *exploration, or* military objectives, rather than economics—there is general agreement that extraterrestrial mining and colonization in *space must now* await launching *of* NASA's orbiting space station, *planned for the* 1990s. Both the technology and economics of mining in space *should then be more attractive.*

14.10 *IM PO RTAN CE AND* SUMMARY

We may summarize the importance and status of each of the novel mining methods as follows.

Existing Methods

I. *Rapid excavation:* Truly continuous extraction and handling systems for hard-rock mining await a breakthrough and remain a distant possibility; but progress is

being made, and the legitimacy of the goal is now widely accepted for both development and exploitation.
2. Automation and robotics: Manless or remote control in mining is attractive for reasons of safety, but widespread adoption depends upon more technological ruggedness, which in turn should produce economic feasibility.
3. *Hydraulic mining:* Applications of water-jet technology are advancing slowly into various unit operations (penetration, fragmentation, and handling), toward a clear goal of an integrated mining system. Extension from coal to harder rock is a companion objective.
4. *Methane drainage:* Signs are favorable for rapid expansion of coat bed degasification throughout the underground coal-mining industry, in part on safety grounds but also with economic justification. Drainage from seams that arc not currently being mined is equally attractive.

Promising Methods

1 . *Underground gasification*: Ripe with promite for difficult conditions, in-situ coal gasification has been fraught with economic risk and technological difficulties.
2. *Underground retorting:* In-situ oil shale retorting faces some technological uncertainties but, more serious, has yet to demonstrate economic viability. Unfortunately, its future is tied to that of the sickly synthetic fuels industry, which is held an economic captive by the international oil cartel.
3. *Ocean mining:* There are intriguing technological possibilities for mining rich unconsolidated deposits in the deep oceans, but political and legal risks are too great until an acceptable treaty of the sea is negotiated by the United States.

Questionable Methods

1. *Nuclear mining:* No applications are likely as long as radiation hazards are uncontained (and the Limited Nuclear Test Ban Treaty remains in effect), in spite of technological promise and economic attractiveness for certain fragmentation applications in underground mining
2 . *Extraterrestrial mining:* The furthest -out of all the novel method, colonization of outer space (most likely site: the moon) is a must to justify risky, untried extraterrestrial mining.

15 MINING METHOD SELECTION AND SUMMARY

15.1 COMPARISON : SURFACE vs. UNDERGROUND MINING

15.2

In this, the final chapter of the book, it is appropriate to offer in-depth comparisons (including costs) of the various mining methods discussed previously and to develop method selection procedures before attempting a general summation. We first compare surface and underground methods, which distinguish the locale of mining.

In the summary chapters on surface and underground mining, some general observations were provided on the strengths and weaknesses of each group of methods (Sections 8.4 and 13.4). We are now in a position to examine and compare in more detail both the deposit conditions and the method features of the two locales.

Deposit Conditions

With each method studied, we have enumerated the principal geologic, spatial. and natural conditions conducive to the use of that method in the mining of mineral deposits (see Tables 8.2 and 13.2). They may be summarized as follows, relative to surface and underground mining:

1. *Ore strength:* Surface suitable for any, although some aqueous extraction methods require soluble, unconsolidated, or rublized deposits; underground ranges from weak (supported or caving methods) to strong (unsupported methods).

2. Rock strength: Similar to ore strength

underground methods limited to specific application

3.Deosit shape: Preferably tabular for surface horizontal; underground more versatile, ranging from tabular or massive to irregular.

4. *Deposit dip:* Surface applicable to any but preferably flat; underground versatile, ranging from flat to steep.

5. *Deposit size:* Usually requires moderate to large for surface; variable for underground but usually smaller and thinner (either surface or underground large-scale methods require larger deposit

6. Ore *grade:* Surface best suited for low-grade deposits; underground requires moderate- to high-grade deposits (except some caving methods)

7. *Ore uniformity:* Both surface and underground range widely, from, uniform to variable (generally, bulk methods require more uniformity.

8.Depth: Surface methods (except borehole).limited to shallow or moderate underground variable, but supported or caving methods can tolerate great depths.

While many deposit conditions exert significant influence in the selection of a mining method, it is evident that the most critical in distinguishing between

surface and underground are ore grade and depth. If the deposit is low-grade, surface is preferable, but at depth, only underground is feasible.!

Method Features

We cannot compare all the features associated with surface and underground methods, but we can note the principal advantages and disadvantages of the two locales (refer to Tables 8.3 and 13.6):

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

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

Overall Considerations

Qualitatively, we may conclude from this verbal comparison that, excluding depth *limitations,* surface mining is usually preferred over underground. There are certain *significant* factors which favor underground over surface mining, however, as noted, and these may govern in unique cases. Our final judgment in a specific case awaits the determination of costs and an economic analysis of competing candidate methods, which we will consider in Section 15.3.

15.2 COMPARISON: NOVEL vs. TRADITIONAL METHODS

Because their characteristics and applications differ drastically, it is more *difficult to* compare traditional methods with novel methods than surface with underground. By their very nature, novel methods have been devised to meet the unique requirements of a particular mineral deposit or to employ the unusual features of a new technology. Nonetheless, some general comparisons are useful and valid and may aid in the selection of one method over another.

Deposit Conditions

The characteristic tic of the mineral deposit which is most discriminating in the selection of a novel mining method is the mineral commodity itself (see Table 14.1). Grouped by commodity, we may compare novel with traditional methods.

1 . *Coal:* Hydraulic mining and underground gasification arc novel methods that compete with conventional underground methods, specifically, room and pillar and long wall mining. When natural conditions are favorable—a steeply dipping deposit for hydraulic mining and a deep permeable deposit for gasification— novel methods may in time become competitive economically.

2. *Coal bed m ethane:* As a source of natural gas, methane drainage competes more with conventional gas-production methods than with traditional mining methods. Already competitive technologically and economically, methane drainage is attractive because it is often a by-product of coal mining.

Additionally, it is advantageous in reducing the explosion hazard in gaseous coal mines, a vital safety feature. Deep coal scams already partially exploited by traditional means or too poor to mine are candidates for methane drainage.

3. *Hydrocarbons:* Kerogen contained in oil shale and bitumen in tar sand are natural hydrocarbons that can be converted to petroleum products following recovery. Traditional mining methods (open pit and stope and pillar) have been employed to exploit them, but oil shale especially is marginal. Underground retorting, a novel method, holds promise but is neither technologically nor economically feasible on a commercial scale at present. Natural conditions conducive to in-situ retorting are complete confinement at depth, high permeability, and a dry environment.

4. *Metallic and nonmetallic minerals:* Three novel methods are applicable to non coal mineral deposits: rapid excavation, ocean mining, and extraterrestrial mining. The most attractive at the present time is rapid excavation, which is limited to low- to moderate-strength rock and uniform conditions. It competes with conventional development techniques underground. Both ocean and extraterrestrial mining are futuristic methods of unknown to likely promise. Ocean mining is most suitable for unconsolidated deposits of considerable areal extent. Extraterrestrial mining has undefined deposit requirements and technological limits.

5. *All minerals.* Automation and robotics and nuclear mining are novel methods applicable to a wide variety of commodities. Automation fits those mining

conditions too perilous for traditional methods or inefficient for human operation; it is applicable to any mining method. Nuclear mining is not now considered a promising prospect but would be attractive for massive, deep-scated, low-grade deposits too difficult or costly to mine by traditional underground methods.

Method Features

While a direct comparison of the features of novel and traditional mining is tenuous, we attempt to identify some of the more significant advantages and disadvantages for the competing methods mentioned in the previous section, " Deposit conditions " .Comparison are based on current knowledge of novel methods. Because of the remoteness of any application, nuclear mining and exterrestrial mining are omitted from this discussion.

1. Mining cost: Lower costs may be realized for rapid excavation, automation, hydraulic mining, and methane drainage than for traditional methods. Underground gasification, underground retorting, and ocean mining are marginal and commercially untested.

2. *Production rate:* Those novel methods with a record of or potential for, large scale production are hydraulic mining, methane drainage, and ocean mining. Underground gasification and underground retorting are promising but commercially untested. Neither rapid excavation nor automation is intented to be compared on a production-rate basis.

3. *Productivity:* While most novel methods exhibit the potential to be highly productive, only hydraulic mining and methane drainage have demonstrated records of productivity that exceed those of traditional methods. Rapid excavation and automation show modest accomplishments, while there are no measures as yet for the others.

4. *Depth capacity:* Automation, methane drainage, underground gasification, underground retorting, and ocean mining arc expected to function *well* at great depths. Rapid excavation and hydraulic mining have depth limitations.

5. *Recovery: None of the* novel methods is expected to surpass competitive traditional methods in recovery, except perhaps hydraulic mining. With some, such as ocean mining, there is no basis for comparison.

6. *Environmental risk:* While environmental hazards arc not assessed in all cases, rapid excavation. automation, hydraulic mining, methane drainage, and ocean mining arc expected to cause little harm. Underground gasification and underground retorting, however, may be potentially damaging to the environment.

7. *Health and safety:* All novel methods, except perhaps underground retorting, *should* compete with traditional methods regarding health and safety. Automation excels.

In addition, there are special feature associated with the individual methods which affect their applicability. For these, refer to the listing of advantages and disadvantages in Chapter 14.

Overall Considerations

With uncertain knowledge of the operating features of the novel methods, we are probably justified in ranking their potential according Table . l4.1 that is, based on the likelihood of their commercial application. In order, as grouped in Chapter 14, they are as follows:

1. *Existing but limited present use:* rapid excavation draulic mining, methane drainage
2. *Promising but not yet in use:* underground gasification underground retorting, ocean mining.
3.Questionable or unlikely use: nuclear mining, extraterrestrial mining.

Attempts to make quantitative comparisons of traditional and novel methods-for example, cost estimates-are fraught with risk and imprecision and are probably valid only in site-specific instances.In the remainder of the chapter, only traditional methods are considered.

15.3 MINING COSTS

Budgeting and Cost Control

The nomenclature and procedure employed in budgeting is prerequisite to an understanding of cost estimation and cost control. We have adopted some of the general terminology in our cost examples, but acquaintance with more of the tails of mine budgeting will assist us in analyzing mine costs.

Table 15.1 compiles the elements of a typical budget, this for an underground coal mine. Customary cost categories, broken down into direct and indirect mining and other production costs, are tabulated for as many cost profit centers as desired and can be isolated. In this case, there are four: mine development, mine exploitation, underground auxiliary, and surface auxiliary. The equipment identified for each center is costed on an ownership (capital) and operating basis, as we did in Examples 8.1, 8.2, and 11.2.

Then other items of operating expense, categorized in the first column, are costed for each center, as applicable. Detailed cost entries arc shown for consumables, ower, and special items. An entire mine budget would itemize all these costs for ach center in a complete computerized tabulation, usually on both a total and unit cost basis, for the period under consideration (month, year, long range, etc.). Thus actual incurred costs can be compared with forecast budget costs and other cost estimates to provide the epitome in cost control. For a more comprehensive treatment of mine budgeting, the reader is referred to wraith (1973), Halls (1982), and Gentry and O'Neil (1984).

TABLE 15.1 Mine Budget Format for Hypothetical Underground Coal Mine

Cost Category	Cost–Profit Centers: Cost Items			
	Mining Unit A: Development (Entry Driving)	Mining Unit B: Exploitation (Longwall)	Underground Auxiliary Operations	Surface Auxiliary Operations
Direct/Ownership Cost	*Equipment*	*Equipment*	*Equipment*	*Equipment*
Purchase depreciation	Continuous miner	Shearer	Locomotives	Shaft hoist
Interest charge	Shuttle cars	Shields	Mine cars	Slope belt
Insurance, tax, etc.	Roof drill	Jacks	Jeeps, cars	Bins, hoppers
Equipment lease	Scoop tram	Panline	Main conveyor	Feeders
Equipment rental	Feeder	Scoop tram	Roof drill	Trucks, cars
Total ownership	Section conveyer	Feeder	Scoop trams	Main fan
Direct/Operating Cost	*Consumable Supplies*	Section conveyer	Cages	Monitoring system
Repairs, maintenance	Roof bolts	*Consumable Supplies*	Face fans	Buildings
Consumable supplies	Timber	Arches	Pumps	Other structures
Fuel, power	Rock dust	Timber	*Consumable Supplies*	Waste disposal
Lubrication, hydraulics	Brattice cloth, tubing	Rock dust	Track	*Consumable Supplies*
Labor, benefits	Miner bits	Brattice cloth, tubing	Conveyor belt	Plant supplies
Special	Drill bits	Shearer bits	Timber, trusses	Office supplies
Total operating	Hand tools	Water sprays	Roof bolts	Safety supplies, clothing
Total exploitation	Miscellaneous	Hand tools	Concrete blocks	Training materials
		Miscellaneous	*Pipe*	Miscellaneous

Cost Analysis

The ultimate basis for decision making selecting a mining method is economic.Assuming that safety and *technological* considerations are satisfied, we prepare cost estimates for all candidate methods in order to make a final choice. Usually the process is done in two stages: (1), If the deposit depth is shallow to moderate, compare approximate costs first for the general categories of surface vs. underground mining; and (2) once that has been resolved with some certainly, compare specific costs for promising, individual mining methods. Generally, overall mining costs including prospecting, exploration, development, and exploitation will suffice, but in some instances, consideration of all production costs (mineral processing, smelting transportation etc.) is necessary.

Surface vs. Underground Mining Costs

Inherently we have assumed underground mining costs to exceed surface mining costs (Sections 8.6 and 13.6). We now need to examine that premise, to analyze the key elements of cost in each case, and to compare typical costs for similar circumstances.

Hedberg (1981) cites reasons for the alleged cost effectiveness of surface mining, based on the hard-rock industry. They include larger equipment, *lower labor* intensity, *simpler* development, higher energy efficiency, and less expensive auxiliary operations (ground control, ventilation, supplies, *etc.). His analysis of unit* operating-cost items shows that for similar ore body configurations and comparable production rates, the major item of difference is labor. Materiel and supply costs (i.e., for explosives, spare parts, fuel, etc.) differ some—about *5III* more—but labor costs are five times higherin underground mining. To reflect this in approximating mining costs, for example, by *the* method of Example 6.1, use a lower labor productivity *(A)* and a lower ratio of labor to operating cost Continuing *his* analysis, using a relative-cost basis, Hedberg estimates operating, capital (ownership), and overall unit costs for comparable but hypothetical surface and underground mines (Fig. 15.1). This time, he investigates operating costs (Fig. 15.1, *left)* on a unit operation basis. Surprisingly, *costs for the* production operations differ little: Rock breakage and loading are lower but haulage is higher in surface mining (the latter because surface trucks on a steep grade are more expensive than underground conveyors or rail *haulage on* a *Hat grade*, *plus* hoisting). The major differences are *for auxiliary* operations and development work, which are many times Hedberg (1981) cites reasons for the alleged cost effectiveness of surface mining, based on the hard-rock industry. They include larger equipment, *lower labor* intensity, *simpler* development, higher energy efficiency, and less expensive auxiliary operations (ground control, ventilation, supplies, *etc.). His analysis of unit* operating-cost items shows that for similar ore body configurations and comparable production rates, the major item of difference is labor. Materiel and supply costs (i.e., for explosives, spare parts, fuel, etc.) differ some—about *5III* more—but labor costs are five times higherin underground

mining. To reflect this in approximating mining costs, for example, by *the* method of Example 6.1, use a lower labor productivity *(A)* and a lower ratio of labor to operating cost. Continuing *his* analysis, using a relative-cost basis, Hedberg estimates operating, capital (ownership), and overall unit costs for comparable but hypothetical surface and underground mines (Fig. 15.1). This time, he investigates operating costs (Fig. 15.1, *left)* on a unit operation basis. Surprisingly, *costs for the* production operations differ little: Rock breakage and loading are lower but haulage is higher in surface mining (the latter because surface trucks on a steep grade are more expensive than underground conveyors or rail *haulage on* a *Hat grade, plus* hoisting). The major differences are *for auxiliary* operations and development work, which are many times costlier underground.

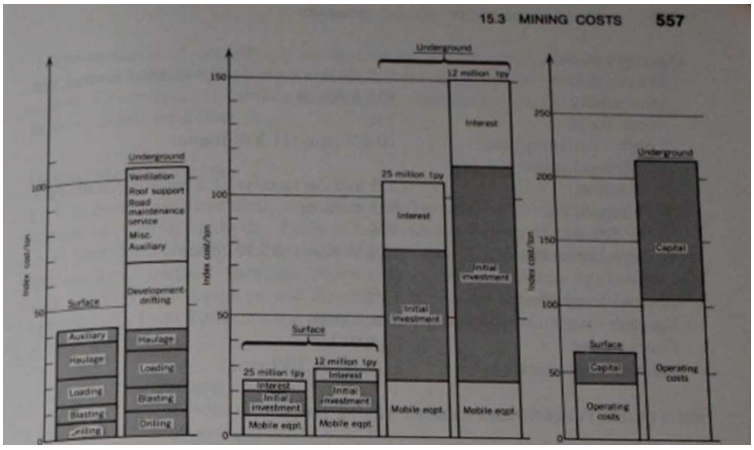


Figure 15.1 Unit cost comparisons on relative basis for surface and underground mining: hypothetical but similar conditions (left) Operating costs (Center) Capital costs. (Right) Overall costs. Note: scales differ. (Modified after Hedberg, 1981.By permission from Mining Magazine, London.)

Comparing unit operating costs for the two categories demonstrates that underground mining typically is over twice as expensive as surface mining.

Capital costs (Fig. 15.1, *center*) are also higher for underground than surface ining. Here Hedberg selects two different production rates; the higher rate is some 25% more cost effective in both cases. He breaks down capital expense into equipment purchase price and utilities and facilities (higher interest accrues for the underground investment because of the greater cost and longer time frame). The result is a unit capital cost that is five times higher for underground than surface mining.

In Figure 15.1 *(right),* overall costs, the sum of operating and capital costs, are represented (based on the higher production rate). Quite clearly, in this hypothetical case, unit overall costs are about four times as high for underground as surface mining. While the ratio may vary, results encountered in actual practice are nearly always in agreement.

A numerical example demonstrates the magnitude of the absolute costs involved for surface and underground hard-rock mining (Nilsson, 1982c): example 15.1. Estimate the overall mining costs for surface and underground mining, given the following:

Open pit mine	
Production	*6.5 million tons/yr (5.9* million tonnes/yr
Investment	$25 *million*
Work force	*130*
Energy consumption	*10 kW/ton (I I kW /tonne)*
Underground mine	
Production	*2.0 m illion tons/yr (I . Я* million tonnes/yr)
Investment	$65 million
Work force	275
Energy consumption	*25* kW/ton (28 kW/tonne)
General	
Annual fixed charge	13%
Wages	520.000/yr
Energy cost	*2{t/kW-hr*
Indirect and other costs	*$l/ton ($1.10/tonnc)*

Solution . Tabulate and *total* all mining costs.

COSTS	*SURFACE M INE*	$million /yr	U N D E R G R O U N D MINE	$m il l io n /y r
Capital	$25 x *0.13*	= $ 3.25	$65 X 0.13	= S8.45
labor	$*20.000* x *130* + 10^6	= 2.60	$20.000 x 275 + 10^6	= 5.50
Energy	6.5 x *10* kW x $0.02	= 1.30	2.0 x 25 kW x $0.02	=1.00
Indirect	6.5 x *$1*	= 6.50	2.0 x SI	= 2.00
Total:	+ 6.5 *m illion tons/yr*	*$13.65*		$18.95
Unit:	$2.10/ton (52.3 I/tonne)		- 2.0 million tons/yr	= $9.50/ton ($l0.47/tonne)

For the conditions assumed in this example, underground mining costs again exceed surface costs by a factor greater than four times (the factor may vary in other branches of the industry). In addition to the reasons cited previously, a higher energy consumption factor prevails underground because of ventilation needs, less efficient energy sources, and less efficient blasting (Nilsson, 1983).

While Hedberg's and Nilsson's examples are hypothetical and the values approximate, others confirm the existence of inherent cost differences between surface and underground mining (Ayler, 1962; Pfleider and Scofield, 1967; Borquez, 1981). All are quick to admit, however, that the magnitude varies and must be evaluated carefully in individual cases. Obvious factors that influence the outcome—other factors being equivalent— arc mineral commodity, stripping ratio, production rates, variability of ore grade with depth, mineral-processing costs, and the features of specific mining methods being compared (see next section).

15.3 MINING COSTS

l he methods of overall cost estimation introduced previously in the text . n be used for comparisons of surface and underground mining; for in- Cfance. Example 6.1 is suitable for an approximation and Example 13.1 for a more accurate estimate.

Mining Method Costs

To this point, w e have dealt only with relative costs in evaluating the economics o f mining methods. Table 3.1 provides a listing of relative mining c0Sls that can be used for comparison purposes. They are, of course, only approximate; further, they are representative of conditions applicable to each method and therefore not directly comparable. The range of relative overall costs from Table 3.1 for the various classes of traditional methods is as follows:

Surface

Mechanical	5-10% (quarrying 100%)
Aqueous	<5-5%
Underground	
Unsupported	30-50%
Supported	60-100%
Caving	20-50%

While these relative costs cannot be converted to absolute costs with any confidence, an approximate conversion factor of 100% ~ S50/ton ($56/tonne) is of the right order of magnitude in the mid-1980s. It is absolute costs, of course, which provide the best measure for an economic analysis or comparison of mining methods. A survey of the literature provides us an overview of these values, keeping in mind that costs have to be referenced by date because of inflation. (As explained in Section
3.8, the consumer price index can be used to adjust prices approximately, to constant or current dollars).

One approach to estimating overall costs is to total item or elemental costs, such as abor, power, explosives, and supplies (as in Table 15.1). The largest and most important of these is labor cost, often expressed in units of productivity; the following are rank-ordered for the principal methods and taken mainly from

Wimpfen (1973) and Hamrin (1982), other than the coal methods (units are tons, or tonnes, of run-of-mine material per face employee-shift):

Open cast mining (coal)	500-1000	(450-900)
Open pit mining	100-400	(90-360)
Long wall mining (coal)	75-180	(68-163)
Room and pillar mining (coal)	30-80	(27-73)
stope and pillar mining	30-50,	27-45
Sublevel caving	20- 40	(18-36)
Block caving	15 -30	14-27)
Sublevel sloping	10- 20	9- 18
Cut and fill stoping	5-10	(4-9)
Shrinkage stoping "	(1-3)	(1-3)
Square sets toping		

For approximate explosives consumption with the various methods, the values (as powder factor in lb/ton, or kg/tonne) listed below are use full (Wimpfen, 1973):

Open pit mining	0,1-1,0	0,05-0,5
Open cast mining	0.2-0.5	0,10-0,3
Block caving	0.1-0.2	0,05-0,1
Square set sloping	0.3-1.1	(0. 5-0..6)
Sublevel stoping	0.3-0.6	(0.15-0.3)
Cut and fill stoping	0.5-1.2	(0.25-0.6)
Sublevel caving	0.6-0.8	0,3-0,4

The same source lists timber consumption for different underground methods *(of less importance* because of the profusion of ground control methods in use today). Other materiel and supply costs are usually computed for a particular mining method from manufacturer s price lists, as are equipment costs (Lawrence and Kim. 1982: Halls. 1982). Energy costs are based on total rated machinery power, load factor, time of usage, and unit energy charge for a given method (Halls, 1982).

Overall mining costs for thc various methods have been estimated or recorded by numerous authors: some of the more recent and reliable are Anon. (1974), Clement et al.. (1978), Halls (1982), Hoskins (1982), and Gregory (1983); *older* but *still* useful compilations are given by Chandler (1959), Ayler (1962), Wimpfen (1973), and Gentry and Hrebar (1976). Unfortunately, cost data rapidly become obsolete or unreliable, even when adjusted for inflation. Further, they are often incomplete, omitting certain methods, or computed on different bases.

Not as precise measures o f mining costs but only as indicators o f absolute cost ranges, the information in Table /5.2 is presented for the traditional methods, with relative costs fo r comparison. Data are estimated for current conditions and applicable only to mining in the United States, recognizing that not all methods are competitive with each other for a given mineral deposit. Values are overall (direct plus indirect) mining costs, including prospecting, exploration, development, and

exploitation, but excluding other costs (processing, transportation, taxes, royalties, etc.). With the cost ranges evident in Table 15.2, it is now understandable why

TABLE 15.2 Estimated Overall Mining Costs for Traditional Mining Methods, on Relative and Absolute Bases[a]

Mining Method	Average Relative Cost,[b] Percent	Range of Absolute Mining Cost,[c] $/ton	($/tonne)
Surface			
Open pit mining	10	3–20	(3–22)
Quarrying	100	25–150	(28–165)
Open cast mining	10	4–20	(4–22)
Auger mining	5	2–10	(2–11)
Hydraulicking	5	2–10	(2–11)
Dredging	<5	1–5	(1–6)
Borehole mining	5	2–10	(2–11)
Leaching	5	2–10	(2–11)
Underground			
Room and pillar mining	30	10–30	(11–33)
Stope and pillar mining	30	8–30	(9–33)
Shrinkage stoping	50	15–40	(17–44)
Sublevel stoping	40	12–35	(13–39)
Cut and fill stoping	60	15–50	(17–55)
Stull stoping	70	20–65	(22–72)
Square set stoping	100	30–125	(33–138)
Longwall mining	20	5–25	(6–28)
Sublevel caving	50	12–40	(13–44)
Block caving	20	5–20	(6–22)

[a] Mining costs *include* prospecting, exploration, development, and exploitation, but *exclude* processing, transportation, taxes, royalties, etc.
[b] Values taken from Table 3.1.

favorable surface costs. To illustrate cost-estimating procedures in determining mining costs, the budget nomenclature of Table 15.1 is utilized, and reference is made to the ensuing tables. In Table 15.3, development costs are compared for several alternatives in driving a drift underground: (1) cross-sectional size (8 x 10 ft vs. 22 x 22 ft, or 2.4 x 3.0 m vs. 6.7 x 6.7 m); (2) type of ground support (raw, bolts, or sets); and (3) haulage method (truck vs. rail). For the conditions assumed, the most economical arrangement, in $/ft, or $/m (or $/yd\ or $/m3), is the larger drift with no support and trackless haulage. In Tables 15.4-6, exploitation costs are tabulated for methods representing the three underground classes: room and pillar mining, cut and fill stoping, and block caving. They are itemized by unit operation and complied from elemental costs for labor, equipment, supplies, and explosives. These examples and others from Hoskins (1982), Halls (1982), and Pugh and Rasmussen (1982), together with formulas and graphs given by Clement et al. (1978), are most helpful in preparing cost estimates.

For cost estimates based on calculations of the incremental costs of the unit operations of mining, see the earlier numerical examples (8.1, 8.2, 11.1, and 13.1).

TABLE 15.4 Exploitation Costs of Stope and Pillar Mining, Expressed on a Unit Basis, $/ton ($/tonne)*

Item	Labor		Supplies		Electric Power	Depreciation	Total
	Operation	Maintenance	Operation	Maintenance			
Development	0.145 (0.160)	0.036 (0.040)	0.425 (0.468)	0.056 (0.062)	0.007 (0.008)	0.059 (0.065)	0.728 (0.802)
Production	0.861 (0.949)	0.492 (0.542)	0.960 (1.058)	0.504 (0.556)	0.065 (0.072)	0.531 (0.585)	3.413 (3.762)
Hauling	0.184 (0.203)	0.046 (0.051)	0.020 (0.022)	0.034 (0.037)	0.045 (0.050)	0.777 (0.085)	0.406 (0.448)
Hoisting	0.125 (0.137)	0.109 (0.120)	—	0.064 (0.071)	0.157 (0.173)	0.089 (0.098)	0.544 (0.600)
Pumping	—	0.018 (0.020)	—	0.021 (0.023)	0.218 (0.240)	0.014 (0.015)	0.271 (0.299)
Ventilation	—	0.018 (0.020)	—	0.007 (0.008)	0.163 (0.180)	0.016 (0.018)	0.204 (0.255)
General Mine	0.429 (0.473)	0.191 (0.211)	0.053 (0.058)	0.086 (0.095)	0.103 (0.114)	0.134 (0.148)	0.996 (1.098)
Supervision	0.725 (0.799)	—	—	—	—	—	0.725 (0.799)
Total:	2.469 (2.722)	0.910 (1.003)	1.457 (1.606)	0.772 (0.851)	0.758 (0.836)	0.920 (1.014)	7.286 (8.031)

* Based on 1977 costs and production rate of 4000 tons/day (3600 tonnes/day).

Source: Halls, 1982. (By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Development Costs of Driving a Drift, Expressed on Total and Unit Bases

	Trackless			Track		
	Unsupported	Rock-Bolted	Steel Sets	Unsupported	Rock-Bolted	Steel Sets
8 × 10 ft (2.4 × 3.0 m) drift						
Excavation						
Labor cost	$1,009,772	$1,080,080	$1,512,809	$1,051,364	$1,138,866	$1,660,275
Expendable supplies and permanent materials	295,689	319,250	918,932	282,809	309,130	909,327
Equipment operating cost						
Plant	497,147	519,082	705,512	250,670	264,693	383,982
Equipment	494,707	494,707	504,659	729,909	729,909	733,555
Passing zones	357,252	357,252	442,385	305,898	305,898	352,628
Demobilization tunnel	201,903	214,871	297,471	97,744	105,568	154,148
Total cost	35,700	35,700	35,700	69,383	69,383	69,383
Cost/ft	$2,892,170	$3,020,912	$4,417,468	$2,788,377	$2,923,467	$4,263,499
(Cost/m)	$386	$403	$589	$372	$390	$568
Cost/yd³	(1,265)	(1,321)	(1,932)	(1,220)	(1,279)	(1,865)
(Cost/m³)	130	136	168	125	132	162
	(4,596)	(4,801)	(5,943)	(4,431)	(4,646)	(5,736)
22 × 22 ft (6.7 × 6.7 m) drift						
Excavation						
Labor cost	$1,482,427	$1,609,302	$2,096,766	$1,622,188	$1,804,358	$2,420,269
Expendable supplies and permanent materials	766,835	828,180	3,708,699	675,560	740,253	3,592,349
Equipment operating cost						
Plant	528,605	554,050	707,085	435,754	489,160	599,025
Equipment	919,187	919,187	937,028	1,503,309	1,503,309	1,519,150
Passing zones	716,910	716,910	805,254	557,813	557,813	639,348
Demobilization tunnel	30,993	33,377	64,175			
Total cost	57,897	57,897	57,897	90,288	90,288	90,288
Cost/ft	$4,502,854	$4,718,903	$8,376,904	$4,884,912	$5,165,178	$8,860,429
(Cost/m)	$600	$629	$1,117	$651	$689	$1,181
Cost/yd³	(1,970)	(2,064)	(3,664)	(2,137)	(2,259)	(3,876)
(Cost/m³)	33	35	55	36	38	58
	(1,183)	(1,240)	(1,943)	(1,283)	(1,357)	(2,056)

Source: Cooney, 1982. (By permission from Northwest Mining Assoc., Spokane, WA.)

TABLE 15.5 Exploitation Costs of Cut and Fill Stoping, Expressed on Total and Unit Bases, $/ton ($/tonne)*

Cost category	Specifications	Unit Cost	Total Cost
Labor	($168/man-shift ÷ 25 tons/man-shift)	$ 6.72 ($ 7.41)	
Extra labor cost, stope preparation	([$168/man-shift ÷ 11 tons/man-shift] − [$168/man-shift ÷ 25 tons/man-shift]) × 950 tons		$ 8,125
Stope preparation, timber	(20.5 bd ft × 950 tons × $220/m bd ft—weighted)		4,285
Chute gates	Two sets/raise ($2500 × 2)		5,000
Subdrift labor	Sand seals: ($168/man-shift × 12 man-shift); no labor change for driving subdrifts since advance rate is equivalent to stoping rate		2,020
Subdrift timber	Per 200-ft section		1,110
Total stope development:	Per 200-ft section		$20,540
	Three sections	$ 0.92 ($ 1.01)	61,620
Drill machines	($0.37 × 3.8 ft/ton)	$ 1.41 ($ 1.55)	
Slushers and tuggers		1.20 (1.32)	
Drill steel and bits	($0.25 × 3.8 ft/ton)	0.95 (1.05)	
Explosives		0.95 (1.05)	
Miscellaneous hand tools and supplies		0.40 (0.44)	
Ground support			
Rock bolting		0.90 (0.99)	
Sandfill		1.05 (1.16)	
Chutes and service lines	Per raise: 2 chutes (2 × 180 ft × $90/ft); 2 crib chutes ([2 × 180 ft] × $54/ft); service lines: ($19.90/ft × 180 ft)	2.48 (2.73)	
Stope repair		0.58 (0.64)	
Pillar extraction		0.29 (0.32)	
Unit direct cost/ton:		$17.85 ($19.68)	

* Based on 1981 costs and stope production of 67,000 tons (61,000 tonnes).
Source: Marks, 1982. (By permission from Northwest Mining Assoc., Spokane, WA.)

TABLE 15.6 Exploitation Costs of Block Caving, Expressed on a Unit Basis, $/ton ($/tonne)*

Activity	Labor		Supplies		Electric Power	Depreciation	Total
	Operation	Maintenance	Operation	Maintenance			
Development	0.754 (0.831)	0.041 (0.045)	0.585 (0.645)	0.036 (0.040)	0.083 (0.091)	0.034 (0.037)	1.533 (1.690)
Drawing ore	0.458 (0.505)	0.056 (0.062)	0.178 (0.196)	0.051 (0.056)	0.083 (0.091)	0.043 (0.047)	0.869 (0.958)
Hauling	0.388 (0.428)	0.082 (0.090)	—	0.048 (0.053)	0.113 (0.125)	0.048 (0.053)	0.679 (0.748)
Hoisting	0.139 (0.153)	0.027 (0.030)	—	0.033 (0.036)	0.133 (0.147)	0.050 (0.055)	0.382 (0.421)
Pumping	—	0.013 (0.014)	—	0.005 (0.006)	0.052 (0.057)	0.005 (0.006)	0.075 (0.083)
Ventilation	—	0.013 (0.014)	—	0.005 (0.006)	0.054 (0.060)	0.006 (0.007)	0.078 (0.086)
General repair	0.175 (0.193)	—	0.108 (0.119)	0.001 (0.001)	—	0.005 (0.006)	0.289 (0.319)
General mine	0.401 (0.442)	0.041 (0.045)	0.026 (0.028)	0.084 (0.093)	0.036 (0.040)	0.109 (0.120)	0.697 (0.768)
Supervision	0.456 (0.503)	—	—	—	—	—	0.456 (0.503)
Total:	2.771 (3.055)	0.273 (0.301)	0.896 (0.988)	0.263 (0.290)	0.554 (0.611)	0.300 (0.331)	5.057 (5.574)

* Based on 1977 costs and production rate of 12,000 tons/day (10,900 tonnes/day).
Source: Halls, 1982. (By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Mine investment Analysis

Gentry and O' Neil (1984) employ the term mine investment analysis for the myriad financial tasks to be addressed in project evalution, whether that be a feasibility report of an entire mineral deposit or the selection of a mining method or machine. Cost estimates and ore reserve estimates are encompassed in investment analysis, but other complex financial decisions are also involved.

We have *already* utilized some of these analytical procedures. In Section 9.2, net present value was the basis for selecting an optimum production rate, and a cost comparison served as the means for selecting a shaft opr slope and for choosing the optimum interval between levels. Nilsson ,1982b) employs a form of investment analysis to determine optimum draw point spacing and recovery rate in sublevel caving , and Chatterjee and just (1981) perform simulation studies in specifying design factors (drill round, draw-point spacing, number of loading machines, etc.) in sublevel stopping. A few examples (devoid of calculations) will suffice to demonstrate the power of mine investment analysis.

Production Rate and Mine Life. Consider an underground gold mine. *characterized* as a high-grade, low-tonnage deposit (Glanville. 1984). Production rates of 100-1000 tons/day (90-90 0 tonnes/day), comparable to mine lives of *15*-1.5 yr. respectively, arc under consideration. After an

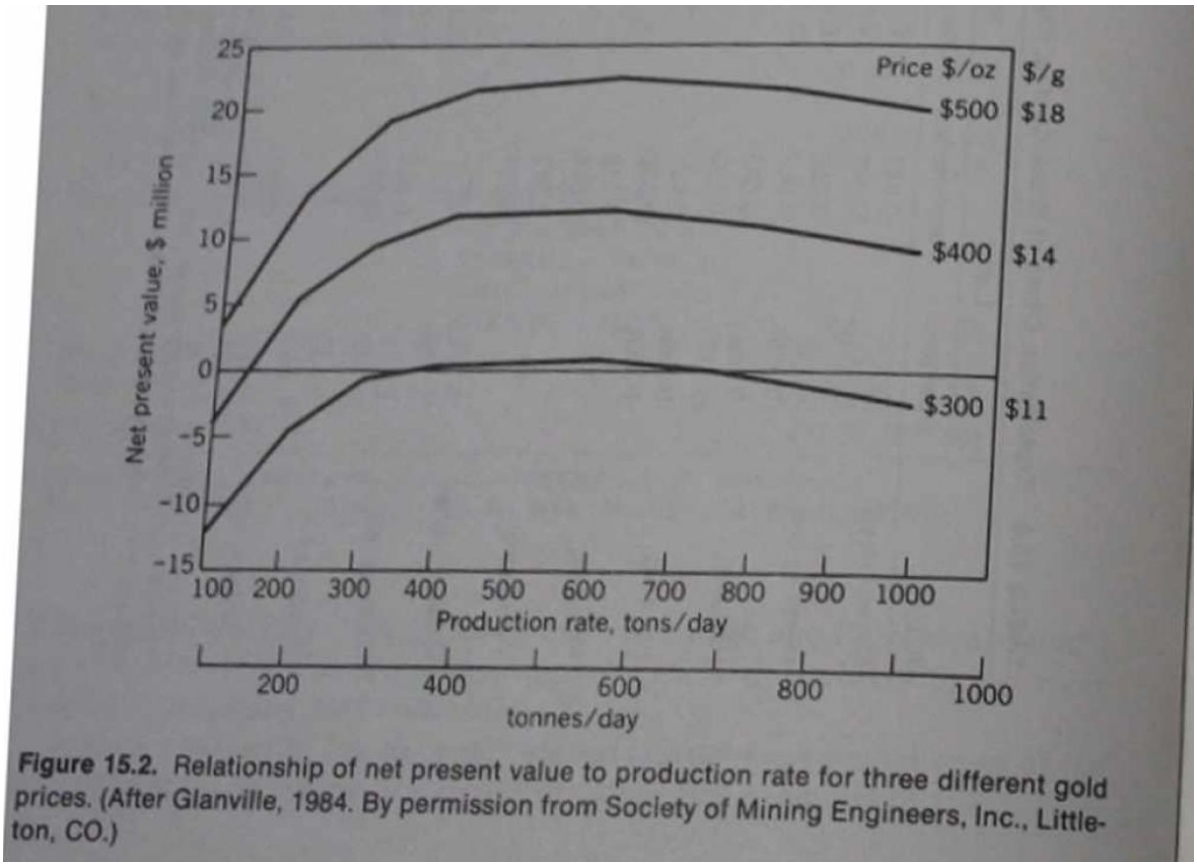


Figure 15.2. Relationship of net present value to production rate for three different gold prices. (After Glanville, 1984. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

Estimate, *of* costs, profits and present values is performed, a " sensitivity graph of net present value production rate for different gold prices ($300-$500/oz), or $10.60-$17.60g) is plotted (Fig. 15.2). In this case the optimum production rate is about 600 tons/day (540 tonnes/day), equivalent l0 a mine life of 2.5 yr; it is essentially independent of gold price and is itself 3 relatively insensitive parameter in this example. Optimization of productio*n* rate based on several other design variables is discussed by Wells
(1978).

Deposit Depth and Grade. In another example, the economics of open pit mining vs. in-situ leaching arc compared for a uranium deposit (Borkert and Gerity. 1978).

The design factors analyzed—considered the most sensitive depositional factors—are depth, size, and grade. To simplify the analysis com pound factors, deposit grade thickness (GT) and depth/GT ratio, are employed . Costs and profits are calculated for reserves varying from 2 million lb (907.0(H) kg) to 10 million lb (4,536.000 kg) U,0*. In Figure 15.3. net profits before taxes are plotted against values of depth/GT ratio. In-situ

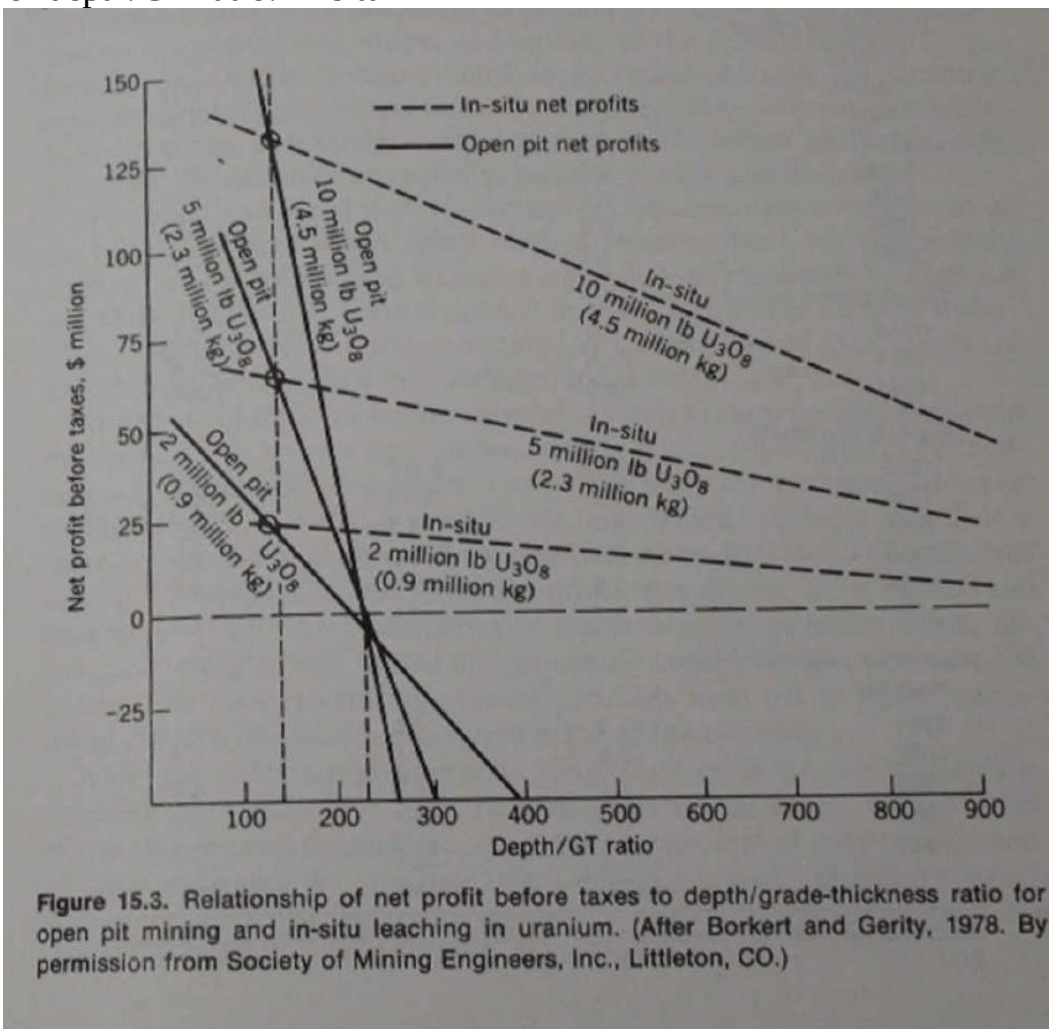


Figure 15.3. Relationship of net profit before taxes to depth/grade-thickness ratio for open pit mining and in-situ leaching in uranium. (After Borkert and Gerity, 1978. By permission from Society of Mining Engineers, Inc., Littleton, CO.)

leaching it clearly more economical than open pit mining when *depth G T* is >150. High, which corresponds to a stripping ratio of 30 yd^3/ton (25.3 m^3/tonne) and a grade thickness of 0,5 ft (0,15m). The analysis also demonstrates that open pit mining is uneconomical, regardless of the competitiveness of in-situ leaching, when depth /GT is >250 (which corresponds to a depth or 250 ft for GT=1 ft, or 75 m for GT=0.3 m

The Return on Investment. The White Pine Copper serves as an actual case history in investment analysis (BOYD, 1967). Table 15.7 summa

TABLE 15.7 Cash Investments and Returns for White Pine Copper, 1929–1965

	Cash Flow, $ million		
	Cash Investment	Cash Return from Operations	Net Annual Return
Property purchased			
1929	$ 0.1		($0.1)
Exploration period			
1930/43	0.1		(0.1)
1944/45	0.1		(0.1)
1946	0.4		(0.4)
1947	0.4		(0.4)
1948	0.1		(0.1)
1949	0.1		(0.1)
1950	0.2		(0.2)
1951	0.1		(0.1)
Development period			
1952	9.6		(9.6)
1953	31.6		(31.6)
1954	18.9		(18.9)
Exploitation started			
1955	12.5	$ 9.7	(2.8)
1956	1.8	11.9	10.1
1957	1.7	4.2	2.5
1958	0.7	5.2	4.5
1959	1.6	5.2	3.6
1960	2.2	3.6	1.4
Operation reached maturity			
1961	1.5	6.0	4.5
1962	2.8	7.5	4.7
1963	2.1	8.3	6.2
1964	2.0	9.4	7.4
1965	2.6	13.5	10.9
Total:	$93.2	$84.5	($8.7)

Source: Boyd, 1967. (By permission from Society of Mining Engineers, Inc., Littleton, CO.)

sizes the annual cash (lows (cash investments, cash returns, and net returns) over a 36-yr history o f the mine. It was in 1966— 14 y r after development commenced and 11 yr after exploitation began— that the mine turned its first profit. Until December 31. 1965. the White Pine mine operated with a negative cash flow or net return; although revenues where realized in 1955, it required another 11 y r before the cumulative investments were offset by cumulative returns.

Reference should be made to the articles cited or the Gentry and O ’Neil , 1984) for more comprehensive treatment o f mine investment analysis techniques.

15.4 METHOD SELECTION

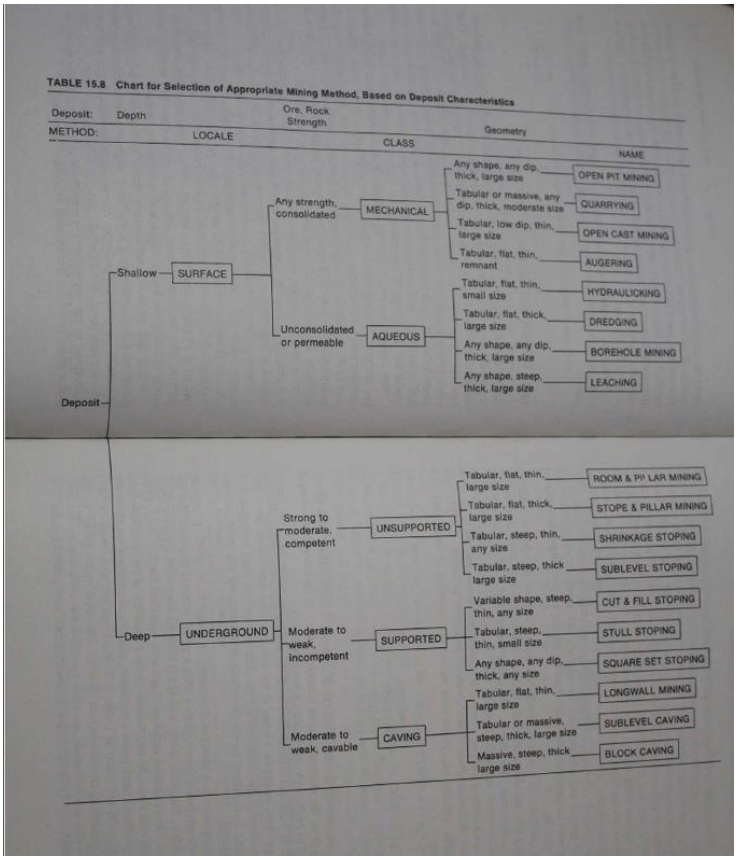
Before devising some comprehensive procedures to select mining methods, we would do well to review the criteria and factors that are relevant. They were enumerated in six categories in Section 3.7. Some of the most important are the depth, size, shape, and attitude of the deposit (geometric factors); geologic and hydrologic conditions; geotechnical properties; economic considerations; technological factors such as recovery, dilution, flexibility, and selectivity; and environmental concerns. Guidelines and a planning model for the design of an exploitation system were also discussed.

We should also recall that the distinctive characteristics or conditions of the mineral deposit which affect method selection have been separately summarized

for both mining locales, surface and underground (Sections 8.4 and 13.4). In addition, some simplified decision matrices to assist in underground method selection were presented in Tables 13.4 and 13.5; others are available for both surface and underground (Lindsay, 1985).

As a result of these earlier discussions, as well as the preceding articles in this chapter, we are in a position to consider more comprehensive method selection procedures. *A qualitative one, developed for the classification of traditional mining methods used in this text, appears in Table 15.8.* It is a chart to aid in the selection of the appropriate method(s), based on important deposit characteristics. All 18 of the principal methods, both surface and underground, and three categories of characteristics, including depth, ore and rock strength, and spatial and geometric considerations, comprise the matrix. The decision-making process proceeds from left to right, successively defining the locale, class, and name of the method.

While this selection procedure is simple and quick to use, it is neither definitive nor quantitative. Only relative terms are employed, and a number of criteria—especially geologic, economic, technological, and environmental—are neglected. We conclude that Table 15.8 is useful as a first approximation and to narrow the choice tentatively to a few candidate methods, but that a more selective, quantitative procedure is needed.



Several sophisticated, numerical schemes have been devised for method Selection, but *probably the* most satisfactory to date is one by Nicholas (1981). All the particulars are not *repeated* here. Nit the main procedural urns are developed and an *example given. In stage I.* to identify the best candidate methods, the deposit is described in terms of geometry , grade distribution, and *geotechnical* properties Using these parameters, the relevant *mining methods* are rank ordered

numerically to determine which two or three are most promising. *In* stage 2. mining and capital costs as well as production *rate, recovery, labor supply, and environmental f*actors are introduced , *Finally, in stage 3, the most likely* candidate methods are costed out, cutoff *grades and* reserves are calculated, and economic comparisons are *completed to determine the most* feasible mining method and overall plan of *exploitation* .

To elaborate *on the methodology o f* stage I. three or four factors each are selected that are descriptive *o f (I) deposit* geometry and grade distribution ,(shape, thickness, *dip. depth,* and grade uniformity), and (2) geotechnical *properties* (rock *strength,* fracture spacing, and fracture shear strength). Next, each mining method is rated *numerically* as to its suitability for each category *of factors. Four* ranks are used: preferred, probable, unlikely, and *unsuitable (eliminated).Numerical* values are assigned to each rank; for instane:

Preferred	3-4
Probable	1-2
Unlikely	0
Unsuitable	-49

(Values are chosen *to* obtain definitive answers; i.e.. 0 neither adds to nor subtracts from *the* chance of using a method, and -49 ensures elimination of a method even (hough other factors are rated high.)

A table of rankings is prepared for all the mining methods under consideration. Any with negative values drop out. and the remaining are rank ordered to reveal the most attractive. The analysis is then completed in stages 2 and 3. resulting in the quantitative selection of the most suitable mining method. Nicholas provides a numerical example of stage I, in which 10 mining methods are analyzed for suitability with a given mineral deposit. The deposit is tabular, flat, very thick, and 425 *ft* (130 m) deep with uniform grade *distribution.* Geotechnical conditions are also specified for the ore deposit and adjacent wall rock. As an example of the suitability of each method for a variety of geometric and grade conditions, numerical ratings (based on the scale of 4 to *-49)* arc *listed* in Table 15.9. Rankings arc also assigned for a range *of* geotechnical conditions (not shown). The appropriate values corresponding to *the* given *deposit* are then specified for each mining method; these are *totaled* in Table 15.10 for geometry/grade and geomechanics. For

TABLE 15.9 Suitability Ratings of Various Mining Methods for Deposit Geometry and Grade Factors

Mining Method	General Shape[a]			Ore Thickness[b]				Ore Dip[c]			Grade Distribution[d]		
	M	T/P	I	N	I	T	VT	F	I	S	U	G	E
Open pit	3	2	3	2	3	4	4	3	3	4	3	3	3
Block caving	4	2	0	−49	0	2	4	3	2	4	4	2	0
Sublevel stoping	2	2	1	1	2	4	3	2	1	4	3	3	1
Sublevel caving	3	4	1	−49	0	4	4	1	1	4	4	2	0
Longwall	−49	4	−49	4	0	−49	−49	4	0	−49	4	2	0
Stope and pillar	0	4	2	4	2	−49	−49	4	1	0	3	3	3
Shrinkage stoping	2	2	1	1	2	4	3	2	1	4	3	2	1
Cut and fill	0	4	2	4	4	0	0	0	3	4	3	3	3
Top slicing	3	3	0	−49	0	3	4	4	1	2	4	2	0
Square set	0	2	4	4	4	1	1	2	3	3	3	3	3

[a] M = massive, T/P = tabular or platy, I = irregular.
[b] N = narrow, I = intermediate, T = thick, VT = very thick.
[c] F = flat, I = intermediate, S = steep.
[d] U = uniform, G = gradational, E = erratic.
Source: Nicholas, 1981. (By permission from Society of Mining Engineers, Inc., Littleton, CO.)

example, the geometry/grade value for the open pit method is 2 + 4 + 3 + 3 = 12, using data from Table 15.9.

Based on the ratings in the grand total column in Table 15 10 the 10 mining methods arc rank-ordered in Table 15.11. The results indicate that the most favorable method is open pit mining with 40 points followed by block caving and top dicing with 34 points each. Eliminated are long wall mining (-18 points) and stope and pillar mining(-20 points).

TABLE 15.10 Numerical Ratings of Various Mining Methods for a Mineral Deposit with Specified Characteristics

Mining Method	Geometry/Grade Distribution	Geomechanics Characteristics				Grand Total
		Ore	Hanging Wall	Footwall	Subtotal	
Open pit	12	9	11	8	28	40
Block caving	13	8	6	7	21	34
Sublevel stoping	10	5	7	2	14	24
Sublevel caving	13	7	6	3	16	29
Longwall	−37	8	5	6	19	−18
Stope and pillar	−38	7	8	3	18	−20
Shrinkage stoping	10	6	6	8	20	30
Cut and fill	7	8	7	10	25	32
Top slicing	15	6	6	7	19	34
Square set	8	8	7	10	25	33

Source: Nicholas, 1981. (By permission from Society of Mining Engineers, Inc., Littleton, CO.)

TABLE 15.11 Rank Order and Rating of Mining Methods Compiled in Table 15.9 for a Specified Mineral Deposit

Rank	Method	Rating
1	Open pit mining	40
2	Block caving	34
3	Top slicing	34
4	Square set stoping	33
5	Cut and fill stoping	32
6	Shrinkage stoping	30
7	Sublevel caving	29
8	Sublevel stoping	24
9	Longwall mining	−18
10	Stope and pillar mining	−20

Source: Modified after Nicholas, 1981.

Having narrowed the preferred choice of methods to two or there, each is now examined in stage 2 for technological, economic, and environmental considerations. The final selection is made during a detailed 3.
The beauty of a quantitative selection procedure such as this is that it lends itself to rapid, longhand evalution of a few factors or equally to computer analysis of myriad complex factors in a more precise study.

15.5 SUM M ARY

Review

Topically, the book's 15 chapters group into seven broad sections:

I. Introduction Chapter	I
11. Stages of mining	2,3
III. Unit operations	4
IV . Surface mining	5, 6, *7,* 8
V. *Underground mining*	9, *10, 11, 12, 13*
V I. Novel methods	*14*
V II. Method selection	*15*

We now review some of the major highlights of each section.
Introduction. Clearly, Chapter I is as much review as it is preview. It is *intended* to *provide* a synopsis of mining and mining engineering. **Included is** *discussion on the cultural* contributions *o f mining* (one of the two **basic** *industries o f mankind),* history spanning nearly 500,000 years, **fundamental** consepts and the languages of mining, four stages of mining activity, and unit operations of mining. Important field adjunct to mining, mineral economics and governmental regulation, are introduced. Lastly, the consequences and side effects of mining-environmental and other –are considered. As a brief overview of the entire book, Chapter is introductive to provide perspective and to serve both as an introduction and a summation.

Stages of Mining. Two stages of mining activity are precursor to the mainstream of mining: prospecting and exploration. Prospecting locates a potentially valuable mineral deposit of coal, ore, or stone, and exploration defines it. Their objective is to narrow the search, to increase the favorability, and to reduce the risk of the area under investigation. Both direct and indirect methods of search are utilized; the principal tools are geology, geoghysics, and geochemistry. Drilling or excavation samples is the final and obligatory step taken to obtain proof of the existence of a hypothetical ore body. Sampling data then must be compiled into a reserve estimate of tonnage and grade, using classical or geostatical methods. The preparation of a feasibility report marks the transition between the second and third stages in the life of a mine.

Mining proper commences with development. The cardinal rule during the final two stages of mining is that an optimal development-exploitation rate to maximize total profit must be determined for each new mine project. In opening up a deposit for mining, three major groups of factors govern development. As many as 11 sequential steps— sometimes scheduled by a CPM or PERT chart— may be required in developing a mine. Exploitation is the production stage of mining and commences near the completion of development. In selecting a mining method, we choose one that best matches the uni4uc characteristics of the deposit, within limits imposed by safety, technology. and economics, to return the greatest profit. Again, a variety of factors (six) controls exploitation. Mining methods are classified generically by locale and class in the scheme proposed here, and cost estimation is broken down into direct, indirect, and overall costs for the four stages of

Unit Operations. The remainder of the book is devoted to engineering aspects of mine development and exploitation. We deal first with certain fundamental operations that are performed to free and transport the mineral from the deposit. They are called the unit operations of mining and include both production and auxiliary steps. In the production cycle occur rock breakage, usually consisting of drilling and blasting, and materials handling, generally comprised of loading and haulage. Auxiliary operations include health and safety, environmental control, ground control, and various supportive activities. Because mining is heavily mechanized today (essentially 100% in the United States), unit operations are closely identified with equipment.

The basic operations and their associated machines are discussed, as well as the formation of cycles and systems and the selection of equipment for both production and auxiliary purpose.

Surface Mining.

Employing our generic classification of mining methods, we discuss first those used in a surface locale. Development for surface mining is directed especially at there are targets: land reclamation, waste disposal, and advanced stripping. Pit planning and design-partly because of the immensity of the scale of operations-is crucial to the success of a surface mine. It is predicated on several objectives and

broken down into short-range and long-range planning. In both phases, the calculation of stripping ratios and the location of pit limits are required.

Two classes of surface methods are employed in mining: mechanical extraction and aqueous extraction. The former is by far the more prevalent, *U. S.* surface production), *the* latter being limited to applications where water *is instrumental to exploitation.* Open pit mining and open cast *mining both mechanical,* are the most commonly used of all mining methods. *Quarrying is employed to mine* dimension stone and auger mining lo recover coal from a high wall once the maximum stripping ratio has been *retched Of the* aqueous *methods,* placer mining is applicable to near-surface deposits *of unconsolidated minerals, while* solution mining is employed *usually for deeper* consolidated *deposits* that can be recovered by dissolving, melting, or slurrying the minerals. Examples of placer mining are hy*draulicking* and dredging, and of solution mining, borehole extraction and *leaching.*

In importance, surface mining clearly ranks ahead of underground mining.
if we compare tonnage or value of current annual production (85% Vs. 15%). In *spite of its* many attractions, however, there are some serious *limitations to surface mining,* not *the least of* which are depth, selectivity and *flexibility, and* environmental constraints.

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

There are three classes of underground mining methods, differentiated by the nature and extent of ground support provided. Unsupported methods are *essentially self-supporting, relying on pillars and* temporary, artificial structural *units* (roof bolts, timber, *light steel* sets, etc.) to resist the superincumbent load. The bulk of U.S. underground mineral production is realized from the four unsupported methods, two for near-horizontal deposits (room and liar mining, stope and pillar mining) and two for near-vertical deposits
Room and pillar and stope and pillar alone are responsible for 75% of that production. The supported class of methods consists of cut and fill stoping, stull stoping, and square set stoping.
Require substantial amounts of artificial support (heavy timber or steel sets, backfill), this class has only limited usage today. The third underground growing in popularity, is caving: long wall mining, sublevel caving, and I κ caving. In these methods, induced, controlled caving of the ore body, overlying rock, or both occurs in conjunction with mining.

Underground mining may be expected to increase in importance in coming years, in part because the easy-to-exploit near-surface deposits have largely been

mined out. White surface mining will probably extend its application somewhat to moderate depths, it is unlikely that surface methods can ever be used for deep ore bodies.

Novel Methods. The so-called novel methods of mining receive their name to distinguish them from the traditional methods already described We class if y them in three categories, based on the likelihood of early, commercial application. Those with limited present application are rapid excavation in hard rock, automation, hydraulic mining, and methane drainage. Methods showing promise but not now in use are underground gasification underground retorting, and ocean mining (the latter especially is attractive for certain metallic resources, provided legal and political barriers can be overcome).

The third group has a questionable future or unlikely use; it consists of nuclear mining and extraterrestrial mining. We conclude that rapid excavation and automation are most promising and versatile, because they are adaptive to most traditional methods of mining. Other novel methods are more restrictive, limited to locale(surface or underground) or mineral commodity. or even site-specific. Automation has probably the brightest future off all the novel methods because it is evolving most rapidly and receiving widest acceptance.

Method Selection . Because exploitation lies at the crux of mining we direct our attention lastly to procedures for selection of the most feasible mining method. Following a summary comparison of surface and underground mining and traditional and novel methods, the final chapter treats of method costing and selection. Sample cost-estimation procedures and tabulations on relative and absolute bases are given for different mining methods.

Some advanced financial procedures that aid in mine investment analysis are also introduced. Two generalized approaches to method selection are presented, one qualitative and the other quantitative. Finally, the chapter and book conclude with a review and future outlook.

Role of the mining engineer

In chapter 1, we defined mining engineering as the art and science applied to the processes of mining and the operation of mines. As practitioners of this branch of engineering, mining engineers plan, design, develop, exploit, evaluate and operate mining. If they acquire additional geologic experience, they may prospect and explore for mines as wall. If they expand their knowledge of processing metallurgy, they may additionally practice mineral processing. Broadening their engineering background to unclude business enables them to manage mines. And should they enter manufacturing or sales organizations that serve the mineral industries-or go into education, research, consulting, or government agencies that relate to mining-they are still involved in the practice of mining engineering.

The mining engineer was once referred to with pride as a jack of all trades. If ever true, it has ceased to be now. Specialists in science, engineering, business, and law who once shunned the mining industry increasingly accept employment at mines.

Mining engineers are now free-and expected to specialize in their own field. They among all engineers are uniquely quail fied under to calculate pit limits, write environmental impact statements, design underground openings, plan ground-control systems, choose equipment for the production cycle and auxiliary operations, select mining methods, ventilate mining, make cost estimates, and cope with the social-economic-political consequences of mining.

A baccalaureate education is barely sufficient for modem mining engineers to begin practice. Supple menting their stud.es with practical experience they are also ell advised to consider graduate work in mining engineering or a related field.

Future Outlook

To a certain extent, every author of a technical textbook is expected to soothsay. It is a fitting departure for an otherwise factual treatise, and I am pleased to add my own predictions for the coming decades to an already voluminous literature 1. *U .S. self sufficiency in mineral production:* Continuing present trends, the United States will increasingly become a net importer of minerals. especially metallics. as it enters period 5 of Love ring's chart (Fig. 1.4). *This does* not mean the demise of the domestic mining industry, but that U.S. mining will grow more selective and competitive. Surprisingly, coal and some fledgling unconventional fuels (coal bed methane, oil shale, tar sand) will survive and grow in the coming two decades as the metals continue *their decline.*

2. Government regulation: The role and influence of government **in the** mineral industries will not diminish; on the contrary, regulation and control are expected to flourish. This is especially true with regard to environmental protection-safeguarding the public's health and security will remain top priority. Mineral depletion and expensing allowances will fall victim to tax reform.

3. Prospecting and exploration: Indirect methods of locating and proving mineral deposits, especially airborne geophysics, will improve in resolution and precision and precision and largely displace direct methods. Ore estimation and evalution will increasingly be conducted be conducted by sophisticated mathematical techniques (geostatistics), carried out by computer.

4. Development and exploitation: Construction tasks for development and exploitation will be performed largely under contract by specialists in the field, surface or underground.

5. *Unit operations and equipment:* surface equipment will tend not to grow larger but more durable, automated, and productive. Underground equipment will evolve in the same way, becoming more mobile as well. Equipment for both locales will achieve more continuous operation: some radical new designs will overcome the cyclic nature that plagues much of mining.

6.Surface mining methods: open cast mining will grow in importance at the expense of open pit mining, largely under pressure of new environmental restrictions and cost cutting. Solution mining, especially in-sute leaching, will expand in usage.

7. *Underground mining methods:underground mining will incrase modestly* in tonnage output, but the number of methods in use will decline. In non coal mining, the high-cost, low-output selective methods (stope and pillar mining, sublevel stopping, sublevel caving, and block caving) will grow. In coal, room and pillar mining increasingly will be displaced by long wall mining.
8.Novel mining methods: Through wider acceptance and use, existing novel methods (rapid excavation, automation, hydraulic mining, and methane drainage) will be accepted as traditional methods. Commercial use of underground gasification, underground retorting, and ocean mining will commence, but the exotic novel methods (nuclear mining and extraterrestrial mining) will still await application.
9. *Demand for mining engineers:* Well-educated, capable mining engineers will remain in demand, those with advanced degres especially so .Increasingly, the demand will be worldwide; as mining activity in the United States declines, it will accelerate elsewhere, and some American mining engineers will once again seek their fortunes abroad.
10.Computer applications. The ubiquitous computer will be in use everywhere in the mining industry, automatic machines, regulating processes monitoring the atmosphere, controlling production and costs, processing records, solving problems, measuring reserves, and the forth. Not that computers will displace people in mines-bur hopefully they will make our job easier.

15.6 SPECIAL TOPIC MINE DESIGN PROBLEM

As a grand finale for an introductory textbook on mining engineering, it is appropriate that we consider an open ended design problem, one that spans all four stages in the life of a mine. The problem must be open ended because, in real life, mine *design is* seldom a well-defined task. Information provided is minimal, requiring assumptions on the part of the student during analysis, and the design procedure is left entirely in his or her hands. A satisfactory solution to the problem requires the follow as minimum, using procedures and material presented in this book:
1.Complete description o f the steps involved in bringing a given mineral deposit through the four stages into a producing mine
2 . Specification o f the unit operations and equipment required for development and *exploitation*
3. Selection of the most suitable mining method, with justification and explanation of operating details
4. Preparation *of* an approximate cost estimate

The amount of detail to be provided varies with the time allowed for and importance attached to the exercise. As usual, suggested problems follow.

PROBLEMS

15.1 For the mineral deposit described below, prepare a specific, detailed outline *of the* steps involved in *bringing* a prospect through the four operational stages into a producing mine. Select the mining method which would be *best suited,* most economical, and safest for the given conditions, defending your choice briefly and citing the main advantages and disadvantages of the method. Sketch the overall mine layout and the mining method. Identify the cycle of operations and major equipment selected for mining, and state the relative cost. Design conditions are as follows:

Deposit	Coal seam, steam quality, calorific value 13.500 Btu/lb (31.400 kJ/kg). thin rock parting
Dip 2°.	Thickness 5.5 ft (1.7 m). average depth of cover 950 ft (290 m)
	Deposit area owned 12 mi2 (3100 hectares), no outcrop
	Coal compressive strength 4000 lb/in.1 (27.6 M Pa)
	Roof (sandstone and shale) average compressive strength 15.000 lb/in.2 (10* MPa)
	Floor (shale) compressive strength MPa) *№00 lb/in.2 (62.1
Topography	Hilly terrain, elevation 800 ft (245 m)
Climate	Warm temperate, mean annual rainfall 60 in. (1.5 m)
production rate	5000 tons/day (4500 tonnes/day) over 30-yr life

15.2 For the mineral deposit described below, prepare a specific detailed outline of the steps involved in bridging a prospect through the for operational stages into a producing mine. Select the mining method which would be best suited, most economical, and safest for the given conditions, defending your choice briefly and city the main advantages and disadvantages of the method. Sketch the overall mine layout and the mining method. Identify the cycle of operations and major equipment selected for mining, and the relative cost. Design conditions are as follows:

Deposit	Coal seam, steam quality, calorific value 13.500 Btu/lb (31.400 kJ/kg). thin rock parting
	Dip 2°. thickness 5.5 ft (1.7 m). average depth of cover 950 ft (290 m)
	Deposit area owned 12 mi2 (3100 hectares), no outcrop
	Coal compressive strength 4000 lb/in.1 (27.6 MPa)
	Roof (sandstone and shale) average compressive strength 15.000 lb/in.2 (10* MPa)
	Floor (shale) compressive strength) 9 000 lb/in^2 (62.1 MPa)
Topography	Hilly terrain, elevation 800 ft (245 m)
Climate	Warm temperate, mean annual rainfall 60 in. (1.5 m)
production rate	5000 tons/day (4500 tonnes/day) over 30-yr life

15.2 For the mineral deposit described below preapare a specific detailed outline of the steps involved in bringing a prospect through the four operational stages into a producing mine. Select the mining method which would be best suited, most economical, and safest for the given conditions, defending your choice briefly and city the main advantages and disadvantages of the method. Sketch the overall mine layout and the mining method. Identify the cycle of operations and major equipment selected for mining, and the relative cost. Design conditions are as follows:

Deposit Limestone bed. average grade 98% $CaCO_3$, uniform, mud seams Dip 5°, thickness 40 ft (12.2 m). average depth of cover 1200 ft (365 m) Deposit area owned 4 mi' (1000 hectares), no outcrop Orc compressive strength 20,000 lb/in.2 (138 MPa) Hanging wall (sandstone, limestone, shale) average compressive strength 16.000 lb/in.2 (110 MPa) Footwall competent limestone

Topography Low mountain ridges, rough terrain, elevation 1800 ft (550 m)
Climate Temperate, mean annual rainfall 40 in. (1.0 m)
production rate 10 000 tons/day (9100 tonnes/day) over 25-yr life

15.3 For the mineral deposit described below, prepare a specific, detailed outline of the steps involved in bringing a prospect through the four operational stages into a producing mine. Select the mining method which would be best suited, most economical, and safest for the given conditions, defending your choice briefly and citing the main advantages and disadvantages of the method. Sketch the overall mine layout and the mining method. Identify the cycle of operations and major equipment selected for mining, and state the relative cost. design conditions are as follows:

Deposit Chalcocite/chalcopyrite ore minerals disseminated in quartz porphyry , average grade 0.5% Cu. ,fairly uniform *Massive irregular deposit, reserves 100 million* tons (90 *million tonnes), average depth of cover* 3000 *ft (915 m)*
Ore compressive strength *12,000 lb/in.*~ (82.7 MPa), *frequent joints*
Metamorphosed capping and ad joining rock (schist), *compressive* strength 8000 *lb /in .* (55.2* M Pa), *fractured. Distinguishable boundary* with ore
Topography Desert and low mountain ridges, rugged terrain, elevation *4500 ft (1370 m)*
Climate Arid , mean annual rain fall 10 in. (254 mm)
Production rate 25*.000 tons/day (22.700 tonnes/ day)* over 20-yr *I,* fc

15.4 Solve Problem 15.1. given the same mineral deposit, except for the *following changes in conditions:*

Deposit Average depth of cover 2000 *ft (610 m)*
Coal compressive strength 2000 lb/in.2 (13.8 MPa)
Roof average compressivc strength 8000 *lb/in.*^{2} (55.2 *MPa)*
Floor compressivc strength 8000 lb/in.2 (55.2 M Pa)
Topography Controlled subsidence permitted

Production rate Controlled subsidence permitted

15.5 Prepare an approximate cost estimate for the development and ex*ploitation operations for the mine of the following* problems:
a. Problem 15.1
b. Problem 15.2
c. Problem 15.3
d. Problem 15.4
Calculate on a unit *basis. Using the estimating* values given in Table 1.3 for *prospecting and exploration, also* determine the range of overall *mining costs for each mine.*

Appendix

CONVERSION TABLE, ENGLISH TO METRIC UNITS (S.I., OR INTERNATIONAL SYSTEM) USEFUL IN MINING ENGINEERING

Acceleration of gravity	32.174 ft/sec^2 = 9.8066 m/sec^2
Angular measure	1 deg = 0.01745 rad
Area	1 in.^{2} = 645.16 mm^2 1 ft^2 = 0.09290 m^2 1 yd^2 = 0.83613 m^2 1 acre = 0.40469 hectare 1 mi^2 = 259.0 hectares
Barometric pressure	1 mm Hg = 133.32 Pa 1 aim = 101.325 kPa
Blasting factor	1 ft^2/drill hole= 0.09290 m^2/drill hole
Drilling factor	1 ft/ton = 0.33598 m/tonne
Energy or work	1 Btu = 1.0551 kJ 1 kW-hr = 3600 kJ 1 cal* = 4.1868 J 1 erg = 0.10000 uJ
Force	1 lb (force) = 4.4482 N 1 dyn = 1 x 10^{-5} N
Friction factor, airflow	1 lb-min^2/ft^4 = 1.8550 x 10^6 kg/m^3}
Gas constant, air	53.35 ft-lb/lb (mass)-^{0}R = 287.045 J/kg-K
Grade of ore	1 oz/ton = 31.25 g/tonne
Head or pressure difference, airflow	I in. water = 25.4 mm water*

1 in. water = 248.84 Pa @ 60°F
1 in. Hg = 3.3768 к Pa @ 60°F

Hear content or enthalpy — *1 Btu/lb* = 2.3260 *kJ/kg*

Heat flow — *1 B t u / h r* = 0.29307 W

Length. — 1 in.=*25.40* mm
/ — ft = 0.30480 m
1 mi = *1.6093 km*

Loading factor, explosives — *1 lb/ft= 1.4882 kg/m*

Mass — *1 lb (mass) = 0.45359 kg*
1 grain = 64.799 mg

5 в 4 appendix
Mass *density* — *1* lb-sec^2/ft^4 = *1 slug/ft^3 = 515.38 kg/m^3*

Mass per unit area — *1 lb/ft^2 = 4.8824 kg/m^2*
1 ton/ft^2 = 9.7652 tonnes/m^2

Moment — *1 ft-lb = 0.13825* m-kg
1 ft-ton = 0.27651 m-tonne

Powder factor — *1 lb/ton* = 0.50000 kg/tonne

Power — 1 *h p = 0.74570 kW*

Pressure or stress — *1 lb/in.^{2} = 6.8948* k Pa
1 lb/ft^2 = 47.880 Pa

Quantity, airflow — *1 ft^{3}min = 0.47195* x *10^{-3}* m^{3}sec

Resistance, airflow — *1* in.-min^2/ft^6 = *1.1170 x 109 N-sec*^{2}/m^8

Specific volume — *1 ft3/lb = 0.06243* m3/kg

Specific weight — *1* lb/ft5 = *16.018 kg/m3*

Stripping ratio — *1 yd^3/ton = 0.84278 m^3/tonne*

Temperature — ^{0} C = H ° (F - 3 2)
K = °C + 273.15

Tonnage factor — 1 *ft^3ton = 0.03121* m^3/tonne

Velocity
1 *in./min = 0.42333* mm/sec
1 *ft/min = 0.00508* m/sec
1mi/hr = 1.6093 *km/hr*

Volume
1 ft^3 = 0.02832 m^3
1 yd^3 - 0.76455 m3
1 gal = 3.7854 L
1 acre-ft = 1233.5 m^3

Volume flow rate
1 *ft^3/min = 0.47195 L/* sec
1 ft^3/min = 471.95 cm^3sec*
1 *gal/min = 63.09 x 10* 6 m^3/sec
1 *gal/min* = 0.06309 L/sec

Weight
1 oz *(avoir.) = 28.35 g*
1 lb = 0.45359 kg
1 ton = *0.90718 tonne*
1 ton (long) = 1.0160 tonnes

Weight flow rate — 1 *lb/hr = 0. 12600 x* 10^{-3} kg/sec

REFERENCES

Anon.. 1976, *A S T M/ IEEES Standard Metric Pratice,* Inst of .Electrical and electronics Engineers. New York, 47 pp

Anon., 1980, *Mining Engineering* Moves to an SI Format, *M n . E n g.,vol 32,* No. 2, Feb., pp. 155-157.

Answers to selected Problems

CHAPTER 2

2.1 468,000 *long ton% (475,500 tonnes)*
56.9% Fe
1.244.400 yd ' (951.400 m)
2.66 *yd*^{3}*long /ton (* 2.03 *m* $^{3/}$*tonne)*

2.3 a. 3.0 *ft (0.92 m)*
b. 253 ft (77 ш)
c. *1.28% S*
d. H4.1 ft/ft (m/m)
c. 2.546.000 tons (2,309,700 tonnes)

2.5 *31.3% Fe*

CHAPTER 4

4.2 a. *-10.6414 g-atoms/kg*
b. No, *it is oxygen-deficient*
c. Increase O_0 or decrease C

CHAPTER 5

5.1 *6.00 yd*^{3}*ton (5.06* m^{3}/tonne)
99 ft (30 m)
300 ft (91 m)
2.25 *yd*^{3}*/ton (1.90 m*^{3}*/tonne)*
357 ft (109 m)
20.0 yd^{3}/lon (16.9 m^{3}/tonne)
6.56 yd^{3}/lon (5.53 m^{3}/tonne)

CHAPTER 6

6.2 Fails
0.96

6.4 a. 1.44
Safe
b. 0.18
Fails

6.6 a. 29 ft
7 holes
b. Keduce parameters

CHAPTER 7

7.1 a. 5060 tons/shift (4590 tonnes/shift)
b. Truck
c. Reduce cycle time, shorten haul distance and/or travel time

7.3 a. I shovel *(fo* 10 yd* (7.6 m3)
6 trucks + I spare *(w* 55 tons (50 tonnes)
b. Truck wait excessive; reduce fleet size or improve excavation working conditions

CHAPTER 8

8.2 $352.60/hr
SI 109.52/hr
$0.67/ton ($0.74/tonne)

CHAPTER 9

9.1 a. 10,944 tons (9928 tonnes)
b. 16.64 kW-hr/skip

9.3 *145 sec/cycle*
6.0 tons (5.4 tonnes)
4 ropes ® i in. (19 mm)
5.0-Π sheave (1.5 m)
Peak 2083 hp (162» k W)
Average 909 hp (67H k W)
R M S 1045 hp (793 kW)
2.14 kW-hr/ton (2.36 kW-hr/tonne)
10.*7ft/ton (II./ tonne)*

CHAPTER 10

10.2 *4 tons/min (3.6 tonnes/min)*
10. *4* a. *25.5%*
h. 79.5%
c. 31.0%
10.6 *25.0 ft (7.6 m)*
10.8 a. *3600 lb/in.^{2} (24.8 MPa)*
1200 ib/in^2 (8.3 MPa)*
c. Top + 1080. 360, 1440. 1800 lb/in.2 (7.5, 2.5, 9.9, 12 4 MPa)
Corner - 11,160, 14,760, 20,160. 31,320 lb/in.2 (-77.0 ,101.8, *139.0, 216.0 M Pa)*
d. R/h ^ 4, fails
e. R/h = I, FS = 1.11
R/h = 2, FS = 1.52

f. R/h = I and 2, metal mine drift
R/h = 4, coal mine opening
R/h = 8. block caving *undercut*

10.10 a. Fails
b. No
c. None

10.12 *14 ft (4.3 m)*

CHAPTER 11

11.1 8.5 in./sec *(216* mm/sec)

11.3 a.4
b $6.72/hr
C . $38.72/hr
d. $0.15/ft ($0.49/m)

11.5 pneumatic (4 drills) Hydraulic (3 drills)

$I79.9l/hr $133.77/hr

$0.57/ft ($1.87/m) $0.42/ft ($1.38/m)

$ 1.07/ton ($ 1.18/tonne) $0.80/ton ($0.88/tonne)
Hydraulic more economical

11.7 a. 5 ft (1.5 m)
b. Per hole 7 ft (2.1 m)
5 ft (1.5 m)
7 lb (3.2 kg)

11.9 b. Per hole, cut, 4 @ 4.1 lb (1.9 kg)
Relievers, 4 @ 0.6 lb (0.3 kg)
No. I enlargers, 4 @ 1.6 lb (0.7 kg)
No. 2 enlargers, 4 @ 3.3 lb (1.5 kg)
Trimmers, 4 @ 3.3 lb (1.5 kg)
c. 20 holes
d. 143 ft (44 m)
e. 9.5 ft/ton (3.2 m/tonne)
f. 52 lb (24 kg)
g. 3.4 lb/ton (1.7 kg/tonne)

CHAPTER 12

12.2 89 ft^3min (0.042 m^3/sec)

12.4 a. 0.19 in. water (47 Pa)
b. 0.60 in. water (149 Pa)

12.6 a. 9.02 in. water (2.24 kPa)

c. 110,000 ft^3/min (51.9 m^3/sec)
7.0 in. water (1.74 kPa)
d. 1140 rpm

CHAPTER 13

13.1 a. 42.0 /lb (92.6 /kg)
b. 3.23% Zn
13.3 a. 107 tons (97 tonnes)/employee-shift
b. Drilling $ 0.70/ton ($0.77/tonne)
Blasting $ 0.16/ton ($0.18/tonne)
Loading $0.42 /ton ($0.46/tonne)
Haulage $0.74 /ton ($0.82/tonne)
c. direct exploitation $2.23 /ton ($2.46 /tonne)
d. indirect exploitation $0.22/ton ($ 0.24/tonne)
e. Overall exploitation $2.45/ton ($ 2.70/tonne)
Overall mining $5.90/ton ($ 6.50/tonne)
f.Overall production $21.70/ton ($ 23.92/tonne)
g.@44.2 lb Zn (97.4 kg), value $24. 04/ton ($ 26.50/tonne)
Gross profit $2. 34/ton ($ 2.58/tonne)
Net profit 1.54/ton ($ 1.70/tonne)
h.Overall exploitation 2.78/ton ($ 3.06/tonne)

REFERENCES

Ahrens , E. H., 1983, "Practical Mining Geology," Preprint, No. 83-371, Fall Meeting. Soc. Mng. Engr.—Л1ME, Salt Lake City, UT, 11 pp.

Alpan, H. S., 1951, " Factors Affecting the Speed of Penetration of Bits in Electric Rotary Drilling, Part 2," *Trans. Instn. Mng. Engr.* (London), Vol. Ill, p. 374.

Alport. P., and R. E. G. Caines, 1984, ''Equipment Adaptation to the Canadian North at Iron Ore Company of Canada," *CIM Bull.* (Montreal), Vol. 77, No. 682, Feb., pp- 39-44.

Anderson. O. E., 1981, " White Pine Separates Safety and MSHA Compliance Functions," *Mng. Cong. J.*. Vol. 67, No. 5, May, pp. 29-33.

Anon., 1962, " Hydraulic Mining with Rotary Drill Unit," *Coal Age,* Vol. 67, No. 7, July. PP 96-99.

Anon., 1967a, *The American Law of Mining,* 5 vols., Rocky Mtn. Mini. Law Fdn., New York.

Anon., 1967b. " Kennecott Sets Sights on Nuclear Test for In Situ Recovery of Copper," *Eng. Mng. J.,* Vol. 168, No. 11, Nov., pp. 116-122.

Anon., 1968, *Rapid Excavation: Significance Needs, Opportunities,* Comm, on Rapid Excavation, Pub. 1690, Natl. Research Council—NAS/NAE, Washington, DC. 48 pp.

Anon., 1970, *Proc. Symposium on Engineering with Nuclear Explosives,* U.S. Atomic Energy Commission and Amer. Nuclear Society, Las Vegas, NV, 1785 pp.

Anon., 1974, *Analysis o f Large Scale Noncoal Underground Mining Methods,* U.S.
Bureau of Mines, OFR 36-74, Dravo Corp., Govt. Pmt. Off., Washington, DC, 605 pp.

Anon.. 1976a, *Surface Mining Supervisory Training Program,* Bucyrus-Erie Co., South Milwaukee, W I, 1:1-5:13 pp.