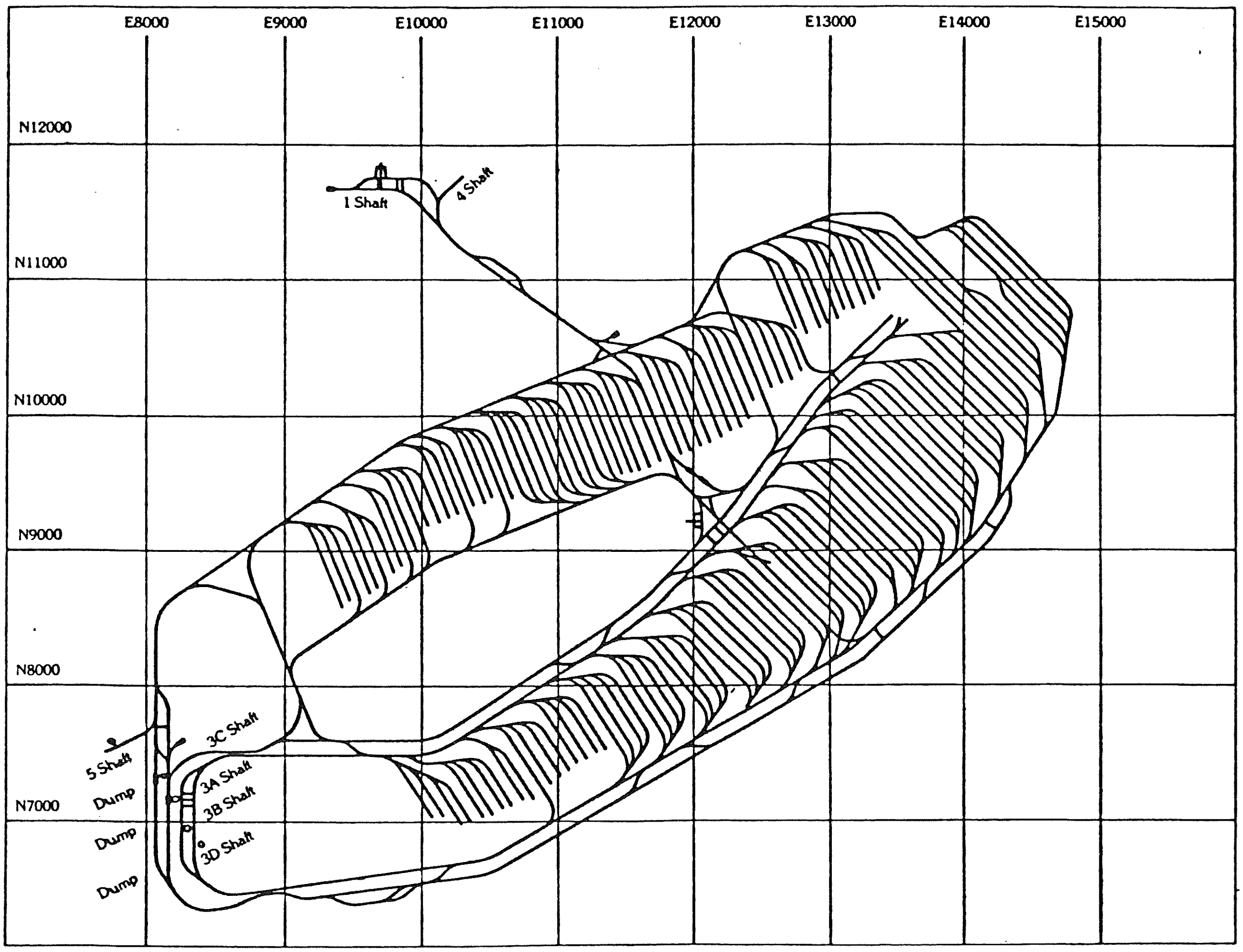
PRELIMINARY REPORT

Underground Mine Model



MEQB REGIONAL COPPER-NICKEL STUDY

A PRELIMINARY REPORT
DETAILS OF THE UNDERGROUND
MINE MODELS

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REGIONAL COPPER-NICKEL STUDY
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This report creates two models of hypothetical underground copper-nickel mines located near the basal contact of the Duluth Complex in northeastern Minnesota (St. Louis and Lake counties). The mine models have been developed by the Minnesota Environmental Quality Board (MEQB) Regional Copper-Nickel Study in order to determine potential environmental impacts, approximate mining costs, and the requirements of an underground mine in terms of equipment, supplies, land, and manpower.

Several assumptions form the framework for development of the hypothetical mine models. The mines are designed to produce 7,938,000 metric tons (mt) (8,750,000 short tons (st)) of ore per year over a mine life of 30 years. An additional 635,000 mt (700,000 st) of waste rock will be produced annually. The cut-off grade (stated in terms of percent copper) is set at 0.60 percent copper. The average grade of the ore is 0.80 percent copper and 0.20 percent nickel.

The mining methods chosen as the most applicable to underground mining of the Duluth Complex are room-and-pillar mining and blasthole open stoping. For modelling purposes, room-and-pillar mining will be used where substantial reserves of fairly flat lying ore exist. The dip angle must be less than 20^{o} and the thickness of the mineralization is restricted to less than 25 meters (m) (82 feet (ft)). Blasthole open stoping will be employed when the height of the ore zone is greater than 25 m (82 ft). Using the models as examples, the report outlines: 1) the activities of the preproduction development period; 2) the major features of the mining methods, including the equipment involved; and 3) the costs associated with each mining method.

A work force of about 1000 people will be necessary for the mine to function at a production rate of 7,938,000 mt/year.

The costs associated with the two mining methods are summarized below.

Operating Costs--$/mt of Cu-Ni ore	Blasthole Open Stoping	Room-and-Pillar Mining
Development	2.05	0.52
Ground Control	—	.26
Drilling	.30	.85
Blasting	.12	.33
Haulage	.76	1.82
Crushing and Hoisting	.35	.33
Power and Fuel	.30	.33
Maintenance (non-allocatable)	.58	.58
Supervision and Services	1.05	1.10
General	.35	.39
Total	$5.86	$6.51
Capital Costs	$130,400,000	$112,200,000

The mining glossary found in Appendix A may aid in understanding some of the mining terms used in this report.

INTRODUCTION

The copper-nickel mineral resources of northeastern Minnesota can be removed from the ground by open pit mining, underground mining, or combinations of both methods. A preliminary report on open pit mining of the Duluth Complex has already been prepared by the Minnesota Environmental Quality Board (MEQB) Regional Copper-Nickel Study. Underground mining will be examined in this report.

Underground mining methods must be considered when the depth of a mineral deposit is such that removal of the overburden makes surface mining techniques unprofitable or when external factors prohibit the operation of a surface mine. Determining the optimum underground mining method requires careful analysis of geologic, economic, and environmental data. By using a less rigorous approach, room-and-pillar mining and blasthole open stoping were chosen as the mining methods most suitable to the development and mining of the mineral resources of the Duluth Complex. Both methods are high productivity mining methods which incorporate the latest developments in underground mining equipment.

The hypothetical mine models developed in this report attempt to typify the mining practices that may be employed in northeastern Minnesota. Although no two mines are alike, there are enough similarities between mines (and mining methods) that much of the information found in this report will remain valid even if different mining methods are actually utilized.

ADVANTAGES OF UNDERGROUND MINING

The advantages of underground mining are basically the disadvantages of open pit (or surface) mining. Similarly, the inverse is true; that is, the disadvantages of underground mining relate to those areas where surface mining holds an advantage. The major advantages of underground mining over open pit mining are lessened environmental impacts, greater selectivity, and reduced exposure to weather.

Environmental Impacts

Much less land surface is necessary for the operation of an underground mine than for an open pit mine of the same production size. At the very least, an underground mine will require that land be available for the mine entrance and access roads. Commonly, additional land is utilized for ventilation shafts, multiple mine entrances, buildings, storage areas, and waste rock dumps. (However, a mining company could decide to dispose of the waste rock in suitable underground openings and thus eliminate the problem of placing waste rock above ground.) Because less land is needed for an underground mine than for an equivalent open pit mine, there should be less disruption of the plant and wildlife species that inhabit or use the surrounding area. With less surface activity occurring, dust and noise generation should be reduced accordingly. Finally, the magnitude of the reclamation program for an underground mine is greatly diminished because of the absence of an open pit and a network of roads, and the reduced size of the waste rock dumps. This results in lower reclamation costs for underground mining than for open pit mining.

Selectivity

Underground mining methods make it possible to economically mine mineral deposits which are too small, irregular, and/or deeply buried to extract using surface mining methods. Mining methods exist which can be applied to irregularly shaped deposits, narrow veins, discontinuous veins, and small pockets or lenses of ore. Using these methods, the mineral laden rock can be removed without having to handle large quantities of low-grade ore and/or waste rock. As a rule, the more selective mining methods are only applicable to deposits where the value of the ore is high enough to justify the generally lower productivity associated with selective mining. On the other hand, underground mining of low-grade deposits must be performed as efficiently as possible in order to be economical, which usually means that large scale total extraction mining methods must be used. However, even these large scale methods offer more selectivity than open pit mining which ultimately removes all the material that overlies the ore at a given depth.

Protection From Weather

To many miners, the environment of an underground mine is more attractive than that of an open pit. An underground mine provides shelter from the elements and offers an unchanging working environment. The temperature in most underground mines does not vary greatly from season to season. Where the natural ambient air temperature of a mine is uncomfortable, the incoming air can be heated or cooled to maintain a constant mine temperature. Likewise, humidity can be regulated. Regardless of what the weather may be like on the surface, the underground climate can remain stable. This is important in terms of a miner's comfort and productivity.

DISADVANTAGES OF UNDERGROUND MINING

Underground mining accounts for only about 15 percent of the metallic and nonmetallic ore (excluding coal, sand and gravel, and stone) produced in the United States (E/MJ June 1977, p.77). Obviously, in cases where both surface and underground mining are physically applicable, surface mining is the favored method. There are several disadvantages of underground mining that can account for this, including higher costs, a longer development period, lower recovery of ore, greater safety risks, shortage of underground labor, and the possibility of subsidence.

Economy

Underground mining is nearly always more costly than surface mining. This is due primarily to the difference in productivity between the two methods. Using tons per man-shift as a measure of productivity, underground mining methods average about one-tenth the productivity of surface mining methods (SME Handbook 1973, p. 17-7). Some of the reasons for this are:

1) The restrictive sizes of the work spaces do not allow for the maximum utilization of existing mining equipment. As an example, consider a raise boring machine which must be partially dissassembled after each raise is drilled so that it can be transported through drifts and shafts with limiting clearance dimensions. This dissassembly takes about four man-shifts.

2) The development of underground equipment and technology has not kept pace with that of open pit. Underground mining could benefit from any new innovations which would make full use of automation and electronics.

3) Underground mining is more labor intensive than open pit mining. The above two reasons are partly responsible for this but the overall nature of an underground mine also affects productivity. When faced with the less than ideal working conditions found in most underground mines, worker productivity suffers no matter what incentives are offered as a countermeasure. Productivity is also lowered because of the number of employees who perform mining duties which do not directly expose or remove ore, but are instead related to controlling the udnerground environment (such as providing ground control, air, water, and light).

Development Period

Mine development follows discovery, title acquisition, and exploration, but only rarely is the size of an initial discovery adequate to justify immediate construction of production facilities. Therefore, the exploratory work (such as geophysics and geochemistry, drill holes, shafts, and drifts) must proceed, phase by phase, until the evidence warrants investment in and construction of mine, plant, and ancillary structures. During this time, unusually large amounts of money are being committed to a project for which there is no guarantee of success. Mining is a capital intensive and high risk industry with the risk being greatest during the exploration and development periods. The possibility exists that after spending millions of dollars on exploration, it may be decided that the mine is not a profitable venture and further activity should be suspended. If mine development is encouraged, it is economically desirable to bring the mine into production quickly. According to a U.S. Bureau of Mines report on the time required to develop Arizona copper mines, open pit development took from one to four years and underground development from four to eight years (USBM Information

Circular 8702). Because there is no return on investment during the development period, the investor will tend to favor an open pit mine over an underground mine because of the shorter lead time involved.

Recovery of Ore

The term "recovery of ore" can be defined as the percentage of the ore grade material within the mining limits that is removed and made available for treatment and upgrading. Table 1 lists the percent ore recovery that can reasonably be expected with each of the major underground mining methods.

Table 1. Percent ore recovery for different underground mining methods.

Underground Mining Method	Percent Ore Recovery Average	Range
Pillared open stoping	75%	60-80%
Room-and-pillar mining	70%	50-90%
Sublevel stoping	75%	65-85%
Shrinkage stoping	80%	75-85%
Stull stoping	Approaches 100%	
Cut-and-fill stoping	Approaches 100%	
Square-set-and-fill stoping	Approaches 100%	
Longwall mining	85%	75-100%
Shortwall mining	85%	75-100%
Top slicing	95%	90-100%
Sublevel caving	80%	60-90%
Block caving	95%	85-100%

It can be seen that caving methods such as longwall and shortwall mining, top slicing, sublevel caving, and block caving; and selective mining methods such as stull stoping, cut-and-fill stoping, and square-set-and-fill stoping achieve the highest recovery of ore. The other stoping methods average about 75 percent recovery. The ore recovery attained by open pit mining approaches 100 percent. Higher recoveries permit the extraction of greater amounts of the metal contained in the ore and provide for the maximum use of natural resources.

Safety

" Underground mining is a hazardous occupation. Work is conducted in a hostile environment, in a structure whose behavior cannot be forecast with certainty, and where insidious hazards to health, such as gases and dusts, often are found as well. Also, work usually is conducted at low levels of illumination, high levels of noise, in cramped quarters, and with a high concentration of mechanical equipment." (SME Handbook 1973, p.12-10) The Mining Enforcement and Safety Administration (MESA) compiles United States mineral industry injury statistics. A summary of the 1976 statistics for metal and nonmetal mines (excluding coal) appears in Table 2. It shows that injuries in underground mines are more numerous than those which occur in surface mines.

Table 2. Mine injury statistics for 1976.

Mining Operations	Injuries Per Year: Fatal	Injuries Per Year: Nonfatal Disabling	Injuries Per Year: Non-disabling	Frequency Rate/ 1,000,000 Hours: Disabling	Frequency Rate/ 1,000,000 Hours: Non-disabling
Metal Mines					
Underground	26	1962	563	37.18	10.89
Surface	6	527	445	9.22	7.66
Mills	2	636	547	10.11	8.81
Nonmetal Mines (excluding coal)					
Underground	9	315	97	23.16	6.76
Surface	5	312	255	11.55	9.52
Mills	8	689	372	13.35	7.14

"The largest single cause of accidents in underground mines of all types has been falls of ground. Other major causes relate to the use of electrical equipment, the haulage system, and the use of explosives. The majority of accidents of all types occur at or in the immediate proximity of the working faces. A mine operator should be ever conscious of adopting operating methodology that will improve the environment in which mine labor works. Such improvements usually result in concomitant improvements in output and cost of production." (SME Handbook 1973, p.12-11.)

Labor Availability

The success of any mining project is highly dependent on the ability of the mining company to recruit and maintain a competent and stable work force. A new mine usually can attract laborers by offering a competitive compensation package, a pleasant and diverse community in which to live, a safe and healthy work environment relative to other mines, and an opportunity to develop a broader range of job skills. Generally, laborers with open pit mining skills are more readily available than those possessing underground mining skills. Most workers can be trained in the rudiments of mine labor but no amount of training can take the place of knowledge gained through years of experience. It has been said that the underground miner is a special breed of person. Not every person is willing to work in the environment of an underground mine. The miner must be able to cope with the artificial environment created and the inherent dangers present underground.

Subsidence

Subsidence is the sinking or lowering of the earth's surface due to the excavation and subsequent collapse of underground workings. Subsidence is affected by the mining method used, span and height of the opening, depth of the excavation from the surface, strength of the rock, and the vertical and lateral pressures in the rock. Underground mining methods which incorporate caving and/or a high percentage of ore recovery are most likely to cause subsidence. "Detailed studies will be needed for each particular underground mine site and mining method to thoroughly evaluate subsidence. Subsidence can occur during or after termination of mining. If subsidence occurs after mining ceases, it could create environmental and economic problems for responsible state and federal agencies since all areas would need to be restricted until reclamation could be completed. Subsidence could alter the surface drainage and hydrology of an area with subsided areas filling with water and possibly discharging acidic surface and ground waters." (Hays, USBM Contract Report S0133089, 1974, p.83.)

MINING METHODS

Underground mining methods should be considered when the depth of a mineral deposit is such that removal of the overburden makes surface mining techniques unprofitable or when external factors prohibit the operation of a surface mine. The process of choosing an underground mining method involves selecting the methods which are physically adaptable to the configuration of the mineral deposit in question, and then, by the process of elimination, determining which is the most advantageous in terms of production rate, cost, safety, and environmental protection.

The various underground mining methods can be classified on the basis of the ground support they provide. Three broad classes of mining systems are recognized as follows:

> 1. Methods in which the underground openings (rooms or stopes) created by the extraction of the mineral are self-supported in that no regular artificial method of support is employed: that is, openings in which the loads due to the weight of the overburden or tectonic forces are carried on the sidewalls and/or pillars of unexcavated mineral or rock. This specification does not preclude the use of rockbolts or other light systems of support, provided that this artificial support does not significantly affect the load carried on the self-supported structure.
>
> 2. Methods in which stopes or rooms require significant support, that is, support to the degree that a part of the superincumbent load is carried on the support system.
>
> 3. Methods in which, because of the spatial and mechanical properties, the deposit is induced to cave under the action of gravity to produce better results than more selective methods.
>
> (SME Handbook 1973, p.9-9.)

Table 3 arranges the existing underground mining methods into these three broad classes.

Table 3. Underground mining methods.

I. Self-Supporting Openings:
   - A. Open-stope mining:
     1. Isolated openings
     2. Pillared open stopes
        - a. Open stoping with random pillars
        - b. Open stoping with regular pillars
   - B. Room-and-pillar mining
   - C. Sublevel stoping
   - D. Shrinkage stoping
   - E. Stull stoping

II. Supported Openings:
   - A. Cut-and-fill stoping
   - B. Square-set-and-fill stoping
   - C. Longwall mining
   - D. Shortwall mining
   - E. Top slicing

III. Caving Methods:
   - A. Sublevel caving
   - B. Block and panel caving

(SME Handbook 1973, p.9-9)

A discussion of these mining methods and the characteristics of a mineral deposit which may limit the applicability of the mining methods can be found in the SME Mining Engineering Handbook (pp. 9-8 to 9-21; 12-45 to 12-253). Appendix B comprises the chapter 9 reference. Gerken also provides descriptions of the different underground mining methods in the appendix of his University of Minnesota Masters Thesis entitled "Feasibility of Different Underground Mining Methods for Copper-Nickel Mining in the Duluth Complex in Northeastern Minnesota From Fracture Data." (1977) It is suggested that anyone who desires more information on these mining methods refer to these sources.

Since the geometric configuration and the geologic and physical (or mechanical) properties of a mineral deposit are fixed by nature, an examination of these factors in order to determine the physical feasibility of the various mining methods is normally the first step taken in narrowing the choice of possible methods. Table 4 summarizes the common applications of the various underground mining methods in relation to the various types of ore bodies, their dips, and the strength characteristics of the ore and adjacent country rock. Table 5 shows the application of various large-scale mining methods to different geologic and mechanical criteria.

Table 4. Applications of underground mining methods.

Type of Ore Body	Dip	Strength of Ore	Strength of Walls	Commonly Applied Methods of Mining
Thin beds	Flt	Stg	Stg	Open stopes with casual pillars Room-and-pillar Longwall
		Wk or Stg	Wk	Longwall
Thick beds	Flt	Stg	Stg	Open stopes with casual pillars Room-and-pillar
		Wk or Stg	Wk	Top slicing Sublevel caving
		Wk or Stg	Stg	Underground glory hole
Very thick beds				Same as for masses
Very narrow veins	Stp	Stg or Wk	Stg or Wk	Resuing
Narrow veins	Flt			Same as for thin beds
(widths up to economic length of stull)	Stp	Stg	Stg	Open stopes Shrinkage stopes Cut-and-fill stopes
			Wk	Cut-and-fill stopes Square-set stopes
		Wk	Stg	Open underhand stopes Square-set stopes
			Wk	Top slicing Square-set stopes
Wide veins	Flt			Same as for thick beds or masses
	Stp	Stg	Stg	Open underhand stopes Underground glory hole Shrinkage stopes Sublevel stoping Cut-and-fill stopes Combined methods
			Wk	Cut-and-fill stopes Top slicing Sublevel caving Square set stopes Combined methods
		Wk	Stg	Open underhand stopes Top slicing Sublevel caving Block caving Square-set stopes Combined methods
			Wk	Top slicing Sublevel caving Square-set stopes Combined methods
Masses		Stg	Stg	Underground glory hole Shrinkage stopes Sublevel stoping Cut-and-fill Combined methods
		Wk	Wk or Stg	Top slicing Sublevel caving Block caving Square-set stopes Combined methods

Wk=weak; stg=strong; flt=flat; stp=steep.

(SME Handbook 1973, p.12-7)

Table 5. Geologic and mechanical criteria in large-scale mining methods.

MINING METHOD	ORE BODY CHARACTERISTICS: ORE STRENGTH Weak	Mod	Strong	WASTE STRENGTH Weak	Mod	Strong	ORE BODY CONFIGURATION: BEDS Thick	Thin	VEINS Nar	Wide	MASS	ORE DIP Flat	Mod	Steep
Room & Pillar[1]		x	x		x	x	x	x				x	x	
Sublevel Stoping[2]		x	x			x			x	x	x			x
Shrinkage		x	x	x	x				x	x			x	x
Cut & Fill		x	x	x	x				x	x	x		x	x
Square Set	x			x	x				x	x	x		x	x
Block Caving[3]	x	x		x	x					x	x			x
Sublevel Caving		x	x	x	x					x	x			x
Longwall	x	x		x				x				x	x	

[1]Uniform thickness and grade.

[2]Regular hanging and foot walls.

[3]Strong fractured rock also can be caved.

(Dravo Corporation, Analysis of..., p. 147)

Spatially, the Duluth Complex can be classified as a massive or multi-layered ore body which possesses strong ore and waste rock. The word massive, when used to describe ore deposits, refers to those which have developed in three dimensions, and have highly variable shapes. Table 4 points out that the mining methods commonly applied to a massive ore body where both the ore and the wall rock are fairly strong are shrinkage stoping, sublevel stoping, cut-and-fill·stoping, and combined methods. Locally, the ore bodies are composed of irregular lenses and discontinuous vein-like deposits, dipping at moderate to near vertical angles. Table 5 emphasizes that sublevel stoping, room-and-pillar mining, and cut-and-fill stoping are the mining methods best suited for underground mining of the Duluth Complex.

Although predicting the cavability of a rock mass is difficult, the very preliminary analysis performed by Gerken (Feasibility of Different Underground Mining Methods..., 1977) indicates that successful caving of the Duluth Complex is questionable and that block and sublevel caving may not be feasible. Until more tests are conducted (preferably under actual mining conditions) it will be assumed that caving methods are not applicable to the Duluth Complex.

After applying physical constraints to the selection of a suitable underground mining method, focus can be centered on room-and-pillar mining, sublevel stoping, shrinkage stoping, cut-and-fill stoping, and any combinations or variations of methods which may also be applicable.

At this point, the techniques used in the search for a mining method become less standardized. The decision making process should involve the mining company's management group, financiers, marketing agents, engineers, and consultants. Using the analytical tools of their respective disciplines, these groups, working together, should be able to select a mining method which is the "best choice" (based on the data that is available for study). In actual practice this may not be the case since the best choice is not always discernable. Because each group views the problem from their own angle, there may not be unanimity between the groups and deciding which is the best overall method can be difficult.

An examination of Table 6 illustrates why the selection of the optimum mining method is not a simple matter. The left-hand column in Table 6 identifies factors which may be considered in the process of selecting a mining method. Any of the methods which can claim these factors as an advantage has received a check in the appropriate column. Note that the factors have not been judged for relative importance and, consequently, are not weighted. There are many relationships and trade-offs indicated in Table 6. An engineer wishing to hold down mining costs would prefer to use a large-scale and generally high productivity mining method which is amenable to mechanization (such as room-and-pillar mining, sublevel stoping, or caving methods). Unfortunately, these methods commonly require that a great deal of development work be performed before mining can commence; and the adaption of mechanization means that a higher initial capital investment must be made. Therefore, if capital is limited, the financiers may decree it necessary to adopt a stoping method which will bring the mine into production quickly and with a small capital outlay.

Caving methods (expecially block caving) are low cost mining methods, but if production must be expanded or contracted due to a change in market conditions, serious problems can occur. In order to increase production, development work must be substantially ahead of mining (however, advance development results in a tie-up of capital). Also, forced production may result in increased dilution. When trying to curtail production from a mine using caving methods, the caving process may be adversely affected. The mine will also have to bear the expense of idle machinery. Another problem with caving methods is that they are not easily modified to other mining methods or variations should this ever be desired. Finally, if ground subsidence is not allowable, then caving methods will have to be dropped from consideration, regardless of their other advantages.

Earlier it was decided that room-and-pillar mining, sublevel stoping, shrinkage stoping, and cut-and-fill stoping were physically applicable to underground mining of the Duluth Complex. Using Table 6 as a guide, the final mining methods selected are room-and-pillar mining and sublevel stoping.

Table 6. Advantages of various underground mining methods.

	Open Stoping	Room and Pillar	Sublevel Stoping	Shrinkage Stoping	Cut and Fill	Square Set	Sublevel Caving	Block Caving
Low Initial Capital Investment (Usually More Rapid Development)	X			X				
Low Cost		X	X					X
High Productivity		X	X				X	X
Amenable to Mechanization (Often Less Labor Intensive)		X	X				X	
Safety		X	X				X	
Selectivity	X				X	X		
Low Dilution	X	X	X	X	X	X		
High Percentage of Ore Recovery					X	X		X
Less Likely to Cause Subsidence	X	X	X	X	X	X		
Easily Modified and Flexible to Different Productivity Rates	X	X	X					

MINE PLANNING

Developing a mine plan is a difficult task. With limited data available for analysis, the mine planner must rely on judgement and experience to select many of the details that comprise the mine plan. Since the apparent economic feasibility of the mine is related to the conclusions arrived at during the planning process, it is imperative that the assumptions and judgements used in the planning stage be clearly stated so that the limitations of the conlusions arrived at are understood. Some of the initial assumptions concerning the underground mine models are:

1) The underground mine will produce 7,938,000 mt (8,750,000 st) of ore annually. Principal underground non-coal mines in the United States range in size from 150,000 to 15,000,000 metric tons per year (mtpy) with the typical size of a new underground mine being approximately 1,500,000 mtpy. The production capacity which has been selected for the mine models is based on the plans of a mining company currently active in the Regional Copper-Nickel Study Area (RCNSA).

2) The cut-off grade (stated in terms of percent copper) for the underground mine models will be set at 0.60 percent copper. The average grade of the ore will be 0.80 percent copper and 0.20 percent nickel. These grades are based on the findings of the Minnesota Department of Natural Resources (MDNR) and on what is believed to represent actual economic cut-off values.

3) It is assumed that the mines will produce 635,000 mt (700,000 st) of waste rock annually. This figure is derived from an estimate of the amount of development work which will be performed outside of the ore boundaries.

4) A 30-year life of the mine will be assumed for the mine models.

5) It is assumed that sufficient manpower, equipment, electrical power, and fuel oil will be available throughout the life of the mine.

6) The mining methods under consideration are room-and-pillar mining and sublevel stoping. Room-and-pillar mining can be used where there are substantial reserves of fairly flat lying ore and when the thickness of the mineralized zone is less than 25 m (82 ft). With thicknesses greater than 25 m, sublevel stoping or modifications of sublevel stoping can be applied to mining the ore body.

7) For the stoping mine model, it will be assumed that the ore is homogeneous and regularly dipping at 25°. Furthermore, it will be assumed that mineralization above the cut-off grade is present and continuous throughout a vertical height of 45 m (148 ft), and that there is only one layer of ore.

The first mining plan that will be examined incorporates the most recent developments in sublevel stoping. The mine will employ large diameter blastholes which extend the full height of the ore zone, transportation by trackless vehicles on the production level and by rail on the main haulage level, ore passes which take full advantage of the force of gravity to assist in ore handling, and the latest in mechanized and automated equipment. Because of the length of the blastholes, sublevels are no longer necessary, and this mining method is often termed blasthole open stoping, the term which will be used in this report.

Room-and-pillar mining will also be studied. As with blasthole open stoping, room-and-pillar mining benefits from the use of mechanized, high-productivity mining equipment.

BLASTHOLE OPEN STOPING

PREPRODUCTION DEVELOPMENT

In order to open up an underground mine and insure a continuous level of production in the early years of a mine's life, a preproduction development stage is necessary. During this time period, the most important (and, usually, the permanent) mine openings are excavated and the extent of the ore is proven. Any unforeseen problems which arise during preproduction development will not adversely effect the production rate, so the most expedient solution to the problem need not be adopted in place of the best solution. When preproduction development is completed, there should be a desired tonnage of ore available for immediate withdrawal by the mining method being employed. This generally means that several working areas must be completely accessible.

Once the mill starts up, continuous ore removal is important so that the mill will not be forced to cut back its production. To make this possible, it usually is necessary for (production) development to be scheduled so that it always precedes the mining operations by a specific time interval or tonnage increment.

The life cycle for the blasthole open stoping mine is shown in Figure 1. Note that mine development is performed at an accelerated rate up through the first years of production, and that production commonly begins before preproduction development is complete. This occurs because of the economic advantage of establishing an early positive net cash flow.

A two-year phase-out period has been scheduled for the mine. There is always the possibility that, with the passing of time, operating conditions

COMPARISON OF DEVELOPMENT RATES DURING PREPRODUCTION AND NORMAL PRODUCTION OPERATION.

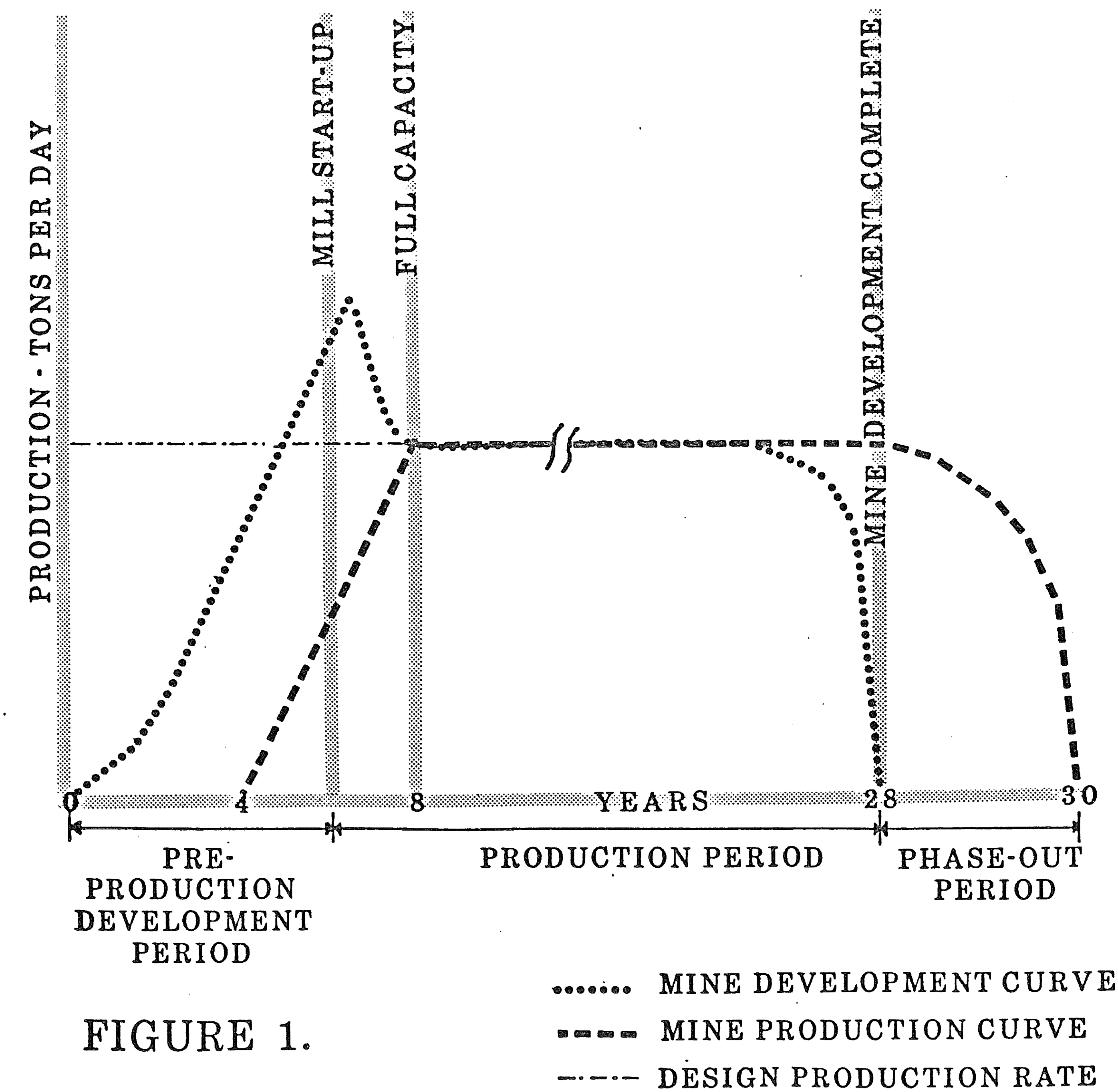


FIGURE 1.

can change so as to bring about either a premature shut-down of the mine or an extension of its productive life.

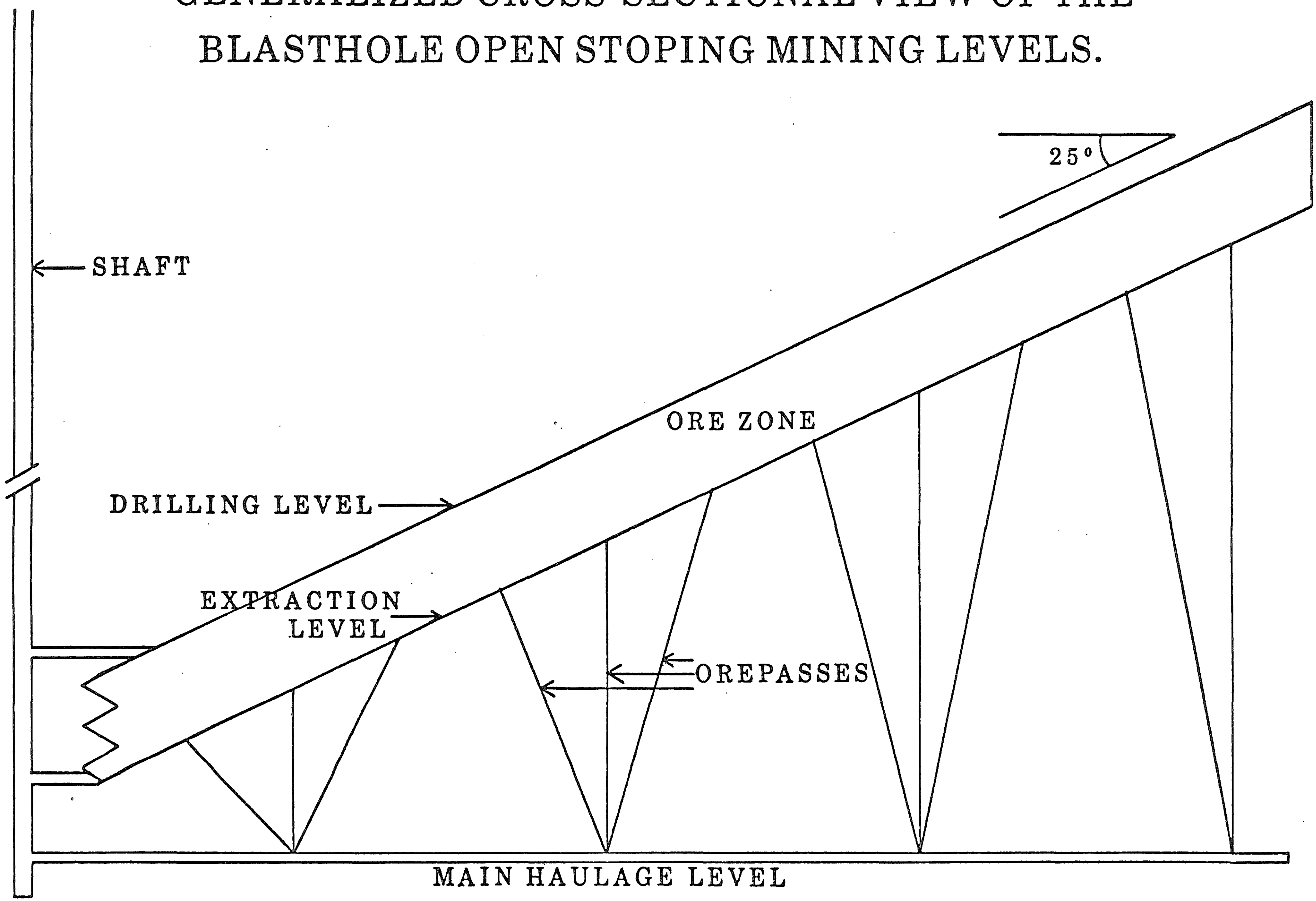
Access to an underground mine is gained by a slope, ramp, or shaft that is normally sunk outside of the ore zone, away from any possible effects from subsidence and blasting or other production operations in the stoping process. This also prevents potential reserves from being tied up because of their location with respect to the mine opening. However, to minimize transportation costs, it is important to provide a location that will permit the lowest average-ton-mile haulage cost for the orebody. (Dravo Corporation, Analysis of..., p. 279)

The major levels of the blasthole open stoping mine are established by driving horizontal openings into the ore zone from the mine opening. With the orebody configuration that has been assumed, three levels will be established--the drilling, extraction, and main haulage levels (see Figure 2). The vertical distance between the drilling and extraction levels will be about 55 m (180 ft) and the vertical distance between the extraction and main haulage levels will vary from 30 to 250 m (100 to 820 ft) due to the dip of the ore.

The uppermost level, the drilling level, corresponds to the top of the stopes. The main purpose of this level is to provide access to the stopes for the blasthole drills. The elevation of the drilling level can be altered in order to follow the structure of the ore as much as possible.

The next level below the drilling level is the extraction level. This level determines the bottom of the stopes. Most of the mining activity occurs on the extraction level. The elevation of this level can be adjusted to follow

GENERALIZED CROSS-SECTIONAL VIEW OF THE BLASTHOLE OPEN STOPING MINING LEVELS.



F'GURE 2.

the bottom contact of the ore.

The main haulage level is located below the lower limit of the minable reserves. A network of railroad track is developed in the portion of this level which underlies the ore to be mined. Ore reaches the main haulage level from the above levels by falling through raises bored in the intervening rock and is then transported by train to the shaft(s). The main haulage level is essentially without grade ($\pm$ 0.5%).

At the outset of the preproduction development stage, access to the blasthole open stoping mine will be through the existing exploration shaft. It may be necessary for the exploration shaft to be extended further so that drifting can begin on the lowest level, the main haulage level. While the haulage level is being opened up, sinking of the production shaft can begin. The development work on the main haulage level will progress toward, and connect with, the production shaft. This will complete the ventilation loop and provide two points of access into the mine. Intense preproduction development of all three levels can now begin in the vicinity of either one of the shafts. A third shaft, the service and waste rock shaft, can now be sunk and linked up with the rest of the mine. At this time, the objectives of the preproduction development program are to prepare the mine for the safe and economical removal of ore at a constant rate over time. To this end, mine service facilities are installed, loading pockets are excavated, crushers are installed, the transportation system is laid out, and several stopes are completely developed and made available for mining.

Development of the mine will take eight years (four years of preproduction development only and four years of combined development and limited production).

If at all possible, initial development will be concentrated in that portion of the mine which is the most attractive economically, since the mining of this portion of the reserve brings about the most rapid return on investment.

Preproduction development expenditures for the blasthole open stoping mine will total approximately $28,000,000 (in 1977 dollars). Labor accounts for two-thirds of the preproduction development cost, and supplies account for the remaining one-third. These costs will be distributed over the first four years of development as shown below.

($ x 10^3)

Years	1	2	3	4	Total $
Labor	1,493	2,987	5,227	8,960	18,667
Supplies	747	1,493	2,613	4,480	9,333
Preproduction Development	2,240	4,480	7,840	13,440	28,000

Shafts and Hoists

The planning of the openings to an underground mine requires careful consideration and engineering analysis. Many factors must be studied in an attempt to determine the design of the opening which best satisfies the following guidelines:

1) Lowest capital cost
2) Lowest operating cost
3) Most dependable
4) Most efficient
5) Most flexible
6) Conforms to mining plan
7) Fastest to construct

Some of the more important factors which warrant careful study are: 1) depth of the orebody; 2) geological characteristics of the orebody; 3) spatial relationship of the opening with the orebody; 4) rock conditions around the opening; 5) ground water around the opening; 6) ore and waste tonnage required; 7) ventilation requirements; 8) multi-level or single level loading; 9) capital and operating costs; 10) underground ore transportation method; and 11) purpose of the opening.

Based on the above, the mine openings selected for the underground mine models will all be circular, concrete-lined, vertical shafts. At mine depths of greater than 610 m (2000 ft), the capital cost of constructing a ramp is from 1.3 to 2.1 times greater than that for a shaft of equivalent capacity. Similarly, construction time for a ramp is from 1.4 to 2.0 times longer than for a shaft.

The circular concrete-lined shaft has many advantages over any other type of shaft. Some of these advantages are:

1) Shaft Sizes. Since a circular shaft can best resist ground pressures, this configuration is less restricted in size than either timber-supported or rectangular concrete-lined shafts.

2) Ventilation. Air flow is much more streamlined, and velocities up to 7.6 meters per second (1500 feet per minute) are common in circular shafts filled with equipment. Velocities up to 10 meters per second (2000 fpm) can be used in ventilation shafts. Shock losses also are much smaller in circular shafts because less of the inside area is filled with equipment. When deep mines and high air flow velocities are involved, tubular steel sets provide a distinct advantage over structural WF shapes for reducing air flow resistance.

3) Production Capability. Since the compartment size is not as restricted, the potential production capability is much greater with circular shafts. Skip capacities exceeding 27 mt (30 st) are feasible. Also, skips can be hoisted on rail or rope guides at faster rates, which may exceed 14 meters per second (2800 fpm).

4) Improved Service Support. This capacity is less restricted because larger cages can be utilized. The larger the cage size, the greater the percentage of equipment and supplies that can be transported inside the cage instead of slung underneath. With bigger cages, large equipment can be lowered fully assembled, under the cage, or diassembled into major components and placed inside the cage. Cages generally are double-decked, and never are slung under muck skips.

5) Flexibility. Optimum flexibility is provided because of the numerous arrangements possible for the various types and sizes of conveyances and shaft equipment that can be installed. If the shaft has been properly designed, changes in mining methods that require higher tonnages, or larger equipment, can be handled easily.

6) Low Maintenance. Because of the concrete lining and steel shaft sets, maintenance costs are very low, and repairs, when necessary, can be accomplished more easily than in timber-supported shafts. The concrete linings are more difficult to replace than the structural steel members, but a deteriorated lining area often can be repaired with welded wire mesh and shotcrete.

7) Fire Safety. Except for any electrical cables, circular concrete-lined shafts are fireproof.

8) Mechanized Construction. Circular shaft construction adapts readily to mechanized sinking operations because of the shape and type of lining. As a result, techniques for decreasing costs and expediting the construction schedule are more likely to be developed.

The principal problem areas experienced with circular concrete-lined shafts include:

1) Space Utilization. A circular shaft opening is not as space efficient as a rectangular shaft. This problem can be overcome through careful planning and design.

2) Salt Damage. Ground water that contains saline solutions can have a very detrimental effect on concrete.

(Dravo Corporation, Analysis of..., p. 219.)

Because of the unique nature of shaft construction projects, the construction will be performed by a contractor. This is a common development practice since the contractor has the experience, equipment, and skilled labor available to him to complete the construction work on schedule.

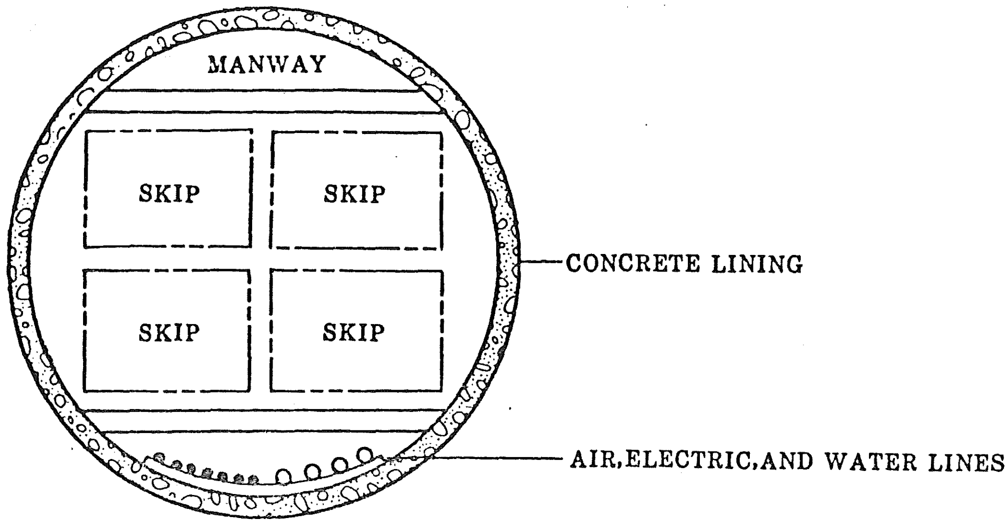
It is normally advantageous to open up a stoping mine from the lowest level of anticipated mining. There should be sufficient proven reserves above this level to indicate that the mine will be able to economically extract ore at the desired rate over the initial 30-year life of the mine. Assuming that any ore which lies within approximately 365 m (1200 ft) of the surface will be mined by surface mining methods, the underground mine will be located between depths of 365 to 730 m (1200 to 2400 ft).

The production shaft will be 730 m (2400 ft) deep and have an inside diameter of 7.3 m (24 ft). The service shaft will be 730 m (2400 ft) deep and have an inside diameter of 6.7 m (22 ft). The construction methods utilized with both shafts will be normal drilling, blasting, and mucking methods. The concrete lining will be one foot thick.

> The design and layout, or internal arrangement, of an opening for development of an underground mine is an important aspect in overall mine design. If all factors are not considered in the initial planning stage, any development opening may actually become a major bottleneck in subsequent mine operations. Openings must be of sufficient size to handle ventilation requirements at a reasonable mine pressure. They also must be capable of handling not only the ore produced but also the materials, equipment, and manpower needed to support the mine. It is advisable to design a certain amount of flexibility into any opening, as insurance against the unexpected. (Dravo Corporation, Analysis of..., p. 205.)

Typical internal arrangements for circular production and service shafts are shown in Figure 3. Circular concrete-lined shafts usually are divided into compartments by structural steel sets, with the arrangement depending on the purpose or function of the shaft. The conveyance guides for the service cage and the skips will be wooden and locked-coil wire rope, respectively. The shafts will also be designed to include water pipes, compressed air lines, electrical transmission lines, and an emergency escapeway. Both shafts will have the secondary function of either intake or exhaust ventilation.

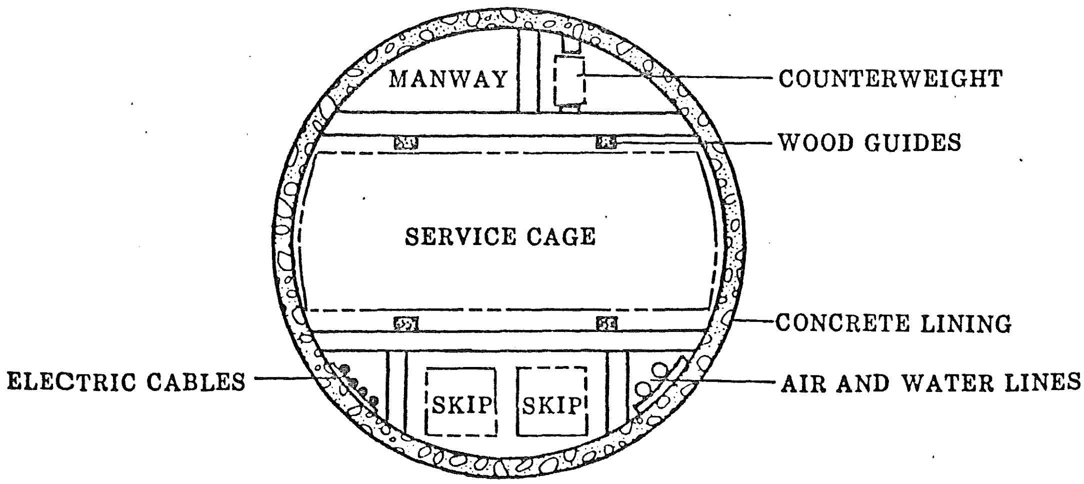
The design parameters for the ore, service, and waste rock hoists are outlined in Table 7.



24' I.D. PRODUCTION SHAFT

TYPICAL CIRCULAR SHAFT ARRANGEMENTS.

(1''=10')



22' I.D. SERVICE AND WASTE ROCK SHAFT

FIGURE 3.

Table 7. Hoist design parameters.

	Ore Hoists	Waste Rock Hoist	Service Hoist
Description	Ground mounted friction hoists. Double hoisting system with skips in balance.	Ground mounted friction 2 skips in balance	Ground mounted friction hoist Cage and counterweight in balance
Capacity	600 mt per hour per hoist 660 st per hour per hoist	180 mt per hour 200 st per hour	18 mt 20 st
Hoist Drum Diameter	3.05 meters 10.0 feet	1.83 meters 6.0 feet	3.05 meters 10.0 feet
Ropes	4 - 38.1 mm flattened strand 4 - 1.50 in. flattened strand	4 - 22.2 mm flattened strand 4 - 0.875 in. flattened strand	4 - 38.1 mm flattened strand 4 - 1.50 in. flattened strand
Weight of Conveyance	17.5 mt 19.3 st	6.4 mt 7.0 st	13.6 mt 15.0 st
Weight of Load	16.3 mt per skip 18.0 st per skip	6.1 mt 6.7 st	0 - 18 mt 0 - 20 st
Rope Speed	11.5 meters per second 2270 feet per minute (fpm)	8.6 meters per second 1700 fpm	7.6 - 2.3 meters per second 1500 - 450 fpm
Horsepower Required	3000 kilowatts per hoist 4000 hp per hoist	750 kilowatts 1000 hp	450 kilowatts 600 hp

The approximate costs (in 1977 dollars) for constructing and equipping the production shaft and the service and waste rock shaft are given below. The costs include the installation of all equipment (including the ventilation system) and the surface and underground facilities.

Production Shaft

Direct Labor	$7,440,000
Materials	5,390,000
Equipment Ownership and Operating Cost	2,600,000
Contractor's Overhead and Profit	3,160,000
	$18,590,000

Service and Waste Rock Shaft

Direct Labor	$4,600,000
Materials	3,110,000
Equipment Ownership and Operating Cost	1,840,000
Contractor's Overhead and Profit	1,960,000
	$11,510,000

The cost of deepening the existing exploration shaft to the main haulage level (say 120 m) and installing a new hoisting system is summarized below.

Exploration Shaft Modification

Direct Labor	$768,000
Materials	557,000
Equipment Ownership and Operating Cost	269,000
Contractor's Overhead and Profit	326,000
	$1,920,000

Total Cost of Shafts and Hoists	$32,020,000

The total cost of installing the shaft facilities for the mine will be equally distributed over the first four years of development as shown below.

($ X 10^3)

Years	1	2	3	4	Total $
Shafts and Facilities	8,005	8,005	8,005	8,005	32,020

It is estimated that the sinking of the production and service/waste rock shafts will each take from 60 to 90 weeks. A labor force of 49 persons will be involved.

MINING OPERATIONS

Scheduling

Ore production from the mine is scheduled for 51 weeks out of the year and 20 shifts per week. This results in 8160 scheduled hours of operation per year or the equivalent of 340 full working days. For underground mines it is commonly assumed that 1.5 hours of an 8-hour shift are lost due to travel time to and from the work place, mismanagement of men and equipment, and breaks for the miners. In order to achieve the desired annual production rate of 7,938,000 mt (8,750,000 st), the mine must extract ore at an average rate of 1200 mt per hour (1320 st per hour).

Mine Design

A theoretical mine layout was assumed for the blasthole open stoping mine model in order to calculate drilling rates, powder usage, cycle times, and manpower requirements. Figure 4 provides a generalized view of one stope of a blasthole open stoping mine. A summary of the dimensions of the major mine openings follows.

VIEW OF A STOPE IN A BLASTHOLE OPEN STOPING MINE.

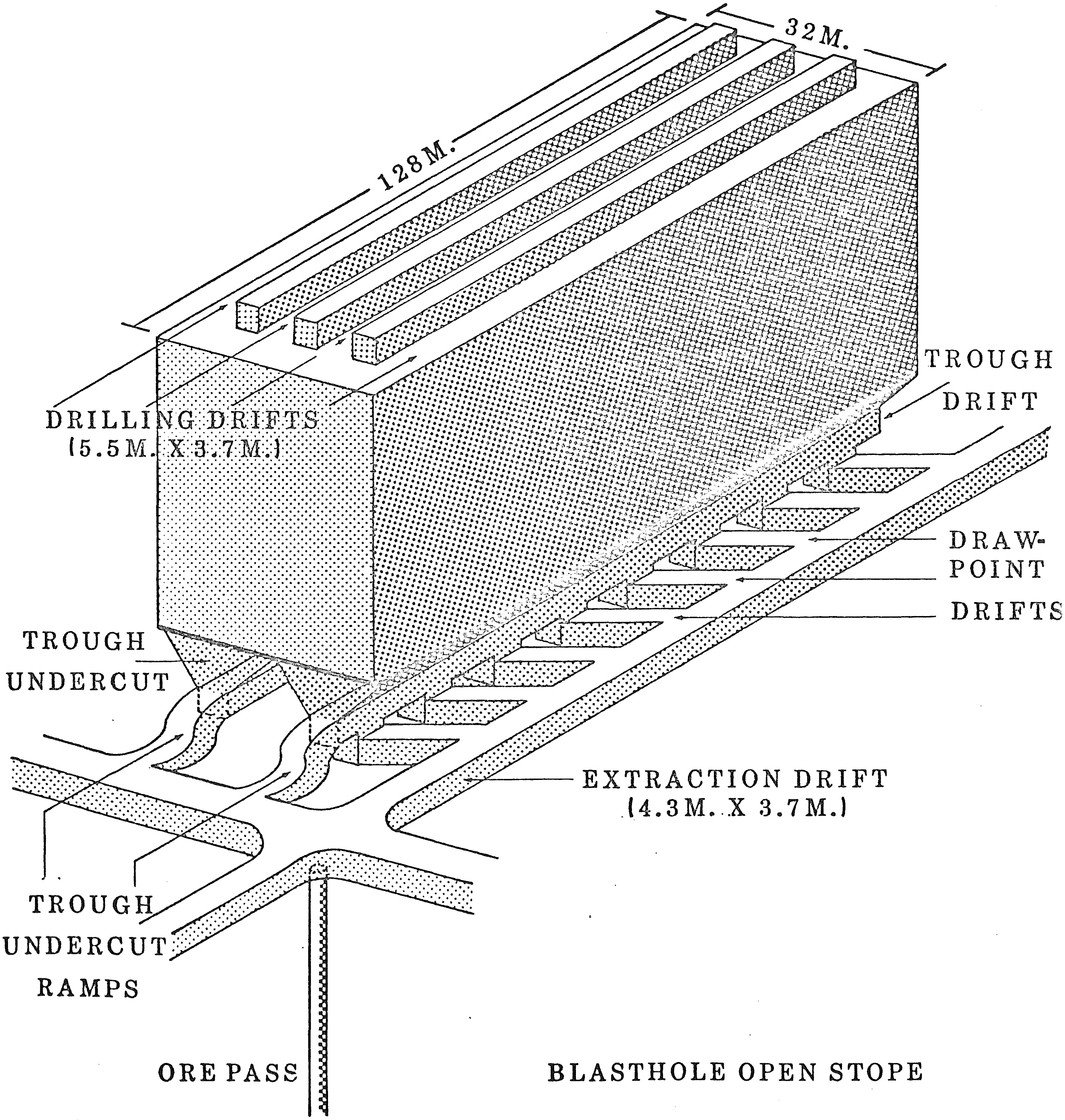


FIGURE 4.

DRAFT REPORT - The reader is cautioned concerning use, quotation or reproduction of this material without first contacting the authors, since the document may experience extensive revision during review.

Stope Size:	128 m long x 32 wide x 45 m high (average) 420 ft long x 105 ft wide x 148 ft high (average)	
Rib Pillars:	12 m wide 40 ft wide	
End Pillars:	12 m wide and 24 m wide, alternating 40 ft wide and 80 ft wide, alternating	
Drilling Level:	Access Crosscuts	3.7 m wide x 3.7 m high 12 ft wide x 12 ft high
	Drilling Drifts	5.5 m wide x 3.7 m high (8.8 m centers) 18 ft wide x 12 ft high (29 ft centers)
Extraction Level:	Trough Drifts	3.7 m wide x 3.7 m high 12 ft wide x 12 ft high
	Drawpoint Drifts	4.3 m wide x 3.7 m high (16 m centers) 14 ft wide x 12 ft high (52.5 ft centers)
	Extraction Drifts	4.3 m wide x 3.7 m high 14 ft wide x 12 ft high
	Access Crosscuts	4.3 m wide x 3.7 m high 14 ft wide x 12 ft high
Main Haulage Level:	Rail Drifts	3.7 m wide x 3.7 m high 12 ft wide x 12 ft high
Orepasses:	1.8 m diameter bored raises 6 ft diameter bored raises	

Several figures were derived from the assumed mine layout and were used for planning and design purposes. Some of these figures follow.

Tons of ore made available by the development and mining of 1 stope (including losses) = 630,700 mt (695,200 st). The number of stopes which must be mined out in a one-year period = $\frac{7{,}938{,}000 \text{ mt/year}}{630{,}700 \text{ mt/stope}}$ = 12.6 stopes/year

Development accounts for 10% of the ore made available.

Annual tonnage figures

Stope mining	7,144,000 mt	7,875,000 st
Development	794,000 mt	875,000 st
Total ore produced	7,938,000 mt/year	8,750,000 st/year

Waste rock (0.08 x tons of ore produced) = 635,000 mt/year (700,000 st/year).

Initial percent ore recovery = 63%. At the conclusion of mining, 38% of the pillars will be extracted. This results in an overall percent ore recovery of 0.63 + ((1.0 - 0.63) x 0.38) = 77%

Stope Development--The stopes are designed to be 45 m (148 ft) high and the distance between the drilling and extraction levels is about 55 m (180 ft). In order to develop the stope undercuts, two 3.7 x 3.7 m (12 x 12 ft) trough drifts are driven the full length of the stope and the troughs are opened up by drilling upward at a 45^{o} angle from the trough drift. This development work creates a funnel-like configuration at the base of the stope which directs broken ore from above into the trough drifts.

Drawpoint drifts which angle from the extraction drift (located in the rib pillar) to a point below the trough drift allow load-haul-dump (LHD) machines to load ore from the stope while remaining under the protective cover of the drawpoint drift. The drawpoints will be spaced at 16 m (52.5 ft) intervals along the extraction drift and there will be eight for each trough drift.

On the drilling level, access to the stopes is gained through crosscuts placed in the 24 m (80 ft) wide end pillars. From these crosscuts, four 5.5 x 3.7 m (18 x 12 ft) drilling drifts are driven parallel to each other for the full length of the stope. After completion of the drilling drifts, a slot raise extending from the extraction level to the drilling level is opened up for the full width of the stope. This open slot provides a free face for subsequent stope blasting.

Stope Mining--Mining of the stope can now begin by retreating toward the 24 m (80 ft) end pillar. One hundred and sixty five millimeter (6.5 in.) diameter blastholes will be drilled vertically in the stope by utilizing down-the-hole (DTH) drills. Since the percussion hammers on DTH drills are located behind the bit rather than up on the drilling rig, there is no diminution of penetration speed with depth and much longer holes can be drilled. The drills are air driven at 56,000 to 70,000 kilograms (kg) per square meter (80 to 100 pounds per square inch (psi)) and provide penetration rates between 3.0 and 4.6 m/hour (10 and 15 ft/hour) using flat-faced tungsten carbide button bits. In addition to the economies of using larger and longer drill holes, other advantages claimed for DTH drills are less drill site preparation, better fragmentation, cleaner holes, improved accuracy, the ability to work in broken ground, less dust in the working environment, and lower noise levels since the hammer is in the hole itself and not adjacent to the operator. The DTH drills are fairly compact (typically 1.4 m (4.5 ft) wide, 3.5 m (11.5 ft) long and 3.4 m (11 ft) high with the mast raised), so they can be transported throughout the mine with relative ease.

The planned spacing, burden, and depth of the blastholes are 4.4 m (14.5 ft), 4.0 m (13 ft), and 37 m (120 ft), respectively. The blastholes will be loaded with ammonium nitrate fuel oil (ANFO) or a water gel explosive, depending on the amount of water present, and will be initiated by detonating cord. The powder factor for stope blasting is expected to be about 0.33 kg of explosives per mt of rock broken (0.65 lb/st), whereas the average powder factor required for development work is expected to be about 0.50 kg/mt (1.0 lb/st) (the overall powder factor will be about 0.36 kg/mt (0.71 lb/st)).

Transportation--The blasted ore will be removed from the bottom of the stopes by LHD machines. The load-haul-dump concept has a distinct advantage over other methods of loading and hauling because only one piece of equipment and one operator are required to perform both operations, and the need to match or coordinate several different types of equipment is avoided. LHD units commonly possess four-wheel drive with hydraulic braking systems, full power-shift transmissions, a low profile, articulation for maneuverability, and good reverse speeds. Their use promotes safety, flexibility, high productivity, and low costs. To reduce ventilation problems, manufacturers are developing more effective exhaust equipment on the diesel units and are also offering some electrically powered LHDs.

Factors generally considered in the selection of LHD specifications include size of drifts, size and type of intersections, mining back heights, shaft and cage dimensions, and lengths and gradients of hauls (most mines have a top haulage grade somewhere between 10% and 20%). (E/MJ, June 1976, p. 161) LHD bucket capacities range from 0.8 to 11 m^3 (1 to 15 cubic yards (yd^3)). The trend at the present time is for a new mine to select the largest equipment which is compatible with the size of the mine openings. Six cubic meter (8 yd^3) units have been selected for the mine model, although 4 and 8 m^3 (5 and 11 yd^3) machines could be successfully applied at some locations and for some purposes.

The LHDs transport the ore from the stopes to 1.8 m (6 ft) diameter bored raises via the drawpoint and extraction drifts. The mean haulage distance will be 91 m (300 ft)(183 m round trip distance). The orepasses will be situated in the 24 m (80 ft) pillars that separate the ends of two stopes. Each orepass will handle an amount of ore equal to the production from two stopes. From the extraction level, the ore cascades down the 1.8 m diameter orepasses to the main haulage level, a drop of from 30 to 250 m (100 to 820 ft).

On the main haulage level the ore will be transferred to an electric train consisting of fifteen 8.5 m^3 (300 ft^3) cars pulled by a 23 mt (25 st) locomotive. Each train will have a capacity of 230 mt (250 st) of rock. The track loops will be 290 m (960 ft) apart. The longest round trip haulage distance should be about 6,100 m (20,000 ft). The trains will travel to the production shaft where the cars will unload while in motion incorporating the bottom-dumping OK car system.

Underground Crushing--Run of the mine ore will be crushed to -150 mm (-6 in) by a 1.37 by 1.88 m (54 by 74 in) gyratory crusher before falling into a storage pocket. The ore can then be loaded into ore skips and hoisted to the surface. The crusher will be designed to handle 1450 mt (1600 st) of ore per hour.

Ventilation--The primary function of mine ventilation is to dilute, render harmless, and carry away dangerous accumulations of gas and dust from the working environment. Because of deeper and larger mines, a higher degree of mechanization, and increasing concern for health and safety, the demands placed on ventilation systems are rising.

> Mine ventilating systems vary greatly and are dependent upon the size and shape of the ore body, system of mining and depth of operations. The most difficult systems to establish and maintain are in deep multilevel operations mining irregular ore bodies. The resistance to airflow of these systems requires the efficient maximum utilization of air volumes necessary to provide and maintain a safe and healthful underground atmosphere. This can be achieved only by the proper distribution and control of adequate air volumes.

There is no ideal or standard system of mine ventilation. The effectiveness and efficiency of the system will be determined by how well certain basic fundamentals are applied and maintained.

Two factors--adequate volumes and proper distribution--along with a working knowledge of the mining system, form the basis for efficient mine ventilation. Unfortunately, there is no simple method of determining adequate volumes. The quantity and physiological effects of all contaminants introduced into the system must be evaluated.

(SME Handbook, 1973, p. 16-51)

Volume requirements for a projected mine sometimes are based on the volume circulated through an active well-ventilated mine of similar tonnage. With this approach, a complete analysis and comparison of mining methods, mining equipment, contaminant-control measures and air distribution should be made between the proposed and active mines. A variance in any one of these factors could lead to under- or over-design of the proposed system unless appropriate adjustments are made. Complete reliance should not be placed on this approach. However, guidelines in certain areas of volume comparisons are very helpful in system design.

Other ventilation experts use a minimum velocity in various active mining areas from which design volume is calculated. This approach to design volumes has been covered in detail by Yourt (Design Volumes for Underground Ventilation, 1965). Some of his rule-of-thumb figures for velocity in feet per minute are:

40-50..........Driving headings, raises, shafts, haulageways, chutes
30-50..........Stopes
50-175.........Scrams
150-200........Enclosures, such as dump doors and crushers
50-200.........Loading pockets.

These figures, no doubt, are based on a source of uncontaminated air. Additional volume for dilution will be necessary where contaminated air is involved.

One area where definite volumes may be designated involves the use of diesel equipment. Regulating agencies usually specify conditions under which such equipment can be used. Approval plates issued by the U.S. Bureau of Mines specify the volume of air necessary for each piece of diesel equipment tested and approved for underground use.

Sampling and subsequent analysis is the one certain method of determining the quality of the working atmosphere from which the volumes necessary for proper quality control can be determined.

In the overall mine-ventilation system, proper distribution is as important as the total volume circulated. Regardless of the volume circulated, air is of little value if not directed in an uncontaminated state to areas where it is necessary to maintain a safe and healthful atmosphere.

The ideal distribution system would deliver the minimum volume of uncontaminated air necessary to maintain a healthful atmosphere directly to the working area, and from there into the return air courses. Admittedly, the ideal system is rarely achieved but with close cooperation between the production, engineering and ventilation officials it can be approached, resulting in increased efficiency, improved safety and better health conditons.

(SME Handbook 1973, p. 16-52)

Underground mines can be ventilated by the force (overpressure) or exhaust (underpressure) method. There is no simple rule to follow in making the selection. Management must weigh the factors involved and choose the method best suited for anticipated conditions. Considering identical ventilation circuts, each adapted for the method employed, the variation in efficiency between force and exhaust systems is negligible.

(SME Handbook 1973, p. 16-55)

Fine particles of dust are a byproduct of the various operations associated with the extraction of ores and the winning of metals. They constitute both a health hazard and a nuisance. The latter condition arises when the dust is dispersed in the form of clouds capable of reducing visibility, adversely affecting morale, and causing undue wear and premature failure of components in mechanical equipment and sensitive electronic systems. A direct result is an increase in mining costs due to increased accident frequency, undue delays while waiting for dust and fumes to clear, and excessive maintenance and repair costs.

> The health hazard involves the inhalation of fine dust particles and their retention in the alveoli of the lungs. The degree of the hazard will depend upon exposure time and nature of the dust, particularly its concentration and physico-chemical characteristics. Pneumoconiosis, a term used to describe all lung diseases caused by dust, is the result of overexposure, and an incidence rate of 1 per 100 underground workers annually is not uncommon in many mining districts.
>
> (SME Handbook 1973, p. 16-56)

Some of the common techniques for control of dusts (such as suppression, dilution, and removal) in an underground mine are reviewed in Appendix C. Dust will be defined as particles having a diameter of less than 10μ.

The ventilation scheme for the underground mine model is based on existing mining systems and the regulations stated in the U.S. Bureau of Mines Schedule 24 for the use of diesel powered equipment in non-gaseous mines. The ventilating system will be an overpressure system with the two main shafts used for intake air. The ventilation requirements for the mine are estimated to be 48,000 m^3/minute (1,700,000 ft^3/minute) at STP (24,000 m^3/minute through each shaft). The air will be heated when necessary by direct-fired propane gas burners. Fresh air will be split between levels depending on the requirements of the equipment and work areas active on each level. Exhaust air will be forced from the mine by secondary fans installed in bored raises which will be located throughout the mine and bored as needed .

Compressed Air--The major power supply underground still is compressed air, but electric-hydraulic systems are gaining in popularity for mucking and drilling equipment. Stationary and portable electric compressors are used throughout the industry, except during start-up stages, or in remote locations where power is unavailable and portable diesel powered units become economical to operate. The stationary compressors are nearly always housed in the hoisthouses. The main air lines will be placed in the shaft compartments. "For optimum operation of equipment, compressors should be connected to aftercoolers, receivers, and water traps to condition the air. The freezing of compressor exhaust ports and a foggy atmosphere retard any mining cycle, and the delays caused by such situations as frozen pumps merely compound the problems. To prevent freeze-ups, methyl glycol, commonly known as tanner gas, is widely used. Caution must be exercised to avoid excessive use of this antifreeze, however, since it flushes out the oils needed for machine lubrication." (Dravo Corporation, Analysis of..., p. 254.)

Pumping Facilities--Water in an underground mine must be controlled. A wet mine results in hazardous working conditions and decreased productivity. Some of the sources of water in an underground mine are:

1) Ground water influx
2) Water used for dust control and suppression
3) Water involved in drilling operations
4) Water used in underground service areas

The pumping system in a mine must be capable of removing any water which collects in the mine. Two key requirements of a well-designed pumping system are adequate sump capacity for settling solids, and an effective method of cleaning the sump. A common arrangement is to have two sumps in parallel with the pump house, which allows one to be cleaned while the other is in use. Water pumped out of the mine will be kept in a closed circuit system with the mill and tailings pond.

Ground Support--The Duluth Complex is a highly competent rock which may not necessitate any intense ground control measures. The most commonly used ground support techniques are expansion-shell and grouted-rebar rockbolts in combination with wire mesh and/or shotcrete. Primary ground support is installed during development (as soon as possible after a blast). Under some conditions, further support may be required for some locations at a later date. To protect permanent or vital mine openings, more extensive ground control measures may be incorporated. These include the use of lagging, concrete, and timber or steel sets. The use of hydraulic backfill as a means of providing ground support is discussed in Appendix D.

There is the possibility that a mine such as the underground mine model will experience problems with rock bursts. Rock bursts are that phenomena which occur when a volume of rock is strained beyond the elastic limit and the accompanying failure is of such a nature that accumulated energy is released instantaneously. They normally do not occur until a depth of 1500 to 3000 ft below the surface is reached. The conditions which influence rock bursts in mines are: 1) the area of the excavation; 2) the shortest roof span; 3) stress pattern and concentration; 4) types of rock involved; 5) directions of planes of weakness in the rock; and 6) the dip of the mineral deposit. Rock bursts can best be controlled by careful design of stopes and pillars.

Underground Maintenance Facilities--In shaft mines, where the equipment cannot easily be brought to the surface for routine maintenance, adequate shops must be installed underground.

> The establishment of a thorough, preventive maintenance program also is essential to increase unit availability and reduce equipment costs.

Complete maintenance facilities are required, including skilled labor, supplies, tools, and work areas. Most underground shops feature comprehensive service-maintenance facilities, including cranes, lubrication and washing units, and specialized tooling to perform all normal mine equipment maintenance, except for complete rebuilding of engines and drive train parts. Warehouses, which are replenished regularly, stock most of the parts subject to frequent failure and replacement to avoid the costly delays involved in unscheduled trips to the surface.

(Dravo Corporation, Analysis of..., p. 313)

MINING COSTS

The costs associated with the mine models will be reduced to an initial capital cost (costs incurred over the first four years of preproduction development), an additional capital cost, and an average operating cost expressed as dollars per ton of copper-nickel ore mined. All costs are from the first quarter of 1977. Initial capital costs include: 1) preproduction development expenditures; 2) the costs of the shafts and facilities; and 3) the cost of the mining equipment purchased during the first four years of the mine's life. Additional capital costs are those which arise from the purchase of equipment--either additional or replacement units--after the fourth year of development. All mining costs which are not subject to depreciation are reflected in the operating cost. This includes labor, supplies, and the cost of operating and repairing equipment.

Equipment Requirements

The equipment selected for use in an underground mine should be chosen with the following considerations in mind. The various units should be physically compatible with each other so that sizes and cycle times are coordinated. Standardization of equipment is important since it increases the operators' proficiency and productivity, simiplifies and expedites equipment maintenance, and reduces warehouse inventories and costs. Equipment that is highly specialized should be evaluated carefully since such units often cannot be fully utilized.

Table 8 is a listing of the initial and additional capital requirements for the blasthole open stoping mine model. The equipment listed under additional refers to units which must be added to those purchased initially to meet the requirements of the mine. Combining the additional equipment list with the replacement equipment needed (as per the depreciation schedule) results in the schedule of capital additions and replacements shown in Table 9. The distribution of the capital cost of the mining equipment required for the first four years of mine development does not appear in Table 9, but is summarized below.

$\$ \times 10^6$

Years	1	2	3	4	Total $
Initial Equipment	962	1,653	2,834	4,916	10,365

Table 8. Underground mine equipment requirements - initial and additional capital requirements.

Item	No. of Units Required: Initially	No. of Units Required: Additionally	Cost/Unit ($\$ \times 10^3$)	Initial Cost ($\$ \times 10^3$)	Additional Cost	Estimated Life-Years
Drills						
Diamond Drilling Machine	2	0	25	50	0	4
One Boom Jumbo-Blockholer	1	2	80	80	160	4
Two Boom Fan Jumbo	2	4	77	154	308	4
DTH Drill	0	10	80	0	800	4
Three Boom Jumbo	10	5	113	1,130	565	4
Raise Borer	1	2	625	625	1,250	8
Ground Control Equipment						
Shotcrete Machine	2	0	145	290	0	4
Roof Bolting Jumbo	3	2	80	240	160	4
Vehicles						
Flat Bed Utility Truck	1	1	37	37	37	8
Water Truck	0	2	40	0	80	8
Cable Reel Truck	0	2	46	0	92	8
Blasthole Loading Carrier	1	2	35	35	70	8
Fuel Truck	1	2	40	40	80	8
Pipe and Vent. Tube Carrier	1	2	40	40	80	8
Self-Contained Mobile Shop	2	1	40	80	40	8
Utility Vehicle	4	8	10	40	80	4
Scissor Lift Utility Vehicle	1	2	43	43	86	8
Underground Grader	1	2	70	70	140	8
Lube Truck	0	5	45	0	225	8
Personnel Transport Vehicle	0	6	40	0	240	8
LHD Utility Vehicle - 2yd^3	12	0	70	840	0	8
LHD Vehicle - 5yd^3	10	0	110	1,100	0	5
LHD Vehicle - 8yd^3	4	8	150	600	1,200	5
Miscellaneous Equipment						
Vent. Tubes (36 in.)	36,000ft	72,000ft	7.64/ft	275	550	
Secondary Ventilation Fans	9	18	7	63	126	10
Refuge Chamber	3	3	5	15	15	
Ore Pass Gates	10	40	34	340	1,360	
Rail System	12,000ft	68,000ft	50/ft	600	3,400	
Rail Haulage System	1	4	400	400	1,600	20
Rail Dump	1	0	80	80	0	
Primary Crusher and Equipment	1	0	1,150	1,150	0	
Booster Compressor	0	6	25	0	150	10
Sump Pumps	1	1	150	150	150	10
Power Line	15,000ft	30,000ft	22/ft	330	660	
Transformers and Controls				75	225	
Underground Repair Shop and Equipment	1	1	300	300	300	10
Underground Fuel Storage	1	1	75	75	75	
Surface Equipment						
50 Ton Rear Dump Truck	1	0	200	200	0	10
Bulldozer	1	0	140	0	0	8
Sub Total				$9,687	$14,304	
Freight @ 3%				291	429	
Tax @ 4%				387	572	
Grand Total				$10,365	$15,305	

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Table 9. Schedule of capital additions and replacements.

Year	Additional Equipment ($ x 10^3)	Replacements ($ x 10^3)	Total Mine ($ x 10^3)
5	3,561	595	4,156
6	1,714	200	1,914
7	1,438	564	2,002
8	1,743	1,430	3,173
9	649	2,846	3,495
10	675	1,324	1,999
11	162	1,802	1,964
12	428	2,212	2,640
13	565	4,308	4,873
14	410	1,956	2,366
15	806	853	1,659
16	486	2,415	2,901
17	96	2,938	3,034
18	364	2,470	2,834
19	96	2,199	2,295
20	364	2,007	2,371
21	524	3,579	4,103
22	449	880	1,329
23	96	2,942	3,038
24	364	2,948	3,312
25	24	2,110	2,134
26	291	2,164	2,455
Total	$15,305	$44,742	$60,047

Supplies

Table 10 is a listing of some of the supplies and materials that are required for the mine model.

Table 10. Supplies and materials requirements.

Description	Quantity
Potable Water	760 liters/minute 200 gallons/minute
Connected Power	21,000 kilowatts 28,000 horsepower
Energy	252,000 Kilowatt-hours/day
Propane Gas	6,800,000 liters/year 1,800,000 gallons/year
Diesel Fuel	14,000 liters/day 3,700 gallons/day
Hydraulic Fluid	1,700 liters/day 450 gallons/day
Rock Drill Fluid	850 liters/day 225 gallons/day
Explosives	8,950 kilograms/day 19,730 pounds/day

Manpower Requirements

The underground mine model requires a work force of approximately 1000 people--850 hourly and 150 salaried employees. Manpower will be distributed according to the following percentages--Development and Production Mining, 40%; Mine Maintenance, 35%; and Mine Services, 25%. The work force will gradually increase throughout the first eight years of development and then remain at a fairly constant level until the last years of the operation. The distribution of manpower is shown in Table 11.

Table 11. Manpower distribution for the underground mine model.

Hourly Employees

Years	1	2	3	4	5	6	7	8	9-27	28	29	30	31
Development	34	76	153	255	238	221	196	196	196	131	0	0	0
Production	--	--	--	--	17	34	59	102	144	144	144	96	0
Services	21	48	95	160	160	160	160	186	212	212	212	141	0
Maintenance	30	67	134	223	223	223	223	260	298	298	298	199	0
Subtotal	85	191	382	638	638	638	638	744	850	785	654	436	0

Salaried Employees

Years	1	2	3	4	5	6	7	8	9-27	28	29	30	31
	30	51	67	113	113	113	113	131	150	139	115	77	0
Total	115	242	449	751	751	751	751	875	1000	924	769	513	0

The productivity associated with the mine can be calculated to be:

$$\frac{7{,}938{,}000 \text{ mt/year}}{1{,}000 \text{ employees} \times 255 \text{ shifts/year}} = 31 \text{ mt/man-shift (34 st/man-shift).}$$

Table 12 outlines the derivation of the average wage rates for hourly and salaried personnel. The mine's annual payroll in 1977 dollars can be estimated for any year by the following process:

Payroll in 9th year

Hourly employees:	850 x $10.38/hr x 2080 hrs/year	
		= $18,350,000
Salaried employees:	150 x $23,400/year =	$3,510,000
Total payroll for year nine		$21,860,000

Table 12. Average wage rates for hourly and salaried personnel.

Hourly

To calculate the average hourly wage rate, use job class 12 as the average mine job. The base rate for job class 12 according to the February 1, 1977 United Steelworkers contract is $6.885 per hour.

Premiums:	Shift differential,	afternoon	$0.20/hour
		night shift	$0.30/hour
	Overtime	>40 hours	@1.5
	Sunday premium	<40 hours	@1.5
		>40 hours	@1.7

Fringe Benefits: Use 40% of base rate before premiums

Total Pay Rate:	Base	$6.885/hour
	Premiums	0.742/hour
	F.B. @40%	2.754/hour
	Total	$10.381/hour

Salaried

For salaried employees, assume an average monthly wage of $1,500 per month.

Average Annual Salary:	$18,000/year
Fringe Benefits: Use 30%	5,400/year
Total Annual Rate:	$23,400/year

Summary of Costs

Tables 13 and 14 present a summary of the costs associated with blasthole open stoping.

Table 13. Blasthole open stoping capital costs.

Initial Capital Cost ($ x 10^3)

Years	1	2	3	4	Total $
Preproduction Development	2,240	4,480	7,840	13,440	28,000
Shafts and Facilities	8,005	8,005	8,005	8,005	32,020
Initial Equipment	962	1,653	2,834	4,916	10,365
	11,207	14,138	18,679	26,361	70,385

Additional Capital Cost ($ x 10^3)

Additional Equipment	15,305	
Replacement Equipment	44,742	60,047

Total Mine Capital Cost (30-year life)	$130,432,000

Table 14. Blasthole open stoping operating costs in $/ton of Cu-Ni ore.

	$/mt	$/st	%
Development	$2.05	$1.86	35%
Drilling	.30	.27	5
Blasting	.12	.11	2
Haulage	.76	.69	13
Crushing and Hoisting	.35	.32	6
Power and Fuel	.30	.27	5
Maintenance (non-allocatable)	.58	.53	10
Supervision and Services	1.05	.95	18
General*	.35	.32	6
	$5.86/mt	$5.32/st	100%

Labor accounts for 47% of the total cost.

---

*The general cost category includes waste rock handling, general supply handling, clean-up work, and the operation of fuel, air, water, and power lines.

ROOM-AND-PILLAR MINING

Room-and-pillar mining is a method in which multiple stopes, or rooms, are mined while the roof and walls are supported by pillars of rock. As the most common underground mining method in the United States, noncoal room-and-pillar mining accounts for over 75 percent of all the mines producing over 1100 mt (1200 st) per day. Figure 5 illustrates a room-and-pillar mining method.

Ore bodies suited to room-and-pillar mining are regular stratiform types that:

1) Have not been greatly folded or deformed.
2) Are strong, or moderately strong.
3) Have moderate to strong backs and floors, since the main roof should span the desired width.
4) Are relatively flat lying.
5) Contain ore that is relatively uniform in both thickness and grade.
6) Are of considerable extent in area.

Massive or dome-type deposits (such as salt) are commonly mined by this method on single or multi-levels. Dipping beds up to 30^{o} and 91 m (300 ft) thick have been mined successfully by driving the rooms horizontally either in the direction of the dip or of the strike, with the pillars on one level superimposed on one another, and with floor pillars between levels.

Rooms from 1.5 to 30 m (5 to 100 ft) high have been opened. The span that will stand unsupported depends primarily on the type, characteristics, and properties of the roof rock. Weaker roofs are usually supported by rockbolts.

The maximum practical depth for room-and-pillar working depends on the strength of the pillars. The deepest room-and-pillar mines in North America are about 980 m (3200 ft) below the surface.

Extraction rates vary from 35 percent at depths below 910 m (3000 ft) to over 90 percent at shallow depths (if the pillars are recovered).

Major Advantages

1) Highly flexible system; easily modified throughout a mine's life to suit the conditions and equipment employed, and to take advantage of new technological developments.

ROOM-AND-PILLAR FULL FACE AND MULTI BENCH MINE.

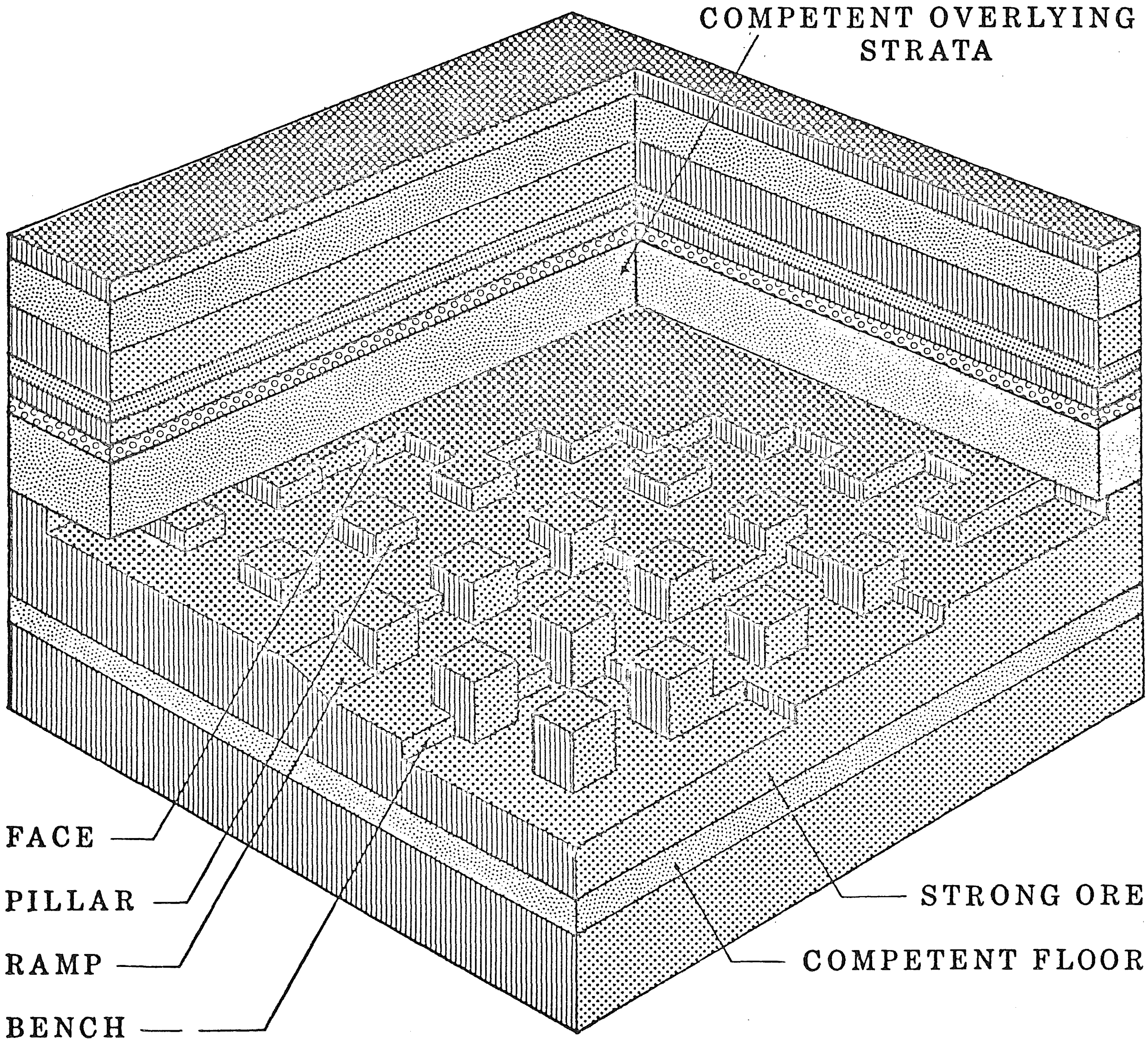

FIGURE 5.

2) High selectivity; new entries can be easily started or stopped without serious effect on either the method or sequence planning.

3) Development of rooms is also a production operation, since most of it takes place in ore.

4) Amenable to a high degree of mechanization, using high-capacity equipment that is currently available.

5) Highly productive method as a result of the large number of working faces provided and the use of high-capacity equipment.

6) Adequate ventilation, which is important, is relatively easy to achieve, even after a mine becomes large and complex as its operations expand.

Major Disadvantages

1) Roof support and maintenance of openings can become costly.

2) Low ore extraction rates because pillars must be left to support the openings and prevent subsidence. In some instances, expecially in the higher grade ores, pillars are recovered and the ground is allowed to cave.

3) Method does not allow for easy underground disposal of mill tailings. In metal mines, all tailings presently are disposed of by conventional methods (tailings ponds and dams). One European room-and-pillar mine is known to be disposing of its mill tailings underground.

Source: Dravo Coproration, Analysis of...p.271.

Development And Mining

In order for room-and-pillar mining to be considered as an alternative to blasthole open stoping, it is required that the ore body be extensive and fairly uniform in both thickness and grade. It will be assumed that the thickness of mineralization is less than or equal to 25 m (82 ft) and that the dip seldom exceeds 15^{o} and never exceeds 20^{o}. (This assumption replaces assumption #7, p. 20. The other assumptions listed under mine planning, pp. 19 and 20, will remain unchanged.)

Entrance to the room-and-pillar mine model will be gained through vertical shafts. The location of the shafts are important since haulage is the highest mining cost associated with room-and-pillar mines.

The room-and-pillar method of mining involves advancing the mining face, or rooms, in cycles that involve the following unit operations:

1) Scaling and local ground support, as needed
2) Drilling
3) Blasting
4) Loading
5) Hauling

With room-and-pillar mining there is very little development work required since the mining faces develop symmetrically around the primary shaft site and advance outward with time. Initially, a room is opened up when a top heading is drilled, blasted, and mucked. The bench created by this first pass can then be mined (see Figure 6). As time goes on, mined out rooms gradually evolve into part of the network of haulage routes.

Production rates and percent extraction with both room-and-pillar mining and blasthole open stoping are similar. The equipment requirements are also quite similar (the equipment mix will vary somewhat).

Mining Costs

With room-and-pillar mining, the early life of the mine is characterized by three years of preproduction development and three years of gradually increasing production (see Figure 7). The initial capital cost for room-and-pillar mining is somewhat less than for blasthole open stoping, but

SECTION THROUGH ROOM OF TYPICAL FULL FACE AND BENCH MINING OPERATION.

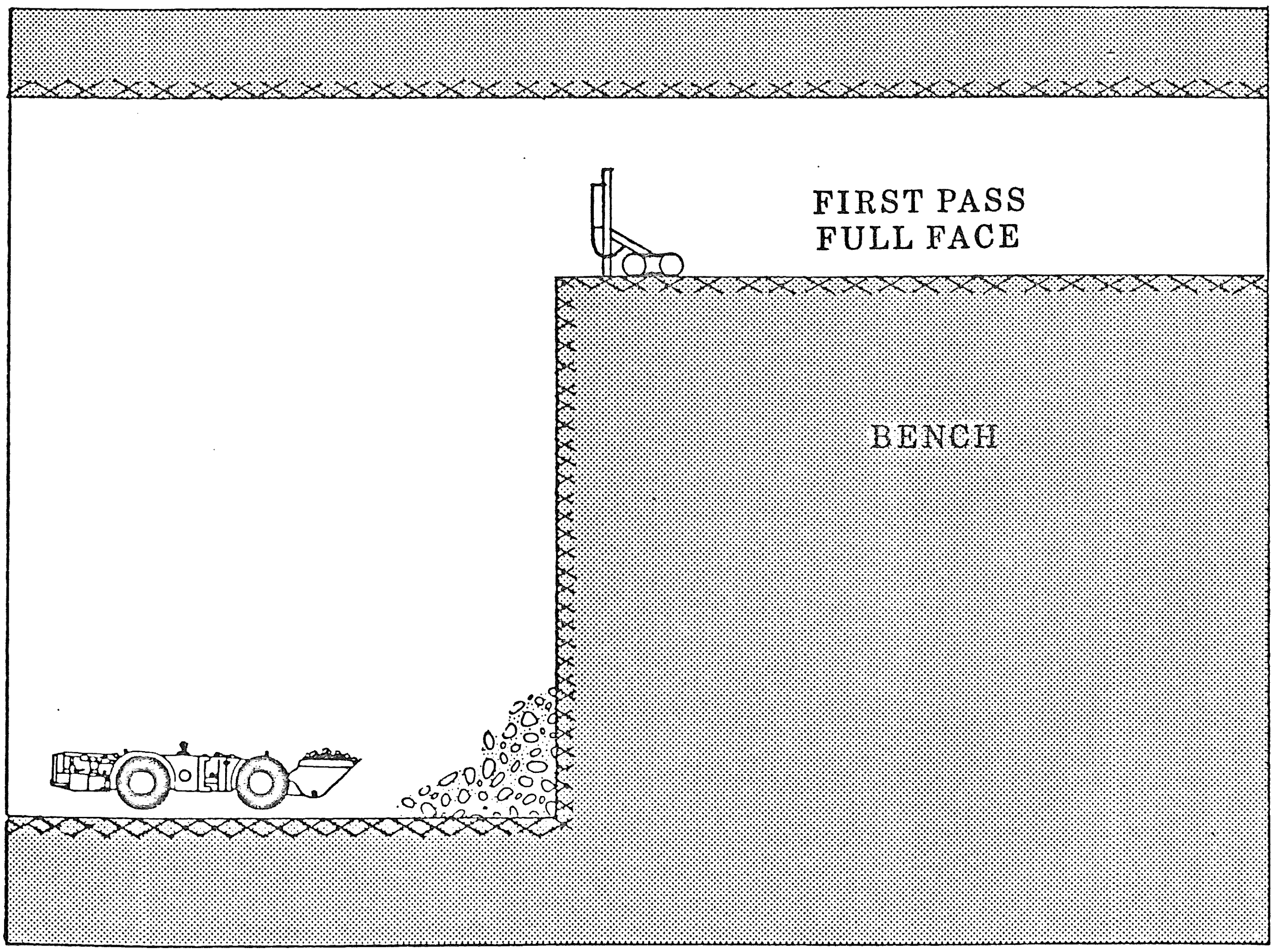

FIGURE 6.

LIFE CYCLE OF THE ROOM-AND-PILLAR MINE

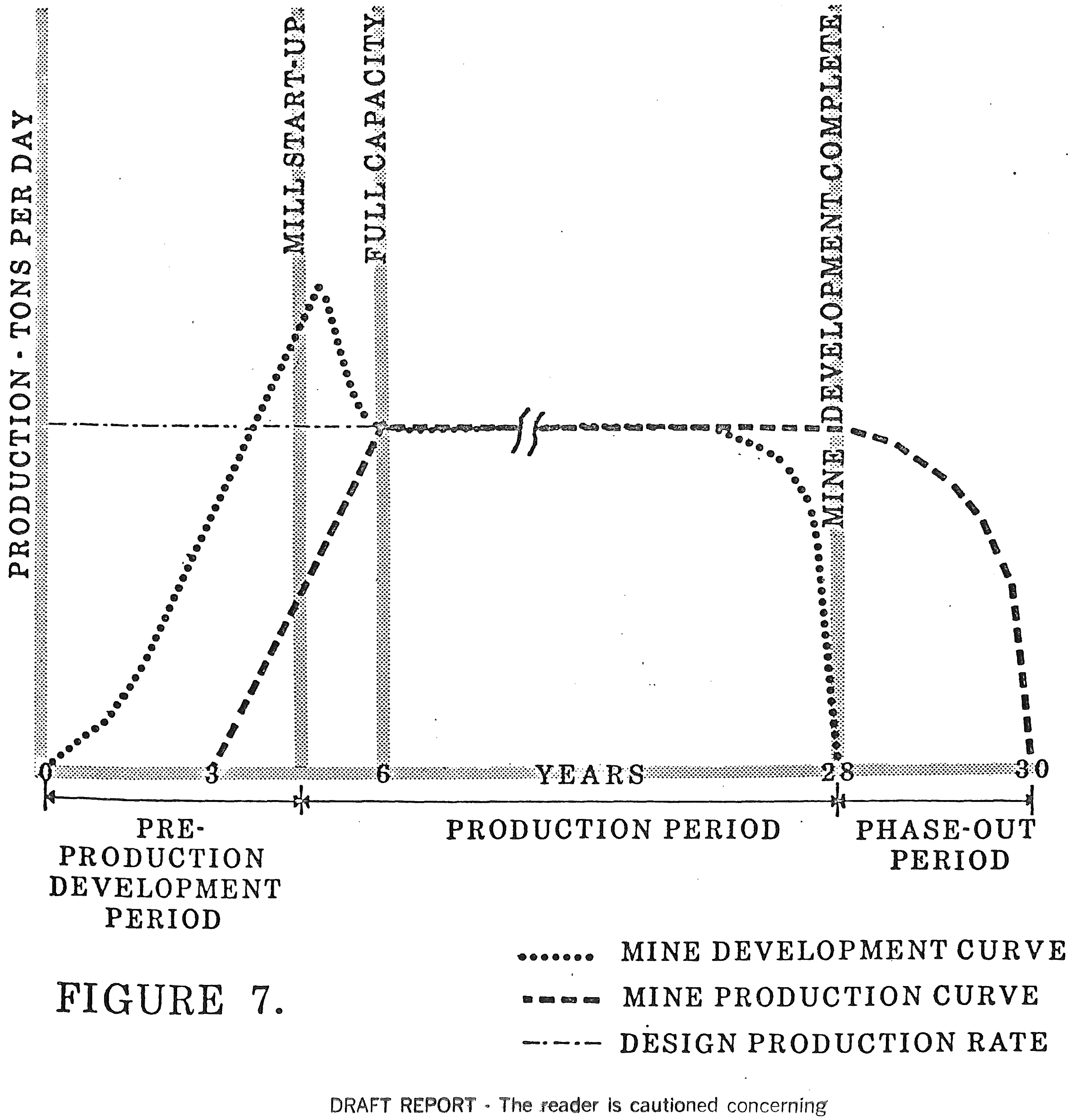

FIGURE 7.

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this is offset by a higher operating cost. Costs for room-and-pillar mining are summarized in Tables 15 and 16.

Table 15. Room-and-pillar capital costs.

	Initial Capital Cost ($ x 10^3)			
Years	1	2	3	Total $
Preproduction Development	2,250	4,500	8,250	15,000
Shaft and Facilities	10,670	10,670	10,680	32,020
Initial Equipment	1,200	2,400	4,400	8,000
	14,120	17,570	23,330	55,020
	Additional Capital Cost ($ x 10^3)			
Additional Equipment	15,746			
Replacement Equipment	41,389			57,135
Total Mine Capital Cost (30-year life)				$112,155,000

Table 16. Room-and-pillar operating costs in $/ton of Cu-Ni ore.

	$/mt	$/st	%
Development	$0.52	$0.47	8%
Ground Control	.26	.24	4
Drilling	.85	.77	13
Blasting	.33	.30	5
Haulage	1.82	1.65	28
Crushing and Hoisting	.33	.30	5
Power and Fuel	.33	.30	5
Maintenance (non-allocatable)	.58	.53	9
Supervision and Services	1.10	1.00	17
General*	.39	.35	6
	$6.51/mt	$5.91/st	100%

Labor accounts for 42% of the total cost.

*The general cost category includes waste rock handling, general supply handling, clean-up work, and the operation of fuel, air, water, and power lines.

SUMMARY

Table 17 summarizes the two mining methods--blasthole open stoping and room-and-pillar mining--as they have been explained and developed in this report.

Figure 8 illustrates how capital cost and operating cost vary with the size of the mine. The upper graph shows the variation of mine capital cost with the total amount of ore removed from the mine. The range of tonnages considered is 80-300 x 10^6 metric tons. The lower graph shows how operating cost varies with annual capacity over a range of 4-12 x 10^6 metric tons per year.

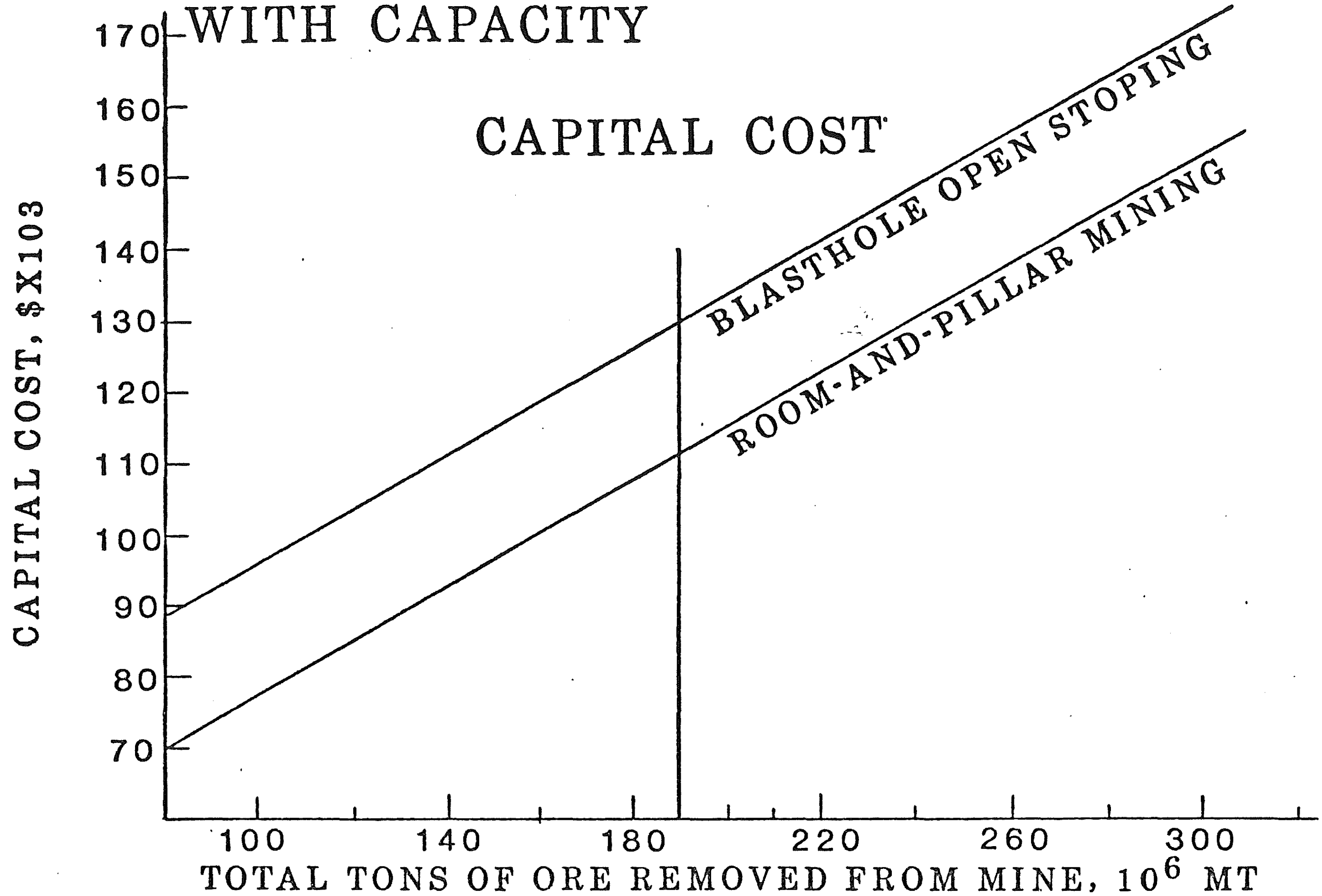
It may also be helpful to re-examine the physical applicability of both of the mining methods as they have been used here, keeping in mind that exceptions are possible.

Ore Thickness	Dip Angle 0-20^{o}	20-45^{o}
< ~ 25 m	Room-and-Pillar	(modifications?)
> ~ 25 m	Blasthole Open Stoping	Blasthole Open Stoping

Table 17. Comparison of blasthole open stoping and room-and-pillar mining models.

Mining Method	Blasthole Open Stoping	Room-and-Pillar Mining
Annual Production	7,938,000 mt	7,938,000 mt
Average Ore Grade	0.80% Cu; 0.20% Ni	0.80% Cu; 0.20% Ni
Waste Rock Produced Annually	635,000 mt	635,000 mt
Mine Life	30 years	30 years
No. of Years Before Production Begins	4 years	3 years
No. of Years To Reach Full Production	8 years	6 years
Recovery of Ore (Percent Extraction)	63% initially 77% ultimately	65% initially 75% ultimately
Electrical Energy Consumption	92×10^6 KWH/year	92×10^6 KWH/year
Propane Gas Consumption	6.8×10^6 liters/year	6.8×10^6 liters/year
Diesel Fuel Consumption	5.1×10^6 liters/year	5.1×10^6 liters/year
Manpower Requirements	1000 employees	1000 employees
Productivity	31 mt/man-shift	31 mt/man-shift
Capital Cost	$130,400,000	$112,200,000
Operating Cost	$5.86/mt	$6.51/mt

VARIATION OF MINING COSTS WITH CAPACITY



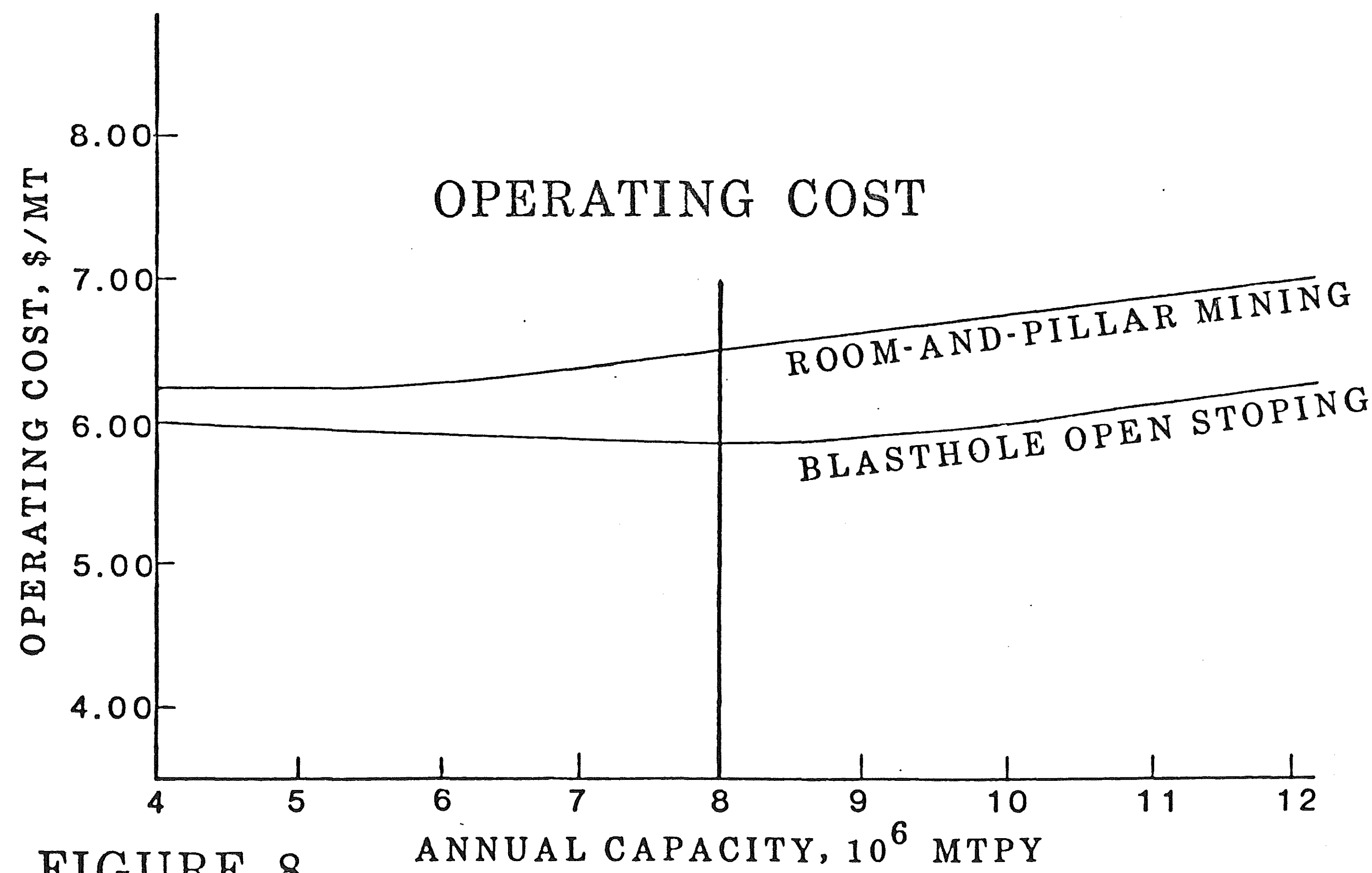


FIGURE 8.

APPENDIX A

Glossary of Mining Terminology

Back - The roof or upper part in any underground mining cavity.

Backfill - Waste sand or rock used to support the roof after the removal of ore from a stope.

Cage - The structure used in a mine shaft for the conveyance of men and materials.

Crosscut - A horizontal underground opening driven across the course of a vein or, in general, across the direction of the main workings.

Dilution - The contamination of ore with barren or low grade rock which is unavoidably removed along with the ore in the mining process.

Drift - A horizontal underground opening in or near an ore body and parallel to the course of the vein or the long dimension of the ore body.

Footwall - The wall or rock on the underside of a vein or ore structure.

Gangue - The undesired minerals associated with the valuable minerals in an ore deposit.

Hanging Wall - The wall or rock on the upper or topside of a vein or ore deposit.

Lagging - Short lengths of timber, sheet steel, or concrete slabs which are wedged behind timber or steel supports to help contain the roof and sides of an opening.

Lean Ore - Rock which contains some valuable minerals but not in sufficient quantities to be processed and marketed at the present time. These materials often become economic and can be utilized in the later years of a mining operation.

Level - Collectively the horizontal, or nearly horizontal, underground passageways or headings at the same approximate elevation; commonly interconnected.

Muck - Rock or ore broken in the process of mining.

Ore - An economic term referring to the portion of a mineral-bearing resource from which a mineral or metal can be extracted, treated, and marketed at a profit. (See Reserve)

Orepass - A vertical or inclined underground passage used for the transfer of ore to a lower level.

Overburden - Unconsolidated surface material, such as soil, sand, and gravel, that generally overlies the bedrock.

Raise - A vertical or inclined underground opening driven upward from a level to connect with another higher level.

Reserve - That portion of an identified resource from which a usable mineral can be economically and legally extracted at the time of determination. (See Ore)

Resource - A concentration of naturally occurring solid, liquid or gaseous materials in or on the earth's crust in such form that economic extraction of a commodity is currently or <u>potentially</u> feasible.

Scaling - The removal of loose rocks from the roof or walls of underground openings.

Shaft - A vertical or inclined underground excavation of small cross-sectional area compared to its depth which provides access to the workings of a mine.

Skip - The structure used in a mine shaft for the conveyance of rock and ore.

Stope - Any excavation in an underground mine, other than development workings, from which ore is being or has been extracted.

Sump - Any excavation in a mine used for collecting or storing water, from which water is pumped to the surface or to another sump nearer the surface.

Waste Rock - Barren or submarginal rock which has been mined but is too low in grade to be of economic value.

APPENDIX B

Underground Mining Methods.
(Source: SME Mining Engineering Handbook, 1973)

9

Selecting a Mining Method—Rock Mechanics, Other Factors

R. G. K. Morrison and Paul L. Russell

9.3—UNDERGROUND MINING METHODS

If the depth of an ore deposit is such that removal of the overburden makes surface mining unprofitable, underground methods should be considered. The problem of recovering the mineral from such a deposit is reduced to selecting or developing a mining system that will exclude other options on a safety and profit basis and at the same time provide adequate ground support to protect working areas and, in some instances, to preserve the surface. Because ground support is a necessary element in this process, the mining systems listed in Table 9-1 and described in the following subsections are classified on this basis. Also, because ground support is so dependent on the spatial characteristics of the deposit and mechanical properties of the mineral and surrounding rock materials, each description includes a specification of the deposit characteristics in which the method is applicable. Three broad classes of mining methods are recognized as follows:

1. Methods in which the underground openings (rooms or stopes) created by the extraction of the mineral are self-supported in that no regular artificial method of support is employed: that is, openings in which the loads due to the weight of the overburden or tectonic forces are carried on the sidewalls and/or pillars of unexcavated mineral or rock. This specification does not preclude the use of rockbolts or other light systems of support, provided that this artificial support does not significantly affect the load carried on the self-supported structure. The design of this class of systems for underground openings can be treated by the methods described in Sec. 7.2.

TABLE 9.1—Underground Mining Methods

I. Self-Supporting Openings:
- A. Open-stope mining:
  - 1. Isolated openings
  - 2. Pillared open stopes
    - a. Open stoping with random pillars
    - b. Open stoping with regular pillars
- B. Room-and-pillar mining
- C. Sublevel stoping
- D. Shrinkage stoping
- E. Stull stoping

II. Supported Openings:
- A. Cut-and-fill stoping
- B. Square-set-and-fill stoping
- C. Longwall mining
- D. Shortwall mining
- E. Top slicing

III. Caving Methods:
- A. Sublevel caving
- B. Block and panel caving

2. Methods in which stopes or rooms require significant support, that is, support to the degree that a part of the superincumbent load is carried on the support system.

3. Methods in which, because of the spatial and mechanical properties, the deposit is induced to cave under the action of gravity to produce better results than more selective methods.

This classification of underground mining methods is essentially the same as that given by Jackson and Gardner.[7] Similar classifications are given by Morrison[8] and Lewis and Clark.[9] Because pillar removal and pillar robbing are secondary mining operations, they are not considered in this section.

Often there is no precise division between requirements for these classes of mining methods. In fact, both open stoping and block caving have been employed in the same mineral deposit. Such a situation might result from a change in properties of the rock materials within or surrounding the deposit, or a change in spatial characteristics of the deposit, such as a thinning out, or the technological or economic changes that make one method preferable to another. However, the choice of a mining method usually is dictated more by the spatial or mechanical characteristics of the deposit than by other factors, and sometimes uniquely.

The principal environmental problems created by underground mining are the discharge of acid mine water into streams, and surface scars produced by subsidence, both concurrent and subsequent to mining. In the United States, the discharge of acid mine water is a problem generally associated with coal mining in the eastern states. Surface subsidence is common in the mining of massive low-grade deposits by caving methods. However, many underground mines, especially those in bedded and large lenticular deposits, and from which a high extraction has been obtained, ultimately subside, often to the degree that utilization of the overlying surface is placed in permanent jeopardy. Coal and evaporite mineral mines are typical examples. Generally, an underground mine does not involve broad environmental problems other than subsidence, although it may present one or more occupational problems.

9.3.1—SELF-SUPPORTED OPENINGS

Self-supported openings—that is, openings in which the superincumbent loads are carried on the sidewalls or pillars of unexcavated rock or mineral—can be mined in any type of mineral deposit except placer. However, the size of opening that can be excavated will depend on the type of rock materials that comprise the sidewalls and pillars. For example, the span (minimum wall-to-wall, wall-to-pillar or pillar-to-pillar dimension) that will stand unsupported may range from virtually zero for closely jointed or thinly laminated rock materials in which there is no cohesive strength across joints or partings, to more than 100 ft for massive bodies of rock. Thus, in general, the size of a self-supported opening will depend on the spacing and strength across mechanical defects in the rock material and on the depth and orientation of the opening.

Mining methods employing self-supported openings can be divided into two broad classes—open-stope mining and room-and-pillar mining. The design of openings of this class are treated in Sec. 7.2.7 through 7.2.12.

Open-Stope Mining—Strictly defined, an open stope is an underground opening from which a valuable mineral has been removed and in which no timber or other material was used to support the walls or roof. More common usage includes as open stopes those in which walls and/or roof may be supported by pillars of ore or waste, or by stulls, roof bolts or other means. We subscribe to the latter definition and, from a structural standpoint, divide open stopes into two classes: isolated (single) openings and pillared stopes (multiple openings) [Sec. 7.2 and Ref. 7].

Isolated Openings—An isolated opening is an unpillared and otherwise unsupported underground opening that is essentially outside the zone of influence (Sec. 7.2.7) of other underground openings. Isolated pockets, lenses and shoots of ore have been mined in this manner. Also, shafts, development drifts and excavations for civil work projects (tunnels, underground chambers for power stations) may be included in this category. The design in this type of opening is considered in Secs. 7.2.7 and 7.2.10.

In general, isolated openings can be mined in any rock type where the physical characteristics permit. The maximum span that can be mined as an isolated opening will depend upon the depth of the deposit and upon the geologic and physical parameters of the rock surrounding the ore. For example, isolated openings with spans up to 100 ft and at a depth of 300 ft have been mined in chert breccias; spans from 50 to 75 ft in fractured jaspilite at depths up to 1,000 ft; and spans from 50 to 60 ft in relatively unfractured dolomitic limestone at depths up to 300 ft. Fig. 9-7 illustrates an isolated opening.

Pillared Open Stopes—Generally, a mineral deposit of considerable areal extent, such as a narrow- or wide-vein deposit or a large pocket or lens of ore, cannot be mined as a single unsupported open stope (opening). To maintain stability, support is required within the limits of the deposit, and if this support is effected by leaving areas of unexcavated ore or waste, the system of mining is referred to as pillared open stoping.

From a structural standpoint such a system of stopes corresponds to a system of multiple openings (Sec. 7.2.8): that is, openings sufficiently close to one another so that the stress distribution around one opening affects the stress distribution around an adjacent opening, and vice versa.

Pillared open stope mining is effected with both random and regular pillar systems (Table 7-2, Sec. 7).

Open Stoping with Random Pillars—Open stoping with pillars that are randomly spaced and/or random in size is illustrated in Fig. 9-8, and the design of this type of opening is treated in Sec. 7.2.11. This method is used in mining the following:

1. Larger pockets or lenses of ore, especially if the ore grade and/or thickness of the deposit is variable. Whenever possible, the pillars are left in leaner ore. If ore is high-grade, pillars usually are extracted in final mining.

2. Bedded and narrow or relatively wide vein deposits that dip at any angle

(usually less than 45°), that is, at an angle such that the broken ore will not flow under the action of gravity. Also, this method is more likely to be employed when the ore grade or ore thickness is variable.

In randomly pillared openings, the span will depend on the quality of the roof rock. If it is massive, openings with spans up to 100 ft have been mined as, for example, in chert breccia, dome salt and thick-bedded limestone. In partially recemented jointed and fractured rock, spans from 50 to 100 ft are not uncommon. In bedded rock in which the roof has formed a parting, the roof span will depend on the thickness of immediate overlying beds. In bed thicknesses of 2 ft or more, openings with spans from 40 to 80 ft have been mined.

In a pocket or lens of ore of limited dimensions, the obtainable areal extraction will depend on the depth of the deposit and on the mechanical properties of both the ore and/or rock material that form the pillars, and the surrounding rock material that forms the roof, floor and sidewalls of the deposit. Generally, at depths less than 2,000 ft, areal extraction ranging from 60 to 80% is obtained. The zinc mines in eastern Tennessee and the lead-zinc mines in the Missouri-

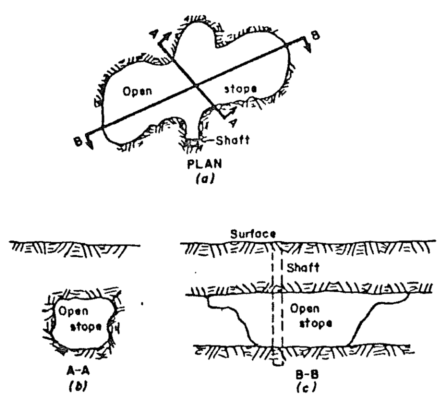


Fig. 9-7—Isolated opening without pillars (Jackson and Gardner[7]).

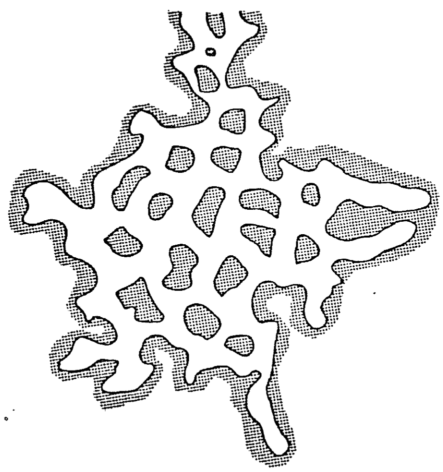


Fig. 9-8—Open stope with random pillars.

Kansas-Oklahoma (Tri-State) District employ pillared open stopes. The average extraction in these mines is 75%.

Open Stoping with Regular Pillars—Generally, in bedded, and sometimes in vein deposits of considerable areal extent in which the ore grade and ore thickness are relatively uniform, regular pillar systems are employed: that is, systems in which the cross-sectional shape and size of pillars and the spacing between pillars is uniform. A typical open-stope mine with regular pillars is shown in Fig. 9-9, and the design of this type of mine is considered in Sec. 7.2.11. Also, in very thick massive deposits in which multilevel mining is employed, regular pillar systems have been used in which the pillars on one level superimpose those of the next lower level. Mining in salt domes follows this procedure as, for example, in the Jefferson Island mine in Louisiana.

In regular-pillared open stopes the span that will stand unsupported depends primarily on the type of roof rock, ranging from 10 to 12 ft for thin-bedded shales to 150 ft in dome salt. Areal and volume extraction obtained with this type of mining depends on the depth of the deposit and on the mechanical properties of the rock or ore materials that form the pillars. Generally, at depths less than 2,000 ft extractions ranging from 60 to 80% are obtained.

Room-and-Pillar Mining—The design of room-and-pillar layouts is essentially the same as that for open stoping with regular pillars (Sec. 7.2.11), except that the former is limited to relatively flat-lying deposits in which the mineral is of comparatively uniform grade and thickness as, for example, coal or evaporite minerals (bed salt, potash, trona and borax). Also, some underground stone quarries employ room-and-pillar systems. This method, illustrated in Fig. 9-10, is effected by mining a grid of rooms separated by pillars of uniform cross section. Many grid layouts have been employed, including systems with rib pillars, and square pillars with checkerboard spacing.

The room widths, which usually are made as large as safety will permit, are limited by the characteristics and properties of the immediate roof rock. The cross-sectional size of the pillars, plus the room widths, determine the extraction,

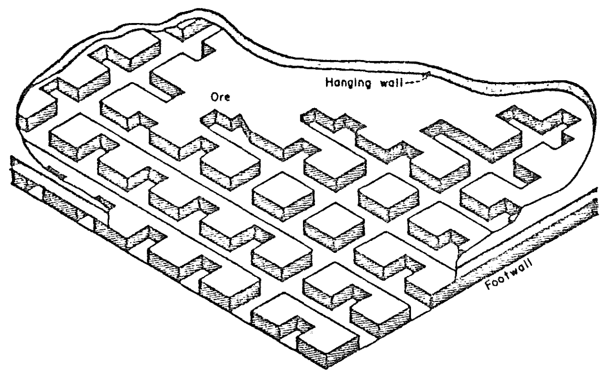


Fig. 9-9—Open stope with regular pillars (Morrison[8]).

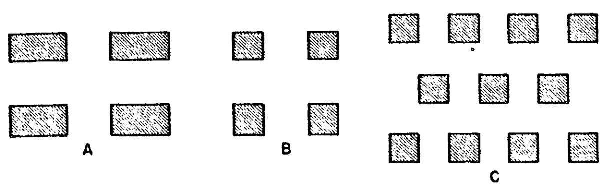


Fig. 9-10—Examples of pillar patterns: (a) rectangular pillars, regularly spaced; (b) square pillars, regularly spaced (checkerboard); (c) staggered checkerboard.

and will depend on the depth of the deposit and the strength and other mechanical properties of the rock material that forms the pillars.

In coal mining, room widths range from 14 to 50 ft, with 30 ft as an average. Extractions vary from 50 to 70%, with 60% as an average. In evaporite mines, the range of room width is comparable, but extraction ratios generally are higher, ranging from 60 to 90%.

As extreme examples of room-and-pillar mining, in the Saskatchewan Canadian Potash District a 7.5-ft bed lying at a depth of 3,100 ft is mined some distance below an aquifer with a room width of 21 ft and an extraction of 30%.[10] In an experimental oil-shale mine, a 54-ft section of oil shale lying under 900 ft of cover was mined with a staggered checkerboard system. The room width was 60 ft and the pillar cross sections 60 $\times$ 60 ft, giving an extraction of 75%.[11]

Sublevel Stoping—Sublevel stoping generally is employed in steeply dipping narrow- and wide-vein and bedded deposits, although this method has been used successfully in relatively flat-lying thick deposits. The deposit thickness may be variable, but the ore grade should be fairly uniform since this method does not lend itself to sorting. The rock material in the hanging and footwall and the ore should be relatively competent—that is, of a type mechanically equivalent to that in which other open-stope methods are employed.

Two basic stope configurations are utilized in sublevel stoping—longitudinal and transverse—and are illustrated in Figs. 9-11 and 9-12. In both stope configurations the ore is mined from sublevels by benching or ring drilling, and the ore should flow by gravity to the drawpoints.

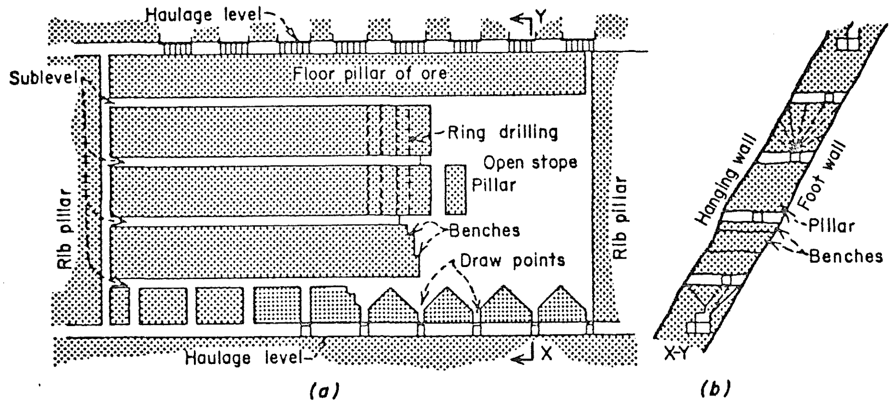


Fig. 9-11—Sublevel stoping with longitudinal stopes in narrow veins (Jackson and Gardner[7]).

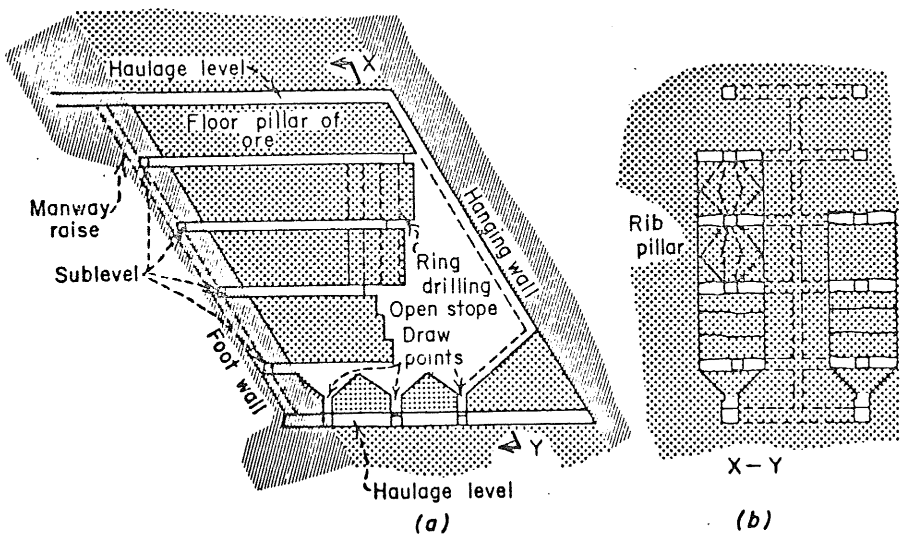


Fig. 9-12—Sublevel stoping with transverse stopes in wide veins (Jackson and Gardner[7]).

Longitudinal stopes are developed in comparatively narrow steeply dipping deposits. The stopes run parallel to the strike of the deposit and are of indefinite length. The width of the stope (or span) is limited by the thickness of the deposit. Either a random or a regular system of rib pillars may be left in stoped areas. Stopes up to 70 ft wide have been mined in this manner. Floor pillars normally are regular, forming the top of the worked-out stope and the bottom for the main haulageways or levels. If the deposit is steeply dipping, these floor pillars support lateral loads. This stope design permits an extraction up to approximately 75%. Some dilution from wall rock may occur, and some sloughing of the floor pillar is permitted, but more general caving must be avoided.

For wide deposits (greater than 70 ft) transverse stoping usually is employed. In this method, the stopes which run perpendicular to the strike of the deposit are limited in length to the thickness of the deposit. These stopes usually are outlined by a regular system of rib pillars, with floor pillars being the same as in longitudinal stopes. The spacing of the rib pillars is determined by the ability of the ore to form an unsupported span, but rarely is greater than 70 ft. If the deposit dips steeply, the floor pillars provide lateral support. The extraction with transverse stopes generally is less than with longitudinal stopes because a greater percentage of the ore is left in the form of rib pillars, but less dilution from the sidewalls is experienced.

Stopes have been mined with spans from 40 to 50 ft in massive sulfides with hanging and footwalls of rhyolite, brecciated tuffs and aggregates or conglomerates;

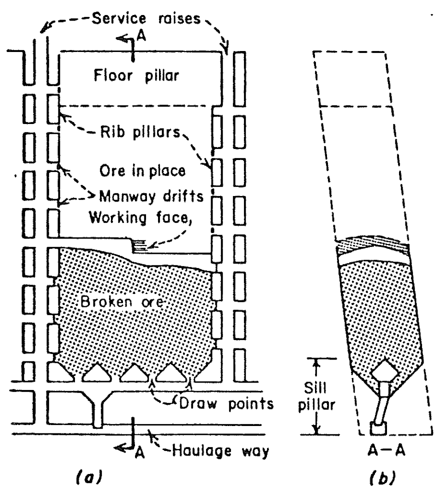


Fig. 9-13—Shrinkage stoping (Morrison[8]).

from 45 to 90 ft in massive sulfide ore with greenstone hanging and footwall; and up to 100 ft in iron sulfides with hanging and footwalls of a highly metamorphosed schist, graywacke and slate.

Shrinkage Stoping—Shrinkage stoping (Fig. 9-13) is used to mine narrow or wide veins, and sometimes bedded deposits that are steeply dipping. This mining method basically is an overhand stoping system in which a portion of the broken ore accumulates until the stope is completed. The increase in bulk as the ore is broken requires that some 30 to 50% must be "shrunk" periodically through chutes or drawpoints to maintain a working floor for additional mining. In general, the vein material must be strong enough to stand unsupported across the width of the stope and, when broken as ore, should not pack to the degree that it cannot be drawn. In vertical to near-vertical deposits, both the hanging and footwall rock should be relatively competent to prevent failure and excessive dilution of ore.

During the period the stope is being mined, both the hanging and footwall rock are stabilized to some degree by the broken ore in the stope. When the stope is mined out and while the remaining broken ore is being drawn, some sloughing from the hanging wall or footwall may occur, but usually the void created by the stoping operation remains open after the draw is completed. The rib and floor pillars remaining, therefore, provide support for the hanging and footwall and can be in the same configurations as described in sublevel stoping.

Shrinkage stoping has been employed to depths greater than 2,500 ft. Extractions from 75 to 85% usually are obtainable. Spans of 34 ft have been mined in a fluorspar deposit with hanging and footwalls of limestone. Spans of 70 ft were obtained in veins of chalcopyrite and pyrite where the hanging wall was defined by a fault and the footwall consisted of an ore-grade cutoff within the veins. The rocks surrounding the veins are slate, schist and graywacke.

Stull Stoping—Stull stoping is a method which employs systematic or random timbering (stulls) placed between the foot and hanging walls of a vein. The vein may be flat-lying to steeply dipping tabular, or narrow in type and usually 12 ft or less in thickness (Fig. 9-14). The stulls provide the only artificial support and usually require that the hanging wall and sometimes both the hanging and footwall be moderately competent as, for example, thin-bedded or partially bonded, jointed and fractured rock types.

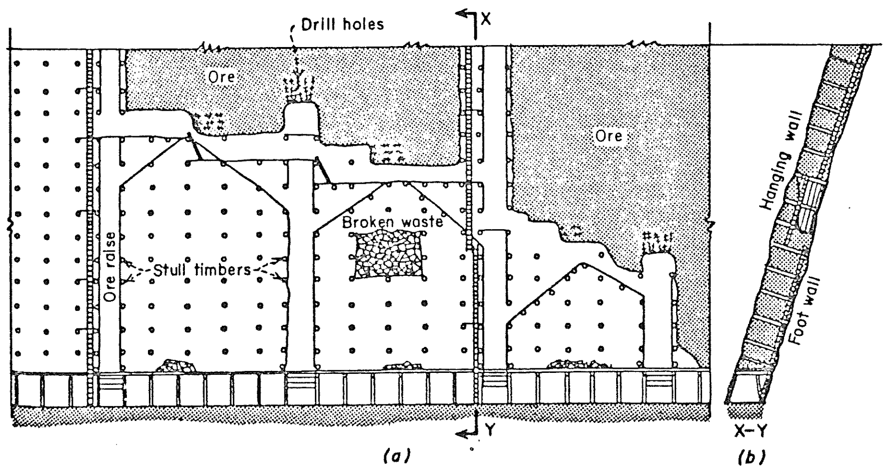


Fig. 9-14—Stull stoping (Jackson and Hedges[1]).

Stull stopes have been used in copper mining at a depth of 3,500 ft in a bed 10 to 12 ft thick dipping ±30° where the ore body occurs in a felsitic conglomerate. For mining thickness that exceeds 12 ft, other support systems are required.

9.3.2—METHODS EMPLOYING SUPPORTED OPENINGS

A supported opening is one in which a significant part of the incumbent load is carried on artificial support systems (props, sets, chocks, packs, backfill, etc.). Because the overlying cover imposes a gravity load of about 1 psi per vertical foot, an opening at a depth of, say, 500 ft will require a support system with a capability of 36 tons per sq ft for total support if no part of the gravity load originally carried by the rock in the opening is transferred to the rock surrounding the opening after excavation. Support systems with this capability generally are impractical except in the later stages of mining with backfill. Even with maximum utilization, stulls, sets and other light support probably will not carry more than a few percent of the gravity load. Hydraulic props (jacks) can provide a greater support. For an opening at a depth of 500 ft, 160-ton hydraulic props placed on 42-in. centers will support about 36% of the overburden weight. Chocks and pack walls, if placed close enough together, may support up to 50% of the gravity load in large stoping excavations. Backfill, on the other hand, can support

100% of the overburden weight after the surface subsides and compacts the fill material.

Cut-and-Fill Stoping—This mining method is best suited to vein or bedded deposits that dip at an angle greater than the angle of repose of the broken rock (Fig. 9-15). The ore may be massive, thick-bedded and partially bonded jointed material, but should be sufficiently competent to maintain stope-width spans. The surrounding rock, mainly the hanging wall, usually is of a type that will not stand for a long period without support.

In this method the ore is broken by overhand mining and removed from the stope. Sorting sometimes is performed, with the waste being left in the stope. After the broken ore is removed, the stope is filled with waste to within working distance of the back and the mining cycle is repeated. The waste fill may be broken rock, sand and/or gravel, soil or classified mill tailings. Pillars, if any, usually can be recovered and extraction may be near 100%.

If the vein being mined is extremely narrow, a method of mining called "resuing" may be used. In this system, the vein and a waste wall are broken separately to provide a working width and the waste is left in the stope for filling.

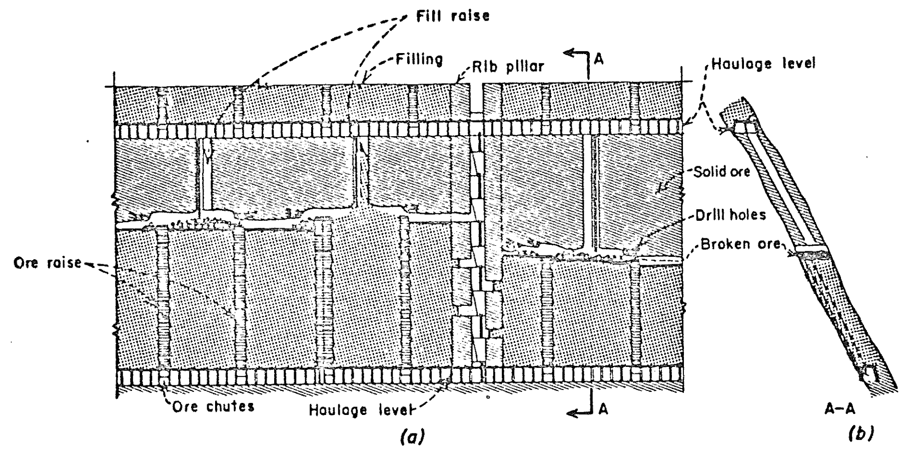


Fig. 9-15—Cut-and-fill (Jackson and Gardner[7]).

Cut-and-fill stoping has been conducted at depths of 8,500 ft in quartz veins 10 ft wide, with the surrounding rock being a lava schist and prophyry. Also, this method has been used at depths of 3,600 ft in quartz and pyrite veins varying from 5 to 100 ft wide.

Square-Set-and-Fill Stoping—Although square-set-and-fill stoping can be employed to mine almost any type of deposit and in most rock types, it generally is used for deposits where the ore is structurally weak and where faulting and fracturing of the surrounding rock has resulted in it also being very weak. The method is adaptable to deposits with irregular boundaries and is extremely flexible where ore varies greatly in short distances. This method can be applied when all others have proved inadequate, and insures a recovery approaching 100%. However, it is high in cost of materials and labor.

Under this system, the ore normally is mined from hanging to footwall for one or more sets of roughly 8 to 10 ft on a side. The placing and filling of the sets is part of the mining cycle. As the mining progresses, all sets are filled except those being used for ventilation, ore passes or manways. Ore sorting may take place in the stope, with the waste forming part of the fill. The square sets usually support only the back and the immediate hanging and footwalls. The fill, when added, will in due course take up a proportion of the total superincumbent load and may eventually assume 100% as caving or subsidence progresses.

This type of mining has been carried to depths greater than 8,500 ft in lead, gold and silver mines with hanging and footwalls of schists, porphyrys, highly altered quartzites, shales, limestones and granites.

An underhand-fill method of recent development, used with or without cemented fill, appears competitive with the square-set-and-fill method.[5,13]

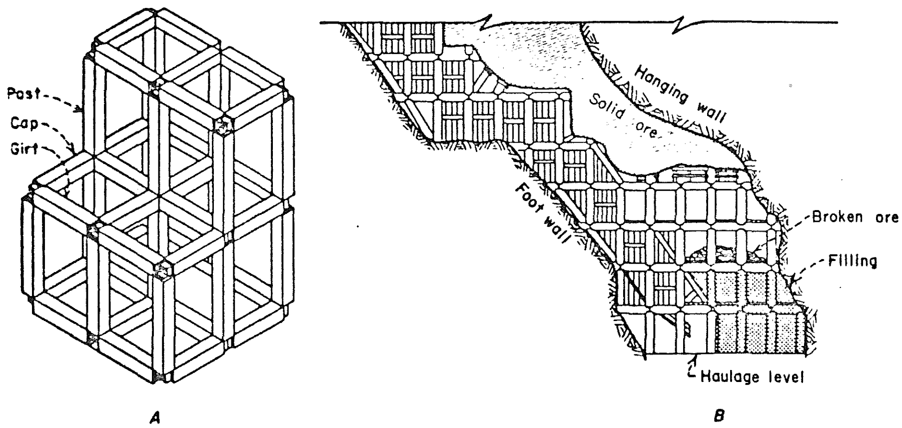


Fig. 9-16—Square-set stoping: (a) square-set timbering; (b) vertical transverse section (Jackson and Gardner[7]).

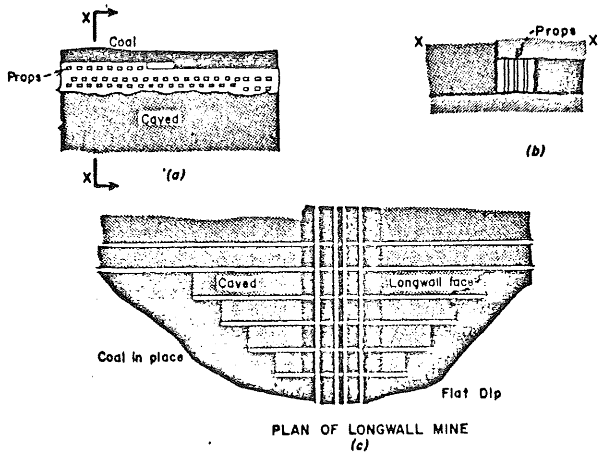


Fig. 9-17—Longwall mining (Stahl and Dowd[14]).

Longwall Mining—Longwall mining, in its original concept, is employed primarily to extract coal, although this procedure, with modifications, has been used to mine metallic minerals—for example, uranium and gold ore.[13] The method is adaptable to deposits ranging in thickness from 3 to 8 ft, dipping at less than 12°, and lying at depths up to 3,000 ft or more, provided the rock materials overlying the deposit are generally thin-bedded, relatively incompetent and cave freely and completely behind the prop line (Fig. 9-17). Also, the rock in the floor should be sufficiently competent to support the prop loads.

Because a massive system of props is used to support the roof over the face and working areas, longwalling is classified as a supported stoping method, although caving occurs in mined-out areas.

A sequential arrangement of stope faces on any dip to give a uniform pressure distribution, generally referred to as modified longwalling, is regarded as a method of ground control rather than a mining method.

In a longwall coal mine the face is fragmented with cutters, rippers, or plows, and the coal is transported from the face by a conveyor. Usually an extraction approaching 100% is possible. Because of virtually complete extraction, the overlying rock caves into the mined void, and the resulting surface subsidence is relatively uniform and complete. However, in some European coal mines in which longwalling is employed, caving and hence surface subsidence is limited by placing rock packs in mined areas.

Shortwall Mining—A shortwall method is utilized in the same type of deposit and in the same rock material as longwall mining. The primary difference in

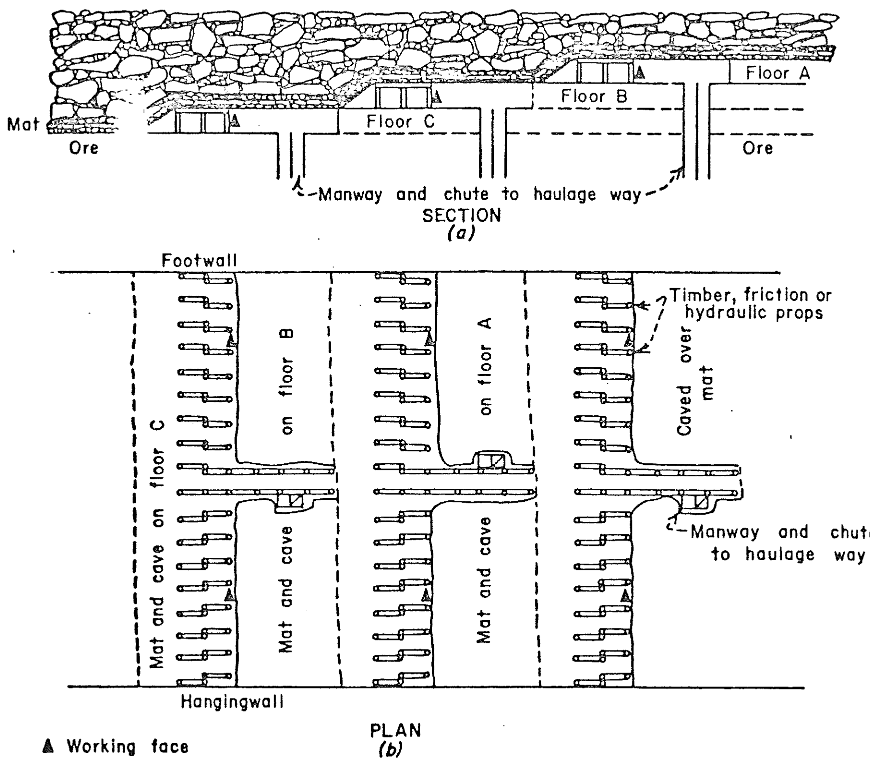


Fig. 9-18—Top slicing with mat (Morrison[8]).

the two methods is the length of the working face. In shortwall mining the maximum working face is normally 150 to 160 ft. Safety and mining laws in the United States require this shortening of the mining face. Also, the shorter face is more flexible, particularly in areas of bad roof. The equipment utilized and the results of this method of mining are the same as given for longwall mining.

Top Slicing—Top slicing is essentially a method for mining massive, thick-bedded or wide-veined deposits (normally greater than 15 ft wide) containing weak ore and walls that will not stand unsupported except over short spans. Ore extraction is in horizontal or near horizontal timbered slices starting at the top of the ore and working downward. A timbered mat is placed in the first cut and the overburden is caved. As subsequent cuts are advanced caving is induced by blasting out props behind the face, but always maintaining working room under the mat (Fig. 9-18).

The high concentration of timber support at the working face and the timber mat place this method of mining under the supported stope classification, even though caving to the surface is an ultimate result. The method is not selective, as ore sorting is not possible. However, hanging and footwall contacts can be irregular. Extraction approaches 100%.

Top slicing has been used successfully at depths of 2,500 ft in a deposit of soft hydrated hematite with a friable and highly fractured cap of chert jasper overlain by glacial fill. The system has worked equally well in other iron ores at depths of from 150 to 400 ft.

9.3.3—CAVING METHODS OF MINING

Three caving methods are generally recognized: top slicing, sublevel caving and block caving. In all cases, the general requirements for use of the method are massive-type mineral deposits of large horizontal area, such as thick beds, masses or wide veins. The ore should be weak, or if hard it should be thoroughly fractured with weak bonding, or a combination of these factors. Overburden may range from firm rock to glacial drift but must cave and follow the ore down as the ore is removed. Top slicing, because of timber requirements and method of support, has been described under supported stopes.

The rock materials in the ore body and overburden should be competent but highly jointed or fractured, and with virtually no bond across joint planes. These rock materials, due to the fracture pattern, should cave when support is removed from a sufficient area, which may range from 10 ft to more than a 200 ft square. In some rock types, assistance is needed to initiate the cave, such as outlining the area to be caved by longhole slots and presplitting. After the area has been detached from the surroundings, the rock should cave under its own weight, move by gravity and form fragments that are small enough to be handled conveniently. The method is most applicable to large low-grade ore deposits. Since the cave progresses to the surface, the method can only be employed where such disturbance can be tolerated.

Sublevel Caving—Sublevel caving can be used to mine massive or large pockets of ore and thick-tabular or wide-vein deposits that dip steeply. The method usually is employed to mine ore bodies of large horizontal extent, often below an open pit where irregularity of the walls or other factors make it preferable to block caving. The rock material in the deposit should be moderately competent, such as a jointed or fractured rock with some joint strength. The rock should not be free-caving, but when it is broken small fragments should be formed. This method, within limits, also can be utilized to mine soft sticky ores which have a tendency to repack. The rock material in the capping should be jointed and fractured, with partial bonding, and follow the ore cave without undue dilution or delay. When the overburden fails, small to medium-sized fragments should be formed but with very little fines.

The sublevels are driven between and parallel to the main levels, with the distance between the sublevels varying between 20 and 40 ft, giving a 20- to 30-ft column of rock to be fragmented by blasting (Fig. 9-19). As the rock is fragmented and collapses into the sublevel for removal, the overburden immediately caves onto the fragmented rock. This method requires the mining of the mineral deposit from the top down. Sublevel caving is nonselective, permitting no sorting in the stope. The hanging wall and footwall can be irregular. This method generally is used to mine low-grade deposits. A 90% recovery with substantial dilution is considered normal.

Sublevel caving has been carried out in ore deposits which contain soft hematite where the footwall is quartzite and siliceous slate, and the hanging wall is a cherty iron formation which is fractured with partial bonding.

Block Caving—Block caving is normally conducted in massive and disseminated low-grade mineral deposits of large horizontal dimensions which are structurally weak (Table 6-5, Sec. 6). The rock material in the deposit and overburden should be incompetent and should thus cave freely. Highly jointed, or fractured or thin-bedded rocks with very low bond strength across joints, fractures and bedding planes are typical caving material. The type of rock material in the overburden normally is not as important as the joint or fracture spacing, the degree of alteration

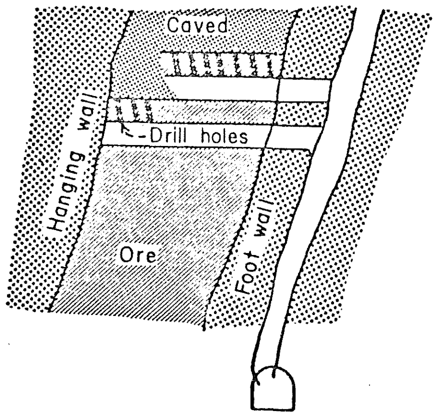


Fig. 9-19—Sublevel caving.

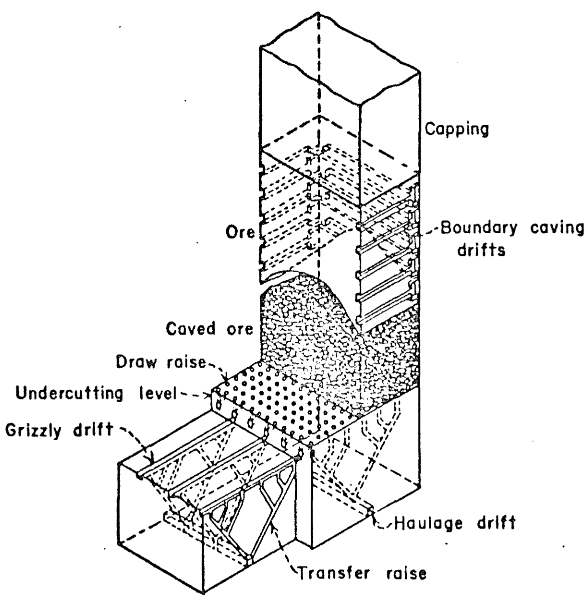


Fig. 9-20—Block caving.

in the rock and the lack of bonding in fractures. These rock materials should break but not repack. This allows for the grinding action of the ore during the caving cycle.

Block caving is a low-cost high-production method. Main haulageways and drawpoints are constructed below the block to be caved. The height of the block normally is greater than 100 ft (Fig. 9-20). The block is undercut and permitted to collapse, with the fragments being drawn off periodically. The initial block to be caved may require some loosening from the surrounding rock to initiate the caving action. Extraction of the mineral deposit with this type of mining is approximately 100%, although dilution may require an earlier cutoff and reduce recovery.

This type of mining is utilized in the porphyry copper deposits in the Southwestern United States, the molybdenum deposits in Colorado, the iron deposits in Northern Michigan, and in asbestos deposits in Canada. It has been carried on successfully at depths greater than 2,000 ft.

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APPENDIX C

Dust Control Techniques.
(Source: SME Mining Engineering Handbook, 1973)

16.5—VENTILATION FOR ENVIRONMENTAL CONTROL—METAL MINES

W. A. Bardswick

16.5.5—DUST-CONTROL TECHNIQUES

Basic Methods—The exposure of workers to harmful dust concentrations may be reduced by a systematic approach to the problem which would include implementing all or some of the following:

1. Reducing the production of dust.
2. Preventing the dispersal of dust clouds.
3. Providing dilution ventilation.
4. Utilizing personal protective measures.

Dust Production—Although most underground operations (such as drilling, blasting, conveying, etc.) produce copious amounts of dust, the amount so produced can be reduced in some instances. The environmental engineer should make a critical survey of all equipment and processes with a view to improving the use of machinery and altering techniques to produce less dust. It is an established fact that dull drill bits, through their grinding action, produce more dust than sharp ones, and that insufficient delivery of water to the cutting edge to remove rock debris also results in larger amounts of dust. In other instances, the grinding or crushing action of falling rock may be reduced by decreasing the distance through which the material falls—for example, a belt-transfer point or a chute used for loading ore trains or feeding a rock crusher. These are but a few examples of the many ways in which the production of dust can be reduced and, needless to say, any success in this approach will greatly enhance the success of the overall dust-control program. In practice, the dust-laden air should be exhausted directly to a return airway or rendered clean before reuse.

Dust Dispersal—The dispersal of dust can be reduced and controlled by a twofold method involving the use of water and control of air currents. Although water and other wetting agents are not significantly effective in allaying airborne dust, they are very useful in suppressing dust at its source. For this reason, every effort should be made to apply the wetting agent directly at the source to ensure that the broken rock is wet in situ and conditioned for ensuing processes tending to disperse dust.

The dispersal of dust can be practically eliminated in most instances by confining the dust-producing operation within an enclosure and controlling the air contained therein. The vitiated air from within the enclosure can be exhausted directly to the upcast airway or, if this is not feasible, it can be delivered to a filtering plant where the dust concentration can be reduced to an acceptable level.

Dilution Ventilation—The use of ventilating currents to dilute and remove dust clouds is the most common method of controlling contaminants at underground operations. This procedure is carried out in producing areas, such as stopes, scraper drifts, etc., by directing an air split from the main ventilating stream through the workings. The design criterion for volume of air required is based on an air velocity of 30 to 50 fpm, depending upon the type of operation and other local conditions. In some instances, volume may have to be increased greatly—for example, high-speed drives or scraper drifts, where the severe dust-producing operations may require as much as 150 fpm or more.

In headings and raises, the design volume also is based on providing an air velocity of 30 to 50 fpm or 30 to 50 cfm per sq ft of face. The type of auxiliary ventilating system employed will be governed by the local conditions at the mine. In most cases, the push-pull (or overlap system) will provide the most satisfactory environment. This system consists of a main ventilating duct which exhausts vitiated air, and a small blowing line kept to within 20 or 30 ft of the face. The length of the overlap of the exhaust and blowing lines will depend upon the size of the drift and, in any case, should be not less than 30 ft. The blowing fan should be rated for the calculated volume and, to avoid recirculation of vitiated air to the face, this figure should not exceed approximately 60% of the exhaust volume.

The overlap system should not be adopted for those working areas subject to a high virgin-rock temperature. Under these conditions, the cool fresh air is heated and becomes saturated with moisture during the time it takes to reach the working face. To avoid this, the forcing or blowing system of auxiliary ventilation should be installed so that the fresh air can be kept dry and delivered to the face with a reserve of cooling power. Ideal conditions will prevail if the discharge end of the blowing system is kept to within 20 to 30 ft of the working face. The sweeping or flushing action can be improved by installing a cone-shaped duct on the discharge end to increase the outlet velocity, or "throw," of the ventilating current.

Blasting damage to the ventilating duct may be avoided by utilizing a semirigid polyethylene duct with a wall thickness of ¼ to ⅜ in. St. Pierre[43] reports that the performance of this type of duct is excellent, with minimum leakage at the joints, a very low friction factor and 100% salvage value. He also reports that conditions were greatly improved during drilling operations in raises by installing a 6-in.-diam polyethylene duct for ventilation purposes. With the discharge end of the duct kept to within 25 ft of the face, dust concentrations were reduced to well below acceptable limits. Huggins[44] reports that dust concentrations in raises were reduced approximately 50% by use of a compressed air venturi and a 3-in.-diam aluminum pipe for ventilation purposes.

The use of exhaust systems only for auxiliary ventilation should be discouraged, since it is impractical to keep the end of the duct near the dust-producing operation, particularly when blasting. The effectiveness of an exhaust duct diminishes rapidly and approaches zero at a distance of approximately two pipe diameters from the inlet end.

Personal Protection—While the use of personal protective measures generally is considered a last resort in a dust-control program, there are occasions when their application can be very effective. One of the indirect methods used in all mines is removal of workers from the area at the time of the blast, and for a sufficient time thereafter for complete removal of the contaminants. A very commendable practice is to schedule all primary blasts, and as many of the secondary ones as possible, for the end of the working shift. The exposure to dust and fumes thereby is considerably reduced because the workers will be out of the zones through which the vitiated air travels on its way to the main upcast airway.

There are occasions when local conditions make it impractical or impossible to confine effectively a severe dust-producing operation. Under these conditions, personal protection may take the form of an enclosure large enough to house the worker and maintained under positive pressure of good-quality air. If this is not feasible, the last resort is use of a dust respirator approved for that particular type of dust by the U.S. Bureau of Mines. However, it is suggested that every effort should be made to utilize some other means of dust control, since experience has shown that the use of dust respirators is difficult to promote, either by a voluntary or a mandatory approach.

Drilling Operations—Drilling is an essential operation in metal mines and the percussive type of drill in common use can be the source of copious amounts of dust. However, a number of precautions can be taken to improve this undesirable feature.

The "collaring" operation produces considerable dust, particularly in the absence of water. Introduction of the automatic back head (interlocking air and water valves) was designed to eliminate the temptation to "collar" dry. Unfortunately, some operators install a hand valve in the water hose near the drill so that dry collaring still is possible—and practiced in many mines. These hand valves should be removed or relocated so that they are not readily accessible to the driller. Dry collaring also can result from careless operation of the automatic control. As the latter is advanced to the "On" position it opens the water valve before the air throttle. If the operator "slams" the automatic control to the open position, the drill steel begins to rotate before any water reaches the cutting edge. This bad practice should be eliminated by instructing the operators in the proper drill operation.

Percussive drills continue to liberate dust after the collaring operation is completed, even though the drill hole is full of water. This is caused by the leakage of small amounts of compressed air past the piston into the front head of the machine, whence it travels down the hollow drill steel as bubbles in the water. At the toe of the hole, these air bubbles entrain dust particles, which are released as the bubbles emerge from the hole. Drill manufacturers have managed to overcome this problem by designing the vented front-head machine, in which the compressed air is permitted to escape before it enters the shank of the drill steel. Although this type of machine disperses less dust than earlier models, it is not used extensively in North America because of the high maintenance cost of the parts in the drill front head. In the standard machine, leakage of compressed air past the cylinder is intentional, since this air is saturated with oil which lubricates the moving parts in the front head of the unit. It would appear that the vented type of drill will not be fully acceptable until the manufacturers implement an alternative method of lubricating the front-head parts of the machines.

A number of other features can adversely affect the operation of drills, including the size and condition of the water tube, the sharpness of bits, the condition of the water hole in the bit and the shank end of the drill steel, and the operating water pressure. These and some other aspects in drill operation, with recommendations to reduce the amount of dust produced and dispersed, are reported by Yourt and Bloomer.[45]

Blasting—Blasting is unlike most other mining operations, since it produces not only dust but other contaminants, such as carbon monoxide and nitrous fumes. Since little can be done to prevent the production of dust during a blast, the emphasis should be on means to control the resultant contaminants.

The first step in controlling dust produced by blasting is to ensure that the area surrounding the blast (the walls, floor and back) is thoroughly wetted beforehand. This precaution will prevent dust settled out during previous operations from becoming airborne. Furthermore, some of the dust created by the blast will adhere to the wet surfaces in the area, thereby reducing the concentration in the air stream.

The second step, which is the most effective measure for controlling the contaminants resulting from a blast, is use of an arrangement called the "air-water blast." This device is excellent in minimizing the dispersal of dust and in reducing the amount of nitrous fumes (because of their solubility in water). Unfortunately, the carbon-monoxide content of the air from a blast is not appreciably affected since it is practically insoluble in water.

The design of the air-water blast device varies from a simple inexpensive model assembled in the machine shop at the mine to the elaborate expensive type of unit available from a number of manufacturers. The effectiveness of this device is determined by the fineness of the mist or fog produced, and the "throw" and "spread" of the mixture. An unpublished report, covering tests made on 11 different types, indicated that the water consumption varied from 0.75 to 8.0 gpm; compressed air, 105 to 300 cfm (free air). It is apparent that a particular design should not be adopted as standard for use in all working places in all mines. A working area with little or no ventilation would benefit most from the unit providing the high rate of airflow. On the other hand, it would be poor engineering to waste expensive compressed air in a place well ventilated by conventional means. Similar reasoning may be applied to the utilization of the water delivered, particularly for blasts in drifts and raises vs. the larger ones in stopes involving much greater tonnages of broken material.

An important feature of the air-water nozzle is the manner in which the compressed air and water are blended before dispersal. The design should incorporate means precluding the drainage of water into the compressed-air line, particularly when the pressure of the latter is lower than that in the former. Two suitable types of nozzle are illustrated in the text published by the ILO,[32] and the details of a number of others are available on request from the Mines Accident Prevention Association of Ontario.[46]

The final step in the control of contaminants resulting from blasting is providing means to cleanse pollutants. A unique type of collector used in South Africa consists of a bed of vermiculite, treated with a solution of sodium carbonate and potassium permanganate, which removes the nitrous fumes, and a flannel filter for the collection of dust particles. After treatment, the air is returned to a fresh-air split where the dilution ratio is at least 5 to 1. Plant details are presented by Rabson.[47]

In the absence of a filtering system, blasting dust and fumes should be diluted and exhausted to surface via an untraveled route, preferably an upcast raise designed for that purpose. If this is not feasible, the blasting schedule should be arranged so that the vitiated air will pass through working places when the miners are absent. Failure to do this creates an undesirable health hazard and reduces production efficiency due to miners leaving their working places for the time necessary to clear the dust and fumes.

Mechanical Loading—Utilization of an air-water blast for a suitable period after blasting obviates the need for applying a water spray immediately before commencing the loading operation. On the other hand, if the rock pile has to be wetted by means of a hand-held hose, the quality of the environment will deteriorate considerably because the application of water under these conditions is similar to directing a stream into a barrel of flour.

Investigation of a rock pile after it apparently is soaked with water will reveal that the depth and extent of water penetration are shallow and limited. In other words, it is practically impossible to wet a pile of rock thoroughly by conventional spraying methods. As a result, dry dusty material is exposed continuously as the loading or scraping operation progresses. Under these circumstances, it is necessary to maintain a continuous mist or water spray at the point where new fragmental material is exposed. Use of a hand-held hose is not recommended, because the operator, on exposing new rock during the process of loading an ore carrier, may be tempted to complete the cycle before applying more water. Some success has been achieved by installing a nozzle spray on the front end of compressed-air-operated loading machines. A nozzle which provides a flat diverging stream of mist-size water droplets produces a good wetting action with minimum water consumption.

The scraping of rock in stopes and scram drifts produces large amounts of dust, particularly if the material is dry and the speed of the operation is high. A successful method of controlling this dust is to install a bar-type water spray where the scraper "bites" into the rock pile, with one or more additional nozzle-type sprays at intervals over the path followed by the scraper. A mist type of spray at the ore-pass or loading point will reduce dust dispersal and condition the rock for ensuing operations (such as crushing and conveying). Details of the nozzle and bar-type sprays are presented by Peacock.[48] The effectiveness of these units in reducing dust dispersal is also discussed.

Ore and Waste Passes—Ore and waste passes are, by the nature of the operation carried on therein, sources of large quantities of airborne dust. The broken rock delivered to the passes contains a considerable amount of inherent dust as a result of the comminution effects of preceding operations, such as blasting, loading, etc. Furthermore, the autogenous-grinding action of the rock as it is dumped and falls down the pass produces more dust, which becomes airborne and subject to dispersal. The extent of the dispersal of dust at these locations is governed by two factors: (a) the displacement of air in the raise by the rock, and (b) the entrainment of air due to the "piston" effect of the falling rock in the raise.

A number of measures can be employed to reduce the dispersal of dust at the entrance to ore and waste passes. The first line of defense is to ensure that the rock is thoroughly wetted before delivery to the dump. As previously mentioned, the use of water-spraying devices in preceding operations will tend to suppress the dust contained in the rock mass. A further wetting effect can be obtained at the dumping site by installing a mist-type atomizer to spray the rock as it falls into the pass. A very fine mist should be used to provide the optimum wetting effect with a minimum amount of water. Excessive use of water at the orepass can be objectionable for two reasons: (a) the excess water in the ore can adversely affect crushing and milling operations, and (b) a large quantity of water may accumulate on top of the ore in the raise, thereby creating a hazardous condition for workers on the lower levels.

The second step in providing a good environment at ore and waste passes is to prevent the escape and dispersal of dust into working areas by confining it within the passes. This can be accomplished by a system of stoppings and airtight doors over the dumps or "tipping" points. The maintenance of these doors is of prime importance, since their main purpose is to prevent the escape of dust-laden air. A number of designs have been developed over the years and a typical one is described by Kneen.[49] This particular unit was designed for Granby-type cars. It is pneumatically operated and easy to maintain.

The third step is providing means to keep the confined ore or waste pass under negative pressure to ensure that all leakage paths are indraft, and to capture the air displaced when rock enters the raise. This can be accomplished by installing a suitable fan to exhaust from a convenient point in the raise. The vitiated air exhausted must be filtered or sent via a direct untraveled route to the return-air raise.

The ideal location for the exhaust fan is at the top of the ore or waste pass. Under these circumstances, the coarse dust is allowed to settle out in the raise, thereby reducing the load on the dust collector and, in the absence of a cleaning device, minimizing the abrasive wear on fan blades. Another advantage of this arrangement is that it facilitates the disposal of dust-laden air by utilizing an exhaust point above the active or producing levels of the mine.

An alternative to the use of a single fan is to install smaller individual exhaust units to service each dumping area. Suitable enclosures, fitted with hinged or sliding doors, can be constructed over the dumping area. The rating of the fan is determined by the volume of air necessary to maintain an indraft velocity of 200 fpm through the door opening during the dumping operation. The air exhausted from the raise usually is very dusty and must be delivered to the return-air system or processed in an efficient dust collector before joining a fresh-air split. The details of a particular design will depend upon the local conditions at the dump site. The reader is referred to typical layouts described by Kneen,[49] Phimister[50] and Gray et al.[51]

Although the dust control measures previously discussed are capable of capturing the air displaced by the rock dumped into the raise, they are not entirely satisfactory for the control of dust dispersed by airflows induced within the raise by the "piston" effect of falling rock. A practical solution to this problem is described by Marshall.[52] The success of this method is due to the provision of "relief valves," created by driving connecting drifts between the ore and waste passes at a number of elevations. In practice, the dust-laden air under pressure because of the falling rock is short-circuited through the connecting drift to the parallel raise, which acts as a "reservoir" for the air exhausted by the fan. In the case study by Marshall, he reports that the surges of dusty air were practically eliminated and that the success of the system enhanced dust-control effectiveness at the underground crusher station and loading pocket.

At first glance, it may appear that the problem of controlling the dispersal of dust from ore and waste passes has been solved. Unfortunately, this is not so, because there is no proven method of accurately determining the fan rating required to overcome the effects of the surges of dusty air. The volumes of the induced airflows depend upon a number of parameters, including: the size and flow rate of the falling rock, the distance through which it falls, the size of the raise, and the openings available for airflow into and out of the system of raises. Although this has been the subject of a number of research projects, the answer still is evasive and there exists a need for further investigation. The nearest approach to a solution has been reported by Anderson,[53] who presents the formula,

$$Q = 10A^{3}\frac{RS^2}{D}$$

where Q is the induced airflow in cubic feet per minute (cfm), A, the opening in square feet in the upstream enclosure, R, rate of material flow in tons per hour, S, height of fall in feet, and D, average particle diameter of material in feet.

Although this formula has limitations and does not provide an accurate solution in all instances, it can be a very useful guide in eliminating some of the guesswork involved in determination of a suitable fan rating for the exhaust system.

Another method of controlling the dispersal of dust from an orepass is to exhaust the induced air from a point near the bottom of the raise. In this approach, the volume of air exhausted must be sufficient to maintain the raise under negative pressures, thereby precluding the buildup of positive pressures, which cause the surges of dusty air. This method can be used to good advantage in a multilevel mine where ore is continuously delivered to secondary orepasses which feed a main orepass (for example, a caving method of mining incorporating scraper drifts). The application of this method is discussed by Foster.[54] One disadvantage is the problem created by the coarse dust particles carried out of the orepass by the air being exhausted. A plenum area or some other provision must be made for the removal and disposal of the coarse-dust particles. Another disadvantage is the abrasive effect of dust particles upon the blades of the exhaust fan, particularly if it is installed in the section of the return airway near the orepass.

Crushers—An underground crushing station is a confined space that can be contaminated readily by dust if preventive means are lacking. The deterioration of the environment in these rooms is caused by the dust contained in the ore as a result of preceding operations, plus that produced by the grinding action of the crusher jaws. These sources of dust can be controlled by applying the usual principles (a) confinement and (b) dilution and removal of the particles which become airborne within the enclosures.

The simplest and most effective approach is to enclose completely the chute which feeds the crusher, the jaws of the crusher and all other openings to the path followed by the ore stream. To prevent the dispersal of dust, the enclosure is kept under negative air pressure by a fan which exhausts from the opening below the crusher jaws. The entry to the exhaust duct in the crusher pit should be as far away from the ore stream as practicable to avoid the capture of coarse particles which have a detrimental effect on all components of the dust-control system.

The rating of the exhaust fan is determined by calculating the volume necessary to provide an indraft velocity of 200 fpm through all unavoidable openings and leakage paths in the enclosures. Since the volume of air required is a function of the area of the openings available for airflow, it is apparent that the fan rating will increase considerably if the jaws of the crushers are not enclosed. Although some objections may be raised to total enclosures of feed chutes and crushers, there are numerous examples in the literature of the successful implementation of this design. Some typical layouts are described by Walker,[55] Slater[56] and Gray et al.[51]

Disposal of the vitiated air exhausted from the enclosures may be accomplished in a number of ways, the choice of which will be governed by local conditions. The ideal method is to exhaust the dust-laden air through a ventilation duct leading to the return-air raise. Unfortunately, this is not always practicable because of the location of the crusher station relative to the raise. Consequently, an alternative method must be employed. In the alternate design, a suitable dust collector is installed in or near the crushing station to filter the air before it is returned to a fresh-air split. If the filtered air is to be recirculated within the crushing plant, it should be diluted by a ratio of three volumes of fresh air to one of filtered air to compensate for the inefficiency of the dust collector. The effectiveness of the latter will depend upon a number of factors, such as the dust loading, the type of filter medium, and the air velocity through the unit.

The type of dust collector recommended for use in underground dust-control systems is one which incorporates a fabric material as the filter medium. It provides good filtering efficiency at relatively low cost. The three types of fabric most commonly used are wool, terylene and cotton. For many years, the latter was very popular and it still is widely used in some of the larger asbestos plants. The main disadvantage of the cotton-type collector is the space requirement due to the low air-to-cloth ratio (velocity through the cloth) required for optimum filtration. If the air velocity through the cotton fabric is increased beyond 2 fpm, the efficiency of filtration deteriorates rapidly.

The use of wool as a filter medium originated in South Africa and it still is used extensively there and in Australia. Permeability is low, resulting in good filtering efficiency at high air velocities. Rabson,[57] Walker[55] and Phimister[50] report on underground installations where good air conditions were maintained with air velocities of up to 26 fpm through flannel bags. However, the life of the bags was limited to about 20 mo in one installation and, in most instances, the filter media is removed from the housing for cleaning purposes. These two features augment the operating cost of flannel-type filters, and the inconvenience of the high maintenance requirement merits due consideration.

In recent years, the development of synthetic fabrics has introduced a number of new cloths suitable for air-filtration purposes. One of these, a heavily napped terylene, has gained much prominence in the Canadian mining industry. Segsworth[58] reported on the use of this material at an air-to-cloth ratio of 7:1. After 7 yr in an underground crushing station, the original terylene filter bags still were in place and delivering clean air with dust concentrations well below the threshold-limit values. Similar performances are being obtained at other Canadian mines under conditions of high humidity, indicating that a heavily napped terylene can be recommended for most underground applications.

One feature of a dust-collecting system often overlooked is disposing of the collected dust. A poor method can result in dispersal of the dust in the working area, thereby negating the value of the control system. Proper disposal of dust will be greatly facilitated by installing the dust collector near the orepass so that the dust hopper can be drained directly to the raise via a duct or totally enclosed chute.

Conveyors—Belt conveyors are prolific sources of dust, particularly at transfer points, where large amounts of dust are dispersed as the conveyed material falls through space. Furthermore, as the return belt passes over the idlers, the dust clinging to the underside of the belt is jarred free and dispersed into the atmosphere. Unless these dust sources are controlled, the conveyor gallery will be subject to high dust concentrations.

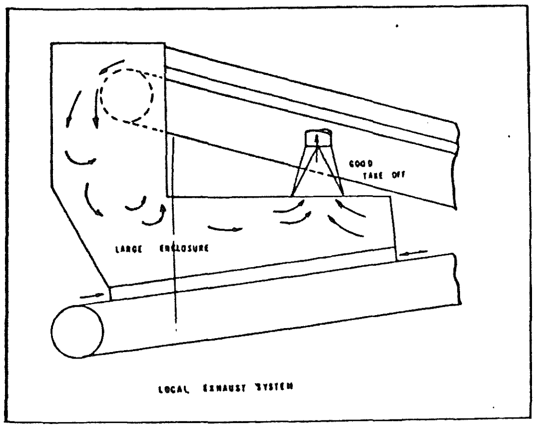

Fig. 16-47—Local exhaust system.

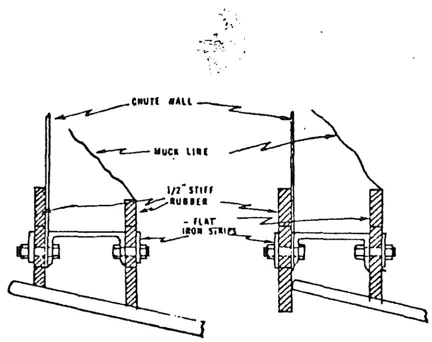

Fig. 16-48—Outside dust skirting inside muck guide.

The ideal way to prevent the dispersal of dust from a belt conveyor is to enclose the conveyor belt completely and exhaust a volume of air sufficient to maintain all leakage paths indraft. For the purposes of maintenance and cleaning spills of material, one side of the enclosure should be hinged for access. This arrangement will effect positive control of all dust sources and provide a suitable working environment.

In some instances it may not be practical to enclose the entire conveyor belt. An alternative consists of installing large enclosures around the path followed by the falling material at the head and tail pulleys of the conveyor system. Enclosure details are shown in Figs. 16-47, 16-48 and 16-49.

The most important feature in the design of an effective dust enclosure is the need to realize that the primary purpose of the ventilated housing should be to control the air currents around the dust-dispersing operation(s). In other words, the aim should be to design an air-control system and not a dust-collecting system. If this is done effectively, the harmful dust particles—those less than

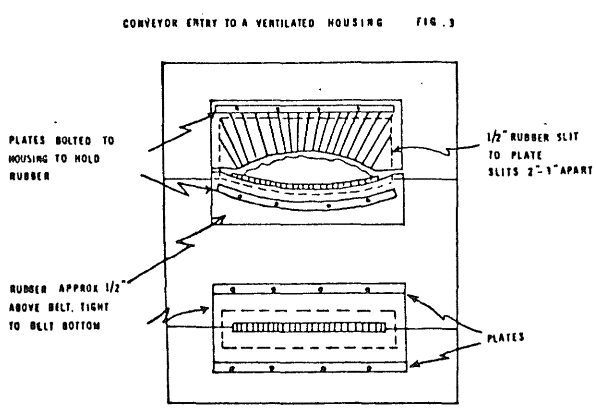

Fig. 16-49—Conveyor entry to a ventilated housing.

10μ in diam—will be captured by the air exhausted from within the enclosure, and the coarse particles, which cause abrasive wear on the elements of the dust-control system, will be left behind to gravitate to the material being conveyed. To satisfy these requirements, the dust-control system should consist of a large enclosure which is effectively sealed and serviced by an exhaust duct installed at a suitable distance from the region of turbulence created by the falling material. Under these conditions, the coarse particles of dust would settle out of the air stream before reaching the inlet of the exhaust duct.

The volume of air which should be exhausted from a belt conveyor enclosure will depend upon the width and speed of the conveyor, the rate of material flow, and the distance of fall for the ore at the transfer point(s). The main criterion for design purposes is to ensure that all leakage paths to the enclosure are kept indraft by air flowing at a velocity of 200 fpm. If the distance through which the ore falls is sufficient to induce airflows within the enclosure, it may be necessary to apply the Anderson Formula[53] in an attempt to determine the exhaust volume of air necessary to prevent the escape of surges of dust-laden air. For standard conveyor enclosures, the recommended volumes of exhaust air are contained in the Manual of Recommended Practice prepared by the American Conference of Governmental Industrial Hygienists.[59] This manual is recommended for use in the design of all industrial fume- and dust-control systems.

The ventilated housings around the head and tail pulleys of the conveyor belt must be complemented by additional measures to control the dust dispersed by the remainder of the conveyor belt. A simple control method is to isolate the conveyor gallery by installing a door stopping at each of the pulleys. In practice, the enclosures around the pulleys should extend to the stoppings. In this way, some air is exhausted from the confined conveyor gallery, and usually is sufficient to maintain indraft airflows through the leakage paths around the conveyor-gallery doors. This arrangement will confine the dust to the isolated conveyor gallery, so that workers entering this area for maintenance or other purposes should be required to wear an approved type of dust respirator.

Shaft-Loading Stations—The dusty nature of the operations at a shaft-loading station tends to produce a poor working environment, aggravated by the confined quarters in such locations. Unfortunately, these stations are difficult to ventilate because, in effect, they are dead-ends, located well below the last active level of the mine and far removed from ventilating circuits. Consequently, the most economical approach usually is the adoption of a "self-contained" dust-control system.

The system recommended consists of the application of the well-known principles relating to confinement of the dust source, removal of the airborne dust from within the enclosure and cleansing of the air by an efficient dust collector. In most instances, the ore chutes and measuring pockets can be partially, if not totally, enclosed. The design principles are similar to those discussed for control of dust in crushing plants and conveyor systems. The details of an ideal design are given by Hall,[60] who reports that dust concentrations were reduced from maximum levels of 1,100 to 216 ppcc.

An alternative approach is to require use of respirators by persons employed in shaft-loading stations. This is not recommended for two reasons: first, there is no guarantee that the operators will conscientiously use respirators; second, some of the uncontrolled dust from this area will be dispersed to other places where men work and travel.

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APPENDIX D

Hydraulic Backfilling

Hydraulic backfilling is the use of the coarse sands from mill tailings in filling the voids left underground after removal of ore from stopes or other underground openings. Backfilling a stope provides some degree of support to the adjacent pillars and overlying ground. When compared to ground support measures utilizing timber or waste rock fill, hydraulic backfill possesses the following advantages:

1) Improved ground control.
2) Faster development, advancement, and extraction of ore from stopes due to the rapid filling with sand.
3) A more effective and economical method of transporting a ground-support material underground.
4) Fuller extraction of ore due to the improved support methods and the reduction in ore loss.
5) Improved ventilation control.
6) Decreased fire hazard.

An additional benefit, which has only recently been appreciated with the utilization of mill waste for underground fill, is the significant reduction in the volume of mill waste requiring impoundment behind surface tailings dams.

However, along with the improvements offered by hydraulic fill techniques over timber setting and gobfill in stopes, several principal support problems and disadvantages are yet to be solved. Among the important drawbacks that result from hydraulic backfill are the following:

1) The large volumes of water used to transport tailings must eventually be pumped out of the mine.
2) Haulageways and drainage ditches are fouled and filled with fine slimes that decant from the filled stopes with the transport water.
3) Spillage of fill from stopes due to piping (erosion) induced by imperfect sealing of the stope, high hydrostatic head, or malfunction of the hydraulic system creates additional maintenance and cleanup costs.
4) During the ore mining process, ore may become mixed with valueless sandfill material creating a finite volume of dead-load material in the ore stream.

> 5) Even though hydraulic fills are a substantial improvement over waste rock fill or open stoping, present hydraulic fills are not providing adequate support in many mining operations. This is evidenced by the development of rock burst in ore pillars and/or instances of additional support (timbers) requirements in order to assure full extraction of the ore. Failure to obtain full extraction promotes poor conservation of valuable mineral resources.
>
> (Hydraulic Sandfill in Deep Metal Mines, 1975, pp. 2-3)

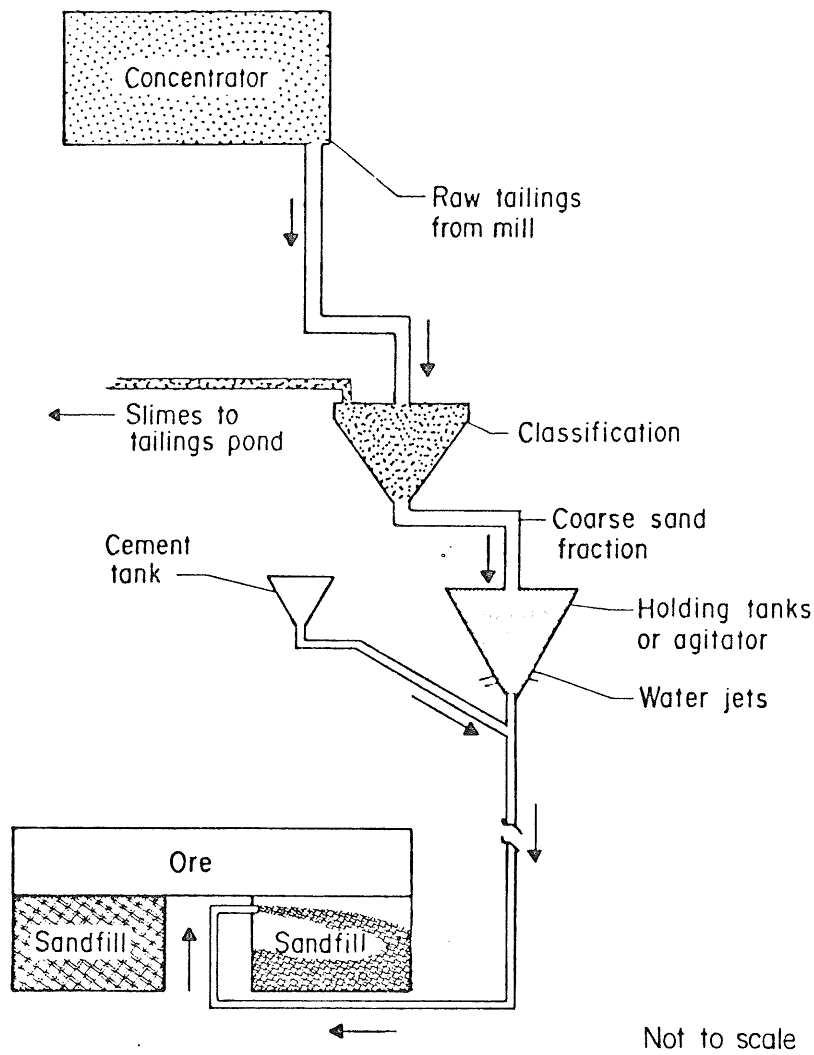
A typical flow diagram of a hydraulic backfill system is shown in Figure D-1. As shown on the diagram, the hydraulic sandfill system begins at the time the raw tailings are separated from the ore concentrate at the mill. The tailings are pumped to a primary classification unit (usually a cyclone) where the coarser size fraction of the tailings are separated from the finer size fraction. The coarse fraction is transported to holding sand tanks; the finer fraction is sent to surface tailings ponds.

The classified tailings are stored in holding tanks until the sand is required underground. Upon demand, the sand is mixed with a predetermined quantity of water; cement, additive, or other modifier is combined with the sand-water mixture to give a stronger fill and to hasten consolidation, and the hydraulic fill is transported to the underground mining area.

When the stopes are to be filled with hydraulic fill, it is necessary to close off all abandoned stope accesses. Bulkheads should be capable of sustaining hydrostatic heads equal to the full height of the stope to be filled. Concrete or heavy timber bulkheads are preferred. Any massive bulkheads should be valved to bleed off liquids from the fill.

Stopes drain by percolation, decantation, or both. Drains are of various designs but all serve the primary function of providing a pathway through

Figure D-1. Flow diagram of a hydraulic sandfill system.



which excess water can be removed from the stope.

> Although hydraulic sandfill has been placed in metal mines for many years, it has only been in the past 15 years that mining companies, educational institutes, and research agencies have been engaged in basic or applied research in this field. To a great extent, however, the majority of articles and presentations have described the physical plant facilities for preparing and transporting the fill material, the preparation of mined openings for sandfill, and general effectiveness of the backfill material at individual mines. Actual research has been primarily directed toward investigating the physical properties of hydraulic backfill and methods of densifying or modifying the sandfill to increase its support capabilities. To complement this laboratory research, field investigations have been undertaken to develop background data pertaining to physical changes experienced in the mine before, during, or following ore extraction. In addition, but to a more limited degree, research has evaluated the factors influencing the hydraulic system used to place the sandfill. Most recently, theoretical modeling of cut-and-fill stopes has enabled researchers to evaluate the effectiveness of various types of sandfill. With additional refinement, this analytical technique should provide a means for predicting the support requirement necessary to alleviate heavy ground conditions associated with stope mining.
>
> (Hydraulic Sandfill in Deep Metal Mines 1975, p. 8)

It must be stressed that even with cemented backfill (tailing sands with cement added), roof closure can occur with surface subsidence being a possible consequence.

> Mine filling costs vary from operation to operation, depending upon type and source of fill, transport distance, cost of sand walls and drainage system, and labor costs. Construction costs for placing burlap sand walls or fences have averaged $0.46 per square foot; costs for concrete bulkheads would be substantially higher. Placement of classified tailings has generally cost less than quarried sand and/or gravel. Other mines have placed smelter slag with the sand fill. Hydraulic sandfill costs have ranged from $0.56 to $4.49 per ton of sand.
>
> (Hydraulic Sandfill in Deep Metal Mines 1975, p. 8)

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