Chapter 4

METHOD SELECTION - A NUMERICAL APPROACH

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INTRODUCTION

In this paper, a numerical process for selecting a mining method, with the emphasis on underground mass mining techniques, such as caving, induced caving, and stoping, is proposed.

In the past, selection of a mining method for a new property was based primarily on operating experience at similar type deposits and on methods already in use in the district of the deposit. Then, the chosen method was modified during the early years of mining as ground conditions and ore character were better understood. Today, however, the large capital investment required to open a new mine or change an existing mining system make it imperative that the mining methods examined during the feasibility studies and the method actually selected have a high probability of attaining the projected production rates.

Although experience and engineering judgment still provide major input into the selection of a mining method, subtle differences in the characteristics of each deposit, which may affect the method chosen or the mine design, can usually be perceived only through analysis of measured characteristics.

The parameters that must be examined when choosing a mining method include:

1) geometry and grade distribution of the deposit;
2) rock mass strength for the ore zone, the hanging wall, and the footwall;
3) mining costs and capitalization requirements;
4) mining rate;
5) type and availability of labor;
6) environmental concerns; and
7) other site-specific considerations.

This paper encompasses a detailed look at the first two parameters since they, plus mining costs, have the greatest impact on the selection of a mining method.

The proposed method selection process is for a project where drilling has defined sufficient geologic reserves, but little or no underground development has been done.

Since each deposit has its own characteristic geometry/grade distribution, and rock mechanics properties, mining method selection should be at least a two-stage process.

In Stage 1, the deposit is described in terms of geometry, grade distribution, and rock mechanics properties. Using these parameters, the mining methods can be ranked to determine which are most applicable; they can then be considered in general terms of mining and capitalization cost, mining rate, type and availability of personnel, environmental concerns, and other site-specific considerations.

In Stage 2, the most likely mining methods are costed out, based on a general mine plan. Mining and capitalization costs are used to determine a cut-off grade from which a minable reserve can be calculated; economic comparisons can then be made to determine the optimum mining method and economic feasibility.

During the mine planning phase of Stage 2, rock mechanics information would be used to provide realistic estimates of underground opening size, amount of support, orientation of openings, and caving characteristics, and open pit slope angles. If ground control or operational problems should be encountered with the methods being considered, modifications could be made. Although planning on paper extends start-up time, it is cheaper to err on paper than to find the error after mining has begun.

METHOD SELECTION - STAGE 1

The main purpose of Stage 1 is to select those mining methods which should be considered in greater detail. The simplest way to do this is by defining those characteristics required for each mining method and then determining whether the characteristics of the deposit are suitable. However, no one mining method is so restrictive that it can be used for only one set of characteristics, as indicated by the classification system proposed by Boshkov and Wright (1973). In the mining method selection proposed, geometry, grade distribution, and rock mechanics characteristics are ranked according to their acceptability for ten general mining methods.

Data Required

The most important data required for selection of a mining method and initial mine layout are geologic sections and level maps, a grade model of the deposit, and rock mechanics characteristics of the deposit, footwall, and hanging wall. Much of this data can be obtained from drill core, and, if it is not collected during the initial core logging or assaying, it will be lost.

Geology. Basic geology interpretation is of major importance in any mineral evaluation. Geologic sections and level maps which show major rock types, alteration zones, and major structures, such as faults, veins, and fold axes, should be prepared. It may be advisable to define the alteration zones on a separate set of maps, which can then be overlain onto the rock type geology maps. These geologic sections and level maps should be prepared at the same scale as will be used for mine planning. Sections should be drawn to true scale, without any vertical exaggeration, because it makes it easier to visualize the relative layout of mine workings. The area included on the maps should extend horizontally in all directions 1.75 times the depth beyond the limit of the orebody. Although an area this size may seem excessive, it will ensure that there is sufficient information for evaluating the limit of ground surface movement due to mining: this information is needed to locate shafts, adits, and buildings, etc.

The importance of a complete set of interpreted sections and level maps cannot be overstated. They are necessary for defining grade distribution, as well as units of similar rock mechanics characteristics.

Geometry of Deposit and Grade Distribution. During Stage 1 of the method selection process, geometry and grade distribution are defined. The geometry of the deposit is defined in terms of general shape, ore thickness, plunge, and depth (Table 1). Grade distribution is defined as uniform, gradational, or erratic (Table 1).

Defining the geometry and grade distribution of a deposit requires development of a grade model. The type of model constructed will depend on the complexity of the geology and how well it is understood, as well as on the drill hole spacing. The grade model should be put on sections and level maps at the same scale as the geology maps and should be contoured by grade, or the blocks should be colored by grade categories. These contoured or colored grade sections and level maps, when overlain onto the geologic sections and level maps, will indicate the dominant rock types, as well as their spatial relationships to the orebody.

Table 1: Definition of Deposit Geometry and Grade Distribution

Geometry of Deposit	
1) General shape	
equi-dimensional:	all dimensions are on the same order of magnitude
platey - tabular:	two dimensions are many times the thickness, which does not usually exceed 100 m (325 ft)
irregular:	dimensions vary over short distances
2) Ore thickness	
narrow:	<10 m (<30 ft)
intermediate:	10 m - 30 m (30 ft - 100 ft)
thick:	30 m - 100 m (100 ft - 325 ft)
very thick:	>100 m (>325 ft)
3) Plunge	
flat:	<20°
intermediate:	20° - 55°
steep:	>55°
4) Depth below surface	
provide actual depth	
5) Grade distribution	
uniform	
the grade at any point in the deposit does not vary signficantly from the mean grade for that deposit	
gradational	
grade values have zonal characteristics, and the grades change gradually from one to another	
erratic	
grade values change radically over short distances and do not exhibit any discernible pattern in their changes	

Rock Mechanics Characterization. In Stage 1 the rock properties need to be classified so that an overall rock mechanics picture of the deposit is provided. A number of classification systems have been presented (Deere, 1968; Coates, 1970; Bieniawski, 1973; Barton et al., 1974; and Laubscher, 1977). All these systems include the basic measurements of rock substance (intact rock) strength, some measurement of the fracture intensity, and some measurement of the fracture strength. The classification systems of Bieniawski, Barton et al., and Laubscher use individual parameters to calculate an overall rock mass quality. The definition of rock substance strength, fracture spacing, and fracture shear strength used in the method selection is presented in Table 2.

Table 2: Rock Mechanics Characteristics

1) Rock Substance Strength

(uniaxial strength[Pa]/overburden pressure [Pa])

weak:	<8
moderate:	8 - 15
strong:	>15

2) Fracture Spacing

	Fractures/m	(ft)	% RQD
very close:	>16	(>5)	0 - 20
close:	10 - 16	(3 - 5)	20 - 40
wide:	3 - 10	(1 - 3)	40 - 70
very wide:	3	(<1)	70 - 100

3) Fracture Shear Strength

weak:	clean joint with a smooth surface or fill with material whose strength is less than rock substance strength
moderate:	clean joint with a rough surface
strong:	joint is filled with a material that is equal to or stronger than rock substance strength

Rock substance strength is the ratio of the uniaxial compression strength to the overburden stress. The uniaxial compression strength can be estimated using the method originally presented by Terzaghi and Peck (1967), which was then modified by Deere (1968), Jennings and Robertson (1960), and Piteau (1970). However, a better estimate of the uniaxial compression strength could be obtained relatively inexpensively by using a point load testing machine. The overburden stress is determined from the depth and density of rock.

Fracture spacing can be defined in terms of fractures per meter or RQD, Rock Quality Designation (Table 2). RQD is the sum length of all pieces of core greater than or equal to two times the core diameter divided by the total length of a drill run. However, I believe the fractures per meter measurement is better because it provides a more quantitative description of the rock fragment size. Fracture shear strength is determined by observation (Table 2).

As part of the geologic log, one should estimate or measure the uniaxial compression strength and the fractures per meter, or RQD measurement, and the fracture shear strength. This data can then be interpreted on sections and level maps at the same scale as the geologic maps. The cumulative sum technique (Piteau and Russell, 1972) can be used to help define zones of similar rock substance strength, fracture spacing, and fracture strength. These maps, when overlain onto the geology and grade outline, will spatially define rock mechanics characteristics.

The use of any of the existing classification systems will also provide the data to determine the classes defined in Table 2.

Method Selection Process

Ten basic mining methods, not including hydraulic or solution mining, should be considered in any selection process:

1) Open pit - a method where mining starts at the surface and waste is removed to uncover the ore; includes strip mining and quarrying.
2) Block caving - a method in which columns of rock are undercut and cave under their own weight; the roof material is expected to cave as well; includes panel and continuous caving.
3) Sublevel stoping - a method of stoping in which the ore is blasted by benching, ring drilling, or long hole; most of the ore is drawn off as it is blasted, leaving an open stope.
4) Sublevel caving - an induced caving method in which the ore is blasted by ring drilling from drifts; overlying rock is expected to cave as the ore is drawn.
5) Longwall - a method in which the deposit, usually a coal seam, is removed in a continuous operation along a long working face; using an extensive series of props over the face and working areas; mined out areas usually cave.
6) Room-and-pillar - a method in which a grid of rooms is developed, leaving pillars, usually of uniform size, to support the roof; the pillars may or may not be removed at a later time;
7) Shrinkage stoping - a stoping method in which most of the blasted ore is left to accumulate in the stope until the stope is completely mined. The broken ore is then drawn off all at once.
8) Cut-and-fill - a stoping method in which each slice of rock is removed after blasting and is then replaced with some type of fill material, leaving space to mine the next slice.
9) Top slicing - a method in which staggered horizontal lifts are mined; the overlying rock is supported by a timber mat and the overlying rock is expected to cave.
10) Square-set - a method in which timber squares are formed to replace the rock mined and to support the surrounding rock; includes other timbered stoping methods, such as stull stoping.

Boshkov and Wright (1973), Morrison (1976), Laubscher (1977), and Tymshare, Inc. (1981) have presented schemes for selecting mining methods. Boshkov and Wright (1973) listed the mining methods possible for certain combinations of ore width, plunge of ore, and strength of ore. Morrison (1976) classified the mining methods into three basic groups, rigid pillar support, controlled subsidence, and caving; he then used general definitions of ore width, support type, and strain energy accumulation as the characteristics for determining mining method (Figure 1). Laubscher (1977) developed a detailed rock mechanics classification from which cavability, feasibility of open stoping or room and pillar mining, slope angles, and general support requirements could be determined. Tymshare, Inc. (1981) developed a numerical analysis that determines one of five mining methods, (1) open pit, (2) natural caving, (3) induced caving, (4) self-supporting, and (5) artificially supporting, and calculates the tonnage and grade for the type of deposit described. This method is meant to be used as a pre-feasibility tool for geologists.

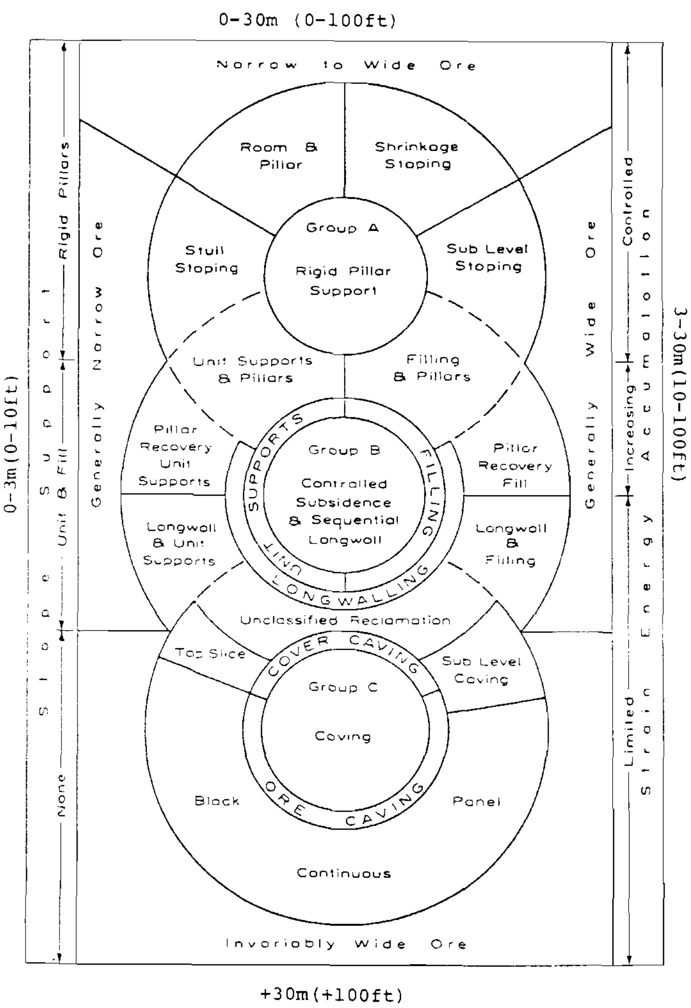

Figure 1: A Method Selection Scheme (after Morrison, 1976).

I propose a selection method which combines portions of all the above methods. The selection process has two steps: (1) determine the characteristics of the deposit, as defined in Tables 1 and 2; and (2) for each mining method, add up the values from Tables 3 and 4 for the combination of characteristics defined in Step 1.

Each mining method has been ranked as to the suitability of its geometry/grade distribution (Table 3), and ore zone (Table 4a), hanging wall (overlying wall rock) (Table 4b), and footwall (underlying rocks) rock mechanics characteristics (Table 4c). There are four ranks:

- preferred: the characteristic is preferred for the mining method;
- probable: if the characteristic exists, the mining method can be used;
- unlikely: if the characteristic exists, it is unlikely that the mining method would be applied, but does not completely rule out the method; and
- eliminated: if the characteristic exists, then the mining method could not be used.

The values used for each rank are listed in Table 5. Values for the eliminated rank were chosen so that if the sum of the characteristic values equalled a negative number, the method would be eliminated. A zero value was chosen for the unlikely rank because it does not add to the chance of using the method, but neither does it eliminate the method. The values used for probable and preferred were chosen so that the characteristics for one parameter could be ranked within a mining method and between mining methods.

Table 5: Rank Value

Ranking	Value
preferred	3 - 4
probable	1 - 2
unlikely	0
eliminated	-49

An example is provided to illustrate the steps in using this selection system and to point out problems with the system. The first step is to list the geometry/grade distribution and rock mechanics characteristics of the deposit (Table 6, column 1). The characteristic columns in Tables 3 and 4 are then identified for the deposit, and the values added up for the geometry/grade distribution, ore zone rock mechanics, hanging wall rock mechanics, and footwall rock mechanics for each mining method (Table 6, columns 2 and 3).

Table 3: Ranking of Geometry/Grade Distribution for Different Mining Methods

Mining Method	General Shape			Ore Thickness				Ore Plunge			Grade Distribution		
	M	T/P	I	N	I	T	VT	F	I	S	U	G	E
Open Pit	3	2	3	2	3	4	4	3	3	4	3	3	3
Block Caving	4	2	0	-49	0	2	4	3	2	4	4	2	0
Sublevel Stoping	2	2	1	1	2	4	3	2	1	4	3	3	1
Sublevel Caving	3	4	1	-49	0	4	4	1	1	4	4	2	0
Longwall	-49	4	-49	4	0	-49	-49	4	0	-49	4	2	0
Room & Pillar	0	4	2	4	2	-49	-49	4	1	0	3	3	3
Shrinkage Stoping	2	2	1	1	2	4	3	2	1	4	3	2	1
Cut & Fill	0	4	2	4	4	0	0	0	3	4	3	3	3
Top Slicing	3	3	0	-49	0	3	4	4	1	2	4	2	0
Square Set	0	2	4	4	4	1	1	2	3	3	3	3	3

	M = Massive T/P = Tabular or Platy I = Irregular	N = Narrow I = Intermediate T = Thick VT = Very Thick	F = Flat I = Intermediate S = Steep	U = Uniform G = Gradational E = Erratic

The three groups of rock mechanics characteristics should be totaled. This total should then be added to the geometry/grade distribution sum (Table 7). Using the above type of characteristic grouping, one can see which grouping(s) reduce the chance of using a particular mining method, or, for cases where the total sum is nearly equal, one can determine which characteristics are the most suitable for the mining method.

After the mining methods have been ranked (Table 8), based on geometry/grade distribution and rock mechanics characteristics, there may be a number of methods which appear suitable.

In our example, the open pit method is the obvious choice from a geometry and rock mechanics characteristics point of view. The next four methods, block caving, top slicing, square-set, and cut-and-fill, are grouped together. It is worthwhile at this time to look at the ranking of all the mining methods by individual characteristics (Table 9). Examination of Table 9 reveals that the choice of a mining method involves compromise. For example, cut-and-fill would be a good method from the rock mechanics point of view, but it has the worst geometry/grade distribution characteristics, whereas top slicing has one of the worst rock mechanics characteristics, but its geometry/grade distribution characteristics are considered the best.

It would not be reasonable to move directly to Stage 2 at this point, since preparing detailed mine plans for all applicable methods delineated in Stage 1 would be extremely time-consuming and costly.

Continuing with our example, the five methods with similar total values should be examined generally in terms of mining costs.

Although all five methods were ranked as applicable, mining costs may be significantly different for each method. Morrison (1976) has ranked the mining methods by increasing unit mining cost, which I have modified slightly, as follows:

1) open pit
2) block caving
3) sublevel stoping
4) sublevel caving
5) longwall
6) room-and-pillar
7) shrinkage stoping
8) cut-and-fill
9) top slicing
10) square-set

On the basis of relative operating cost, the methods would be ranked as follows:

1) open pit
2) block caving
3) cut-and-fill
4) top slicing
5) square-set

Based on this simplified ranking by mining cost, I would evaluate open pit and block caving first. Cut-and-fill would then be considered if neither of these two methods proved feasible.

Table 4: Ranking of Rock Mechanics Characteristics for Different Mining Methods

Key:

Rock Substance Strength

W = Weak
M = Moderate
S = Strong

Fracture Spacing

VC = Very Close
C = Close
W = Weak
VW = Very Weak

Fracture Strength

W = Weak
M = Moderate
S = Strong

4a: Ore Zone

Mining Method	Rock Substance Strength			Fracture Spacing				Fracture Strength		
	W	M	S	VC	C	W	VW	W	M	S
Open Pit	3	4	4	2	3	4	4	2	3	4
Block Caving	4	1	1	4	4	3	0	4	3	0
Sublevel Stoping	-49	3	4	0	0	1	4	0	2	4
Sublevel Caving	0	3	3	0	2	4	4	0	2	2
Longwall	4	1	0	4	4	0	0	4	3	0
Room & Pillar	0	3	4	0	1	2	4	0	2	4
Shrinkage Stoping	1	3	4	0	1	3	4	0	2	4
Cut & Fill	3	2	2	3	3	2	2	3	3	2
Top Slicing	2	3	3	1	1	2	4	1	2	4
Square Set	4	1	1	4	4	2	1	4	3	2

4b: Hanging Wall

Mining Method	Rock Substance Strength			Fracture Spacing				Fracture Strength		
	W	M	S	VC	C	W	VW	W	M	S
Open Pit	3	4	4	2	3	4	4	2	3	4
Block Caving	4	2	1	3	4	3	0	4	2	0
Sublevel Stoping	-49	3	4	-49	0	1	4	0	2	4
Sublevel Caving	3	2	1	3	4	3	1	4	2	0
Longwall	4	2	0	4	4	3	0	4	2	0
Room & Pillar	0	3	4	0	1	2	4	0	2	4
Shrinkage Stoping	4	2	1	4	4	3	0	4	2	0
Cut & Fill	3	2	2	3	3	2	2	4	3	2
Top Slicing	4	2	1	3	3	3	0	4	2	0
Square Set	3	2	2	3	3	2	2	4	3	2

4c: Footwall

Mining Method	Rock Substance Strength			Fracture Spacing				Fracture Strength		
	W	M	S	VC	C	W	VW	W	M	S
Open Pit	3	4	4	2	3	4	4	2	3	4
Block Caving	2	3	3	1	3	3	3	1	3	3
Sublevel Stoping	0	2	4	0	0	2	4	0	1	4
Sublevel Caving	0	2	4	0	1	3	4	0	2	4
Longwall	2	3	3	1	2	4	3	1	3	3
Room & Pillar	0	2	4	0	1	3	3	0	3	3
Shrinkage Stoping	2	3	3	2	3	3	2	2	2	3
Cut & Fill	4	2	2	4	4	2	2	4	4	2
Top Slicing	2	3	3	1	3	3	3	1	2	3
Square Set	4	2	2	4	4	2	2	4	4	2

Table 6: Example of Numerical Method Selection Process

Geometry/Grade Distribution	(Column 1)	(Column 2)	(Column 3)	
		open pit	block caving	etc.
		(values from Table 3)		
General shape:	tabular or platey	2	2	
Ore thickness:	very thick	4	4	
Ore plunge:	flat	3	3	
Grade distribution:	uniform	3	4	
depth (used later):	130 m (425 ft)	--	--	
		12	13	
Rock Mechanics Characteristics		(values from Table 4)		
Ore Zone				
Rock substance strength:	moderate	4	1	
Fracture spacing:	close	2	4	
Fracture strength:	moderate	3	3	
		9	8	
Hanging Wall				
Rock substance strength:	strong	4	1	
Fracture spacing:	wide	4	3	
Fracture strength:	moderate	3	2	
		11	6	
Footwall				
Rock substance strength:	moderate	4	3	
Fracture spacing:	close	2	3	
Fracture strength:	weak	2	1	
		8	7	

Table 7: Example - Characteristics Values Totaled for Different Mining Methods

Mining Method	Geometry/Grade Distribution	Rock Mechanics Characteristics				Grand Total
		Ore	HW	FW	Total	
Open Pit	12	9	11	8	28	40
Block Caving	13	8	6	7	21	34
Sublevel Stoping	10	5	7	2	14	24
Sublevel Caving	13	7	6	3	16	29
Longwall	-37	8	5	6	19	-18
Room & Pillar	-38	7	8	3	18	-20
Shrinkage Stoping	10	6	6	8	20	30
Cut & Fill	7	8	7	10	25	32
Top Slicing	15	6	6	7	19	34
Square Set	8	8	7	10	25	33

Table 8: Ranking Results

Total Points	Method
40	open pit
34	block caving
34	top slicing
33	square-set
32	cut-and-fill
30	shrinkage stoping
29	sublevel caving
24	sublevel stoping
-20	room-and-pillar
-18	longwall

Having narrowed the preferred mining methods to two, each should now be generally examined in terms of mining rate, labor availability, environmental concerns, and other site-specific considerations, in order to determine whether these parameters will eliminate any method from further consideration.

Mining rate should be dictated by the mining method chosen and the size of the deposit. However, in instances where a mill already exists in the area, a production rate that is perhaps higher or lower than that dictated by the least costly mining method may be required. Therefore, a compromise must be made.

Other factors affecting the mining method selected would be the market for the resource being mined and the available labor pool. If the labor pool is large and unskilled, a method that is highly mechanical or technical and requires skilled personnel should not be chosen, of course. Environmental concerns are more and more becoming a controlling factor in method selection. Also, the environmental conditions underground must be considered. Whether or not subsidence is permitted can determine what methods are feasible.

LCDs

native land

China

Remember, the purpose of this numerical method selection system is not to choose the final mining method. It is intended to indicate those methods that will be most effective given the geometry/grade distribution and rock mechanics characterisitics, and which will require more detailed study in Stage 2. If nothing else, this selection system will allow miners/engineers to consider what characteristics are important for the mining methods being considered.

METHOD SELECTION - STAGE 2

The purpose of Stage 2 in the method selection process is to lay out general mining plans for those methods delineated in Stage 1, determine cut-off grades, and then calculate minable reserves so that economic analyses can be made in order to determine which mining method will provide the greatest return on investment.

It is not the purpose of this paper to discuss determination of cut-off grade or minable reserves. Rock mechanics characteristics of the deposit that are critical for mine planning

Table 9: Ranking of Mining Methods by Each Characteristic

Geometry/Grade Distribution	Ore	HW	FW	Rock Mechanics Total	Grand Total
top = 15	pit = 9	pit = 11	c&f = 10	pit = 28	pit = 40
bcv = 13	bcv = 8	r&p = 8	sqs = 10	c&f = 25	bcv = 34
scv = 13	lng = 8	sst = 7	pit = 8	sqs = 25	top = 34
pit = 12	c&f = 8	c&f = 7	shs = 8	bcv = 21	sqs = 33
sst = 10	sqs = 8	sqs = 7	bcv = 7	shs = 20	c&f = 32
shs = 10	scv = 7	bcv = 6	top = 7	lng = 19	shs = 30
sqs = 8	r&p = 7	scv = 6	lng = 6	top = 19	scv = 29
c&f = 7	shs = 6	shs = 6	scv = 3	r&p = 18	sst = 24
lng = -37	top = 6	top = 6	r&p = 3	scv = 16	lng = -18
r&p = -38	sst = 5	lng = 5	sst = 2	sst = 14	r&p = -20

pit = open pit
bcv = block caving
sst = sublevel stoping
scv = sublevel caving
lng = longwall
r&p = room & pillar
shs = shrinkage stoping
c&f = cut & fill
top = top slicing
sqs = square set

and selecting a mass mining method will be discussed.

Rock Mechanics Data

In order to estimate cavability of a deposit, stope widths, pillar sizes, and slope angles, more rock mechanics data is required for Stage 2 than for Stage 1. Most of this additional data should have been collected at the same time as the data for Stage 1. Design of pit slopes and underground openings depends largely on the geology of the area, the strength of the rock mass, and the pre-mine stress. Strength of the rock mass is a function of the strength of the intact rock, the strength of the geologic structures (joints, faults, etc.), and the characteristics of the geologic structure (orientation, length, spacing, etc.). Once the geologic structure data are available, potential failure geometries can be defined and stability analyses can be made using the strength properties.

Strength Properties. Basic strength properties needed for Stage 2 of the method selection process are uniaxial compression strength, stiffness (Young's Modulus), Poisson's ratio, tensile strength, intact rock shear strength, natural fracture shear strength, and fault gouge shear strength. Rock units, such as salt, shales, etc., may require creep testing under controlled temperature and humidity.

All the strength properties, except perhaps the fault gouge strength, can be measured using unsplit drill core specimens. The number of specimens required for representative testing depends somewhat on variability of the rock unit; however, three to six samples per rock type per test type should be sufficient for Stage 2. During drilling, unsplit core samples must be saved for rock testing. We recommend collecting three samples per rock type per test type per drill hole (Call, 1979). By sampling each hole, a collection of samples will be built up, from which samples for testing can be selected.

Geologic Structure. Rock mass strength also depends largely on the characteristics of the geologic structures, orientation, spacing, length, strength, etc. Fracture shear strength has already been discussed in the rock strength section. For Stage 2 of the method selection, areas with similar joint orientations are defined as structural domains; distribution of the fracture set characteristics and potential failure paths are defined for each domain.

Geologic structures are divided into two categories: major structures and rock fabric. Major structures are faults, folds, dikes, etc., which have lengths on the order of the deposit size and are usually considered individually in design. Rock fabric is predominantly joints and faults that have a high frequency of occurrence and are not continuous.

Structural data can be obtained by using detail line mapping (Call et al., 1976), cell mapping, or oriented core mapping. Detail line mapping is a technique that involves the measurements of fracture characteristics of all joints which intersect a line. This mapping technique is a spot sample within a structural domain; it provides the data for determining distribution of joint set characteristics on a joint-by-joint basis. Cell mapping, which involves measuring the mean orientation and fracture characteristics for each fracture set within a 10 m to 15 m (30 ft to 50 ft) wide cell, can be done by the geologist during his mapping of surface and underground rock exposures. This method provides the data needed to evaluate variability in geologic structure on an areal basis and is, thus, a means of delineating structural domains.

Cell mapping and detail line mapping are used in those instances where some type of rock exposure exists. However, in cases in which structure data can be obtained only from drill core, a few oriented core holes should be included in the drilling program. Oriented core holes provide the same information as detail line mapping, except that oriented core data will not provide joint length characteristics. The oriented core data can, also, aid the geologist in his interpretation of the geology.

Pre-mine Stress. Pre-mine stress is one of the most difficult parameters to determine. Because of the complex tectonics associated with many mineral deposits, the stress field will probably be variable, depending on proximity to the nearest major geologic structure. Techniques such as stress-relief overcoring and hydrofracturing are available, but they are generally expensive and diffcult to justify until the feasibility of mining the deposit has been established. The pre-mine stress field can be estimated using the geologic history, orientation of geologic structures, and type of fault movement (Abel, personal communication). Although this method is indirect and could be misleading about the pre-mine stress field, it is probably better to use it or assume a hydrostatic stress field than to assume the elastic theory.

Hydrology. Hydrologic conditions can affect strength properties of the rock, as well as the cost of mining. Information needed includes a water table map, location of water sources, and locations of geologic structures that would be water-bearing. Because a pump test would provide a quantitative estimate of the pumping requirements necessary during mining, one should be made.

Rock Mechanics Input for Selection of Mass Mining Methods

If the engineer has the necessary information, as discussed above, he can provide realistic estimates on size of openings, support requirements, cavability, and slope angles for selecting a mining method. Attempting to determine these parameters will enable the engineer to see which data is critical in the analysis or is lacking; therefore, when development starts or further exploration is in progress, the data collection program can be properly set-up.

Open pit. Although this symposium is concerned primarily with underground mass mining methods, the open pit method should be considered during the method selection. At what depth of overburden to go underground is primarily a function of the mineral value and the stripping ratio. Using a method similar to that presented by Soderberg (1968), an estimate of the maximum stripping ratio for a given mineral value was calculated (Figure 2). The mineral value is a function of the market price and the cut-off grade. In order to estimate stripping ratio, the slope angle and the limit of the ore zone in section are needed (Soderberg, 1968). Slope angle can have major impact on the stripping ratio; consequently, rather than simply using a 45° slope angle, the most realistic slope angle should be determined from the available data. An assessment of the final slope angles can be made by defining potential failure geometries from the orientation of the geologic structures and then choosing a slope angle that minimizes the number of daylighted structures. If shear strength, length, and spacing data are available, a stability analysis can be made. With the estimates of the mineral value and the stripping ratio, whether an open pit method should be considered can be determined (Figure 2).

Block caving. During Stage 2, the cavability of the deposit should be examined in greater detail than during Stage 1. Once the cavability is determined, the minimum drawpoint spacing, supportable drift size, and subsidence limit should also be determined.

The cavability of a deposit is determined by the fragment size distribution at the drawpoint and the undercut width required to sustain a cave. If the fragment size is coarse, the undercut width may be greater than the width of the deposit, or the drawpoints will be plugged much of the time, thereby reducing mining rate and increasing secondary blasting cost.

A two-dimensional fragment size analysis was developed by White, Nicholas & Marek (1977). The analysis results in a distribution of fragment size based on fracture spacing, but it does not include the effects of attrition. However, by comparing fragment size distribution with existing caving deposits, using the same fragmentation analysis (Figure 3), cavability of the deposit being examined can be determined. The fragment size distribution curve can be generated from detail line data or from fracture per foot data (Table 10). Details of the analysis can be found in White (1977). Because the analysis is two-dimensional, orientation of the drill holes or cross-sections analyzed should be considered. The fragmentation can also be evaluated using RQD and the cavability index (Figure 4) or Laubscher's Rock Mass Reading System (1977).

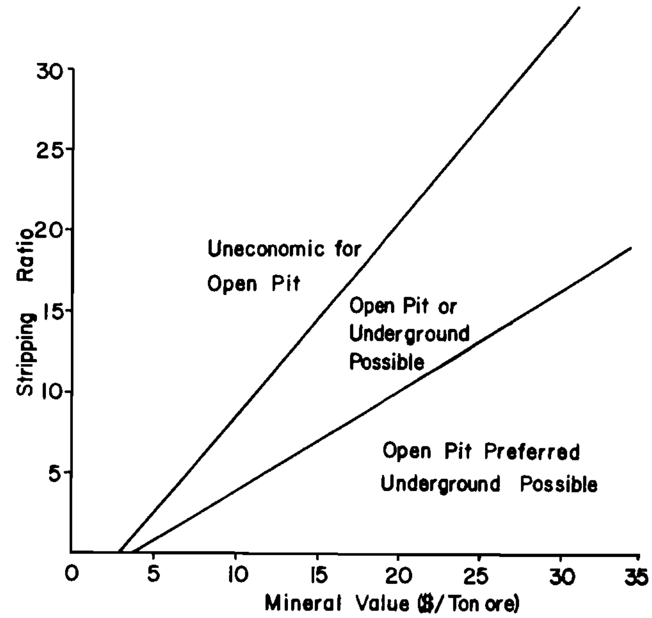

Figure 2: Stripping Ratio vs. Mineral Value.

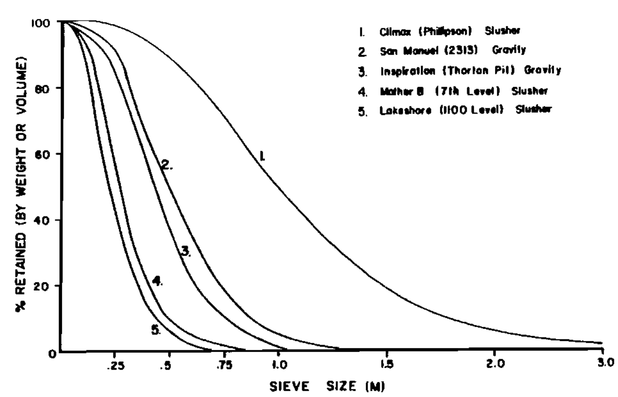

Figure 3: Fragment Size Distribution Curves of some Existing Block Caving Mines (after White, 1977).

NOTE: Data are from limited areas and do not necessarily represent an average for that mine.

Table 10:

$$\text{Percent retained at size } X = \frac{V_f}{V_t}$$

$$\text{where } V_t = \text{total volume} = \frac{6N}{B^3}$$

$$V_f = \text{volume greater than size } X$$

$$= N6\ e^{-BX}\ \frac{X^3}{6} + \frac{X^2}{2B} + \frac{X}{B} + \frac{1}{B^3}$$

B = (1/fracture spacing)*$\sqrt{2}$;
N = number of fragments in sample; and
X = fragment size to be analyzed.

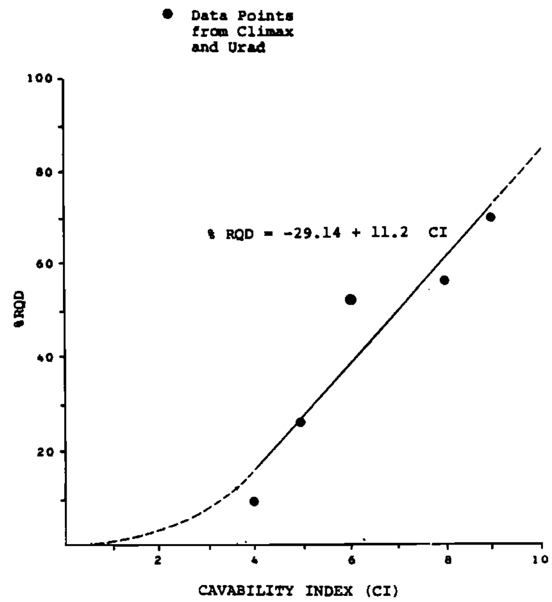

Figure 4: RQD vs. Cavability Index (after McMahon and Kendrick, 1959).

Undercut width required to sustain a cave is most critical for those deposits where the fragmentation is coarse and the average undercut width of the deposit is less than approximately 150 m (500 ft). Using Laubscher's classification (1977) or the pressure arch concept (Alder et al., 1951), the undercut width required to sustain a cave can be estimated. Laubscher provides an hydraulic radius, area/perimeter, for his five classes of rock. In the pressure arch concept, the rock is considered to have a maximum distance that it can transfer the load (Figure 5). The ability of the rock to transfer a vertical stress in a lateral direction over an underground opening depends on the shear strength of the rock, the horizontal stress, and the strength of the rock pillars. Although each deposit has its own maximum transfer distance, a correlation between depth and maximum transfer distance has been determined (Figure 6). Based on the pressure arch concept, if the undercut width does not exceed twice the maximum transfer distance then only the rock under the pressure arch (Figure 5a) has the potential for caving. However, the maximum transfer distance can be reduced by some type of boundary weakening.

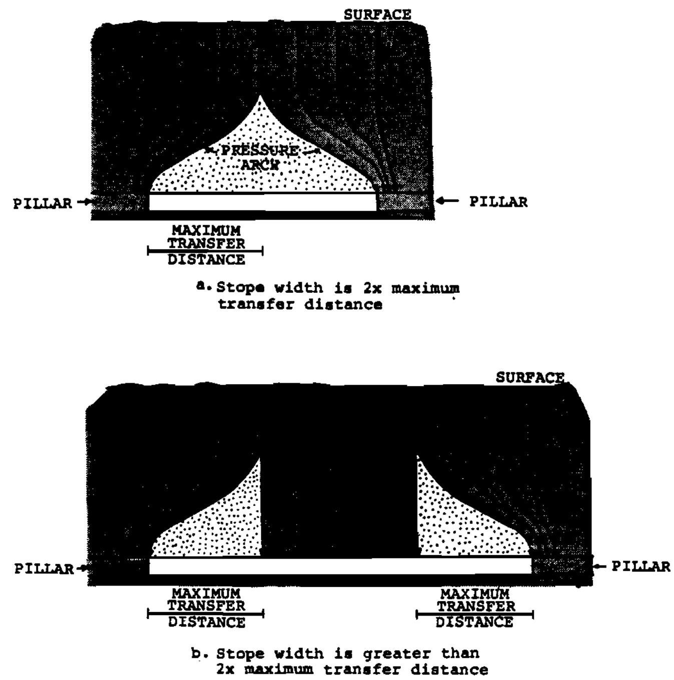

Figure 5: Pressure Arch Concept.

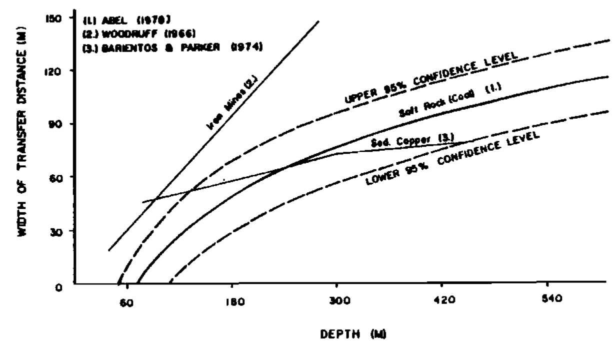

Figure 6: Transfer Distance vs. Depth.

Once it has been determined that the deposit is cavable, drawpoint spacing and gathering drift size should be determined for the general mine design.

Drawpoint spacing is primarily a function of the ore and overlying waste fragment size distribution and the pillar strength. The general consensus has been that the smaller the fragment size the narrower the width of draw, consequently, the closer the drawpoint spacing. Also, when the overlying material is more fragmented than the ore, the drawpoint spacing should be closer in order to minimize dilution. However, comparison of existing properties indicates the correlation between fragment size and draw width area is weak (Figure 7); especially considering the indications from the Henderson Mine where the ore is moderately to well fragmented, the drawpoint spacing is wide, 12.2 m X 12.2 m (40 ft X 40 ft), and the ore recovery appears to be good. The ground between the drawpoints can be considered a pillar (Figure 8), and, if analyzed as such, it can be used to determine the minimum drawpoint spacing. The load on the pillar is the most difficult parameter to determine. The worst loading condition occurs when the undercut is within 100 ft of the pillar and the rock is being loaded by the abutment stresses. Kendorski (1975) estimates that this abutment loading is two times the overburden stress, while Panek (1978) estimates that it is three times the overburden stress. Using three times the tributary-area-load to determine load on the pillar and Wilson's (1972) pillar analysis to determine load carrying capacity, a minimum drawpoint spacing can be estimated. Using the fragmentation curves and the graph in Figure 7 and the pillar analysis, an estimate of the drawpoint spacing can be made.

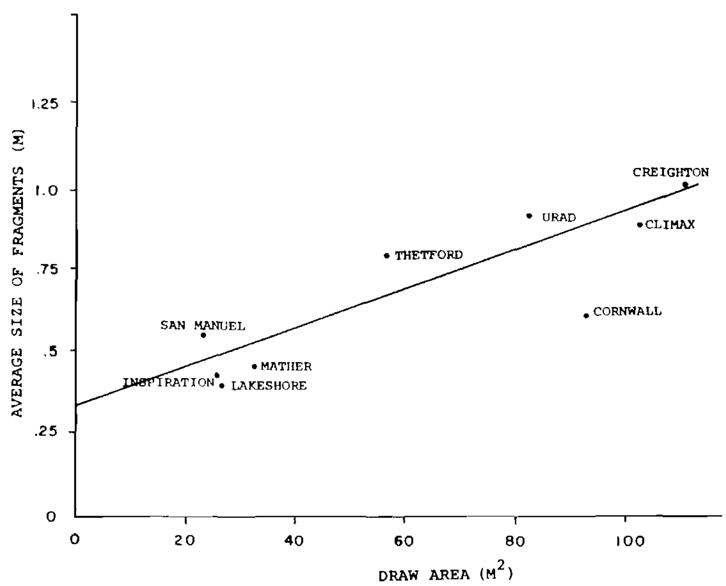

Figure 7: Fragment Size vs. Draw Area (after White, 1979).

The ore gathering drift size and support required are important in estimating cost of the mining method. The drifts should be oriented so as to minimize potential failure geometries, which are usually normal to the strike of the predominant structures. Laubscher (1977) and Barton and Lunde (1974) have correlated their rock classifications to support requirements. Because Barton's work was primarily on tunnels, which generally have more support than a drift in a mining operation, his work may not be applicable to

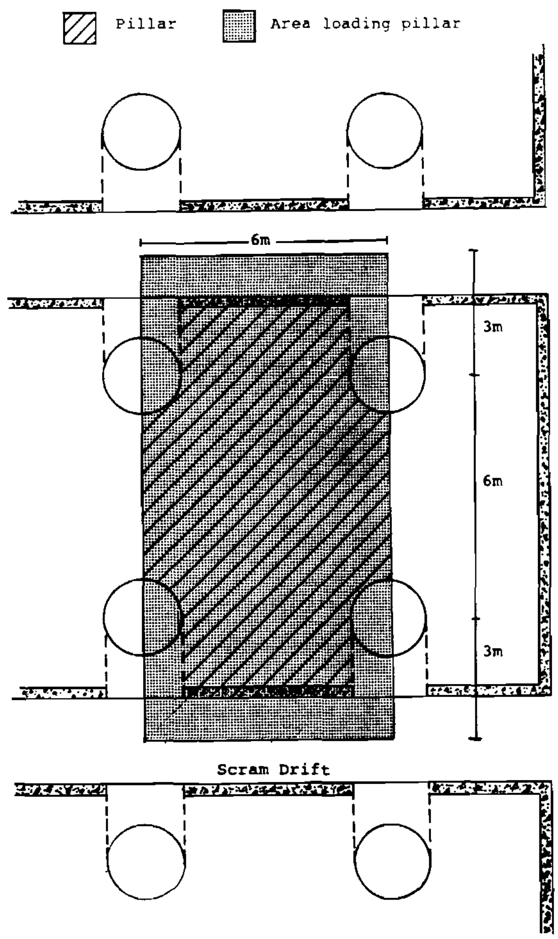

Figure 8: Definition of Pillar between Drawpoints.

determining support requirements for a mine. If one of these classification systems is used, the rock classes can be interpreted on the level maps, where the gather and haulage drifts are shown. From these level maps, the percent of area that the different support systems will be required can be determined and the support cost estimated. If neither of these systems have been used but information on the orientation, spacing, shear strength, and possibly length of the joint sets and fault systems is available, the support required for the drifts can be determined by (1) identifying potential failure geometries, (2) determining the load at the edge of the opening, and (3) determining which support system(s) can carry the load calculated in step 2.

The subsidence limit should be defined for locating buildings and shafts which are to last the life of the deposit. In the absence of a major geologic structure, a 45° angle projected onto the surface from the bottom of the ore zone is usually considered the closest to the deposit one should locate long-term facilities. However, most actual ground movement takes place within a 60° angle from the deposit. If a major fault exists, it will probably control the limit of subsidence.

Stoping. The two important parameters in the economics of a stoping method for which a rock mechanics study can provide estimates are the width of the stopes and the size of the pillars.

In sublevel stoping, the width of a stope is a function of the immediate and intermediate roof (Alder and Sun, 1968). The immediate roof is characterized by the pressure arch concept already discussed. The maximum stope width is twice the maximum pressure arch. Pillars spaced this distance must be able to carry tributary-area-load. The immediate roof is that ground under the pressure arch which will behave as beam, plate, or arch. Joint orientation, spacing, and length can be used to define the stope width. In many instances, the beam developed by bolting can be used. The pillars within twice the maximum transfer distance do not have to carry tributary-area-load, but rather the load under the pressure arch, halfway to the next support. Using Wilson's pillar analysis (1972) and the potential failure geometries through the pillar, the pillar load carrying capacity can be determined (Nicholas, 1976).

For shrinkage stoping, the same type of analysis needs to be made as for sublevel stoping, except that the cavability of the overlying rock has to be evaluated. Support requirements can be estimated, as discussed under block caving.

Sublevel caving. For sublevel caving, rock mechanics data on the cavability of the hanging wall, the sublevel drift size, the support needed, and the spacing between the sublevel drifts is required. Janelid and Kvapil (1966) have presented guidelines for the layout of a sublevel mine. The hanging wall must come in behind the ore zone; otherwise sublevel caving will not work. Using analyses similar to those in block caving will provide an estimate of the dimension needed to initiate the cave and the fragment size distribution. Janelid and Kvapil also related drift size to the required width of draw. Another aspect of a sublevel design is the support required for these drifts. If extensive support is required, the method may not be feasible. Support requirements can be estimated, as discussed previously.

Vertical spacing of drifts is mainly a function of equipment, but the horizontal spacing between drifts is determined by the width of the draw ellipsoid and the stability of the rock. Janelid and Kvapil related drift spacing to the distance between sublevels and the eccentricity of the ellipsoid (Figure 9). The ground between the drifts can be considered pillars (Figure 10) and analyzed as such.

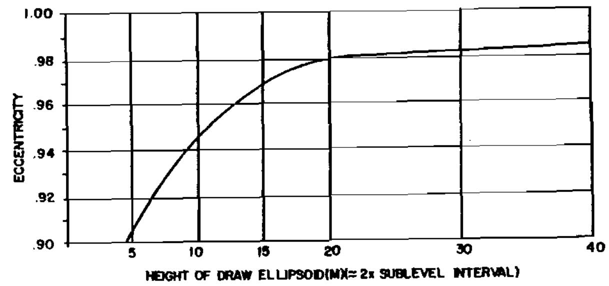

Figure 9: Eccentricity vs. Height of Draw Ellipsoid (after Janelid and Kvapil, 1966).

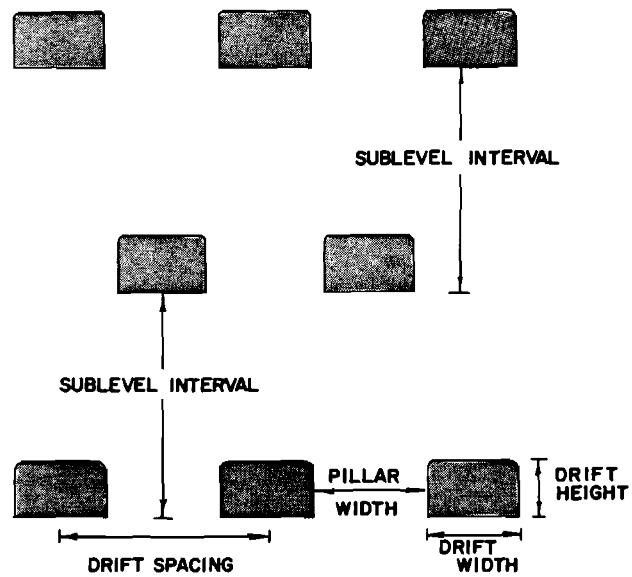

Figure 10: Sublevel Caving Geometry.

The worst load condition occurs for the ground nearest the cave. There is some abutment loading occurring, which can be estimated, and the stability of pillars can be determined.

Concluding Comments

Mining method selection should be based primarily on the geometry and grade distribution of the deposit, the rock mechanics characteristics of the ore zone, hanging wall and footwall, and on the mining and capitalization cost, with first priority given to the rock mechanics characteristics. Selection of the mining method should occur in two stages.

Stage 1: Define the geometry/grade distribution and rock mechanics characteristics of the deposit and rank the mining methods according to their ability to accommodate these characteristics.

Stage 2: Develop an initial mine plan of the two or three highest ranking mining methods to provide a better estimate of the mining and capitalization cost and to determine cut-off grade and minable reserves.

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