NORTHWEST MINING ASSOCIATION SHORT COURSE
MINE FEASIBILITY - CONCEPT TO COMPLETION

Geotechnical Considerations

by

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SECTION 1

GEOTECHNICAL DATA REQUIRED FOR A FEASIBILITY STUDY

Introduction

A rock mechanics analysis for a feasibility study requires the same basic data for both an underground mine or an open pit; however, some data is more critical to one method than to the other.

The two test cases used for this short course provide some of the basic geology but do not provide the numerical data needed to make an appropriate evaluation. In the past, once the reserves were determined, most open pits were mined at a 45° slope, and the method used throughout the district was chosen for underground mines. Today, however, the large capital investments required to open new mines make it imperative that the mining method chosen enhance the probability of attaining the projected production rates. Consequently, in order to perform numerical analyses, appropriate data is needed during the early stages.

This is not to say that experience and engineering judgment should be ignored, rather that they should be used in conjunction with numerical analyses.

The amount of data required is a function of the accuracy required for the feasibility and complexity of the geology. These parameters are difficult to determine until the drilling has been completed; therefore, all data that can be collected from drill core should be done as part of the ore definition drilling program. The basic parameters needed are interpreted geologic sections and level maps, joint set characteristics of the ore zone, hanging wall, and footwall, intact rock, fracture and fault shear strengths of the different rock types, the pre-mine stress conditions, and the hydrologic conditions.

Geology

Basic geologic interpretation is of major importance in any mineral evaluation. Interpreted 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. The 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, to make it easier to visualize the relative layout of mine workings. The area included on the maps should extend horizontally beyond the limit of the orebody in all directions 1.75 times the depth. Although an area this size may seem exces-

1.75 times the depth. 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.

Geologic Structure

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 and are included as part of the geologic maps. 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.

Strength Properties

The rock strengths needed are the intact rock shear strength, the fracture shear strength, and the fault gouge shear strength. For slope design studies, primary strengths needed are those for the fracture and fault gouge; however, for an underground analysis all three are needed.

The intact rock shear strength is obtained from uniaxial and triaxial compression tests. As part of the uniaxial compression testing, Poisson's Ratio and Young's Modulus should be measured. The intact rock shear stregth can be estimated if only disk tension and uniaxial compression tests are performed. The natural fracture and fault gouge shear strengths are determined by performing direct shear tests on natural fractures and fault gouge.

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 an initial feasibility study. 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.

Pre-mine Stress

Pre-mine stress is one of the most difficult parameters to determine. It is more critical to an underground design than it is to a slope design. 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 difficult 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. To provide a quantitative estimate of the pumping requirements necessary during mining, a pump test should be made.

Mining Method Selection

Determination of the appropriate mining method is one of the most difficult aspects of a feasibility study. The first decision to make is whether the deposit should be open pit, underground, or a combination of the two. The best approach is obviously the one with the best return on investment. If both underground and open pit are feasible, then a feasibility study of each approach is required.

A quick evaluation that can be made prior to the feasibility study is a comparison of stripping ratio to mineral value (Figure 1-1), which will indicate whether open pit mining is economical.

These curves were generated by calculating the breakeven stripping ratio using a method similar to that presented by Soderberg (1968). To use Figure 1-1 you must estimate your recoverable mineral value and determine from your sections the breakeven stripping ratio and plot it on the graph. Because the stripping ratio is sensitive to slope angle, a quick and dirty analysis should be performed rather than just using a 45° angle.

If the results of the open pit versus underground evaluation indicate that underground mining is the best way to recover the ore, then the appropriate mining method must be determined. The parameters that must be examined when choosing an underground 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.

In a feasibility study, mining method selection should be at least a two-stage affair. The first stage is mainly the elimination of those methods that are not obviously applicable, by using some type of classification system; e.g., Nicholas, 1981; Laubscher, 1981; Boshkav and Wright, 1973; Morrisson, 1976. The remaining possible mining methods can then be ordered, based on general mining cost and other site-specific considerations: environmental conditions, required production rates, and market conditions, for example. With this ranking we can go on to the second stage. This involves making a preliminary layout of the two most probable mining methods in order to calculate the mining cost and capitalization from which a cut-off grade can be determined and a minable reserve calculated. As part of the mine layout, rock mechanics would be

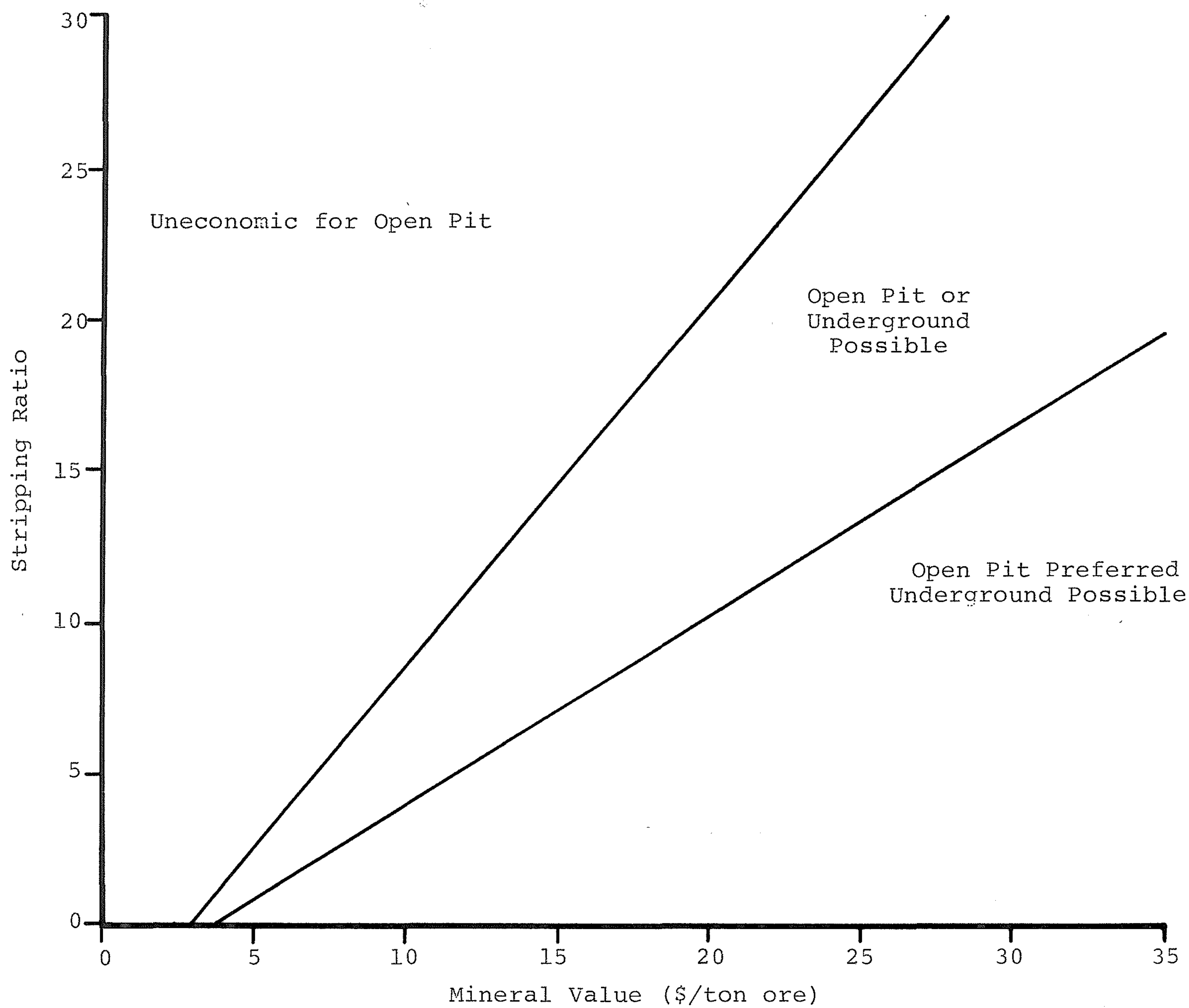

Figure 1-1: Quick Method to Evaluate Feasibility of Open Pit vs Underground.

used to evaluate the required size of openings, types and amount of support, caving characteristics, and expected subsidence. During the mine planning stage, problems with the chosen methods might be encountered, at which time modifications could be made.

After the mining method(s) for the feasibility study have been determined, the rock mechanics evaluation is similar for both underground and open pit. Figure 1-2 is a flow chart of how a rock mechanics analysis should progress. The extent of the work would be determined by the accuracy required in the feasibility study and the complexity of the geology.

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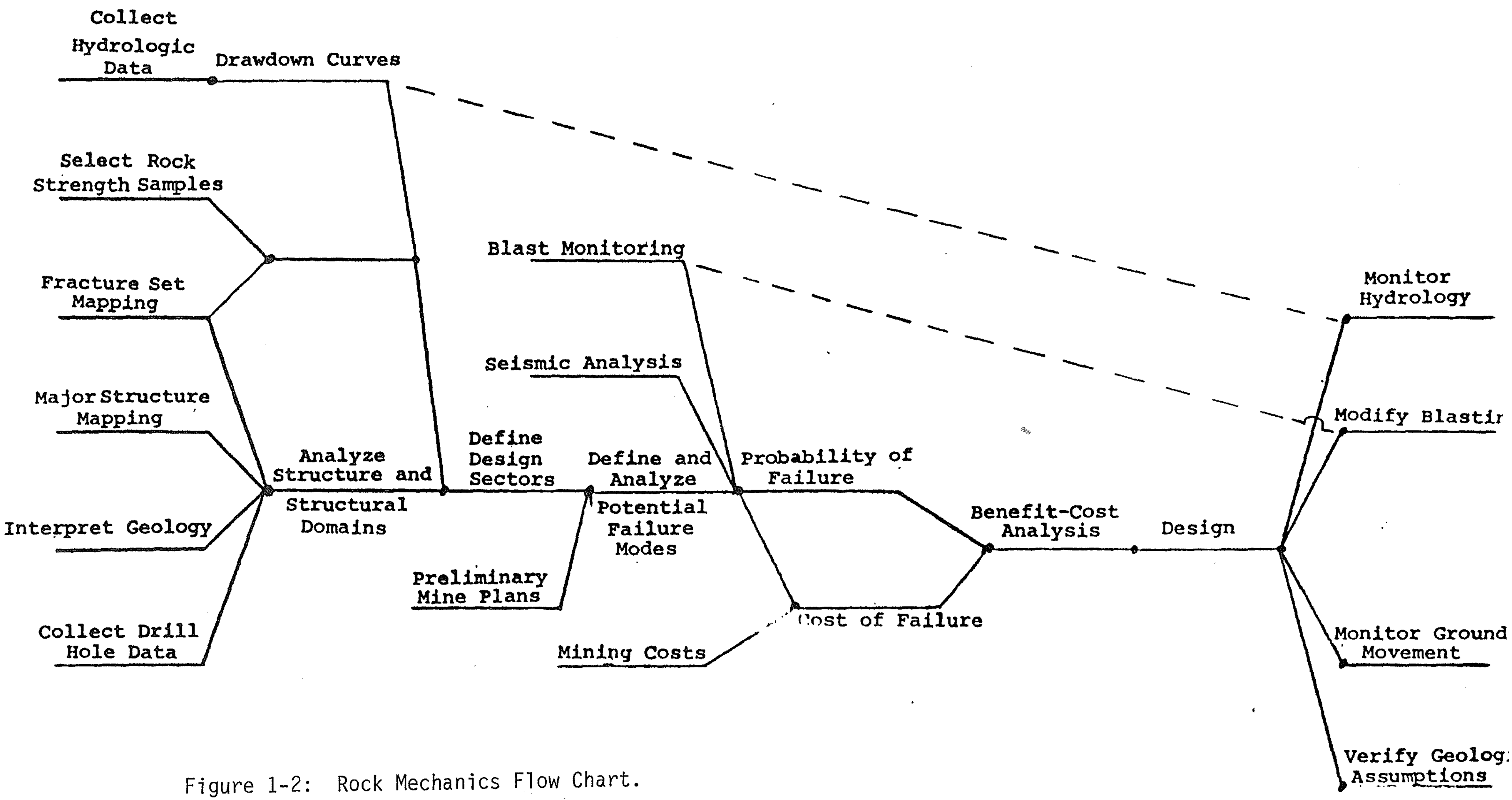


Figure 1-2: Rock Mechanics Flow Chart.

SECTION 2

GEOTECHNICAL INPUT TO THE FEASIBILITY STUDY OF THE ZINC PENNY DEPOSIT

As indicated earlier, additional data would have or should have been collected during the drilling program. For this case study, typical data for this type of deposit has been selected, based on personal observations of the Pre-Cmbrian Belt Series and data from similar rock units.

Base Data

The cross section provided indicates that there are three basic rock units, quartzites, argillites, and ore, and that the only major structure is a near vertical fault that has displaced the ore zone. The drill hole data indicates there are smaller faults with the same orientation as the one indicated, and these smaller faults have a 100 ft spacing, are slickensided but with little fault gouge, and have displacements of 1 to 5 ft.

As part of the drilling program, the RQD (Rock Quality Designation) or fracture spacing and rock hardness were collected. Based on the RQD data, cross sections were generated (Figure 2-1) showing the distribution of RQD values. Personally, I would prefer using fracture spacing because RQD can be misleading; however, RQD has become widely accepted and is commonly used for comparison purposes. At the zinc penny deposit, all of the rock is predominantly hard, so a hardness section is somewhat meaningless; however, in deposits with alteration zones, the hardness is useful in identifying relative differences in rock strengths between the alteration zones. The RQD plot does identify the width of the fractured ground on either side of the fault.

Rock fabric data of the quartzites and argillites were obtained from surface outcrops and from an oriented core hole. For the quartzite and argillite four joint sets were identified: parallel to bedding, normal to bedding (cross joint), at right angles to the strike of bedding (vertical set) and parallel to the major fault. Figure 2-2 shows these joint sets on a lower hemisphere Schmidt plot, and Table 2-1 summarizes the mean orientation, spacing, and lengths of these sets. For the ore zone, only two joint sets were identifiable: parallel to the bedding and parallel to the major fault system. Figure 2-2 shows these sets on a Schmidt plot, and Table 2-1 summarizes the joint set characteristics.

From the drill core, a limited rock testing program provided the results listed in Table 2-2.

The pre-mine stress is based on the most recent tectonic activity, which is the major fault. Dr. Abel's empirical correlation predicts the following stresses:

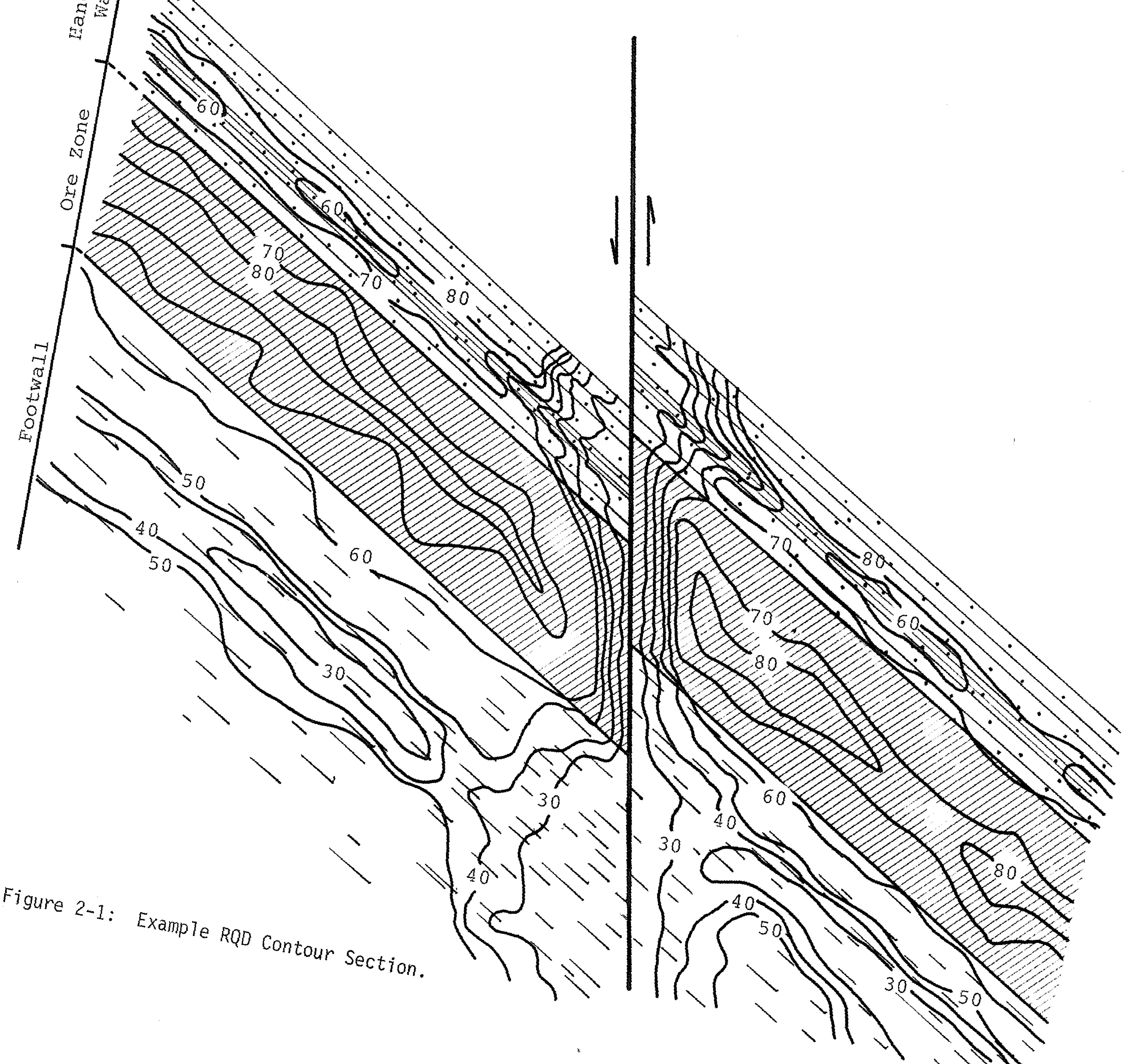


Figure 2-1: Example RQD Contour Section.

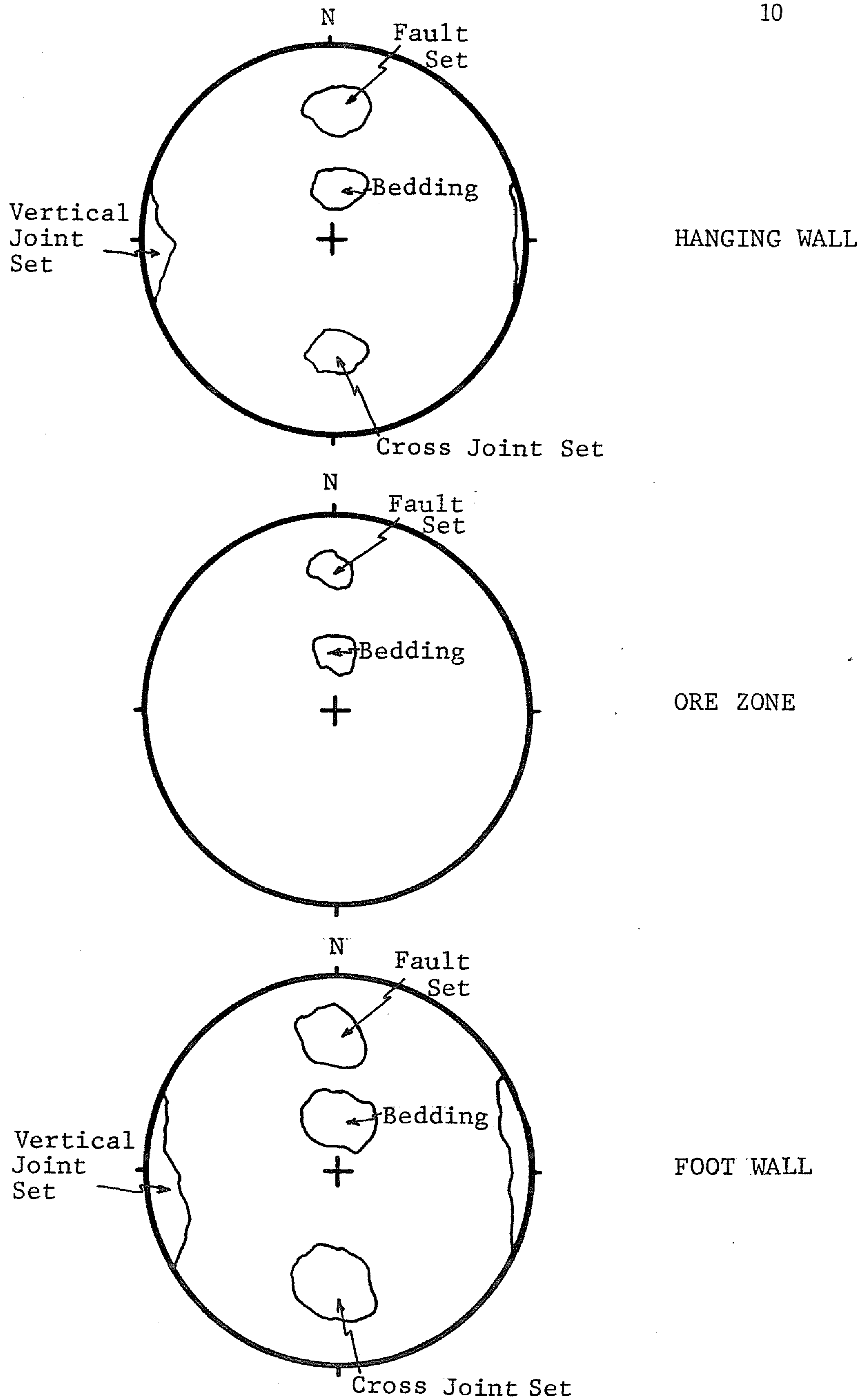


Figure 2-2: Lower Hemisphere Schmidt Plots of Joint Sets.

Table 2-1: Rock Fabric Properties

Joint Set	Mean Dip Direction (deg)	Mean Dip (deg)	Mean Spacing (ft)	Mean Length (ft)
		Hanging Wall Quartzites		
Bedding	180	30	3.0	8
Cross-Joint	0	60	5.0	3
Vertical	90	88	5.0	4
Fault	180	80	3.0	3
		Footwall Argillites		
Bedding	180	30	0.4	2
Cross-Joint	0	60	0.8	.5
Vertical	90	88	2.0	2
Fault	180	80	1.0	2
		Ore Zone		
Bedding	180	30	1.8	ND
Fault	180	80	2.5	ND

Table 2-2: Rock Strengths

Intact Rock Strength

Rock Type	Mean Uniaxial Compression (psi)	Mean Friction Angle (deg)	Mean Cohesion (psi)
Quartzite	19,500	52	3400
Ore	13,800	43	3000
Argillite	3,200	37	800

Fracture and Fault Gouge Shear Strengths

Rock Type	Mean Friction Angle(deg)	Mean Cohesion (psi)
Quartzite	28	22
Ore	33	18
Argillite	21	5
Fault (peak)	18	12

Stress	Orientation		Magnitude
	Bearing	Plunge	
Major	90°	0°	$1.5 - 2 \times \sigma_{ovb}$
Intermediate	0°	10°	$1 - 1.5 \times \sigma_{ovb}$
Minimum	180°	80°	σ_{ovb}

where σ_{ovb} = density x depth

Unless disking had been observed in the drill core, we would assume that the three stresses were all near equal in magnitude, but would consider the orientation of stress in our analysis.

The hydrologic data provided is a flow of 1000 gpm, which is the expected flow rate once the mine has reached full production. The water pressure is not known, but assuming that the beds daylight on surface and that the water table is near surface, the water pressure at depth could be up to 650 psi, which is significant, especially during the early stages of the mine operation.

Method Selection

As already indicated, method selection is basically a process of elimination where the obviously unfeasible methods are eliminated and a method or combination of methods is determined from the remaining method(s). Using the numerical approach for method selection (Nicholas, 1981), the geometry of the deposit was first evaluated (Table 2-3) based on the information below.

Shape:	tabular
Ore Thickness:	intermediate (60 ft)
Plunge:	intermediate (30°)
Grade Distribution:	uniform

Using these parameters, the ranking values were calculated (Table 2-4). The rock mechanics characteristics were evaluated (Table 2-3) based on the following characteristics.

	Rock Substance Strength	Fracture Spacing	Fracture Strength
Quartzite	moderate	very wide	moderate
Ore	moderate	very wide	moderate
Argillite	weak	close	moderate

The ranking values were then calculated for these characteristics (Table 2-4). Although square set and top slicing were considered, they did not rank significantly higher than any of the other methods. These two methods have significantly higher mining costs than all of the other methods; therefore, they were

Table 2-3: Classification for Mining Method Selection
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) Grade Distribution

uniform:	the grade at any point in the deposit does not vary significantly from the mean grade for that deposit
gradational:	grade values have zonal characterisitcs, 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 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

Table 2-4: Numerical Classification Ranking

Method	Geometry (16 pts max)	Rock Mechanics Characteristics (12 pts max) HW	Ore	FW	Combined Total (52 pts max)
Cut and Fill	14	7	7	12	40
Square Set	12	7	5	12	36
Room and Pillar	10	9	9	4	32
Shrinkage Stoping	8	4	9	7	28
Top Slicing	8	4	9	7	28
Sublevel Stoping	8	9	9	1	27
Sublevel Caving	9	5	9	3	26
Block Caving	8	2	4	8	22

excluded, resulting in the following ranking.

cut and fill
room and pillar
shrinkage stoping
sublevel stoping

At this point, the rock mechanic needs to interact with the mine planner to incorporate operational considerations in developing an actual mining method. This was done for this project, resulting in consideration of two mining methods:

1) a longhole blasting with delayed fill, and
2) a mechanized post-pillar cut and fill.

The method selection and consequent mine plan require constant interaction between the mine planner and rock mechanic to ensure that decisions on layout to optimize operations do not cause ground stability problems. However, for this case study, Mr. Pacey of Redpath and I were only able to discuss the layout during an initial meeting.

Our discussion on analysis will be specific to these two methods; however, the basic questions that must be addressed in most any mining method are:

1) what is the best orientation for drifts and stopes;
2) how wide an area can be opened with limited support or how wide an area must be undercut in order to cave;
3) what support is needed in drifts and/or stopes;
4) what pillar size is needed; and
5) will there be subsidence and if so how much?

Drift and Stope Orientation

The best orientation is dependent on the dominant structure and stress orientation. The preferred orientations, based on the geologic structures for drifts, are with the long axis normal to the strike of the dominant joint sets. However, for stopes the long axis should be parallel to the strike of the dominant joint sets. In addition, the long axis of drifts and stopes should be aligned parallel to the principal stress direction if possible. If the prefered orientation is different for the two situations, I generally prefer to use the structural control unless there is convincing information that the principal stress is significantly greater than, >1.5 times, the overburden pressure.

For the Zinc Penny deposit, the preferred orientation of the long axis of the major drifts should be north-south, ±30°, i.e., normal to the strike of bedding and the major fault system. Drifts in other orientations will certainly have to be driven;

however, they may require more support. The preferred long axis of stopes should be in an east-west direction, ±30°, i.e., parallel to the strike of bedding and the major fault system. If stopes have to be oriented in some other direction, then one dimension would have to be modified or the amount of support increased.

Width of Stopes

Width of opening is primarily a function of the fault spacing and orientation and joint set characteristics, and the extent of support that can be afforded. By generating a fracture section(s) and determining the area of material that would be expected to fail (Figures 2-3 and 2-4), the appropriate stope span can be predicted. Repeating the process of determining the failure area for different opening widths, a probability of failure area being less than a certain amount can be calculated (Figures 2-5a and 2-6a). An estimate of the tonnage was made assuming a unit length; however, the analysis is primarily intended to identify the width at which the amount of failure increases significantly. Dividing the failure area by the width of opening normalizes the different widths and allows determination of the point at which the amount of failure shows a significant increase Figures (2-5b and 2-6b).

From this analysis, the expected height of failure is also obtained to aid in designing a support system.

Pillar Size

The appropriate pillar size depends on loading conditions. Some pillars are expected to carry the full load to the surface for nearly the life of the mine while other pillars provide only a temporary support. Therefore, we usually generate a series of curves for various expected pillar dimensions and then evaluate expected loading conditions after meeting with the mine planner. The method used to calculate the load-carrying capacity is A. H. Wilson's (1972) method, which is based on a confined core carrying most of the load and the outer skin of the pillar yielding. The longhole blasting methods being considered would require a long pillar in the range of 50 ft wide and 69 ft high (vertical). Figure 2-7 shows the load-carrying capacity (LCC) for a long pillar at various pillar widths. For the post-pillar cut and fill method, the pillars are likely to be square, between 10 and 30 ft wide. As part of the post-pillar method, the height at which failure should occur is not well-defined; therefore, the LCC curves were calculated for various pillar widths and heights (Figure 2-8).

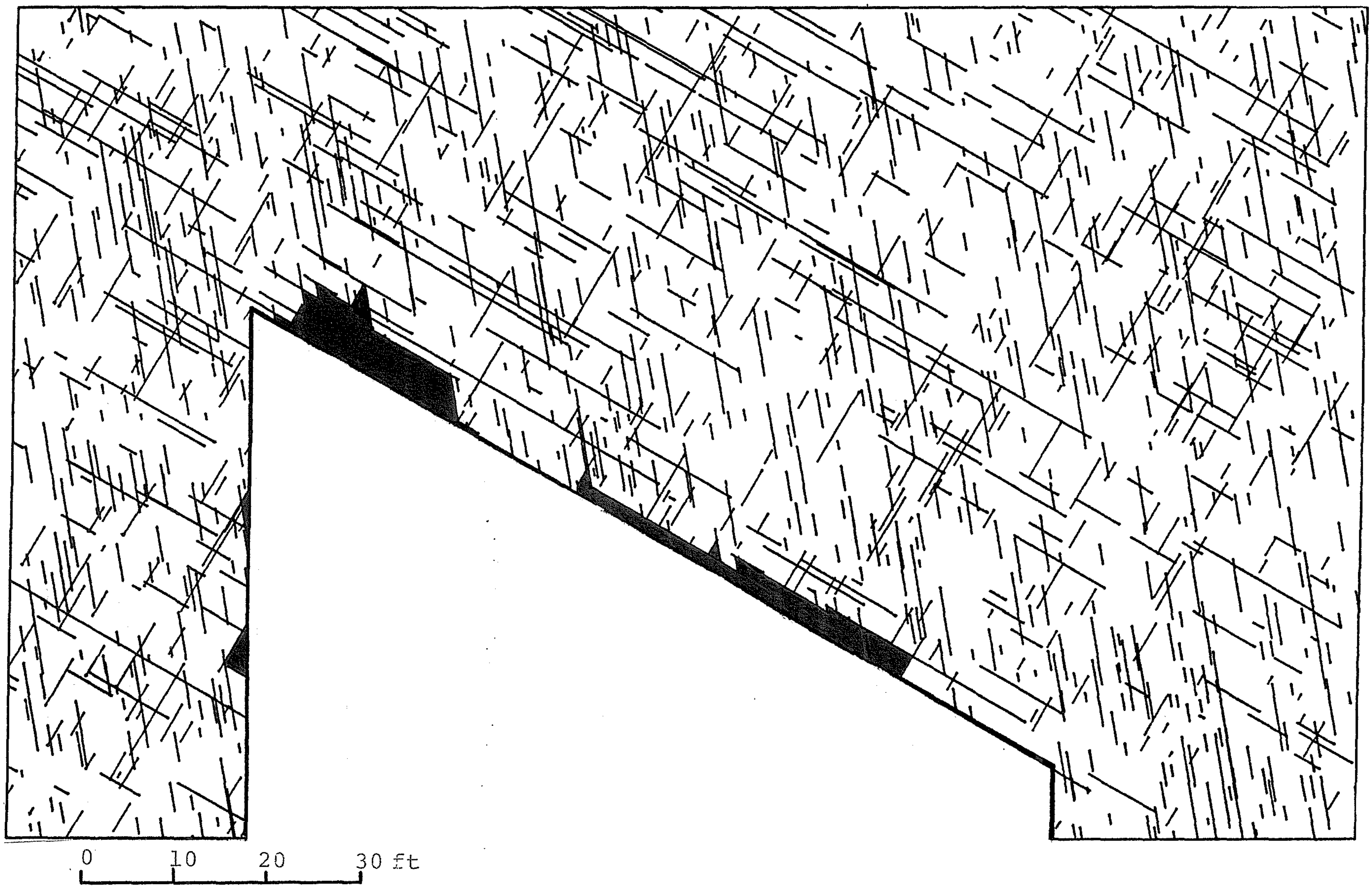


Figure 2-3: Quartzite Fracture Cross Section Looking Down Strike of Bedding with an Angled Back (Longhole Blasting Method).

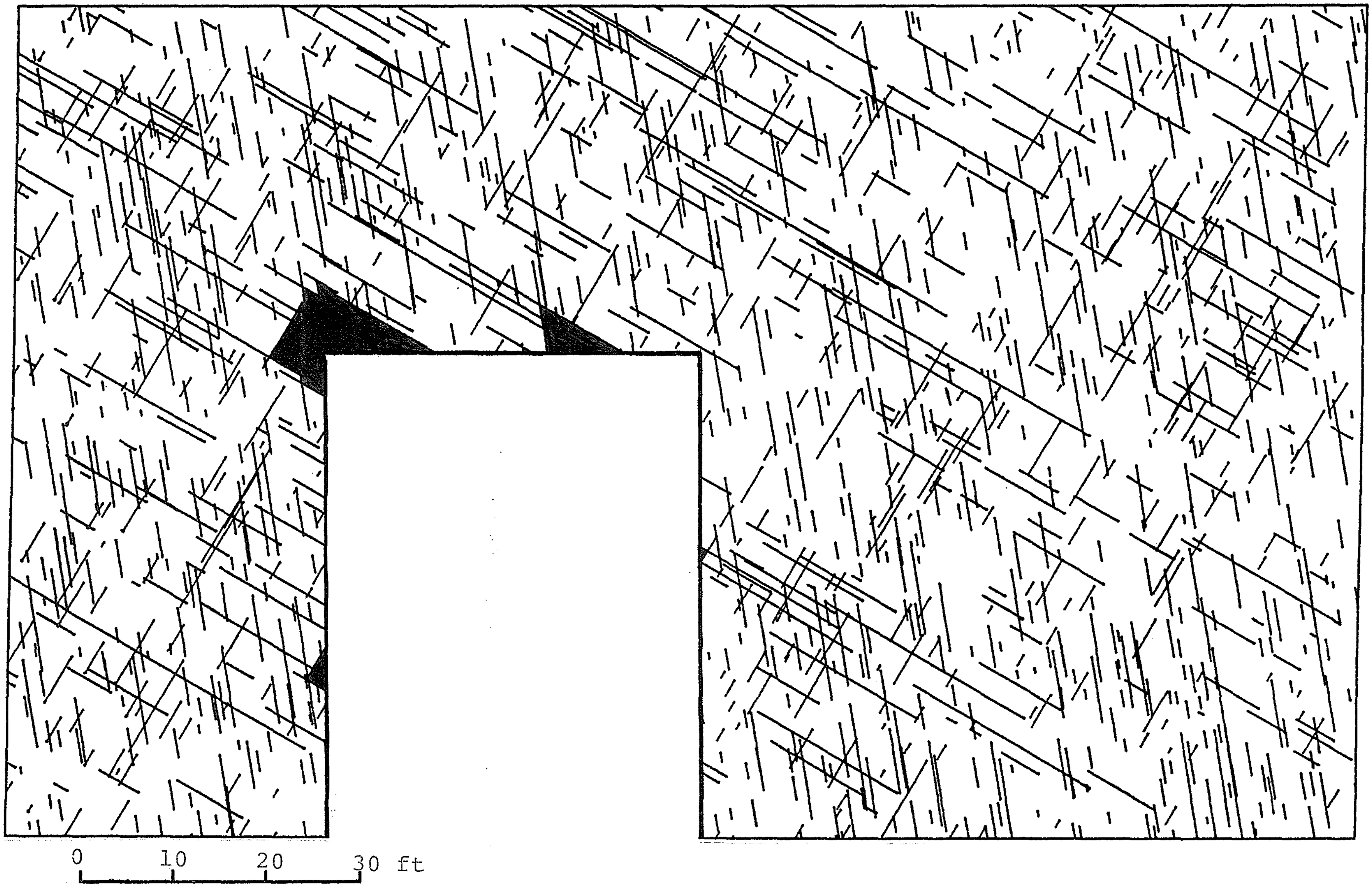
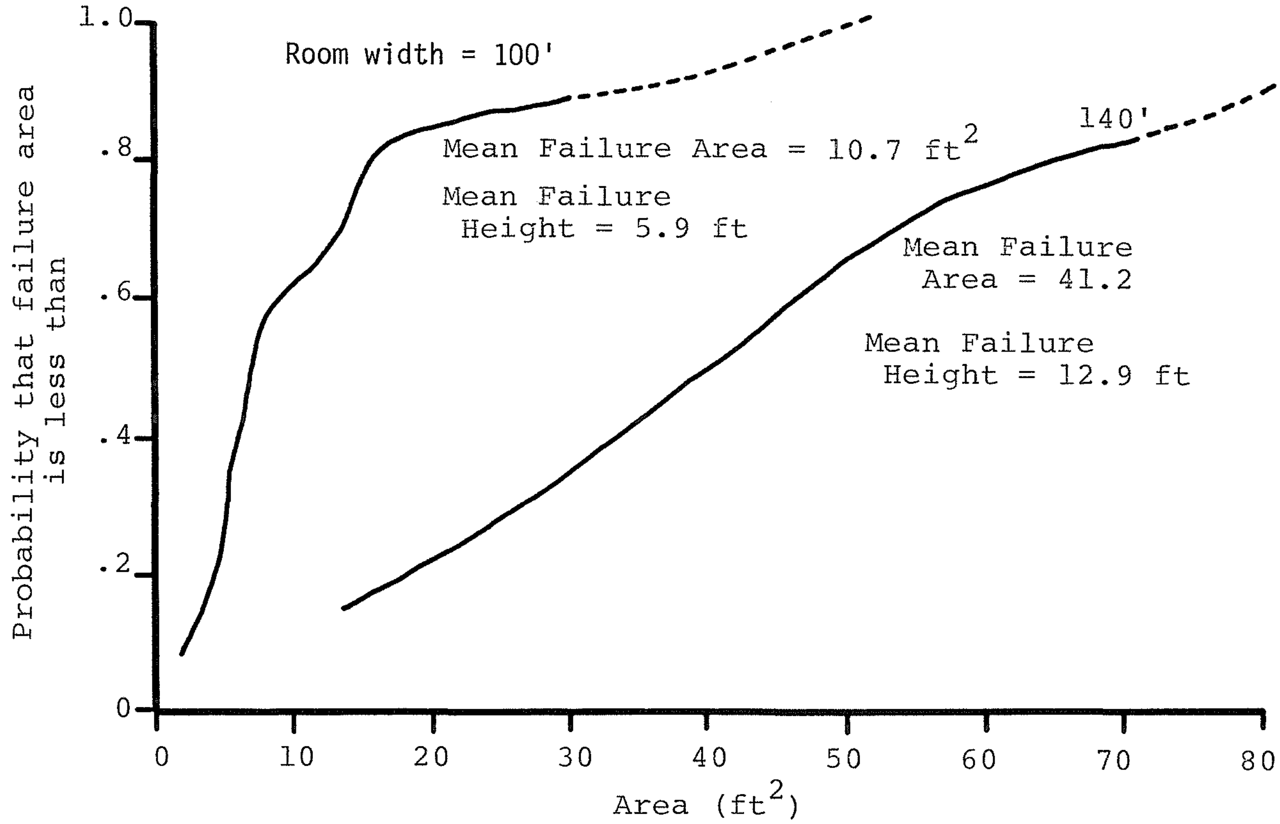


Figure 2-4: Quartzite Fracture Cross Section Looking Down Strike of Bedding with Flat Back (Post-Pillar Cut and Fill).



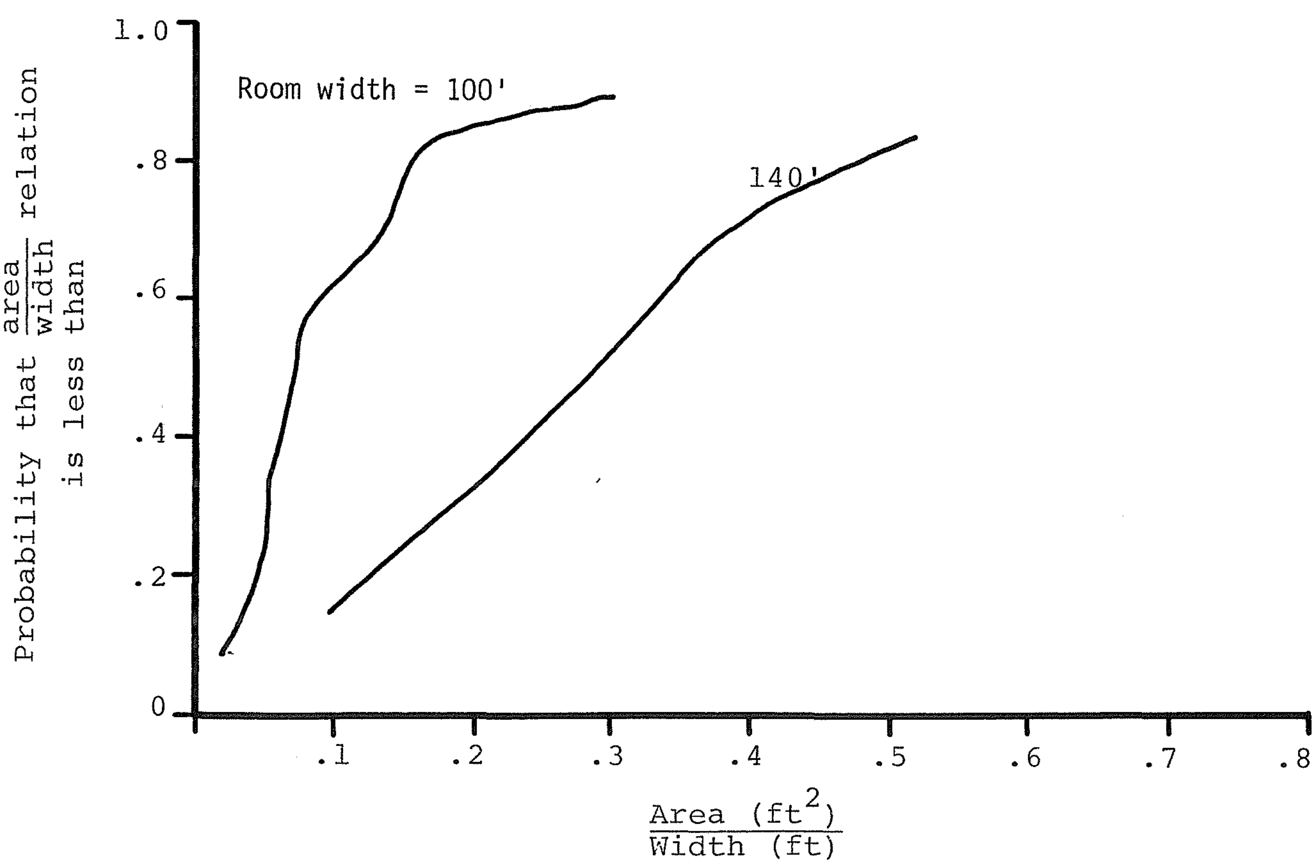


Figure 2-5: Probability Distribution of Failure Areas for Various Stope Spans with an Angled Back (Longhole Blasting Method).

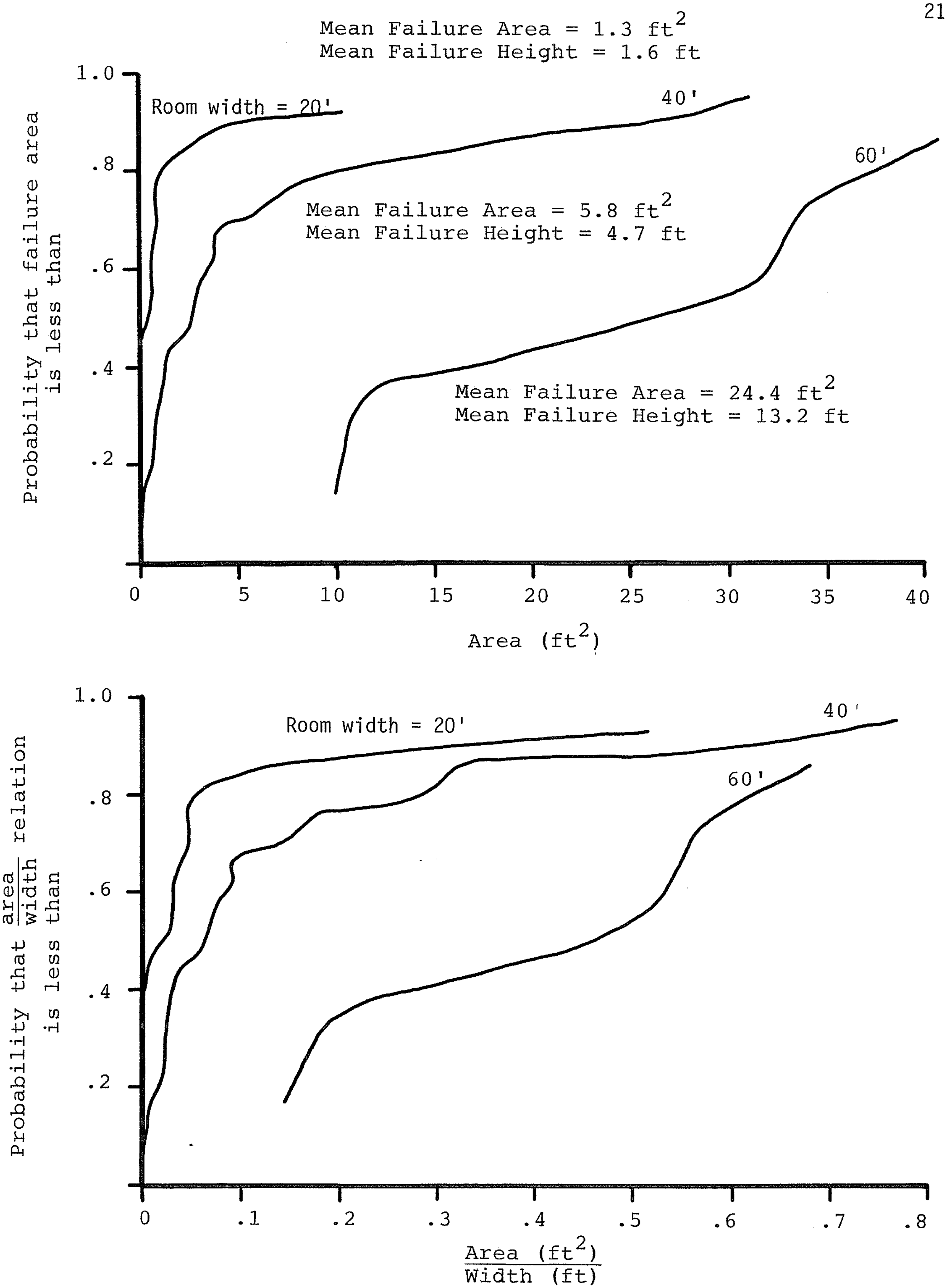


Figure 2-6: Probability Distribution of Failure Area for Various Stope Spans with a Flat Back (Post-Pillar Cut and Fill).

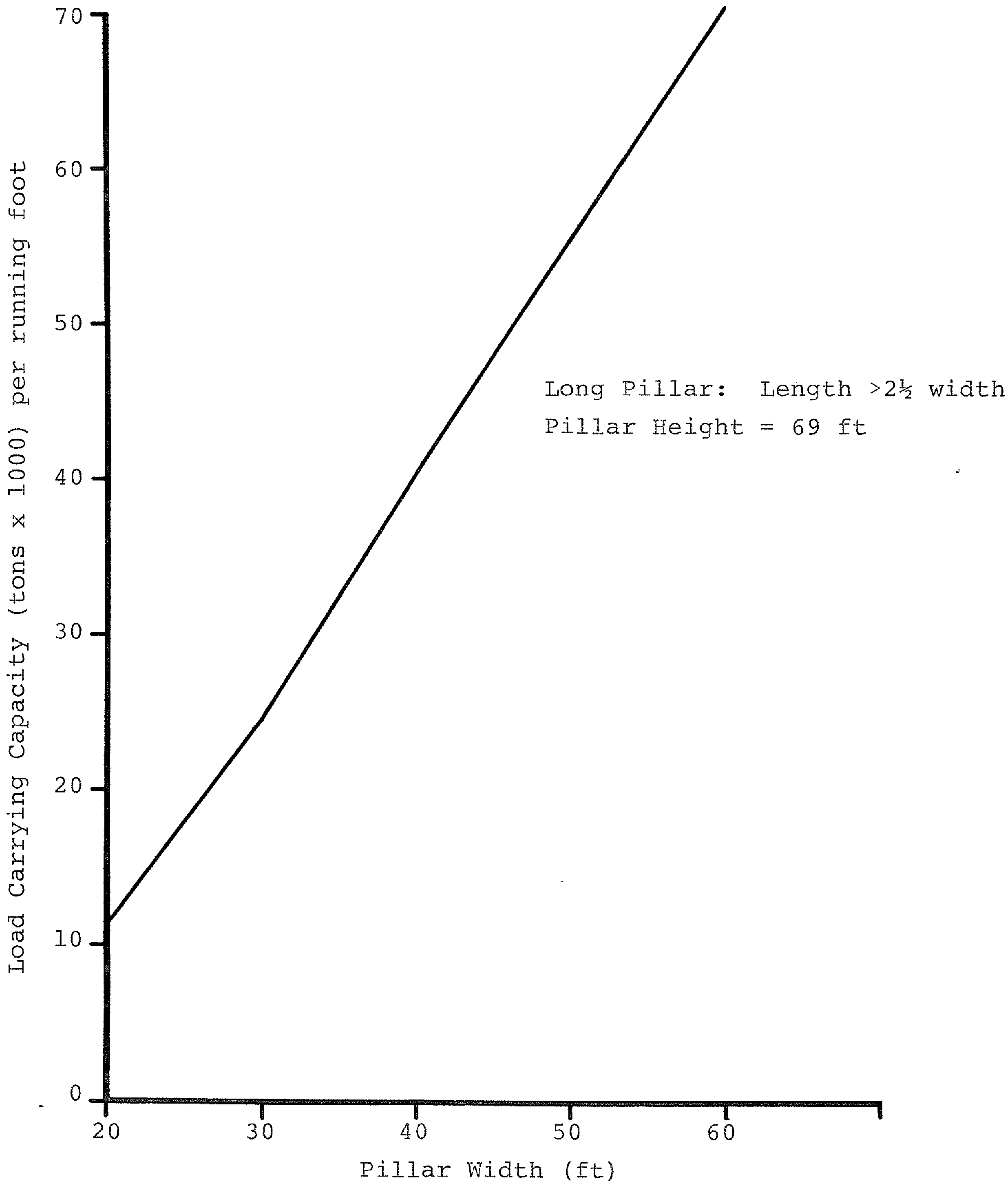


Figure 2-7: Load Carrying Capacity for Various Pillar Widths.

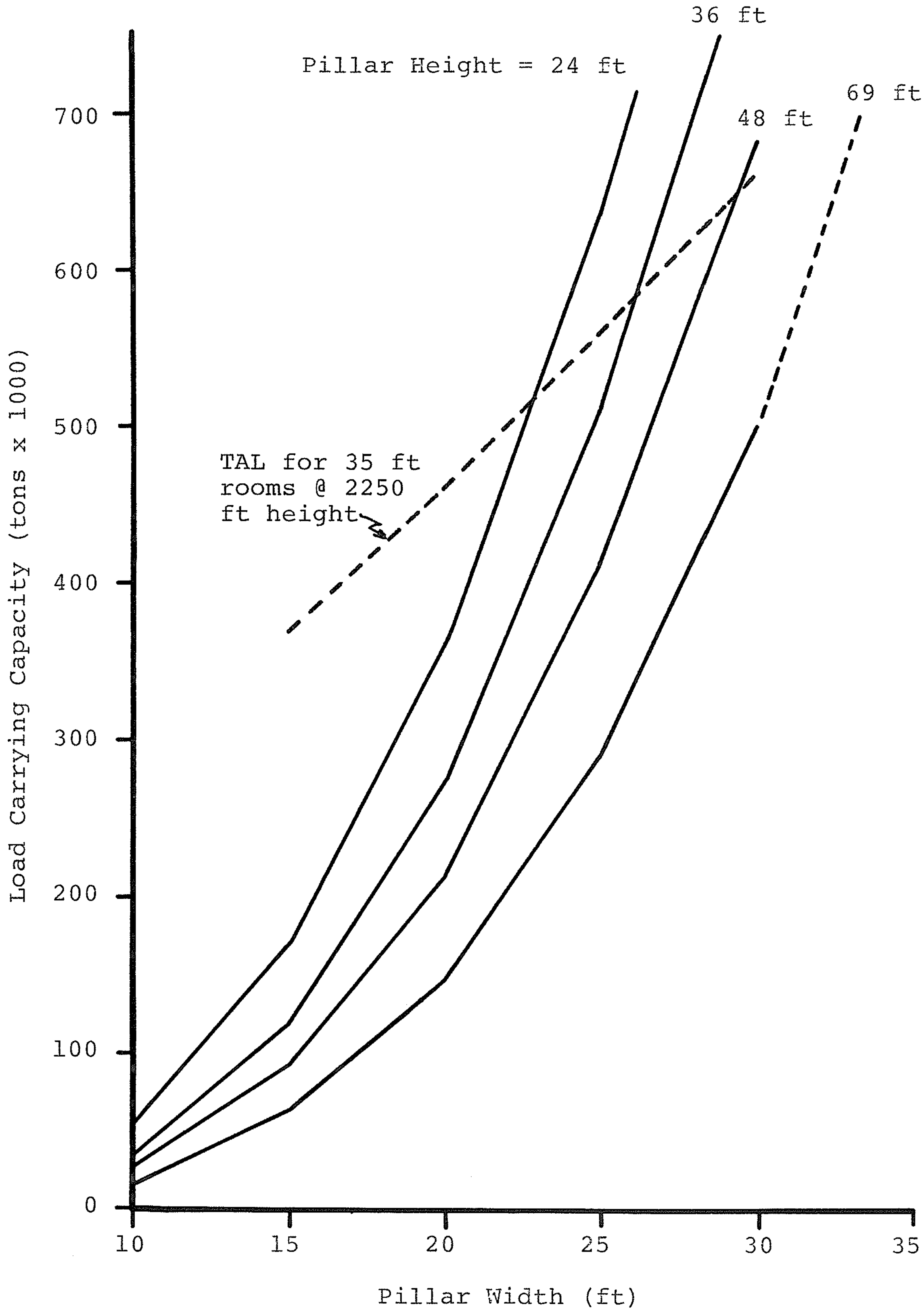


Figure 2-8: Load Carrying Capacity for Various Pillar Widths and Tributary Area Load for a 30 ft Room.

Subsidence (Backfill)

Subsidence is a greater problem now than in the past because of environmental controls and concerns about affecting aquifers. Most of the guidelines on predicting subsidence are based on coal studies, but are being applied to hard rock mines because there is nothing else available. We have not contacted the environmental people; however, we have assumed that subsidence can be tolerated but needs to be kept to a minimum. **Minimizing subsidence can be done by either backfilling or leaving a couple of barrier pillars. As both proposed mining methods are based on a 65 to 80 percent recovery and the use of backfill subsidence,** some surface subsidence can be expected.

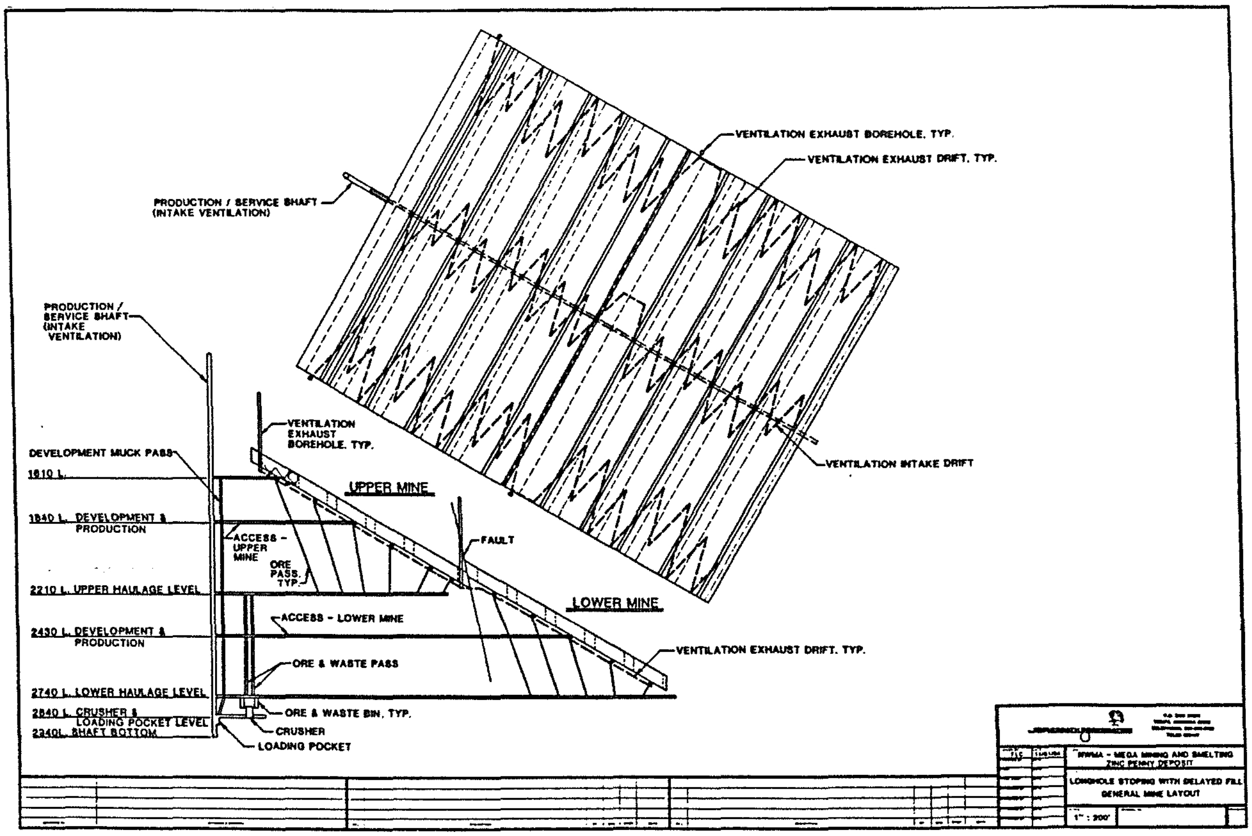
Longhole Blasting Method

This method will be explained by Neil Pacey, as part of the underground mine planning talk; however, the basic components of the layout are stopes that are 150 ft wide with their long axis parallel to strike of the bedding. The stopes are separated by pillars, in the range of 50 ft (Figure 2-9). These pillars are to be mined out after the stopes are filled. Mining is to be done by fan drilling from the footwall; therefore, men should not be exposed to the hanging wall.

In the stope span analysis 100 and 140 ft spans were evaluated (Figures 2-3 and 2-6a). The total failure area for the 140 span is larger than the 100 ft span. Using the average failure area per unit length, the 100 ft span would have around 0.9 tons of failure per running foot of stope, which is less than 1 percent dilution, and the 140 ft span would have around 3.4 tons of failure per running foot, whch is also less than 1 percent dilution. Based on the average dilution, the choice of 150 ft is reasonable. The failures are controlled by the length of the cross joint and fault joint; therefore, in future work these lengths should be mapped as accurately as possible.

Most of the access drifts will be in the argillite, which is significantly more fractured than either the ore or the hanging wall. Utilizing some of the classification methods to predict drift support (Laubscher, 1977; Bieniawski, 1976; and Barton, 1974), the drift support was predicted. If the water pressure is reduced by draining, then 5-ft rock bolts on 4-ft centers, with 2 to 4 in. of shotcrete should support the ground. However, if the water pressure cannot be reduced, light steel with lagging may be required. Further work is needed to determine the water pressure in the argillites.

The proposed 50 ft pillars are temporary and will only carry part of the load under the pressure arch. In order to allow time for the backfill to drain, the pillar would not be mined until the second stope was filled and the third stope was being mined. Stability of the pillar adjacent to the stope being mined must be



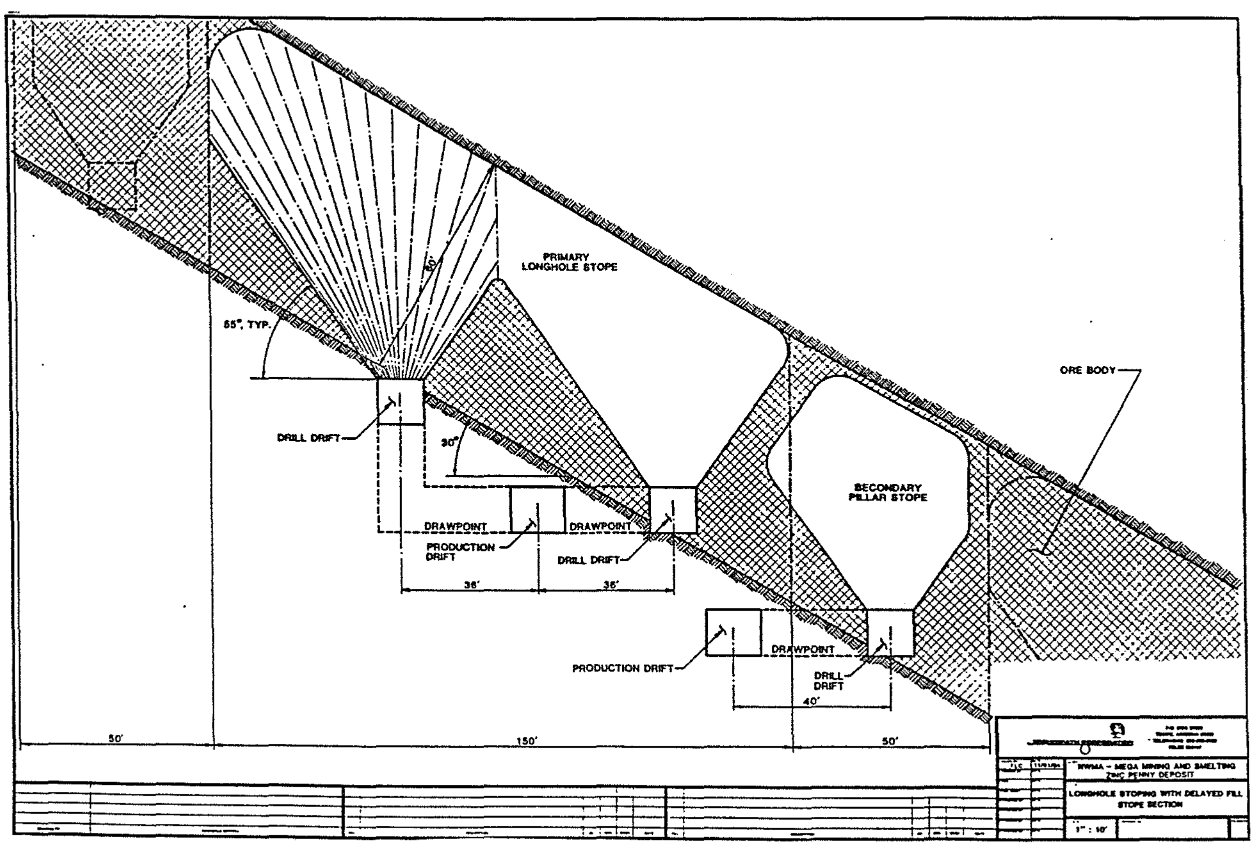


Figure 2-9: Longhole Blasting Plans (provided by Neil Pacey of J. S. Redpath).

ensured. The near worst loading condition is tributary area load to the surface. Using a depth of 3000 ft, this load is around 49,500 tons. Using Figure 2-7, the pillar width that has a LCC of 49,500 is 46 ft wide; i.e., a 46 ft wide pillar has a safety factor of 1 at the deepest point. Therefore, the proposed 50 ft wide pillar is an appropriate choice for the feasibility study. Some consideration to varying pillar dimensions for different depths should be considered, thereby maximizing ore recovery at the shallower depths. The pillar in the fault zone will have to be wider because the ground is more fractured and the strength of the fault gouge is less than that of the rock. A similar analysis of this material results in a pillar width around 80 ft.

Surface subsidence is expected. If we assumed that all the void was filled, with an uncemented fill, we would predict the maximum subsidence of 11 ft; however, the area will not be 100 percent filled, nor will all of the ore be extracted. For a feasibility study, the 11 ft subsidence is the best estimate. Further work would be needed, and a surface monitoring program would be recommended. In addition to maximum subsidence, the extent of the area disturbed by the subsidence must be estimated. Based on the coal data and a correction for the dip of the deposit, Figure 2-10 shows the estimated limit of subsidence. The existence of the major fault may control the area of subsidence. The ventilation boreholes will be affected by the subsidence and, perhaps, the main shaft would have to be moved farther away.

Post-Pillar Cut and Fill

The post-pillar method consists of room and pillar type mining where each lift is filled before the next lift is mined. The pillars are intended only to support the immediate back and, therefore, are expected to fail prior to reaching full height. The failure, though, will occur down in the fill, which will prevent a total collapse. Figure 2-11, provided by Neil Pacey, shows the proposed plan for the post-pillar cut and fill method.

The appropriate roof span and pillar size are interrelated for a post-pillar design. Based on the fracture analysis (Figures 2-4 and 2-6), the maximum span recommended would be 40 ft because the area/width plot (Figure 2-6b) shows a significant increase between a span of 40 and 60 ft. If the chosen span causes the pillar to fail prematurely, then the span should be reduced. The pillar generally fails at mid-height; therefore, if the mining lifts are 12 ft, then failure could occur any time after the pillar is 36 ft high. For design purposes, the appropriate pillar width will be designed at a height of 36 ft with a safety factor of 1. The actual pillar dimensions should vary depending on their distance from the nearest abutment. However, for the feasibility study, near maximum loading condition will be used for design. The maximum loading is likely to be TAL with

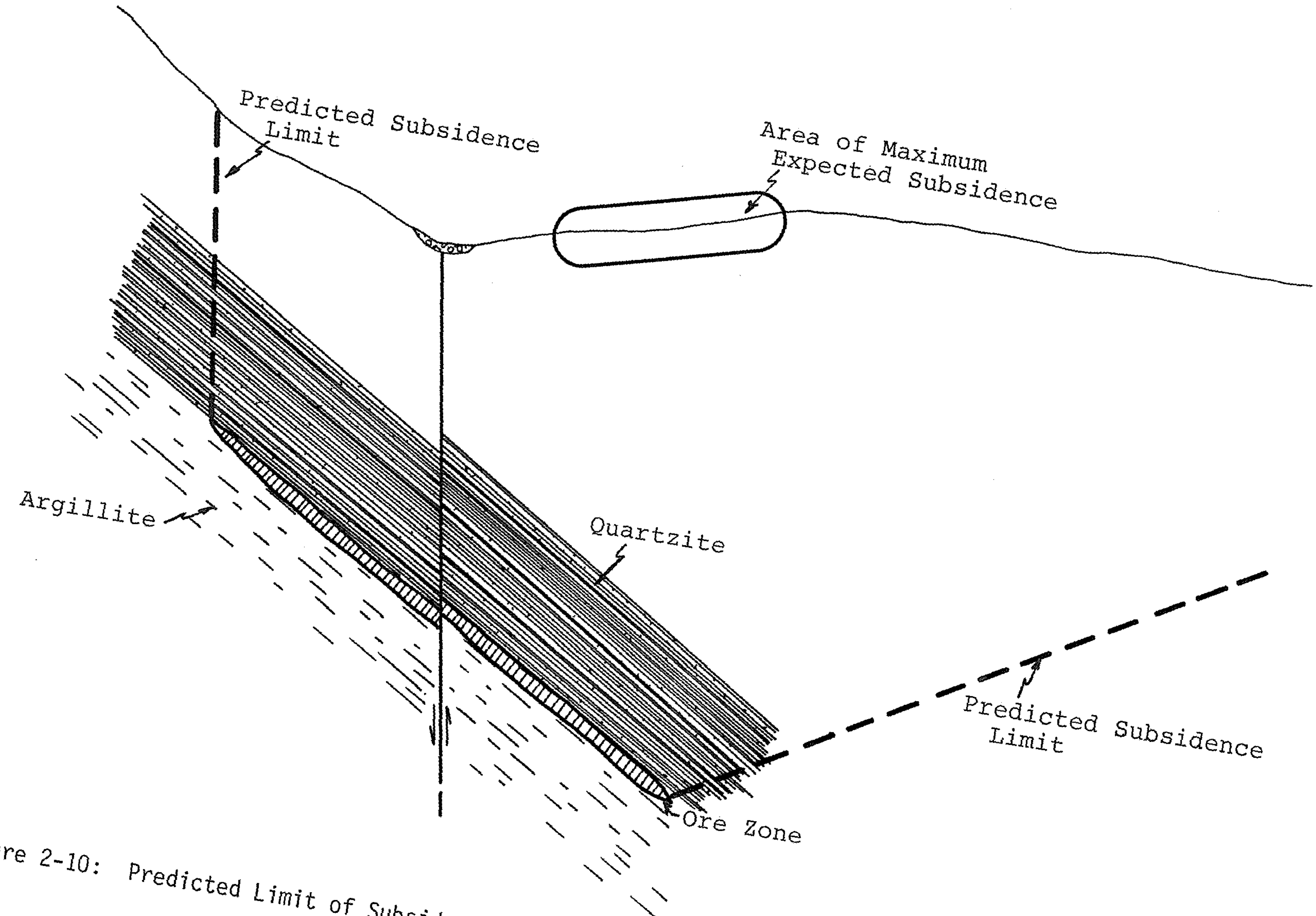
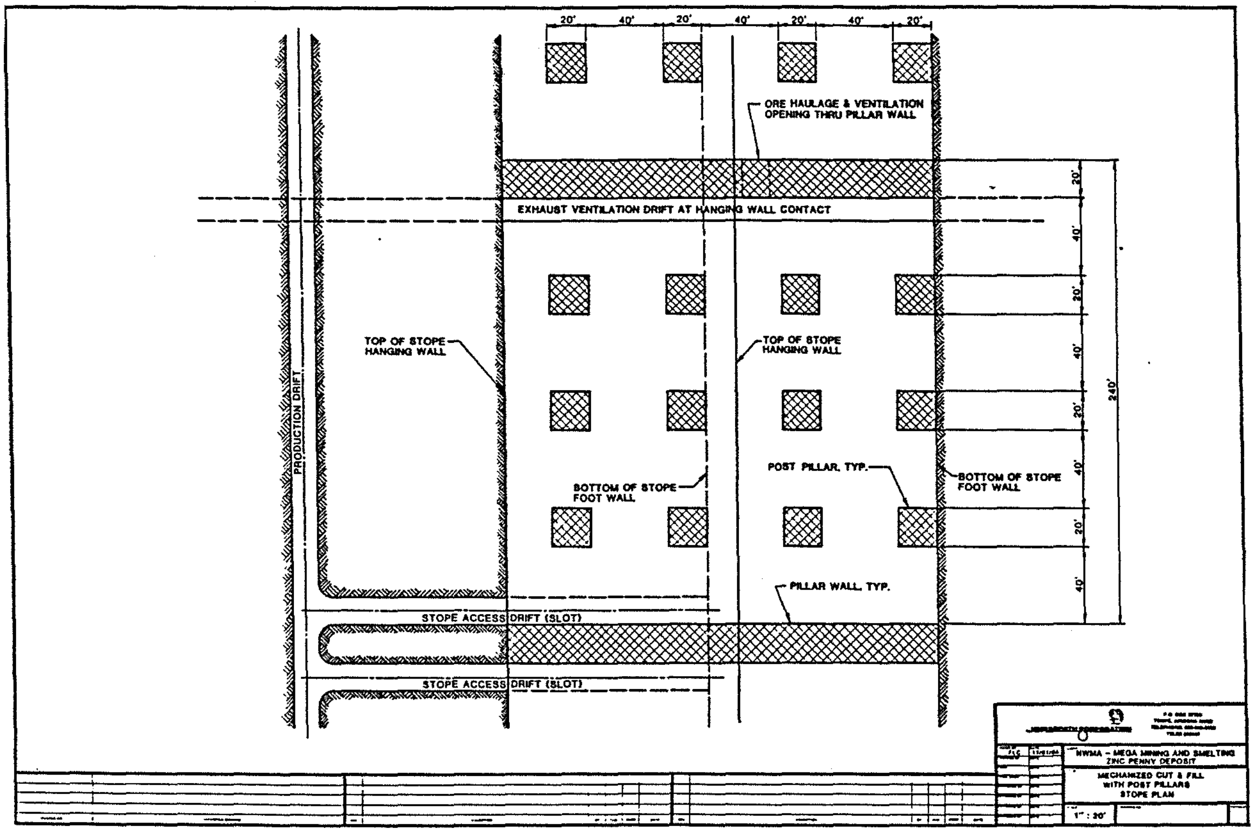


Figure 2-10: Predicted Limit of Subsidence.



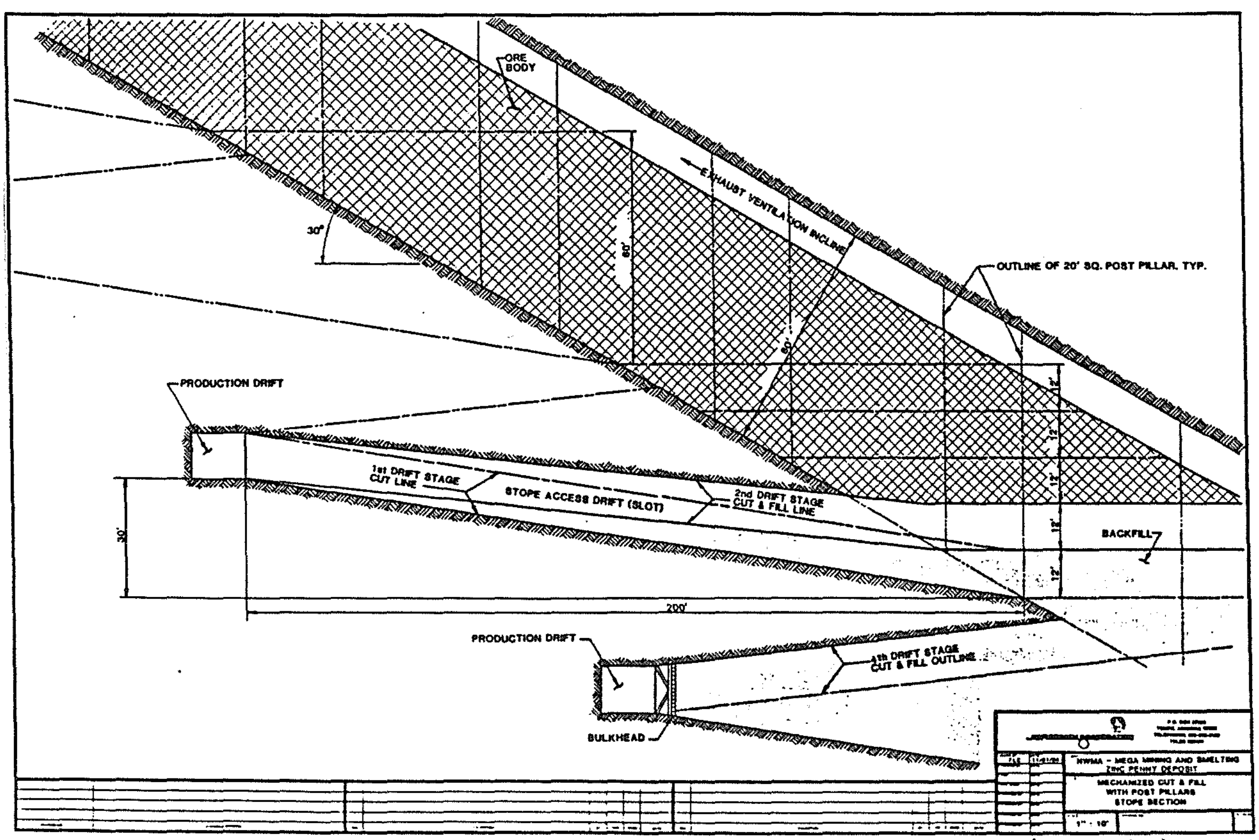


Figure 2-11: Mechanized Post-Pillar Cut and Fill Mine Plans (provided by Neil Pacey, J. S. Redpath).

an overburden height of 2250 ft. Using Figure 2-8, where the TAL curve intersects a 36 ft high LCC pillar curve is the appropriate pillar width. A curve was generated for various room widths versus pillar widths (Figure 2-12) for the design conditions listed above. From Figure 2-12 it is obvious that the best design is the one which maximizes pillar room width. Although the calculations were not carried out to a 40 ft span, projecting the curve results in a 32 ft wide pillar. This width is different from that drawn in the mine plan (Figure 2-11). The initial recommendation made by the rock mechanic for room widths and pillar widths was made prior to any significant calculations.

The subsidence and drift support are the same for the cut and fill method as they are for the longhole blasting method.

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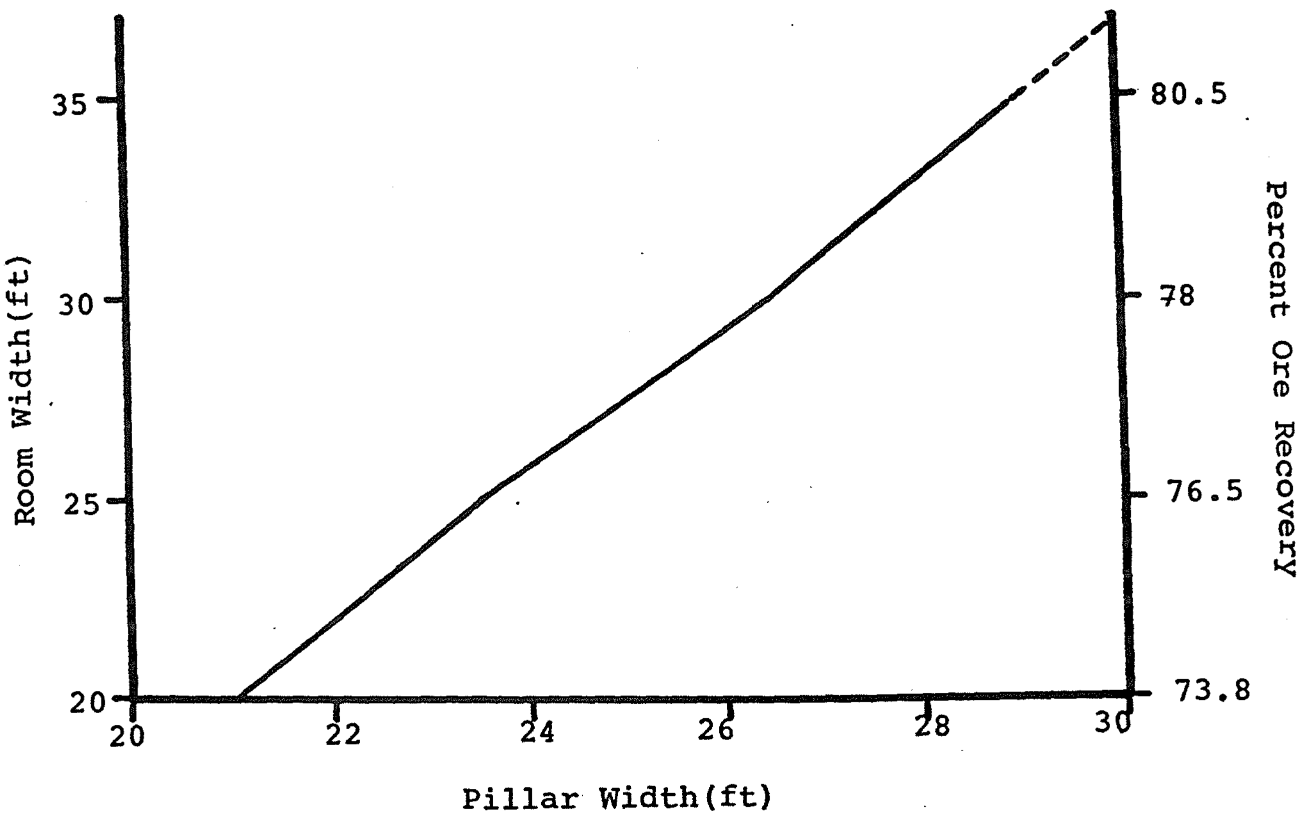

Figure 2-12: Room Width and Pillar Width Combinations for a Post-Pillar Cut and Fill Method.