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EXPATRIATE RESOURCES LIMITED

WOLVERINE JOINT VENTURE

WOLVERINE DEPOSIT - GEOMECHANICS ASSESSMENT

FINAL REPORT

PROJECT NO.: 0263-001-01
DATE: NOVEMBER 6, 2000

Project No. 0263-001-01

November 6, 2000

Mr. Brad Marchant
Vice President of Mining and Development
Expatriate Resources Limited
701 - 475 Howe Street
Vancouver, British Columbia
Canada V6C 2B3

Re: Wolverine Deposit Geomechanics Assessment

Dear Mr. Marchant,

Please find attached a copy of our above referenced report dated November 6, 2000.

Should you have any questions or comments, please do not hesitate to contact me at 684-5900 Ext. 114.

Yours truly,
BGC ENGINEERING INC.
per:

Mr. Scott Broughton, P.Eng.
Project Manager

encl. Final Report
BL/bl

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LIMITATIONS OF REPORT

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1. INTRODUCTION

The Finlayson Project, operated by Expatriate Resources Ltd. (EXR) is a zinc-rich, polymetallic project in southeast Yukon. The project consists of the larger Kudz Ze Kayah (KZK) deposit and the Wolverine deposit, a distance of approximately 30km away. The KZK deposit is accessible by a gravel road from the Robert Campbell Highway. The Wolverine deposit is currently accessible by fixed wing aircraft which can land on a nearby airstrip or Wolverine Lake.

1.1 Project Background

The KZK and Wolverine deposits (jointly referred to as the Finlayson Project) are proposed to be developed simultaneously with a concentrator constructed near the KZK deposit. The KZK deposit will be developed as a 3000 mtpd open pit mine, while the higher grade Wolverine Project will be developed as a 1250 mtpd underground operation. A road will be constructed connecting the two operations and ore from Wolverine will be hauled by truck to the concentrator at KZK. A camp, office complex, warehouse, and other infrastructure will be constructed at KZK to service the Wolverine operation.

1.2 Scope of Work

Expatriate is currently engaged in a pre-feasibility level study of the Finlayson Project. BGC Engineering Consultants (BGC) was engaged by EXR to review the geotechnical data collected to date and to prepare geotechnical guidelines that will form the basis of an underground mine design for the Wolverine deposit.

A two-day site visit was made to the Wolverine project site by Mr. Brennan Lang of BGC to observe first-hand the ore, hangingwall, and footwall conditions in core samples. The hangingwall and footwall rock was observed in outcrop but not the ore. Geological sections were reviewed onsite with Mr. Jeff Bradshaw of Expatriate who was able to provide good insight into the geology of the deposits. Other tasks carried out as part of the site visit included assessing potential borrow pit areas for mine backfill, observation of potential fault zones in the mine area, and assessing conditions at the proposed portal site.

Previous geotechnical data collection was directed by Golder Associates. As part of this study, we have reviewed project correspondence and a geotechnical summary report by Golder Associates (1997).

The following report provides geotechnical guidelines for stope excavations and mine development headings. The guidelines include recommendations for maximum spans, stand-up times, ground support requirements, mining methods and backfill. It should be noted that these recommendations are based upon our assessment of typical conditions as observed in drill core. The proposed test-mining phase will enable a better understanding of the geomechanical behaviour of the rock mass. These recommendations should be updated if necessary to reflect actual conditions encountered during the test-mining phase.

2. GEOTECHNICAL CONDITIONS

2.1 Geology

The Wolverine Deposit is characterized as a polymetallic, volcanogenic massive sulphide orebody hosted by a thick sequence of fragmental rhyolite volcanic rocks at the interface between a lower feldspar and a quartz porphyritic rhyolite and an overlying non-porphyritic rhyolite. The hangingwall sequence of non-porphyritic rhyolite contains interbeds of argillite plus carbonate and magnetite iron formation exhalite units that serve as regional stratigraphic markers. The mineralization includes lenses of massive sulphides containing pyrite, sphalerite, chalcopyrite and galena with lesser silver rich tetrahedrite. The massive sulphide lenses are 2 to 14m thick, and average 5.1m in thickness. In some locations, the massive sulphides are divided into two zones by an internal lens of argillite up to 8m thick. The deposit dips 30 to 50^{o} to the northeast with 35^{o} being average. The footwall rocks immediately below the deposit are mineralized with stringers of pyrite, chalcopyrite, sphalerite and minor pyrrhotite. The footwall rhyolites near the deposit are also intensely altered with chlorite and sericite. Table 1 provides a summary description of the principal rock units that will be encountered during stoping and mine development.

Table 1 Rock Type Summary

Immediate Hangingwall	Interbedded carbonaceous to siliceous argillite and rhyolite. Some minor improvement in rock quality in western sections of the deposit are noticeable in the core. Faulting (gouge) is observed at portions of the hangingwall contact. Hangingwall rocks are universally strongly foliated.
Iron Formation	Hard, blocky magnetite exhalite unit 2-10m thick, located approximately 50m above the hangingwall contact.
Ore Zone Rock	Massive to semi-massive sulphide (pyrite, sphalerite, galena).
Inter-Zone	2-8m thick zone of carbonaceous to siliceous argillite.
Immediate Footwall	Rhyolite tuff and argillite, chloritic and sericitic alteration. Strongly foliated.

Reserves calculated by J. Nilsson, 2000 (all categories) on the Wolverine deposit are reported to be 5.5 million tonnes grading 12.83% Zn, 1.25% Cu, 1.47% Pb, 315.0g/t Ag, 1.56 g/t Au. Reserves currently exist over a strike length of approximately 650m and approximately 400m in the dip direction. The mineralized zone contains two thicker lenses known as the Wolverine Zone to the east and the Lynx Zone to the west. For the purposes of this report, the deposit as a whole will be referred to as the Wolverine deposit.

2.2 Geotechnical Data Collection

Geotechnical data has been collected from exploration drill core on the Wolverine and Lynx deposits. The geotechnical database consists of:

- 1996 geotechnical core logging on holes WV96-25 to WV96-72;
- Year 2000 geotechnical core logging on seven holes (WV00-113 to WV00-120); and,
- Core photos of the ore intersections and the immediate hangingwall and footwall zones.

A geotechnical data collection program was prepared by Golder Associates with the data collected by EXR geology staff. During the data collection, Golder visited the site to review and guide the data collection procedures. Geotechnical parameters that were measured in the core include recovery, RQD, fractures per run, point load index or field strength index, weathering index, presence of faults or broken core, joint set number, joint alteration number, and joint roughness number. The Y2000 geotechnical logs are not included in any other reports so they are appended herein for completeness.

2.3 Intact Rock Strength

The intact rock strength was estimated using standard field index tests and by point load index testing. Point load tests were not done very often because of the high joint frequency. A summary of intact strengths for the various rock types is provided in

Table 2 Intact Rock Strengths

Rock Type	Strength	Description
Hangingwall Rocks	R1-R3 1-50 MPa	Very weak to medium, generally weak
Iron Formation	R3-R4 50-100 MPa	Medium to Very Strong, Generally Strong
Ore	R3-R5 50-250 MPa	Medium to very strong
Footwall Rocks	R1-R3 1-50 MPa	Very weak to medium, generally weak

2.4 Rock Mass Classification

Rock mass classification has been carried out previously by Westmin Resources and Golder Associates (1997) using the NGI Tunnelling Quality Index (Q) system. Rock mass classification is a useful tool for describing and categorizing different rock types for the purpose of assessing stability and support requirements in underground excavations. A number of empirical design procedures have been established to predict for example, stable excavation spans, cable bolt support requirements, and dilution levels based on rock mass classification.

Two of the most common classifications systems for mining applications are the CSIR Geomechanics Rating (RMR) and the Norwegian Geotechnical Institute's (NGI) Tunnelling Index (Q).

A) The Geomechanics Rating, developed by Bieniawski (1973), calculates a Rock Mass Rating (RMR) based on five parameters as follows:

- strength of intact rock;
- rock quality designation (RQD);
- spacing of joints;
- condition of joints; and,
- groundwater conditions.

Ratings are applied to each parameter based on the observed conditions and engineering experience. An overall rating is obtained by adding the individual ratings for each of the five parameters. The overall rating ranges from 0 (Very Poor Quality) to 100 (Very Good Quality) and is then adjusted to account for joint orientation with respect to the excavation.

Rock mass ratings were carried out on drill core with some engineering judgement applied for parameters such as joint length, large-scale roughness, and water conditions. A typical rock mass rating for the ore and hangingwall/footwall are provided in Table 3.

Table 3 Typical Ore Rock Mass Rating (RMR)

Parameter	Description	Rating
Intact Strength	R4 (100-250 MPa)	12
RQD	50-75%	13
Joint Spacing	200-600mm	10
Joint Condition	Rough, planar, tight joints, assumed 3-10m, no weathering on joints unweathered	23
Groundwater	wet	7
Joint Orientation	fair joint orientation relative to excavation	-5
Total RMR	FAIR to GOOD	60

Table 4 Typical Hangingwall/Footwall Rock Mass Rating (RMR)

Parameter	Description	Rating
Intact Strength	R1 (5-25 MPa)	2
RQD	<25%	3
Joint Spacing	<60mm	5
Joint Condition	Rough curved, joint open <1 mm, joint length about 3-5m, weathering on joints, some joints infilled	12
Groundwater	Wet	7
Total RMR	Poor	29

B) The NGI rock mass classification system (Barton et al., 1974) calculates a rock tunnelling index (Q) from six parameters based on:

- rock quality designation (RQD);
- joint set number (Jn);
- joint roughness number (Jr);
- joint alteration number (Ja);
- joint water reduction factor (Jw); and,
- stress reduction factor (SRF).

The parameters are combined to obtain Q as follows:

$$Q = \left(\frac{RQD}{Jn}\right)\left(\frac{Jr}{Ja}\right)\left(\frac{Jw}{SRF}\right)$$

Typical ore intersection properties are provide in Table 5.

Table 5 NGI -Q Classification of Ore Zone Rocks

Parameter	Description	Rating
RQD	50-60	60
Jn	2 joint sets	4
Joint Roughness Jr	Rough, planar	1.5
Joint Alteration	Fresh to slightly altered	1.0-2.0
Joint Water	Assume drained	1.0
Stress Reduction Factor	Low Stress-Medium Stress	1.0
Q		4-20 (Fair to Good)

Note: Large-scale joint roughness not distinguished from drill core. Conservatively estimated as planar.

Typical hangingwall and footwall conditions are provided in Table 6.

Table 6 NGI Q Classification of Hangingwall and Footwall Rocks

Parameter	Description	Rating
RQD	Generally 0	10
Joint Set Number, Jn	2 joint sets plus random	6
Joint Roughness, Jr	smooth, planar	1
Joint Alteration, Ja	Low friction coating, sericite, graphite, gouge	3-6
Joint Water, Jw	Assume drained	1.0
Stress Reduction Factor, SRF	Low Stress-Medium Stress	1.0
Q		0.1-4 Very Poor to Poor

Note: 1. Large-scale joint roughness not distinguished from drill core. Conservatively estimated as planar.
2. RQD of 10 normally used to estimate Q when RQD is less than 10.

RMR values have been calculated for rock in the hangingwall, ore zone, and footwall and are also summarized in Table 7. The RMR ratings were calculated from the Q rating based on known relationships between the two systems.

BGC has reviewed the core logs and examined many of the ore intersections and immediate hangingwall and footwall host rocks. We conclude that footwall and hangingwall rocks of both the Lynx and Wolverine Zones can be classified as very poor to poor. The quality of the ore zone is classified as "Fair" (Q=10-30).

Table 7 Rock Mass Classification Summary

*Note: Qualitative descriptions accompanying numerical ratings are assigned according to each classification system and may vary between systems.

	RMR Range*	Q Range
Hangingwall	20-40 (Poor)	0.1-4 (Very Poor to Poor)
Iron Formation	50-60 (Fair)	4-10 (Fair)
Ore	50-65 (Fair to Good)	4-20 (Fair to Good)
InterZone	20-30 (Poor)	0.1-0.4 (Very Poor)
Footwall	20-40 (Poor)	0.1-4 (Very Poor to Poor)

2.5 Major Geologic Structures

A number of faults can be observed in the drill core. These faults have not been interpreted on the geologic sections, however it is thought that they trend parallel or subparallel to the stratigraphy. These faults are up to several metres thick and contain crushed rock and gouge. The faults are expected to be highly water-bearing.

2.6 Hydrogeology

There have been no hydrogeological studies done on the Wolverine deposit. The geology staff reports there were no adverse water conditions encountered during exploration drilling collared from surface, although one hole temporarily exhibited artesian flow. It is understood from discussions with the site geologist that existing holes have all been grouted through the ore zone.

The sedimentary and volcanic rock suite hosting the Wolverine and Lynx Zones is intensely fractured and strongly foliated. Being an exhalative, the general direction and dip of the foliation is consistent with the dip of the orebody. These units could be expected to have relatively high hydraulic conductivity (>10^{-6} m/s). Significant inflows should be expected during the initial mine development, particularly when crossing major faults which are known to be present. When intersected, this water could quickly erode friable fault material and seriously slow the mining advance. During the proposed test-mining phase, it is recommended that test holes be drilled ahead of the face to establish the location of faults and to drain them if possible. If faults do not drain rapidly, a grouting program should be undertaken to aid in advancing through the fault zones.

The Goodman equation (1992) has been used to provide a preliminary estimate of the steady state inflow.

$$Q = \frac{2\pi KH}{\ln\left(\frac{2L}{r}\right)}$$

Where Q is the flow per unit length of stope, K is the hydraulic conductivity, H is the hydraulic head, L is the average depth, and r is the radius of a circle whose area equals the stope cross section. Using the Goodman equation, and an average K of 10^{-5} m/s the steady state groundwater inflow near the end of the mine life is estimated to be 90 litres per second.

It is recommended that if any future drilling is carried out, the depth to the water table be measured. In addition, it is recommended that slug tests be performed to estimate the hydraulic conductivity of the formation at various depths. An estimate of the steady state groundwater inflow is needed in order to size the water treatment plant that will likely be necessary to treat discharge pumped from the mine. Some of the water to be discharged will include water drained form the backfill.

Given the likelihood of initially high groundwater inflows, it is recommended that the mine be developed well down-dip of the proposed mining areas before mining begins, thereby allowing drainage of faults and other water bearing fissures intersected by the development. This approach will be more desirable than an advancing mining sequence where the mine would be continuously tapping undrained strata as the mine advances.

2.7 In-Situ and Mining Induced Stress

The in-situ or pre-mining stress conditions at Wolverine will have an influence on mine pillar stability particularly during the late stages of the mine life. There have been no in-situ stress measurements conducted at the site and the authors are not aware of any direct measurements made in the local vicinity. For the purposes of this report, in-situ stress magnitudes have been assumed based on typical shallow mining conditions. The vertical stress is assumed to be due entirely to the weight of overlying rock. The unit weight of the rock is on average 27 kN/m^3, yielding a vertical stress gradient (σ_v) of 0.0275 MPa/metre. For preliminary mine design purposes, the major (σ_1) and intermediate (σ_2) principal stresses are assumed to be 2.0 times and 1.5 times the vertical stress respectively. The direction of the major principal stress is assumed to be northeast which is consistent with a number of measurements made on the west side of the North American plate. This direction is also consistent with the direction of stress needed to form the northwest striking mountain ranges in the Finlayson area.

The Wolverine project site is located in an area of low seismic activity and lies in Seismic Zone I as characterized by the Geological Survey of Canada. Given the proposed life of the mine and the mining method, natural seismic activity will have a very low impact on underground mining operations.

3. MINING

3.1 Stand-up and Exposure Time

The strongly foliated and friable nature of rock (particularly the carbonaceous argillite) will require that excavation spans be maintained as narrow as possible for as short a time as possible to avoid costly ground support. The "stand-up or exposure time" of the argillite and rhyolite is estimated to be on the order of hours (Figure 1) for spans greater than 3m. The stand-up time is defined as the time that the span will stand unsupported before failing. Failure of the argillite and rhyolite would occur by gradual spalling and ravelling of the small pieces of rock, generally 150mm in size or less. If it were allowed to progress, the failure would eventually develop a stable arch above the opening to a depth of about one-half the span.

During the mine development, strict instructions will need to be given to the mining crews not to exceed the tunnel design size. An allowance should be made in the drilling to include overbreak. Tunnel intersections should be located wherever possible in higher quality rhyolite or siliceous argillite rather than carbonaceous argillite.

The limited stand-up time of the hangingwall rock dictates that in development headings, and stopes where the hangingwall is exposed, ground support should be installed immediately after the round is mucked out. In addition to their high degree of foliation and weak structure, the argillites are naturally friable and will erode quickly. This will be exacerberated by the ventilation causing the rock to dry-out rapidly. As the material dries, it becomes more friable and this is another reason for immediately covering it with shotcrete in permanent openings. In stopes, the behaviour of the argillite in the hangingwall or "Inter Zone" is expected to degrade rapidly after approximately one-month exposure time.

In extremely poor ground such as fault intersections, it may be necessary to shorten the rounds to about 1.5 to 2.0m to prevent the ground from caving before the ground support is installed.

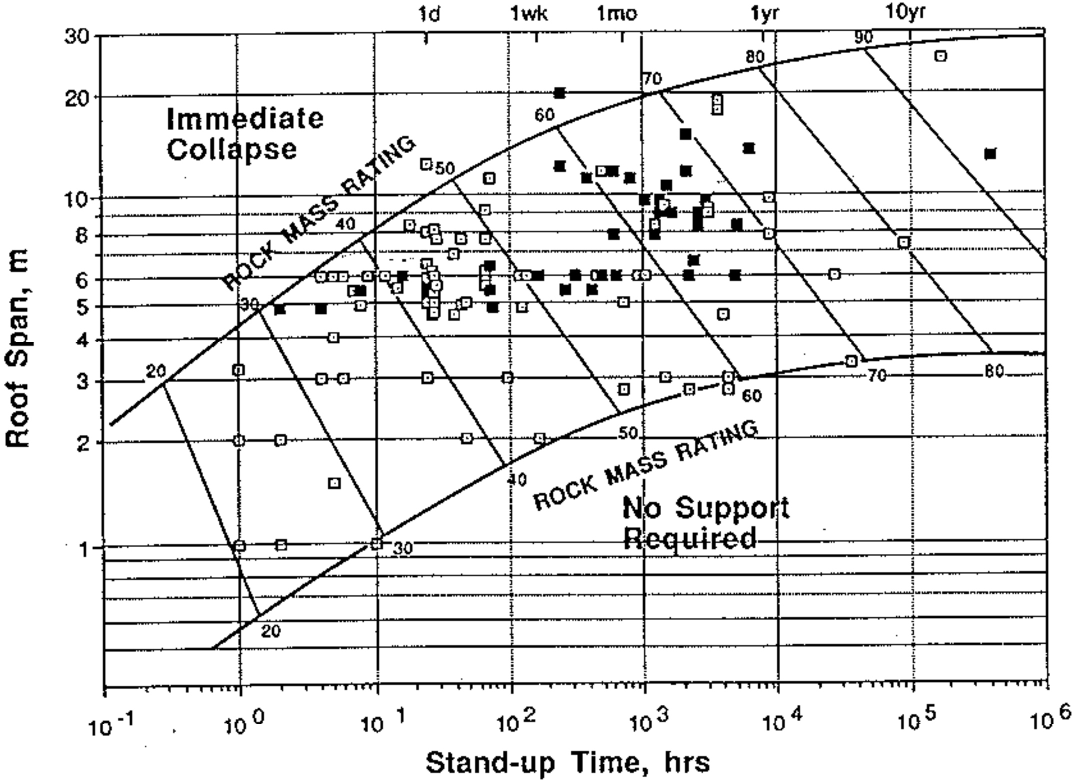
By comparison, the calculated stand-up time in ore is in the order of years for spans in the range of 3-9m. In areas where the back is comprised of massive sulphides, the face could advance several rounds in front of the supported ground if necessary. The above assessment is appropriate for non-structural rock mass failure. In the more competent massive sulphides, wedge failures caused by intersecting structures are more likely. If discrete wedges are defined by jointing with the potential to fall out of the back, they should be supported immediately.

Figure 1 Stand-up Time vs. Rock Mass Rating

3.2 Span Design

In the stopes that expose the hangingwall, the ground conditions, hangingwall dip, stand-up time and equipment constraints have combined to limit the allowable span (measured along the dip) to 4m (Figure 5). Limiting the exposed hangingwall span to 4m will allow sufficient time to install temporary support. Installation of the temporary support (see below) is estimated to be adequate for supporting this span over approximately one month. This span is also sufficient to permit mining using reasonably sized mechanized equipment. After one month, it is anticipated that the hangingwall rocks will become so friable that the mesh and ground support will become ineffective in controlling the ravelling of small pieces.

The size of the permanent mine development headings is dictated for the most part by equipment sizes, clearance regulations and ventilation requirements. In discussions with Nilsson Mine Services Ltd. (NMS) who are carrying out the pre-feasibility mine planning, it was indicated that a 4.5m wide x 5m high decline size would be adequate. It is recommended that



1d
1wk
1mo
1yr
10yr
Immediate Collapse
ROCK MASS RATING
No Support Required
ROCK MASS RATING
Roof Span, m
Stand-up Time, hrs

the mine development be excavated with an arch as shown in Figure 4 because it is an inherently more stable shape and it will enhance the performance of the shotcrete.

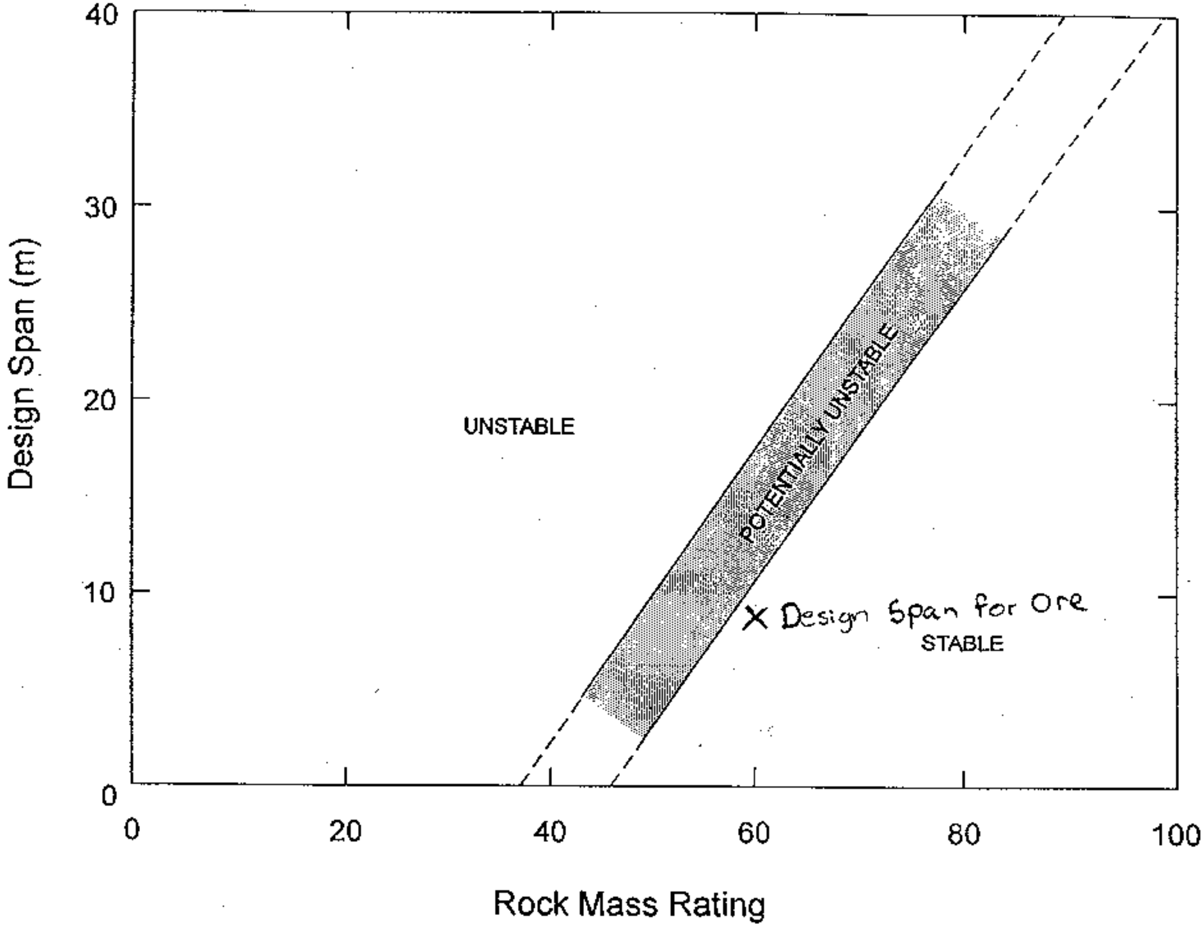
In wider sections of the orebody, where the back is entirely in massive sulphides, larger spans can be excavated, (Figure 7). In such cases, the Stability Graph for Entry-Type Excavations (Lang et al., 1991) provides a useful empirical method for predicting the stability of temporary entry-type excavations. The Stability Graph for Entry-Type Excavations was developed from 172 case histories taken from temporary underground excavations covering a wide range of rock conditions.

The Stability Graph for Entry-Type Excavations shown in Figure 2 indicates that for the ore (60% RMR) stable spans up to 10m can be achieved with pattern rockbolting. These spans are only possible in the absence of continuous, adversely oriented geological structures which will normally control structural stability. For design purposes we recommend that horizontal spans in ore, not exceed 9m. With better experience, which comes only through mining, it may be possible to increase this allowable span.

Figure 2 Stability Graph for Entry-Type Excavations

3.3 Ground Support

In both permanent and temporary openings, ground support should be installed immediately after the round is mucked out. This is especially important for exposures of hangingwall argillite. The following recommendations are to be considered applicable to average conditions at Wolverine. Actual ground support may be varied on a round by round basis according to the observed conditions and the long-term stability requirements of the particular heading.



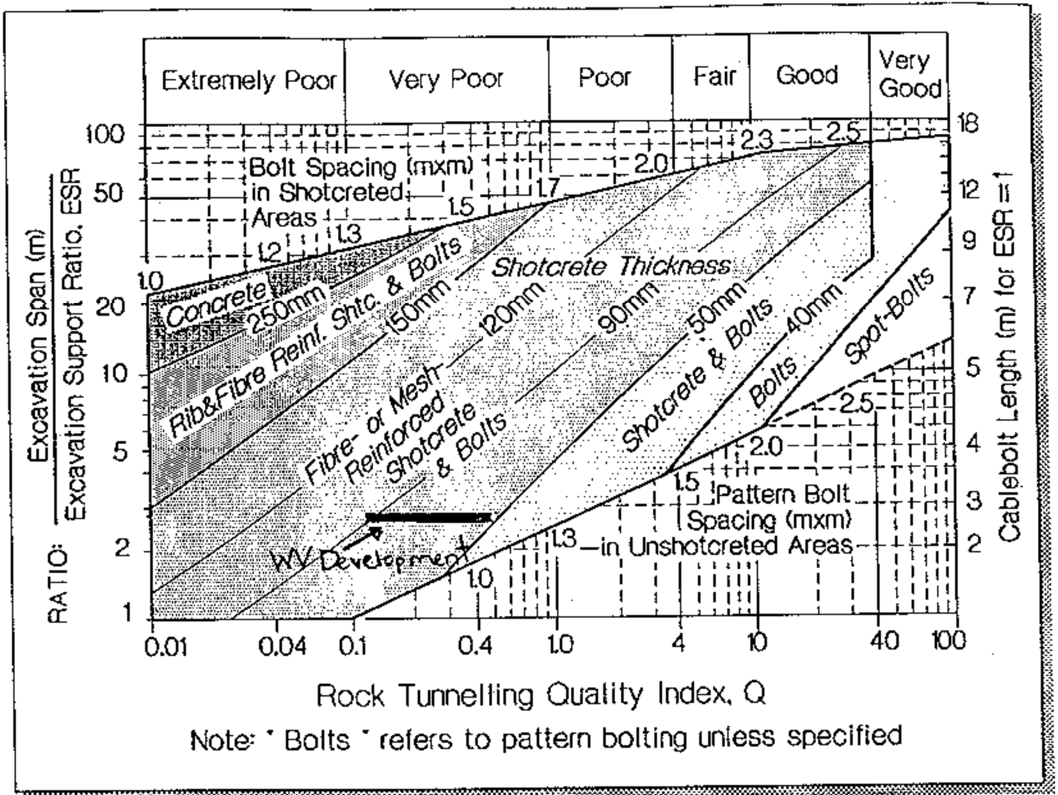
40
30
20
10
0
0
20
40
60
80
100
Design Span (m)
Rock Mass Rating
UNSTABLE
POTENTIALLY UNSTABLE
STABLE
X Design Span for Ore

3.3.1 Permanent Mine Development Ground Support

The purpose of the ground support in the development headings will be to provide permanent support that will resist deformation due to stress changes as mining progresses. Re-installing ground support in active development headings is much more expensive than doing a good quality job immediately after it is excavated. The support system should also be resistant to corrosion, which is a common problem in sulphide orebodies. Permanent mine development is defined for the purposes of this report as excavations required for more than one month.

The ground support requirements have been estimated using an empirical method proposed by Barton et al. (1976) which makes use of the rock tunnelling quality index, Q and the Excavation Support Ratio (ESR) (Figure 3). The ESR is a factor that accounts for the type of excavation (i.e. permanent vs. temporary). For permanent mine development headings, an ESR value of 1.6 is appropriate. Permanent mine development headings are planned to be 4.5m wide x 5m high. In Figure 3 a Span/ESR ratio of 2.8 for a tunnel with a Q value of 0.1-0.4 requires ground support consisting of fibre-reinforced or mesh-reinforced shotcrete and bolts. The shotcrete thickness should be 75mm in the back and 25mm on the walls. Shotcrete should be applied as soon as possible after mucking out each round. In particularly weak areas such as fault intersections, the shotcrete thickness may need to be increased to 100mm. Rockbolts should be installed on a 1m pattern in the back and corners of the tunnel and on a 2m pattern on the walls. A typical support installation for the mine development is shown in Figure 4.

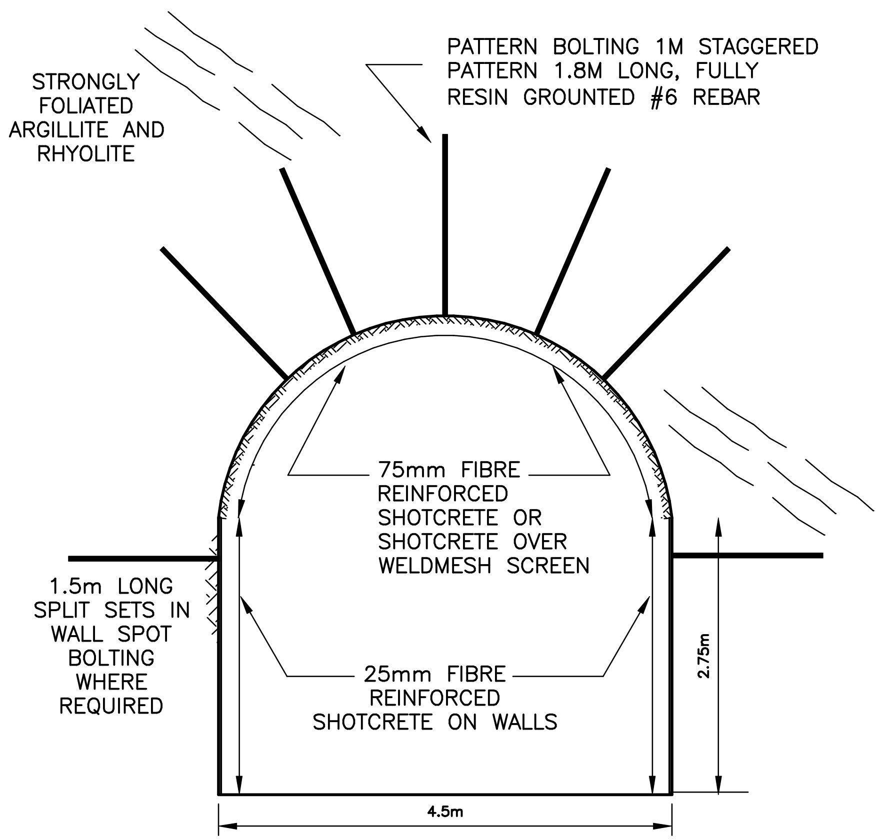
Figure 3 Ground Support Requirements Based on Norwegian Tunnelling Quality Index



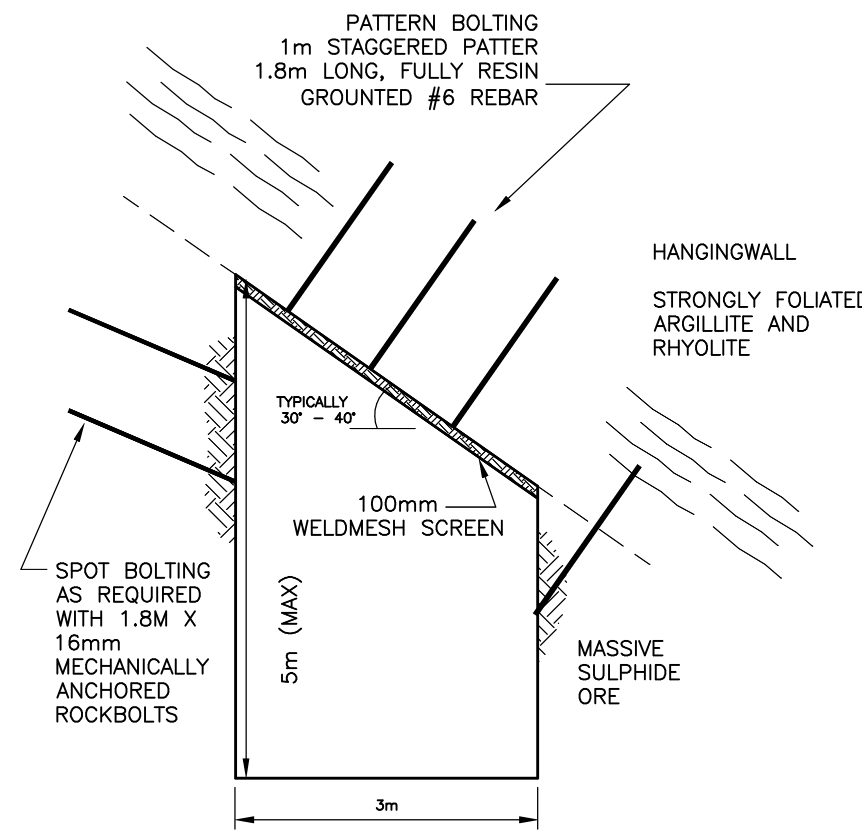
Extremely Poor
Very Poor
Poor
Fair
Good
Very Good
Bolt Spacing (mxm) in Shotcreted Areas
Concrete
250mm
Rib&Fibre Reinf. Shtc. & Bolts
150mm
Shotcrete Thickness
120mm
90mm
50mm
40mm
Fibre- or Mesh-Reinforced Shotcrete & Bolts
Shotcrete & Bolts
Bolts
Spot-Bolts
Pattern Bolt Spacing (mxm) in Unshotcreted Areas
WV Development
RATIO: Excavation Span (m) / Excavation Support Ratio, ESR
Cablebolt Length (m) for ESR = 1
Rock Tunnelling Quality Index, Q
Note: " Bolts " refers to pattern bolting unless specified

Figure 4 Typical Ground Support Requirements

TYPICAL SUPPORT IN DEVELOPMENT HEADINGS

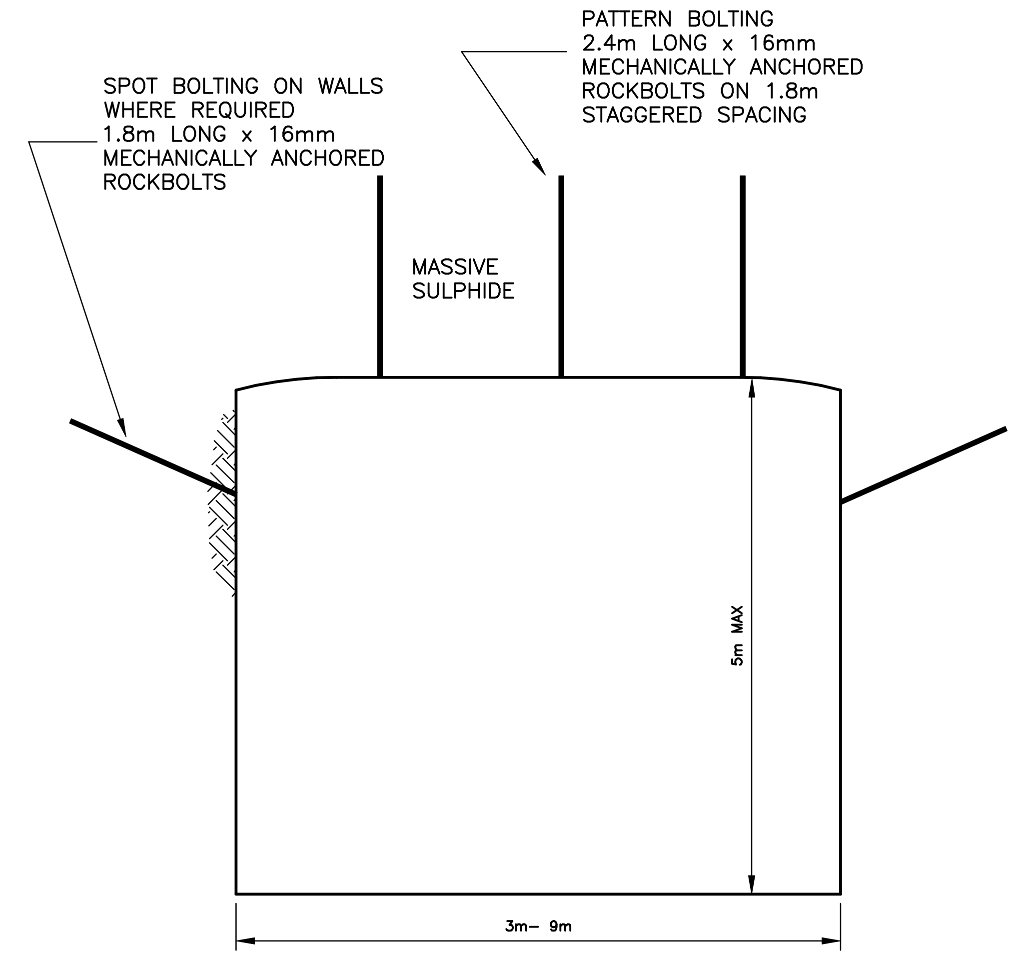


TYPICAL HANGINGWALL SUPPORT IN STOPES



NOTE:
IN EXTREMELY POOR GROUND, APPLY 25mm SHOTCRETE OVER WELDMESH SCREEN

TYPICAL SUPPORT IN ORE



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SCALE:	1:75
DATE:	SEPT 2000
DRAWN:	KB
DESIGNED:	BDL
CHECKED:	–
APPROVED:	–

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AN APPLIED EARTH SCIENCES COMPANY
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CLIENT: EXPATRIATE RESOURCES LTD.

PROJECT: WOLVERINE J/V – GEOMECHANICS ASSESSMENT

TITLE: TYPICAL GROUND SUPPORT REQUIREMENTS

PROJECT No. 0263–001

DWG. No. FIGURE 4

REV. A

Shotcrete will be a regular part of the excavation cycle at the Wolverine Mine, particularly in development headings. A dry mix shotcrete machine and a boom mounted shotcrete nozzle are recommended.

3.3.2 Temporary Ground Support

Temporary ground support will be installed in stopes and is intended to hold the ground for approximately one month, the duration needed to mine a section of the stope before backfilling. When the hangingwall is exposed in the back, support should consist of 1.8m long resin grouted #6 rebar on a 1m staggered spacing (Figure 4). Weld mesh screen should also be installed in the back. Resin grouted rebar provides bonding over the full length of the bolt and do not rely solely on the end anchor and plate to provide support. This is advantageous in this type of ground where spalling around the plate could cause a loss of support if mechanically anchored rockbolts are used. In addition, the hangingwall rocks are too weak to provide adequate anchorage.

Spot bolting with 1.8m long mechanically anchored rock bolts should be carried out in the harder ore rock in the wall as required. In very poor ground, it may be necessary to apply a 25mm thick layer on the hangingwall, covering the bolts and screen.

3.4 Stoping Technique

The dip of the ore is too shallow to permit longhole or open stoping methods and too steep to permit travel of mechanized equipment on the footwall. The most suitable method for mining the Wolverine deposit is a combination of mechanized, overhand cut and fill, and drift and fill. In shallower sections of the orebody, this technique will more closely resemble drift and fill stoping. These techniques will provide the flexibility necessary to mine the Wolverine deposit at ore thicknesses ranging from 2 to 14m or more and at dips ranging from 30-50^{o}.

Figure 5 illustrates the technique for ore thicknesses (HW-FW) of 2 and 4m. The ore is mined to a minimum width (side to side) of 3m to accommodate 5 yd LHD's. A 2m high wall is maintained on the down-dip side and the back is excavated at the angle of the hangingwall. It is unlikely that any special controlled blasting techniques will be required as the back will want to naturally break to the hangingwall contact. As the drift advances, temporary support shall be installed on a round by round basis. The length of the stope should be limited to prevent the hangingwall from being exposed for longer than one month.

After the level is completed, the bottom 2m should be backfilled with uncemented backfill. The remainder of the level should be backfilled with cemented backfill. This cycle is then repeated on the next level above.

Figure 6 shows the excavation sequence for an 8m thick ore zone. The initial cut is made 6m wide in ore. When the cut is completed along strike, it is backfilled with uncemented fill. The ore directly above the initial pass is divided into two 3m wide cuts in order to limit the exposed hangingwall span. The up-dip cut is mined first and then backfilled with a cemented backfill. After allowing sufficient time for the backfill to gain strength, the down-dip cut is mined. This cycle is then repeated on the level directly above and beside the first cut.

Figure 7 shows how the stoping technique will be applied in 10m thick portions of the orebody. The ore is initially mined 9m wide and 5m high and backfilled with uncemented fill. To mine the ore above the initial cut, a 3m wide centre cut is first made and backfilled with cemented backfill. When adequate strength is developed in the backfill, a 3m wide cut is mined down-dip of the first cut and then backfilled with uncemented fill. Another 9m wide cut is then excavated on the "up -dip" side of the cemented fill and the cycle is repeated.

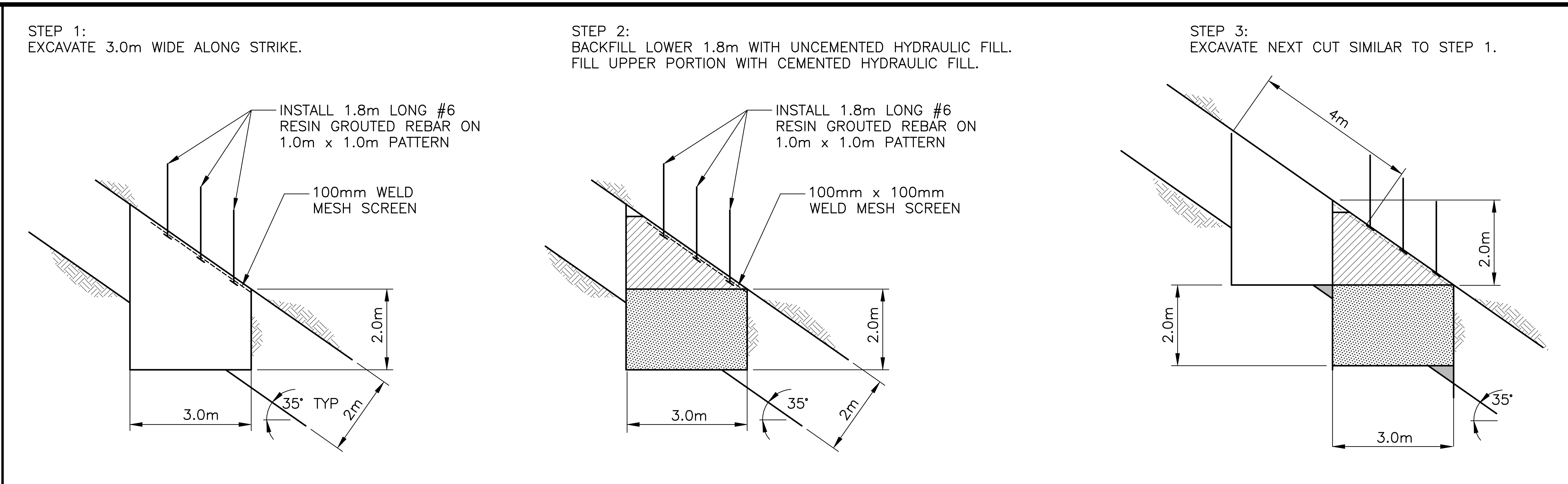
Similar techniques can be used to mine the ore at any angle or ore thickness. In practice, the ore thickness varies along strike, which adds additional complexity, however with good planning, this method is workable, provides a high degree of extraction, and minimizes the cement content in the backfill.

3.5 Mine Access

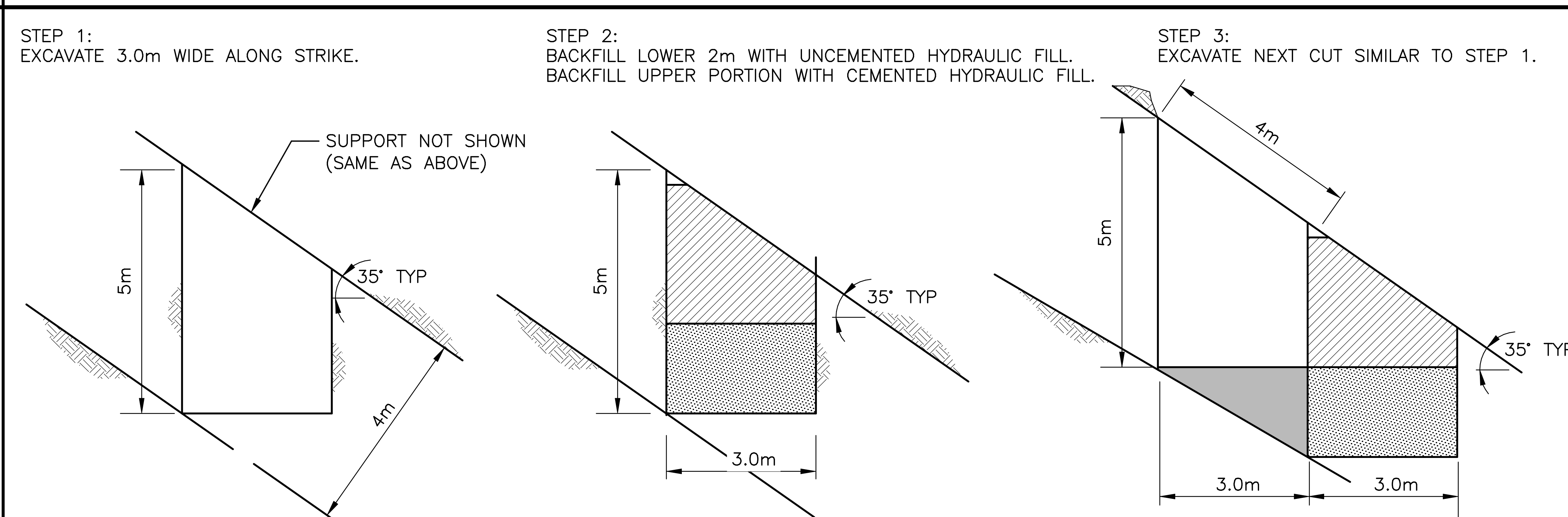
It is recommended that the portal be collared in the hangingwall iron formation and the decline stay in the iron formation to El. 1275m, thereby avoiding strongly weathered rhyolite and argillite near surface. Below 1275m, the access decline should be developed in the relatively thin ore zone separating the Wolverine and Lynx zones.

Figure 5 Stoping Sequence for 2m and 4m Thick Zones

2m TRUE THICKNESS



4m TRUE THICKNESS



NOTES:

1. IN EXTREMELY POOR GROUND, THE HANGINGWALL SHOULD BE TEMPORARILY SUPPORTED WITH 25mm SHOTCRETE OVER REBAR AND SCREEN.

LEGEND

- UNCEMENTED HYDRAULIC BACKFILL
- CEMENTED HYDRAULIC BACKFILL

REV.	DATE	REVISION	DRAWN	CHECKED	APPROVED

SCALE:	1 : 100
DATE:	SEPT 2000
DRAWN:	MT
DESIGNED:	BDL
CHECKED:	
APPROVED:	

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY
Vancouver, B.C. Phone: (604) 684 5900

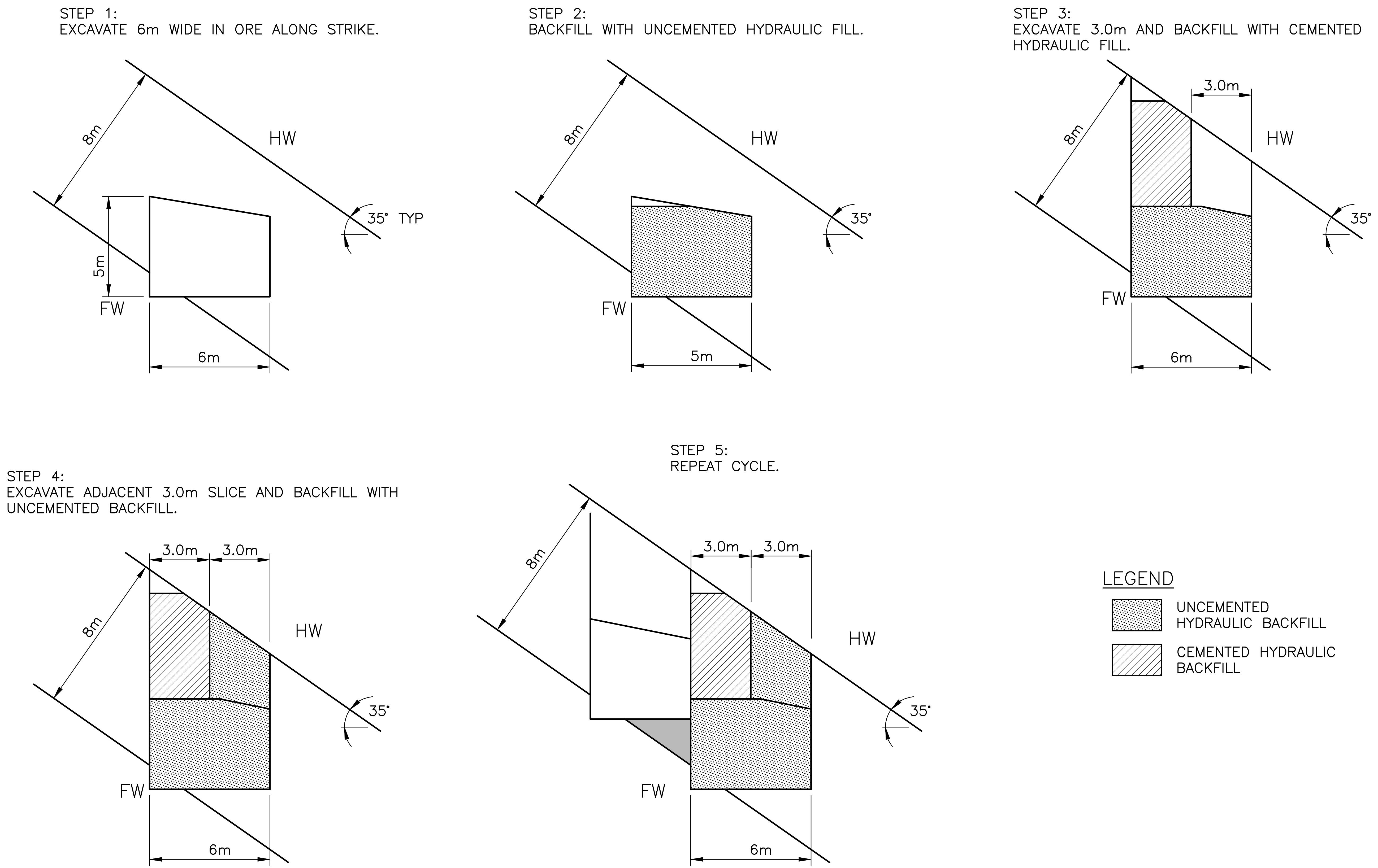
CLIENT: EXPATRIATE RESOURCES LTD.

PROJECT: WOLVERINE J/V – GEOMECHANICS ASSESSMENT

TITLE: STOPE SEQUENCING FOR 2m AND 4m THICK ORE ZONE

PROJECT No.	DWG. No.	REV.
0263 – 001	FIGURE 5	A

Figure 6 Stoping Sequence for 8m Thick Zone



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TITLE: STOPE SEQUENCING FOR 8m THICK ORE ZONES

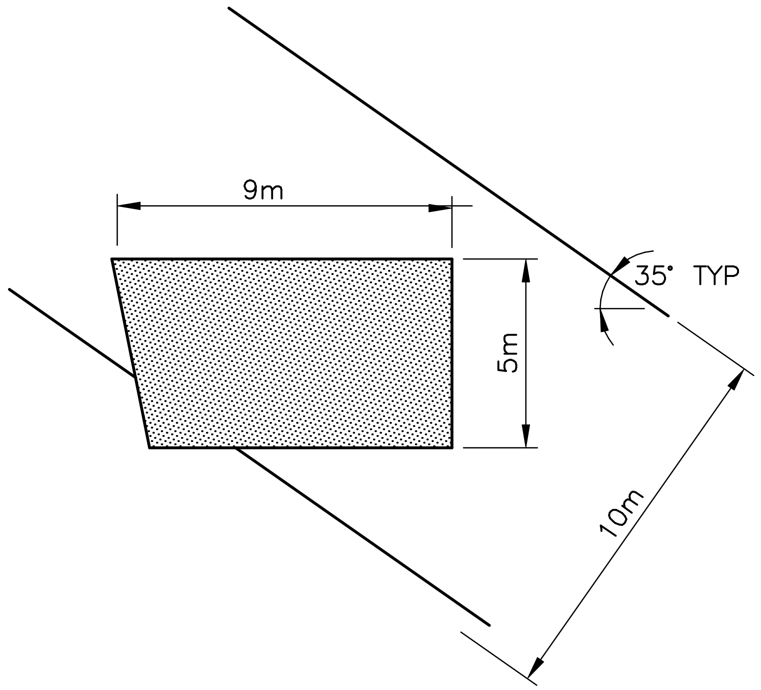
PROJECT No. 0263–001

DWG. No. FIGURE 6

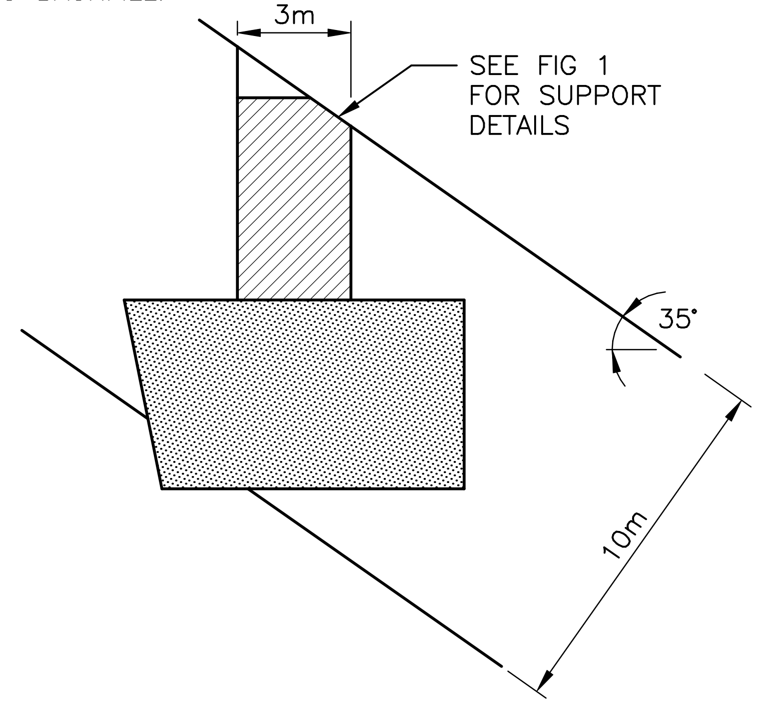
REV. A

Figure 7 Stoping Sequence for 10m Thick Zone

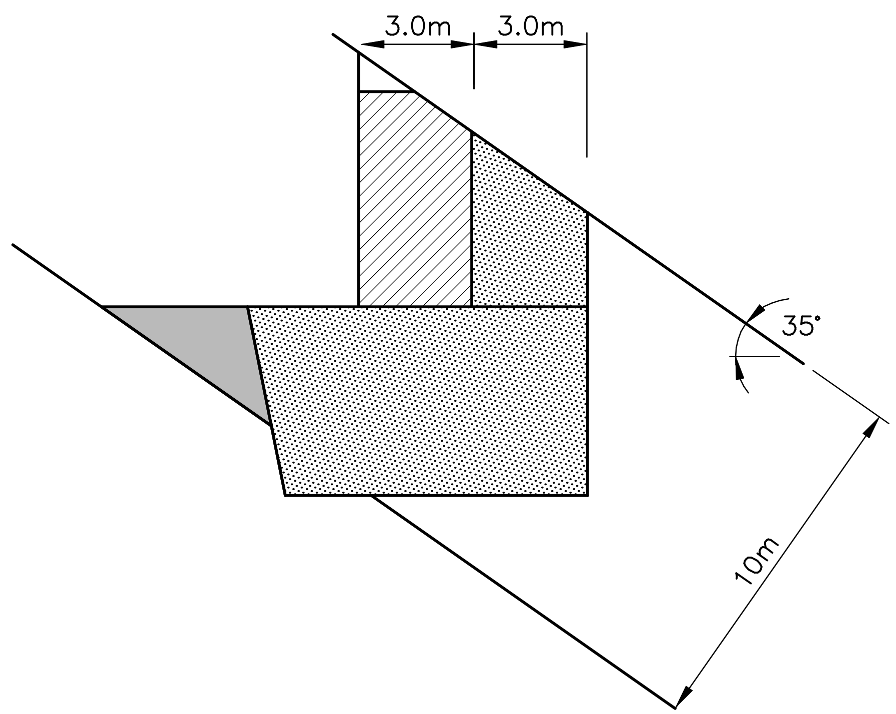
STEP 1:
EXCAVATE 9m WIDE CUT IN ORE AND BACKFILL WITH UNCEMENTED HYDRAULIC FILL OR UNCONSOLIDATED RUN OF PIT FROM SURFACE BORROW PIT.



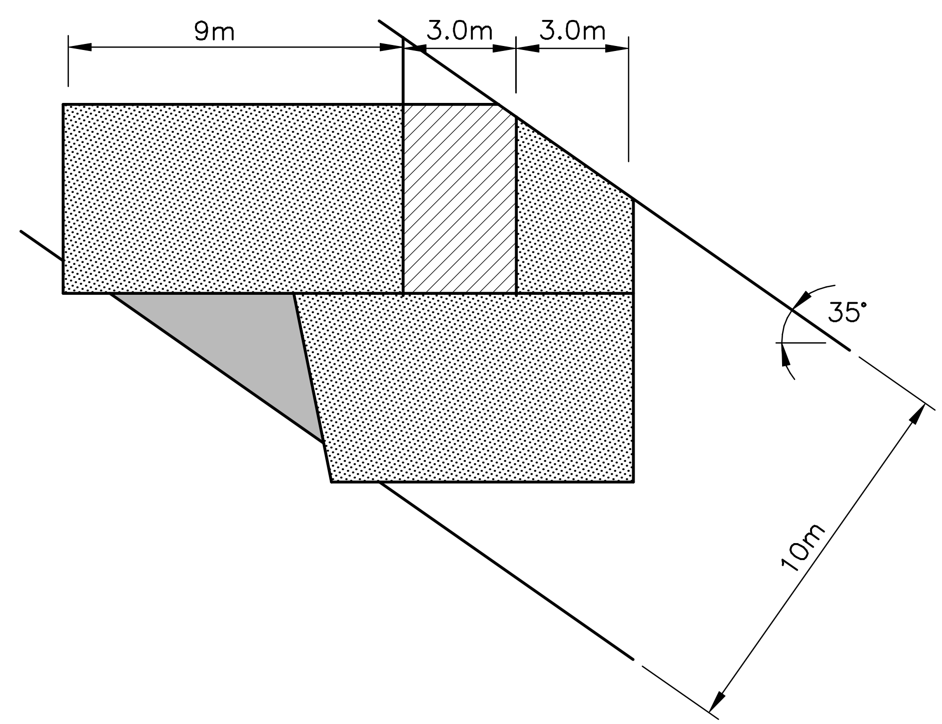
STEP 2:
EXCAVATE 3.0m WIDE CUT TO EXPOSE HANGINGWALL. BACKFILL WITH CEMENTED HYDRAULIC BACKFILL.



STEP 3:
EXCAVATE 3.0m CUT AND BACKFILL WITH UNCEMENTED HYDRAULIC FILL.



STEP 4:
EXCAVATE 9m IN ORE AND BACKFILL WITH UNCEMENTED FILL. REPEAT CYCLE.



LEGEND

UNCEMENTED HYDRAULIC BACKFILL

CEMENTED HYDRAULIC BACKFILL

REV.	DATE	REVISION	DRAWN	CHECKED	APPROVED

SCALE: 1 : 200
DATE: SEPT 2000
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CLIENT: EXPATRIATE RESOURCES LTD.

PROJECT: WOLVERINE J/V – GEOMECHANICS ASSESSMENT

TITLE: STOPE SEQUENCING FOR 10m THICK ORE ZONES

PROJECT No. 0263–001

DWG. No. FIGURE 7

REV. A

4. BACKFILL

Mine backfill is an integral part of the proposed mining method and will serve three principal purposes:

- To allow a high level of extraction of the ore;
- To permit extraction of the orebody in slices by limiting exposed spans in the hangingwall to 4m;
- To enhance regional mine stability by limiting convergence and relaxation of stresses in the hangingwall and footwall; and,
- To provide a working platform as mining progresses.

Much of the backfill required for mining of the Wolverine and Lynx deposits will be uncemented. Some cemented fill is required for sill pillar recovery and for obtaining high recovery in wider sections of the deposit.

4.1 Backfill Options

A number of backfill options have been considered including:

- Paste backfill using unclassified mill tailings;
- Hydraulic backfill using local sand borrow sources;
- Rockfill using gravel and cobble from local borrow sources.
- Hydraulic backfill using classified tailings;

Paste backfill is considered to be the least practical option for this project for the following reasons:

- The mill is located approximately 30km (trucking distance) from the mine. Transporting the paste this distance by truck or pipeline would be very costly. Dewatering the unclassified mill tailings to transport them as a filter cake would also require very large and expensive filtering equipment.
- Paste backfill plants are generally 8-10 times more expensive than conventional hydraulic backfill plants.
- Paste backfill requires cement in order for it to consolidate. This means that cement would be added to the paste backfill simply to make it hard enough to walk on, which would be an unnecessary use of cement.

During the site visit, sources of sorted sand were investigated for possible use as hydraulic backfill. If such deposits exist, they could be mined during the summer months to produce a sand stockpile that could be used for backfill throughout the year. There were no borrow sources identified between the airstrip and the camp that would provide a high yield of sand. The most extensive unconsolidated soil deposits near the mine are at the airstrip (see Photo 1). This material consists of mixed sand, gravel to cobble and boulder size material. A grain size analysis is of this material is provided in Figure 8, showing that the sand size fraction accounts for only 22% of the material. The sample contains 3.4% minus 200 mesh, which should be acceptable for hydraulic fill. However, it is not known if this is consistent throughout the deposit. A washing plant would probably be needed to ensure the backfill was consistently clean and well draining. It would be possible to set up a screening and wash plant near the airstrip to produce the required backfill.

It is possible that during construction of the road to KZK, a clean, reworked sand deposit is identified. This material could be trucked as a backhaul to the mine. Such a source, if present, would be more desirable than tailings from a cost and scheduling perspective because:

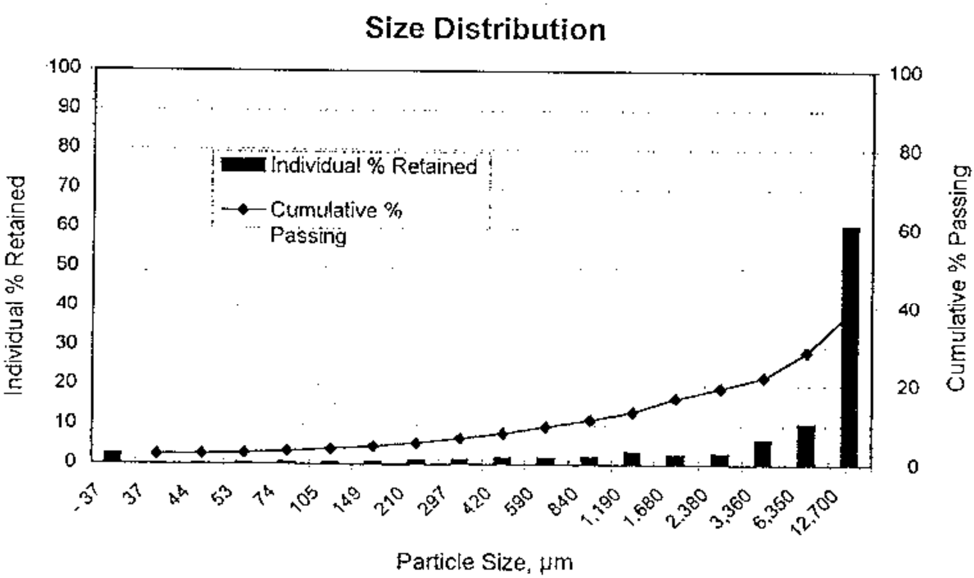
- Borrow sand mining could be carried out for approximately one month each year by a contractor on a campaign basis eliminating the need for purchasing screening and stockpiling equipment;
- the material would likely be dry, whereas the classified tailings would be wet and would could potentially freeze without special heating; and,
- acid generating tailings may become acid generating during storage thus necessitating a special storage area at the portal site.

It is recommended that as part of the KZK-Wolverine road engineering, a survey of potential sand borrow sources be carried out.

Another option is for the unsorted sand, gravel and cobble material to be mixed with a cement slurry and placed in the stopes as a concrete-like- backfill placed by truck and rammer-jammer. This option is considered technically feasible for portions of the stope requiring cemented backfill, however where if uncemented fill is required, the loose sand and gravel could not be compacted dense enough to produce a sufficiently stiff working surface that equipment could travel on. A mixture of the airstrip borrow material and development waste on the bottom half of the stope, with a hydraulic fill on the top half of the stope would ensure tight filling and good trafficability.

Figure 8 Airstrip Borrow Material Grain Size Distribution

Sieve Size		Individual	Cumulative
Tyler Mesh	Micrometers	% Retained	% Passing
1/2"	12,700	60.9	39.1
1/4"	6,350	10.4	28.7
6	3,360	6.4	22.3
8	2,380	2.9	19.4
10	1,680	2.5	16.8
14	1,190	3.3	13.5
20	840	2.1	11.4
28	590	1.8	9.7
35	420	1.8	7.9
48	297	1.3	6.6
65	210	1.1	5.5
100	149	0.8	4.7
150	105	0.7	4.0
200	74	0.6	3.4
270	53	0.5	2.9
325	44	0.2	2.7
400	37	0.1	2.6
Undersize	- 37	2.6	-
TOTAL:		100.0	



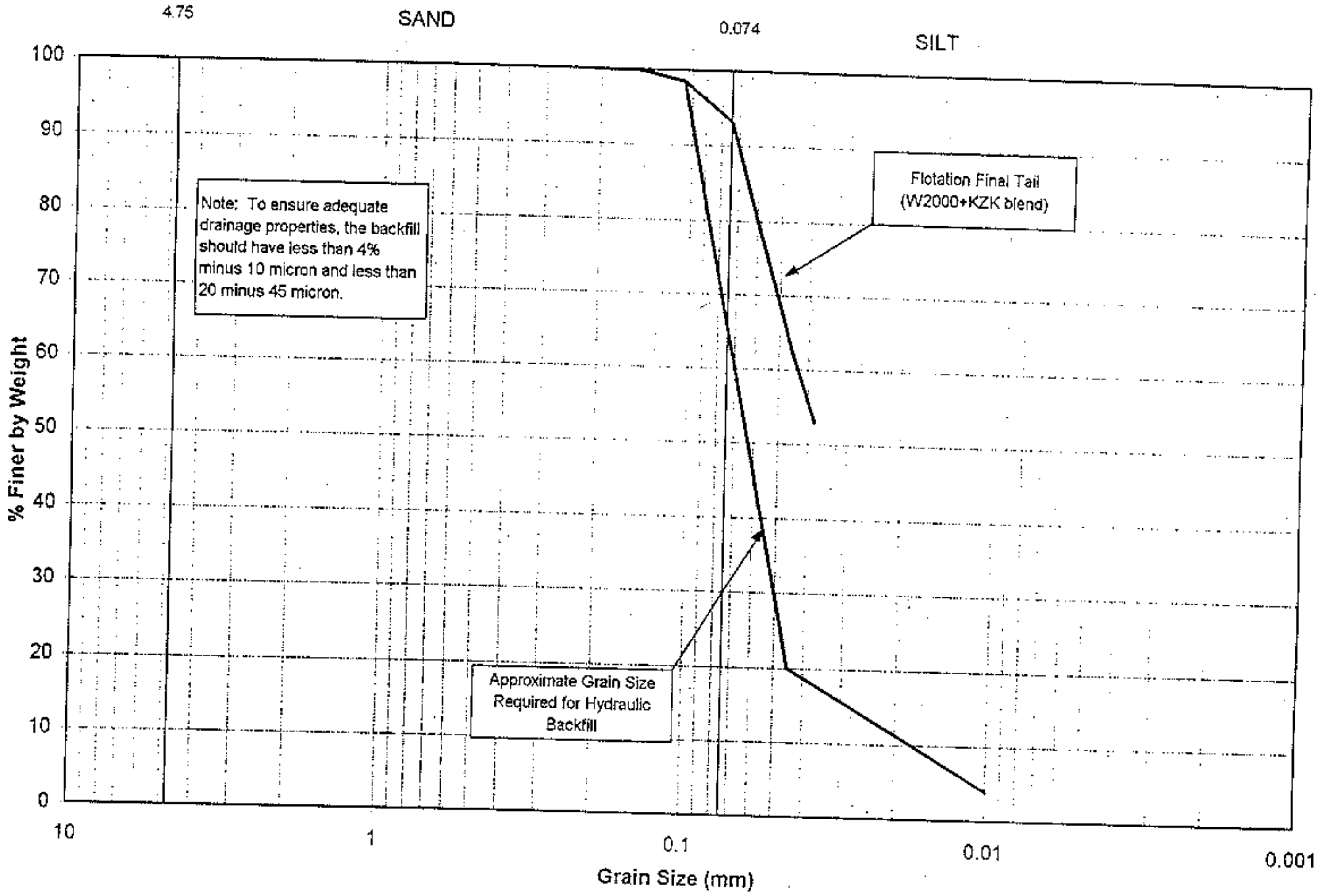
Hydraulic backfill has the benefit of being inexpensive in terms of capital cost and delivery cost. Although cement usage is higher than for paste fill for equivalent strengths, the strength requirements for Wolverine stopes are not significant. Furthermore, the stoping technique does not require a large proportion of cemented backfill. Hydraulic backfill is the most favoured backfill option for the Wolverine deposit. Unless a suitable source of borrow sand is located it is recommended that the feasibility study be based on hydraulic backfill derived mainly from classified mill tailings.

Since the mill capacity is 4250 mtpd, it is believed that sufficient sand can be produced from the tailings stream. Use of classified tailings has the additional benefit of reducing the volume of tailings that must be stored on surface. Most of the sand fraction will be pyrite and therefore advantageous to store underground as backfill. The mine will be developed with a decline access and all workings will be below the elevation of the portal, making the backfilled areas naturally flooded upon mine closure.

A preliminary hydrocyclone simulation performed by Krebs Engineers Ltd. on the tailings grain size distribution indicates a recovery of approximately 26% of the tailings as sand backfill may be possible (Appendix III). The grain size distribution of the tailings and backfill are provided in Figure 9. Assuming a tailings flowrate of 126 tonnes per hour, the classified backfill sand production will be 32.8 tonnes per hour.

Figure 9 Hydraulic Backfill Grain Size Distribution

Estimated Grain Size Distribution for Hydraulic Backfill Using Classified Tailings



4.2 Backfill Design Criteria

4.2.1 Backfill Quantities

The estimated amount of backfill required has been calculated and is summarized in Table 8. The most recent waste development schedule calls for between about 25,000 and 75,000 tonnes per year of waste. For design purposes, it is assumed that on average, 25,000 tonnes of development waste rock is placed as backfill. This rock will be end-dumped in the larger mined-out stopes, prior to placing hydraulic fill.

Table 8 Backfill Quantity Estimate

Tonnes Mined Per Year	456,250 tonnes
Specific Gravity (Ore)	4.0
Dilution (10% assumed)	45,600 tonnes
Specific Gravity (HW/FW rock)	2.7
Void Space to be Filled	130,900 m^3
Waste Rock Fill Tonnage	25,000 tonnes
Waste Rock Fill Volume	12,900 m^3
Hydraulic Backfill Required	118,000 m^3
Dry Bulk Density of Placed Hyd. Fill	2.5 tonnes/m^3
Tonnes Classified Tailings Required	295,000 tonnes/yr

This classified tailings requirement of 295,000 tonnes/yr closely matches the estimated production of 302,000 tonnes per year. Any short-term backfill requirements for additional fill that cannot be met by the mill, can be supplied by end dumping run of pit borrow material in stopes prior to placing hydraulic fill.

4.2.2 Backfill Cement Requirements

Hydraulically placed classified tailings have the advantage over paste backfill that when drained provide an adequate working surface without the addition of cement. A significant portion of the backfill placed in the Wolverine and Lynx Zones will be uncemented. For portions of stopes requiring cemented fill, the cement content will vary according to the following criteria. Backfill must serve four main purposes in the stopes. First, it must stand up as a vertical wall from 3 to 6m high depending on the ore thickness (refer to Figures 5 to 7). Secondly, the bearing capacity of the fill must be high enough to provide a working surface on which equipment can operate. Thirdly, the backfill should be placed sufficiently tight to the back to confine any potential displacement and to effectively limit the exposed span. Fourth, the fill must be capable of developing sufficient strength when cemented, to facilitate sill pillar recovery.

The bearing capacity of the uncemented fill is estimated 100 kPa using an assumed friction angle of 33^o and zero cohesion. The estimated pressure exerted from the tires of equipment is 345 kPa. Therefore, the uncemented fill will not support the 5 yd LHD's and trucks during tramming and mucking without penetrating the surface of the fill. They will likely start to wear ruts in the fill. It is recommended that on uncemented fill surfaces, mining advance by leaving a 0.3-4m thick surface of ore on the floor of the stope on which equipment would work. Just before the stope is to be filled, this ore would be carefully removed with an LHD. Over the life cycle of a stope, this procedure should minimize the loss of ore in the backfill and dilution of backfill in the mucked ore. Alternatively, a 0.3m thick crust of cemented fill with a cement content of approximately 6% could be poured on the top of the fill surface. Both options have advantages and disadvantages and should be tested when the mine is in production.

The uniaxial compressive strength (UCS) required for a vertical face of cemented backfill can be estimated using the formula below developed by Mitchell (1983).

$$UCS = \frac{\gamma H}{1 + H/L}$$

Where γ is the unit weight of the fill, H is the fill height, and L is length of the fill exposure.

Assuming a backfill unit weight of 24.5 kN/m^3, the cement requirements for various backfill pours are provided in Table 9. The cement contents necessary to achieve the strength to resist rotational and planar failure, and also to withstand the blasting and equipment impact, have been estimated from similar operations. As part of the final feasibility, it is recommended that cylinders be cast with varying classified tailings/cement ratios and tested in uniaxial compression. It is traditional at mining operations to express the cement content on a weight percent basis. Given that the fill may consist of either tailings or borrow sand, with dramatically different S.G.'s, we have provided the estimated cement requirements in units of kilograms cement per cubic metre of backfill. The cement contents in Table 9 are expected to produce the desired strength in 2 weeks.

Table 9 Backfill Cement Requirements

Fill Height	UCS Required by Mitchell Equation	Cement Content (kg/m^3)
2	50 kPa	50
4	98 kPa	50
6	147 kPa	75

4.3 Backfill Water

There is no year-round source of water near the proposed decline portal where a backfill plant would be located. A settling pond will need to be established near the portal to settle suspended solids from the mine water. Water could be reclaimed from the pond when needed to make backfill slurry. It is anticipated that a water treatment plant will be necessary to treat mine water.

4.4 Mine Backfill Plant and Re-Slurrying Plant

Due to the very fine grind of the tailings, it is recommended that the backfill preparation plant at the mill consist of a two-stage hydrocyclone plant. After cycloning, the cyclone underflow should be filtered to reduce the moisture content to approximately 15%, and stored in a bin with a truck load-out capability. Trucks hauling ore to the concentrator from the Wolverine deposit will backhaul the classified tailings to the mine and store it on a covered concrete pad near the re-slurrying plant. Any runoff from the stored backfill would be collected from the pad and directed into the mine discharge settling pond.

The re-slurrying plant at the Wolverine Mine should be located approximately on the current exploration road, above the main access decline such that the slurry will flow by gravity through a lined borehole to the access decline. The re-slurrying plant would consist of a loader fed vibrating sand hopper, conveyor belt, agitation tank, water tank, and an instrumentation package to control the slurry density within a set range. The backfill distribution lines should be sized to maintain a flow velocity of 3m/s.

During operation, the re-slurrying plant would require 1 loader operator and one plant operator. Backfilling would be carried out generally on dayshift. Actual backfilling would occur from 7-8 hours each day.

5. TEST MINING PHASE

A test-mining program has already been proposed to assess the ground conditions in greater detail and to confirm the proposed mining method is suitable for the ground conditions. This program is considered especially important for the Wolverine deposit given the generally poor quality of the hangingwall and footwall rocks. During the test mining, provision should be made for backfilling of the stopes using screened sand from the area near the airstrip. Some special provisions for long term storage of the ore and waste rock from the test-mining program will need to be made.

5.1 Dewatering Prior to Test Mining

It is recommended that a dewatering program be undertaken in the area of test mining before actual stoping is carried out. This is intended to simulate actual mining conditions since the mine development (in ore) will normally drain much of the mine in advance of production. It is recommended that the test-mining schedule include a period where the stope is developed and then allowed to drain for one month. During this time, the groundwater inflow quantities would be monitored. Depending on the monitoring results, additional drain holes would be drilled to further drain the formation, or if it drains quickly, mining could resume in less than one month.

5.2 Geotechnical Monitoring Program

Prior to test mining, six standpipe piezometers should be installed in exploration drill holes across the deposit. Two of the holes should be located in the area of the proposed test mining. Each piezometer should have an intake zone at least 50m below the water table. Prior to installing the piezometers, slug tests should be performed to estimate the hydraulic conductivity of the rock mass at various depths.

During the test-mining phase, it is recommended that an experienced rock mechanics engineer be on site to:

- Perform geotechnical mapping and rock mass classification to characterize the various rock types
- Observe the performance of the installed ground support and modify the ground support recommendations as necessary;
- Install and monitor geotechnical and hydrogeological instrumentation;
- Carry out numerical modelling of the excavation using finite element or boundary element techniques using measures stress changes to calibrate the numerical model. Once calibrated, the model can be used to predict conditions later in the mine life.

Six vibrating wire stress meters and 12 ground movement monitors should be purchased for installation during the test mining phase. The stress meters will be installed in the hangingwall of the stope to monitor the relaxation of the stope back as the stope advances. The stress gauges will also be installed in sill pillars to monitor the stress increase as mining advances. This data will allow the engineering staff to adjust the size of stopes and pillars.

At least four flat jack stress cells should be installed in the mine backfill to monitor the stress increase in the fill as the stope advances. The degree to which stress is transferred (if at all) from hangingwall to footwall will be an important influence on hangingwall and pillar stability.

The geomechanics investigation could be carried out be one full time rock mechanics engineer at the site during the test mining. An allowance of approximately $18,600 for geotechnical instrumentation should be budgeted.

Table 10 Test Mining Geotechnical Instrumentation

12 Ground Movement Monitors	$4,800
4 Flat Jack Stress Cells	$4,000
Six vibrating wire stress gauges	$4,800
Vibrating Wire Readout Box	$1,500
Piezometer Pipe and Tips	$1,500
Installation Tools and Accessories	$2,000
Total	$18,600.00

6. CONCLUSIONS AND RECOMMENDATIONS

6.1 Conclusions

The following conclusions are made with respect to ground conditions and backfill options for the proposed Wolverine Mine:

1. Ground conditions in the hangingwall and footwall of the deposit are classified as poor to very poor. Almost all permanent mine development headings will require rockbolting and shotcrete support.
2. Ground conditions in the ore are classified as fair to good. Ground support will consist of pattern rockbolting and some cable bolting to support local wedge blocks.
3. The mine access decline should be collared and driven in the hangingwall iron formation to an approximately Elev. 1275m before crossing through the rhyolite and argillite to the ore.
4. Mine planning should attempt to limit the exposure time of the stope hangingwall rocks to approximately one month.
5. A preliminary estimate of the steady state groundwater inflow to the mine is 180 litres per second. A treatment plant constructed at this capacity should be capable of handling expected high inflow rates during initial mine development.
6. All mine workings should be backfilled as tightly as possible to improve stability within the stope, reduce mine subsidence, and reduce stress build-up in mine pillars. The backfill will be a combination of development waste rock and hydraulic backfill. The hydraulic backfill will consist of primarily of classified mill tailings, however screened borrow sand can also be used.
7. Preliminary test work performed on the combined KZK-Wolverine tailings indicates that approximately 26% of the tailings stream can be recovered as sand backfill. Additional laboratory testing is required to confirm the percolation rate and drained bulk density of the classified tailings.

6.2 Recommendations

The following recommendations are made for future work that would form part of the full feasibility study:

1. Carry out an interpretation of the faulting observed in drill core, and plot these on the geologic sections. This will be particularly useful in planning the mine development and test mining.
2. The increase in stress as mining progresses may lead to a gradual deterioration of the access decline pillar. During feasibility level studies, numerical modelling should be carried out to assess the affect of these changes. This in turn will guide the long-term mine planning.
3. Carry out a survey of potential sand borrow sources within approximately 1km of the proposed KZK-Wolverine road alignment. Ideally, the source would be as close to Wolverine as possible, a high sand content, and less than approximately 3% -200 mesh. This material would be used as a back-up source of sand in the event that the mill cyclone plant cannot produce the required quantity. Some gravel and cobble size material would be acceptable as it could be screened and utilized for road crush in the pit, blast hole stemming, and other purposes.
4. Perform a laser particle size analysis on the concentrator tailings stream to determine the grain size distribution in the minus 400 mesh size range. This information is necessary to more accurately assess the amount of backfill that can be recovered from the tailings.
5. A mine water treatment plant and a backfill re-slurrying plant will be required at the mine site. Consideration should be given to combining these facilities along with offices and other site facilities into a single building.

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Per:

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Senior Mining/Geotechnical Engineer

Scott Broughton, P.Eng.
Project Manager

References

Franklin, J.A., L., Zhang, (1992). Water Inflow Prediction in Mines. Page101-Page107. Proc. Canadian Rock Mechanics Symposium, Sudbury, ON

Golder Associates (1997). Report on Preliminary Geotechnical Assessment of the Wolverine Project, Report Prepared for Westmin Resources Ltd., Vancouver, B.C.

Lang, B (1992) Master of Applied Science Thesis, Dept. of Mining and Mineral Process Engineering, University of British Columbia, Vancouver.

APPENDIX I

SITE PHOTOGRAPHS

APPENDIX II

YEAR 2000 GEOTECHNICAL DRILL LOGS

APPENDIX III

BACKFILL PLANT HYDROCYCLONE SIMULATION