Wolverine Project

MINE DEVELOPMENT AND OPERATION

VERSION 2006-01

Prepared by:
Yukon Zinc Corporation
Vancouver, British Columbia

April 2006

Table of Contents

List of Tables

List of Figures

1 Mining

1.1 Introduction

The Wolverine Mine will be a ramp-accessed underground operation with a daily production rate of approximately 1440 t/d. The current reserve of 5.2 Mt provides a projected mine life of approximately 10 years.

The current reserve extends from 1350 to 1080 m elevation, with the orebody open in the down-dip direction.

The mine currently has one portal at 1345 m elev providing access to the main ramp, which is sized at 5 m x 5 m. Approximately 450 m of development was driven in 2005 as part of an exploration program. A second portal, the 1360 Portal will be driven during the operations phase. This will be the primary entrance and exit for all men, equipment and supplies.

The ramp will be located centrally between the Lynx and Wolverine zones above and in the thinner 'saddle zone'. The ramp will continue in the hanging wall of the deposit for an additional 315 m, then it will be driven in the thin ore of the 'saddle zone' for the remaining 1150 m.

Drift and Fill will be the mining method. Three variations on the method were developed for application to different orebody thicknesses.

Paste backfill will be the primary fill for the mine. The paste plant will be located at the north end of the industrial complex, closest to the portals. Mine development waste rock and DMS flotation rock will also be used as loose unconsolidated fill in applications where it will not be exposed by future adjacent mining.

Mining will be undertaken using diesel-powered mobile equipment. Drilling will be accomplished using electric hydraulic drill jumbos. A rockbolting machine will be used for bolting. Scooptrams will have 6.1 m^3 and 4.6 m^3 buckets. The ore and waste haulage trucks will be 50 t units.

A maximum of 165 m^3/s of ventilation will be required based on a projected equipment list for the mine. An upcast system is anticipated, with fresh air introduced to the mine via one central fresh air raise, with exhaust flowing out of two exhaust raises and the portals. Direct-fired propane heaters will be installed above the two fresh air raises to maintain above-freezing temperatures in the mine throughout the year.

Power to the mine will be supplied by the main generators at the industrial complex. Power will be delivered in cable run down the main ramp via the 1345 Portal. An industrial Ethernet cable will be employed for mine radio and other communications.

Compressed air will be delivered from the main compressors located adjacent to the industrial complex.

1.2 Geotechnical Evaluation

1.2.1 Introduction

Dr. Khosrow Aref, P.Eng. of Rockland Ltd. provided a geomechanical mine design assessment of the Wolverine deposit. He was retained to provide geotechnical input and, in particular, to analyze the collected geotechnical data, assess stope and pillar sizes and recommend ground support systems. The following section summarizes the results of various geotechnical investigations and provides the geomechanical mine design guidelines. Appendix A contains the complete report and includes the guidelines for the stope excavations and mine development headings, including recommendations for maximum span designs, stand up times, ground support requirements, pillar design and geomechanical aspects of the mining method.

1.2.2 Data Collection

1.2.2.1 Site Visits

Three site visits were undertaken during the course of this investigation. The first visit was preliminary in nature and intended to review typical drill hole cores and collect geotechnical data. The second site visit, carried out at the early stage of the test-mining program, was designed to assess the geotechnical aspects of decline and suitability of the ground support system. Three instruments and equipments: a tape extensometer, the point load test apparatus and pull test equipment were purchased for the rock mechanics program. During the site visit, training of the staff with these instruments/equipments for their applications was carried out. The third site visit was planned towards the end of the test-mining program and was directed to evaluate the geotechnical aspects of the decline and test stope area. During each site visit a number of meetings and discussions were held with the project geologists, geotechnical technicians, engineers, mine planners and mining contractors.

1.2.2.2 Data Available

The following information was made available for use in this investigation:

- Geological drill hole logs: from WV 00-113 to WV 00-120, from WV 04-121 to WV 04-131, from WV 05-132 to WV 05-188, from 95-01 to WV 97-99, from WV 00-01 to 00-03
- Geotechnical drill holes logs: 9501GEO to 95GEO24, WV 96-25, WV 97-103, GeoTech WV 05-133 to GeoTech WV 05-189
- Drill hole information (coordinates, dip and azimuth)
- Wolverine deposit drilling map (Scale of 1:1500)
- Preliminary geology of decline and test stope area map (Scale of 1:500)
- Decline excavation and test stope area
- Typical mining methods for the Wolverine project

Geological characterization for the geotechnical analyses is included in Appendix A.

1.2.2.3 Drill Hole Data

YZC geological staff collected geotechnical data based on a document prepared by Golder Associates. Geotechnical parameters that were measured in the core include recovery per run, RQD, fracture per run, point load index or field strength index, weathering index, presence of faults or broken core, joint set number, joint alteration number, minimum average discontinuities spacing, condition of discontinuities and ground water. A review of the supplied data indicated that these parameters were not collected for all holes. A total number of 57 drill holes from the 2005 drill program were used to assess the rock mass quality. The location and distribution of these drill holes in the Wolverine, Saddle and Lynx zones are shown in Appendix A, along with the compiled drill hole data. In order to characterize and compare the rock mass quality across the Wolverine deposit, each zone was divided into the upper, center and lower levels. Figure 4 of Appendix A shows the geomechanical domains for the Wolverine, Saddle and Lynx zones.

YZC staff collected the rock mass classification parameters (RQD, Jn, Jr and Ja) during the geotechnical core logging. The majority of these ratings were initially collected based upon the core run. Subsequently, YZC staff adjusted the information based upon the rock units, to reflect the geomechanical domains. According to the point load strength tests, examination of drill hole cores and review of core logs, the ore zone has a higher uniaxial compressive strength and is much more competent than the immediate hanging wall and footwall zones.

1.2.2.4 Test Mining Program

A test-mining program at Wolverine was carried out to provide detailed information on the geological, geotechnical, mining, recovery and cost aspects of the project. The ground support requirements, and the mine design itself could be estimated on the basis of geotechnical information gathered from the exploration drill core. However, the reliability of geotechnical data collected in drill holes could be compared to the actual conditions underground during the test mining.

The program consisted of a decline, which was collared in the southwest of the deposit. The decline is approximately 450 m at a -15% grade and has dimensions of approximately 5 m wide by 5 m high in cross section and was sized for production. The working passed through a number of rock types and extended into the ore to expose the hanging wall (HW) and to evaluate the proposed mining method. The passage is considered to be representative of other areas of the deposit.

1.2.3 Rock Mass Rating and Barton’s Q

The two most widely used rock mass classifications in the mining industry are Bieniawski’s RMR (1976, 1989) and Barton et al. Q (1974). The RMR and Q rock mass classifications were employed to assess stability at Wolverine. Both classifications, through extensive case histories, specify the maximum stable unsupported span against various rock mass qualities. Each method incorporates geological, geometric and design / engineering parameters to derive a qualitative value of the rock mass quality. The similarities between RMR and Q stem from the use of identical or very similar parameters for calculating the final rock mass quality rating. The differences between the systems lie in the weight given to similar parameters in each scheme. Ideally, both rock mass classification systems should be used and compared in any project.

The geotechnical parameters were reviewed to characterize the rock mass quality of the Lynx, Saddle and Wolverine zones. The objective was to identify any distinct geomechanical differences between these zones and their HW, ore and FW (footwall) zones. The geomechanical domains of approximately 5 m above and below the ore zone were considered for the HW and FW. The range and the most typical rating of each parameter were established.

Barton et al. (1974) and Barton (1988, 1994) described the application of the Q system for rock mass classification as the determination of no-support limits for various types of excavations. Assuming dry to minor inflow condition (Jw =1) and a medium stress environment (SRF = 1), Q values were calculated. Table 1-1 presents Q values and corresponding rock mass descriptions for various zones.

Table 1-1: Summary of Typical Rock Mass Quality for the Wolverine Deposit

Zone	Levels	HW/ORE/FW	Q	Description
Lynx	Upper	HW	1.3	Poor
		Ore	120	Extremely good
		FW	0.6	Very poor
	Center	HW	0.8	Very poor
		Ore	135	Extremely good
		FW	0.7	Very poor
	Lower	HW	2.3	Poor
		Ore	168	Extremely good
		FW	2.5	Poor
Saddle	Upper	HW	0.4	Very poor
		Ore	64	Very good
		FW	1.6	Poor
	Lower	HW	1.3	Poor
		Ore	75	Very good
		FW	2.6	Poor
Wolverine	Upper	HW	1.1	Poor
		Ore	22	Good
		FW	0.5	Very poor
	Center	HW	0.03	Extremely poor
		Ore	75	Very good
		FW	1	Poor
	Lower	HW	1.4	Poor
		Ore	90	Very good
		FW	2	Poor

The rock mass quality ratings highlight the distinct separation between ore and immediate HW/FW. The ore rock quality can be described as ‘good to extremely good’. With the exception of the ‘extremely poor’ HW rock quality in the center level of the Wolverine zone, the rock mass quality in the immediate HW and FW is rated as ‘very poor to poor’. In the upper level, Saddle’s HW is poorer than Wolverine and Lynx, however; it has a higher rock mass quality for the FW. In the center level, Saddle’s HW has a higher rock mass quality than Lynx and Wolverine. In lower levels, Lynx and Wolverine have similar HW and FW. A comparison of the ore rock mass quality indicates that Lynx has a higher rating than Saddle and Wolverine zones for all levels.

1.2.3.1 Empirical Evaluation

Based upon the geotechnical core logging, the typical Q ranges of the immediate HW, ore and FW zones have been established (Table 1-1). With the exception of the ‘extremely poor’ HW rock quality in the center of the Wolverine orebody, the rock mass quality in the immediate HW and FW has a range of 0.4 to 2.3 (Table 1-1). Using an ESR of 1.6, the maximum unsupported span ranges from approximately 2.5 to 5 m respectively. The ore has Q values ranging from approximately 22 to 168 (Table 1-1). Using an ESR of 1.6, the maximum unsupported span ranges from 11 to 24 m.

These spans are possible in the absence of continuous, adversely-oriented geological structures, which normally control stability. In other words, the suggested unsupported span is applicable where the Q values are consistently equal or greater than specified values. Areas with lower Q values are obviously unable to sustain the suggested spans. In such conditions, provision must be made for additional ground support.

1.2.3.2 Rock Mass Quality of the Test Mining Area

Several underground visits were made to identify various geomechanical zones along the decline and test stope areas. The major lithologies encountered during the test mining were: ARMS (Argillite), ARGR (Graphitic Argillite), RHFS (Siliceous Siltstone), EXCP (Calcite Pyrite Exhalite), SSMS (Massive Sulphide), ARCL (Chlorite Altered Argillite) and RHSR (Sericite Altered Rhyolite or Argillite). The rock mass quality was estimated based on the Q (Barton 1974) and the RMR (Bieniawski 1989) rock mass classifications. A total number of eight locations, representing typical rock mass quality along the decline and test stope areas, were selected. Table 1-2 presents a summary of rock mass classification results at the selected locations. The photographic records of underground observations and their locations are identified in Appendix A.

Table 1-2: Rock Mass Quality along the Decline and Test Stope Area

Location	Rock Type	Q	Description	RMR	Description	Photograph
1	ARMS	0.01-1	Extremely poor – very poor	18-35	Very poor - poor	1 to 4
2	EXCP	1.9-9	Poor - fair	52-70	Fair - good	5 to 7
3	EXCP	1.9-9	Poor – fair	52-70	Fair - good	8 to 10
4	ARMS	0.08-0.6	Extremely poor – very poor	20-35	Very poor - poor	11 to 13
5	SSMS	4.1-6.6	Fair	49-67	Fair - good	14 to 16
6	ARMS-SSMS	4.1-6.6	Fair	49-67	Fair - good	17 to 21
7	SSMS	4.1-6.6	Fair	49-67	Fair - good	22 to 25
8	SSMS	4.1-6.6	Fair	49-67	Fair - good	26 to 29

1.2.4 Ground Support

Rock mass classification and experience gained during the test-mining program were utilized to prepare ground support guidelines. Table 1-3 presents the support guidelines based upon the geomechanical zones/rock types for the back and walls. The main ground support systems in the ore and hanging wall are resin rebar/mesh and split set/fibre reinforced shotcrete (FRS) respectively. In the very poor ground conditions, provision will be made for steel set arches and timber support. The main type of ground support on the walls is split set. The experience during the decline excavation in the ‘very poor to poor’ ground condition suggests that the back will be supported immediately after the blast. The application of a thin layer of FRS to the rock surfaces will improve the stability.

FRS will be a major component of the ground support system in the poor rock mass quality. Since the performance of shotcrete depends on its quality, the application of high quality FRS will only be considered.

The stability of pillars was evaluated based upon a number of assumptions and empirical methods. Numerical modeling will be employed for pillar and stope stability assessment during the mining operation. The modeling will allow pillar stresses, and in particular where multiple openings are present, to be more accurately evaluated.

Table 1-3: General Guidelines on Support Type and Density for the Back and Walls

Geomechanics Zone/ Rock Type	Back/ Walls	Function	Opening	Operating Life	Span m (ft)	Support Type	Bolt Length m (ft)	Spacing[1] cm (in)	Shotcrete[2] cm (in)	Comments
Argillite or similar rock type	Back	Access	Decline	Long	5 (16’)	Split set + mesh + some resin rebar	2.4 (8’)	1.2 x 1.2 (4’ x 4’)	50 (2”)	Also requires timber support, steel set arches 5 m (10’) span and straps for occasional use.
		Intersection	Decline intersection	Long	>5 (16’)	Split set + mesh + some resin rebar	2.4 to 4.5 (8’ to 15’)	1.5 x 1.5 (5’ x 5’) to 1.8 x 1.8 (6’ x 6’)	50 (2”)	Use of final support depends on geometry of intersection.
	Walls	Access	Decline	Long	5 (16’)	Split set + some mesh	2.4 (8’)	1.5 x 1.5 (5’ x 5’)	25 (1”)	n/a
		Intersection	Decline intersection	Long	>5 (16’)	Split set + mesh	2.4 to 4.5 (8’ to 15’)	1.5 x 1.5 (5’ x 5’)	25 (1”)	n/a
EXCP or similar rock type	Back	n/a	Decline	Long	5 (16’)	Resin rebar + mesh + some split set	2.4 (8’)	1.2 x 1.2 (4’ x 4’)	n/a	n/a
	Walls	n/a	Decline	Long	5 (16’)	Split set + some mesh	2.4 (8’)	1.8 x 1.8 (6’ x 6’)	n/a	n/a
Ore	Back	Stope	Herringbone Primary	Short	4 (13’)	Resin rebar + mesh + some split set	1.8 to 2.4 (6’ to 8’)	1.2 x 1.2 (4’ x 4’)	n/a	Close spacing because men constantly working under exposed back.
			Herringbone Secondary	Short	4 (13’)	Resin rebar + mesh + some split set	1.8 to 2.4 (6’ to 8’)	1.2 x 1.2 (4’ x 4’)	n/a	Close spacing because men constantly working under exposed back.
	Walls	Stope	Herringbone Primary	Short	4 (13’)	Split set + some mesh	1.8 to 2.4 (6’ to 8’)	1.8 x 1.8 (6’ x 6’)	n/a	n/a
			Herringbone Secondary	Short	4 (13’)	Split set + some mesh	1.8 to 2.4 (6’ to 8’)	1.8 x 1.8 (6’ x 6’)	n/a	n/a

Notes: [1]Spacing values quoted represent minimum. Condition will occur where specific block of ground support density require additional support, resulting in higher densities
[2]Shotcrete is the fibre reinforced shotcrete
Split set is SS33 – Resin rebar is #7 – Mesh is #8 gauge

1.2.4.1 Rock Mass Classification System

Ground support requirements were estimated using the empirical method by Grimstad et al (1993). According to the proposed mining method at Wolverine (see Section 1.3), the permanent mine development headings will be 4 m wide. In Figure 6 of Appendix A, a drift with Span/ESR ratio of 2.5 and Q value of 0.4-2.3 require ground support consisting of fibre reinforced shotcrete (FRS) or mesh reinforced shotcrete and bolts. FRS and bolts are required in the poorer ground condition where Q values drop to approximately 0.4. The shotcrete thickness will be 50 mm (2") in the back. Further information on the application of shotcrete is given in the next section. In a better ground quality (Q>1), pattern bolting is expected to be sufficient. In the ore zone, where Q values range from 22 to 168 and using the Span/ESR ratio of 2.5, application of pattern bolting will be sufficient.

The suggested ground support recommendations are applicable where the Q values are consistently equal or greater than specified values. Areas with lower Q values require additional ground support. It should be noted that the excavation support ratio is related to the use for which the excavation is intended, and the extent to which some degree of instability is acceptable. The calculated span and ground support recommended by this method does not apply to multiple opening situations. This is primarily due to the effect of stress redistribution. Barton’s method is not sensitive to such changing conditions and therefore only provides an appraisal of initial conditions prior to full production mining.

1.2.5 Geomechanical Aspects of Mining

Three different variants of the Drift and Fill mining methods: Drift and Fill Mining with Side Slash, Drift and Fill with Retreat Panels, Drift and Fill with Primary and Secondary Panels, are proposed for the Wolverine Deposit. All three are geomechanically viable and are discussed in Section 1.3.

All mine workings will be backfilled as tightly as possible to improve stability within the stope and reduce stress build-up in mine pillars. The backfill will be combination of paste fill and waste fill.

Knowledge of orebody geometry and major structures ahead of the advancing stope/panels will be an important aspect of the ground support specification, mine planning and dilution control at Wolverine. Recommendations are based upon an interpretation of geotechnical data from drill core and geomechanical assessment of the decline. As more information becomes available, the interpretation must be updated. If significant changes in interpretation occur, the results of the analysis will be reviewed.

1.2.6 Instrumentation and Monitoring

As the mine is developed, a program of instrumentation and systematic monitoring will be instigated. In addition, a record of stoping performance will be maintained. This information will be used to adjust the various geomechanical mine design parameters in order to improve prediction. The main goal of the program is to verify the panel/pillar design and ensure the adequacy of the recommended ground support. The instrumentation and monitoring results can be compared with empirical and/or numerical model predicted displacement and stresses. The calibrated model may then be used to optimize the mine layouts and the design of similar geomechanical domains at the mine. Specific monitoring recommendations are available in Appendix A.

1.3 Mining Method

Drift and Fill is the mining method selected for the project. The primary reasons for selecting Drift and Fill is that a high percentage extraction of the deposit can be achieved, as no permanent pillars are required and thinner zones are mineable.

Stope lifts will be 4 m high and each stope block will be comprised of five stope lifts, a vertical extent of 20 m. Paste backfill with loose waste from the development program and float rock from the DMS plant will be used to fill the mined voids. When one lift is mined and filled, the next will be mined at an elevation 4 m higher, using the backfill of the previous lift as the new floor of the stope. Mining will proceed in this fashion in the up-dip direction until the stope block is completely mined out and filled. The fifth and final lift of each stoping block will be mined beneath the backfill of the adjacent upper stope block, exposing the backfill.

Three distinct variants of the Drift and Fill mining will be employed: Drift and Fill with a Side Slash (DFSS), Drift and Fill with Retreat Panels (DFRP), and Drift and Fill with Primary and Secondary Panels (DFPS). Mining method selection for any portion of a stoping lift will be determined by horizontal ore thickness, as shown in Table 1-4.

Table 1-4: Selection of Mining Method by Ore Thickness

Horizontal Ore Thickness (m)	Mining Method
<4	Drift and Fill with Side Slash (DFSS)
4 to 7	Drift and Fill with Retreat Panels (DFRP)
>7	Drift and Fill with Primary and Secondary Panels (DFPS)

Drawing MO-X-020 shows the three mining methods on a cross-sectional view. Table 1-5 shows the overall percentage of ore by mining method.

Table 1-5: Percentage Mining by Method

Method	Tonnes	% of Reserve	Average tpd	Average Horizontal Thickness (m)
DFPS	3,634,399	70	1005	12.2
DFRP	930,211	18	257	5.9
DFSS	643,736	12	178	2.9
Total	**5,208,346**	**100**	**1441**	**9.9**

Using these proportions, the average steady-state daily production of 1441 tpd will be typically met by 4 drift rounds from DFPS zones, one from DFRP zones, and 4 m of DFSS drifting and slashing. To maintain production rates, the mine will operate as many as four stope blocks simultaneously.

1.3.1 Drift and Fill with Primary and Secondary Panels (DFPS)

For wider zones of ore (>7 m horizontal thickness) the footwall stope drift will be driven 4 m wide in ore along the footwall contact of the ore. Stoping panels will then be excavated at 5 m widths from the footwall stope drift in a ‘herringbone’ fashion at an angle of approximately 45° to the footwall stope drift. These panels will be driven into and expose the argillitic hanging wall contact.

- Most of the mining backs will be in ore, providing a competent back for most stope headings.
- The poor ground of the hanging wall will have minimal exposure, controlling external dilution and enhancing safety for the workers.
- High productivity can be maintained due to multiple working faces.

The panels will be extracted using a primary and secondary sequence. The primary panels will be mined first with solid ore backs and walls. These will then be tight-filled with the fill bulkheads placed as close to the footwall stope drift as possible to minimize the unsupported span. The secondary panels will then be mined between the backfilled primary stopes, with ore in the back and the exposed backfill of the two adjacent primary panels as walls. The secondary panels will then also be filled as tightly as possible. This will be done in a retreat fashion, filling the footwall drift simultaneously.

The hanging wall is primarily composed of very poor graphitic argillite. Excessive dilution is anticipated at the ends of the herringbone panels once it is exposed. It is possible that the last round will be drilled and blasted to double length with extension steel, such that the ore is completely blasted without having to control the back. The drift end will be mucked as completely as possible, using a remote controlled scooptram if required. Continuous unraveling of the hanging wall is anticipated. Mucking will continue until dilution is excessive, rendering the muckpile uneconomic, at which point the panel will be closed for filling.

Panels will be paste backfilled individually, which has the potential to be a finicky and tasking exercise due to the numerous bulkheads required and the small size of each individual pour. This will be mitigated by using pre-fabricated mechanical bulkheads designed to be placed and sealed rapidly and recovered after the pour for later use. This process will also allow more complete backfilling of each individual panel than a muckberm and fill fence system, enhancing the stability of the stope by minimizing the open span.

1.3.2 Drift and Fill with Retreat Panels (DFRP)

For ore 4 to 7 m horizontally thick, the method will be modified by extracting the panels one at a time in a retreat fashion rather than using a primary and secondary sequence.

The first panel will be mined at the furthest extent from the access, then it and that portion of footwall drift will be backfilled. After curing for a period of 3 to 7 days, the next adjacent panel will be mined, exposing the backfill of the previous panel along the floor. In this fashion, the stope will incrementally retreat towards the stope access.

The backfill bulkhead for each stope pour will be placed in the footwall drift and both the panel and a portion of the footwall drift will be filled.

1.3.3 Drift and Fill with Side Slash (DFSS)

For ore <4 m horizontally thick, the ore will be mined in two separate passes. The first will include drifting along the footwall to the extent of mineable ore. Upon reaching the economic extent of the stope, the mineralized wall will then be slashed using horizontal drill jumbo holes starting at the end of the stope and incrementally retreating toward the stope access. The hanging wall exposed by the slashing will not be bolted. As such, the broken ore will be mucked remotely and no access will be allowed in the slashed area.

The maximum width of DFSS stopes will be determined by the reach of the jumbo drill, as indicated in Drawing MO-X-020. A 4 m drill steel is assumed, limiting this mining method to a maximum horizontal ore thickness of 4 m at the average dip of 34º.

The individual blasts will range in strike length exposure (6 to 8 m is assumed). When the exposed hanging wall inside the stope becomes unstable, which will be primarily dependent on the length of exposure, a bulkhead will be placed in the footwall drift and the stope void will be filled as tightly as possible with paste backfill.

The mining method will be determined on the basis of ore thickness, as shown in Table 1-3.

During operations, test holes will be drilled regularly in the backs and walls of the footwall stope drifts as they are driven to locate the hanging wall contact and determine the ore thickness. This will assure stope back competence and help set the transition points from one stoping method to another. Assaying the cuttings from the test holes will not be required, as the ore/waste hanging wall contact will be defined by the cuttings colour and penetration rate.

1.3.4 Hanging Wall Lenses

There are occurrences of thin lenses in the hanging wall of both the Lynx and Wolverine zones. These vary in thickness from 1 to 9 m horizontal thickness and are oriented parallel to the main lens with between 1 and 11 m of barren waste separating the lens from the main orebody. A total of 269,000 tonnes of the mining or 5% of the mining reserve is comprised of ore from the hanging wall lenses.

These lenses will be mined in one of two ways:

- Using Longitudinal Drift and Fill mining (LDF), the hanging wall lens will be accessed by a short stope access driven from the footwall ore drift of the main lens in the waste that separates the two orebodies. Mining would be accomplished by entering the thin lens roughly perpendicular to its strike then mining in both directions, filling the stope with a single pour on completion of the 4 m vertical stope lift.
- In cases where there is very little separation between the hanging wall lens and the main orebody, the two will be mined together as a single DFPS stope, including the waste between them as internal dilution.

1.3.5 Naming Convention

Figure 1-1 shows a typical stoping block is shown on plan and cross sectional views. The naming convention shown on the figure is explained as follows:

Stope Blocks will be named for the zone (‘L’ for Lynx, ‘W’ for Wolverine or ‘B’ for the barrier pillar zone in the Saddle) followed by the elevation of the stope block center in meters above sea level. The stope in the figure is W1210, where ‘W’ represents the Wolverine orebody and ‘1210’ represents the center elevation of the block.

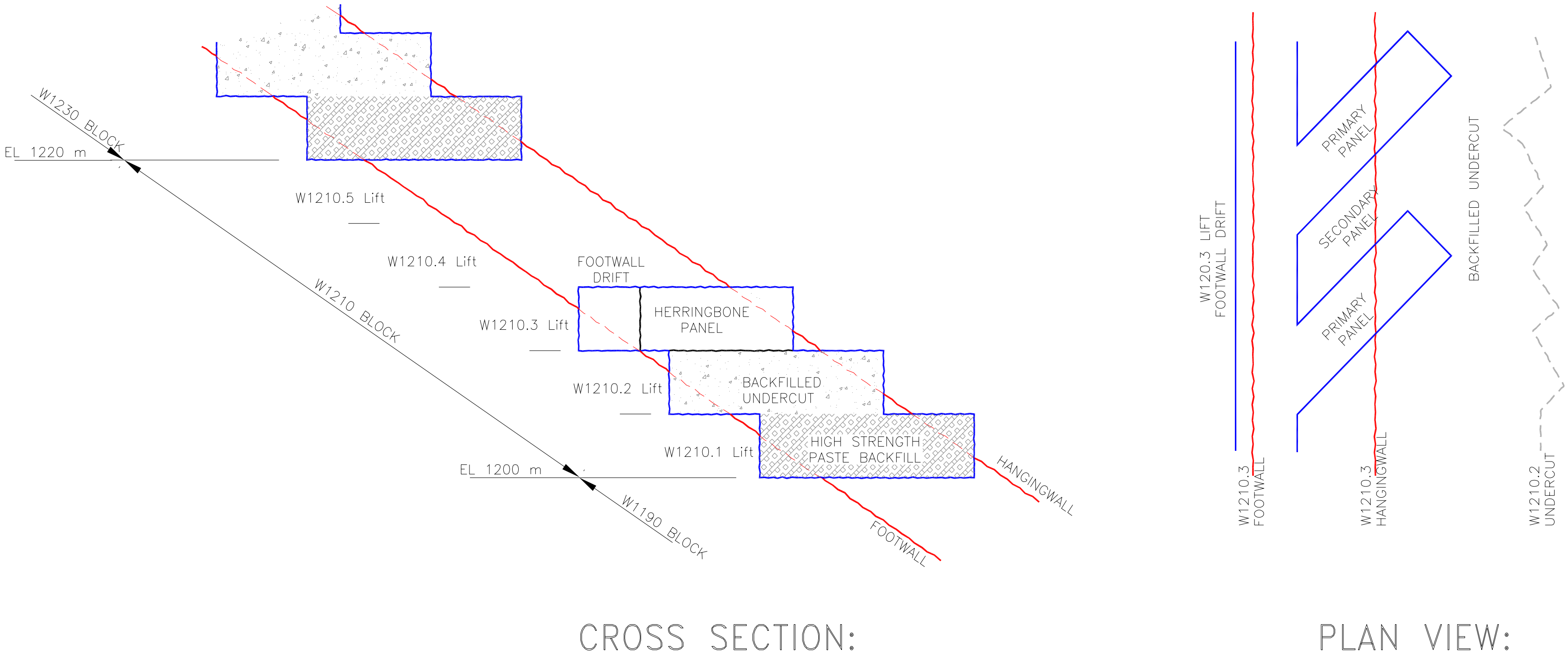
Lifts will be named for the Block followed by the lift number, counting from the lowest to the highest. In Figure 1-1, the active lift is W1210.3, signifying the third lift of W1210 Block located from elevations 1208 to 1212 m. In the figure, Lifts W1210.1 and W1210.2 have already been mined and filled and W1210.3 is being mined atop the backfill.

As shown, W1210 Block would be located above W1190 Block and below W1230 Block. The uppermost lifts of W1190 Stope have not been mined yet.

W1210 Block would be on the same elevation and share a stope access with L1210 and H1210 Blocks.

The plan view shows two primary panels that have been mined on the stope lift, but no secondary panels as yet. During operations, each panel will be named as an extension of the stope lift name (i.e., W1220.2.5). Odd numbers will designate primary panels and even numbers will designate secondary panels. For simplicity, the panels were not named on any figure in this study.

Figure 1-1: Reference Terms for Stoping Blocks



1.3.6 Block Mining Sequence

For any given stope lift, the footwall stope drift will start in the thin ore of the saddle area, centrally located between the two orebodies. This ore will be left as a barrier pillar to protect the main ramp until the ramp is no longer needed.

The ore will generally thicken as the stope drift advances toward the centers of either orebody, then taper again toward the extremity of each stoping lift, ultimately to a non-economic thickness. As such, regular transitions will be made between the stoping methods along a stoping lift in response to changes in ore thickness. Drawing MO-X-21 demonstrates the stope sequencing at the extent of a typical stope.

Drawing MO-X-022 shows a detailed level plan for the 1210 Blocks, showing the transition of mining methods across the level for all three zones.

The footwall drift will be the main production heading, driven as fast as possible to the extent of mining. Primary DFPS panels will augment production from this heading. The panels will be filled immediately after mining, saving the occasional one for use as a remuck or gear storage area. The DFSS mining will then start at the extent of mining and retreat toward the stope access. DFRP will also be done on a retreat basis, filling the footwall drift as well as the mining panel.

To aid in determining transitions between methods, test holes will be drilled regularly in the backs and walls of the footwall stope drifts, as they are driven to locate the hanging wall contact and determine ore thickness. This will also assure integrity of the stope. Assaying the cuttings from the test holes will not be required for this determination, as the ore/waste hanging wall contact will be defined by the cuttings colour and penetration rate.

1.3.7 Roadbed

Roadbed in the stope blocks will be an ongoing concern. The floor in the stopes will be comprised of ore, footwall rock and paste backfill from previously mined and filled stope lifts. The footwall rock is a soft rhyolite schist that is anticipated to be easily gouged by the mine equipment tires. Paste backfill is a very poor mucking floor, as the tires of the equipment (particularly the scooptrams) tend to bite into the paste and churn it. To help both mucking floors, a 0.5 m layer of broken ore will be left on the floor of the entire stope block until the completion of mining, estimated to be at most a three month period. The broken ore will be recovered just prior to filling.

1.4 Mine Access and Development

All lateral development in the mine will be done using drill jumbos, rockbolting jumbos and scooptrams sized appropriately for the heading dimensions.

The estimated mine development requirements are shown in Table 1-6. Each development type is broken out by exploration phase, pre-production development period, and Life-of-Mine (LOM). A complete LOM layout for the mine is shown in Drawings MO-X-001 through 005. Individual level plans are shown in Drawings MO-X-007 to 019.

Table 1-6: Mine Development Requirements by Phase

Heading Description	Dimension (m)		Development Period		
	width	height	pre-pro	operations	LOM
Lateral					
Ramp	5	5	495	1318	1813
Stope Access	4	4	248	1655	1903
Diamond Drill Drift	4	4	166	1269	1435
Vent Drift	4	4	77	967	1044
Total Lateral			**986**	**5209**	**6195**
Raising					
Main Intake/Exhaust	3	Dia	228	100	328
Level Exhaust	3	3		320	320
Manway	1.5	1.5		483	483
Total Lateral			**228**	**903**	**1131**

The pre-production development totals represent the capitalized development completed prior to production.

These figures do not include the existing 450 m of development completed during the 2005 test-mining program. The as-built drawing detailing this work is shown in Drawing MO-X-006.

1.4.1.1 Main Access Ramp

The mine will be accessed by a single ramp driven at -15% grade, located centrally between the Lynx and Wolverine deposits in a zone of weaker mineralization (predominantly sub grade ore) called the 'saddle zone'. The ramp has been sized at 5 m wide x 5 m high, with an arched back to accommodate a future production fleet of 50 t diesel haulage trucks. A typical ramp profile is shown in Drawing MO-X-027.

During the 2005 test-mining program 250 m of ramp was developed via the 1345 Portal in the hanging wall of the deposit (see Drawing MO-X-006). The ramp will continue in the hanging wall of the deposit for an additional 581 m to the 1263 m elevation. At this point the ramp will enter the ore of the saddle zone. The ramp will then be driven in the ore of the saddle zone an additional 1400 m to its deepest elevation of 1080 m.

The footwall of a deposit is the usual location for the main access ramp. However, for the Wolverine deposit, the footwall rock is schistose and too weak to host large long-term excavations.

Ramps that are located in the hanging wall of a deposit run the risk of having the ground de-stabilize when the ore beneath them is mined. However, for the Wolverine deposit the only competent waste rock hosting the deposit is found in the hanging wall. Both are exhalites, the iron formation (EXMT) and a calcareous exhalite unit (EXCP). The latter was selected to host the initial ramp, as it is the stronger of the two and nearer the orebody. It occurs at a nominal horizontal distance of 40 m from the orebody at a thickness of between 2 and 8 m. The ore beneath the portion of ramp that is in the hanging wall EXCP formation is the weakest grade of the reserve and will be left as a barrier pillar until the very end of mine life. It is mined last in the production forecast, mitigating the risk of de-stabilizing the main ramp.

Ramps located in the ore itself often limit the production capability of the operation by necessitating a linear and inflexible mining schedule designed to maintain integrity of the main ramp. However, for the Wolverine Mine this location is appropriate for the following reasons:

- Remaining in the EXCP unit is not possible, as it diverges from the orebody with depth.
- There is no competent rock unit between the massive sulphide and the EXCP unit. Mining beneath the ramp, even during barrier pillar extraction, is liable to cave up to the ramp.
- Having the ramp in ore will allow numerous stope access take-offs from the main ramp during barrier pillar extraction, increasing the production capability of the final years. Otherwise it would be difficult to maintain positive economics during the final years of mine life.
- Access to the Wolverine and Lynx stoping blocks will be achieved by breasting the stope accesses progressively for each 4 m stope lift. With the ramp in the hanging wall, access to each lift would have to be achieved twice: once for primary mining and once for barrier pillar extraction. This would incur considerable waste development costs for very little ore when the barrier pillar is being extracted.
- Separate accesses are driven for Wolverine and Lynx Zones, allowing schedule flexibility between the two zones. With the hanging wall stope access, the entire 4 m stope lift must be completed in both zones before the next lift is started.
- The occurrence of the mineralized zone is expected to be more predictable and more competent than any waste rock unit.
- With the stope accesses in ore, many can be driven at 5 m wide to accommodate the 50 tonne haulage truck width into the stope.
- A hanging wall access scheme could not be followed for the entire deposit even if it was desired, as it would cross the property boundary at the lower elevations.

1.4.1.2 Stope Access Drifts

Stope access drifts will be located at 20 m vertical increments from the main ramp. Stope accesses will be driven at 4 m wide x 4 m high with an arched back. A typical stope access profile is shown in Drawing MO-X-027.

For the upper levels of the mine where the ramp is in the EXCP unit of the hanging wall, stope access drifts will be driven a nominal length of 45 m at 4 m wide to accommodate 4.6 m^3 scooptrams. The orientation will be roughly perpendicular to the stope strike to minimize the length of the drive and maximize stability. It will also be perpendicular to the bedding and cleavage planes of the hanging wall unit. The initial crosscut to access each stope will be driven downgrade at -15% to the first and lowest stoping lift. Once the initial lift is mined and filled, the stope access back will then be breasted to provide access to the next mining lift, 4 m higher than the previous one. Access to both the Lynx and Wolverine Zones will be provided from the same stope access. As such, mining must be completed across the entire 4 m stope lift for both zones before the access is breasted for the next, upper stope lift.

For the elevations of the mine where the ramp is located in the ore, stope accesses will be driven directly from the ramp separately for the two zones. Much of this development will be in ore.

The gradient of both the main ramp and the stope access drifts will be maintained at -15% through stope access intersections to minimize development requirements. This is not usual practice; normally both headings are flattened for an intersection. However it is not unique; Goldstream Mine (also developed and operated by Procon) maintained this practice. Falconbridge's Kidd Creek Mine in Timmins, Ontario maintains a steeper average gradient of -17% through all intersections.

1.4.1.3 Diamond Drill Drifts

Six diamond drill drifts will be excavated in the hanging wall above the deposit, three for the Wolverine zone and three for the Lynx zone for orebody definition by underground diamond drilling prior to mining. These will be located at approximately the 1300, 1220 and 1160 m elevations. The W1272 DD, will be excavated as part of the preproduction capital.

The ventilation raise access drifts will be extended into the hanging wall of the deposit and a 'tee' intersection established at the EXMT formation. The diamond drill drifts will then be driven at 4 m wide with a 4 m high shanty-back profile parallel to the bedding of the EXMT formation. Drill drifts will be driven to near the lateral extent of the ore body to provide complete coverage for definition drilling.

1.4.1.4 Ventilation Drifts

Ventilation drifts will be driven at 4 m wide x 4 m high. These will generally be short drifts driven perpendicular to structure in the hanging wall of the deposit to access vertical ventilation raises driven between stoping blocks.

1.4.1.5 Miscellaneous Development

Allowances for miscellaneous development were added to ramp footage as follows: a 10 m passing bay every 500 m, a 20 m sump every 300 m, a 6 m remuck bay every 150 m, and a 4 m electrical bay every 1000 m. These allocations result in 8% miscellaneous development, which is applied to the linear distance of all ramp development for costing and scheduling.

Miscellaneous development for stope accesses was estimated at 4% for remuck stations, sumps and electrical bays. No allowance was made for other development.

1.4.1.6 Raising

Raising will be constructed for ventilation and emergency egress from the mine. Three different types of raises will be excavated over the life of the mine: bored raises, drop raises, and conventional inclined raises.

The bored raises will be the main intakes and exhaust conduits for the mine ventilation system. There will be three: the B1260 Fresh Air Raise (FAR) at 128 m long, the W1290 Return Air Raise (RAR) at 130 m long, and the L1300 RAR at 100 m long. All will be vertical with 3 m diameters and sprayed with a 0.1 m thick layer of shotcrete to prevent unraveling of the walls during operations. The B1260 FAR and the W1290 RAR will be excavated during the pre-production period. The B1260 FAR will also be equipped with a steel manway with landing platforms every 6 m to act as an emergency egress.

Vertical drop raises will be excavated to connect ventilation transfer cross-cuts of the stope blocks. The drop raises will typically be square in cross section, 3 x 3 m wide x 16 m long (based on a stope block interval of 20 m vertically less the 4 m height of drift). A small longhole drill will be employed to drill the drop raises. These will not be bolted and will therefore not serve as escapeways.

Conventionally driven inclined raises will be excavated as manways between stoping blocks. These will be typically 1.5 x 1.5 m and excavated in wider sections of the orebody such that they can be located in the ore. The raises will be excavated at 50° dip or less so that landings are not necessary.

Drawings MO-X003 through 005 show the LOM layout, including all raises in longitudinal, sectional, and 3-dimensional views.

1.5 Ore Reserves

1.5.1 Statement of Ore Reserves

The estimation of ore reserves for the Wolverine orebody is shown in Table 1-7.

Table 1-7: Ore Reserves Estimate

Category	Tonnes	Zn %	Cu %	Pb %	Ag g/t	Au g/t
Measured	583,043	9.92	0.94	1.20	240.30	1.21
Indicated	4,625,303	9.68	0.93	1.27	289.72	1.39
Total	**5,208,346**	**9.71**	**0.93**	**1.26**	**284.19**	**1.37**

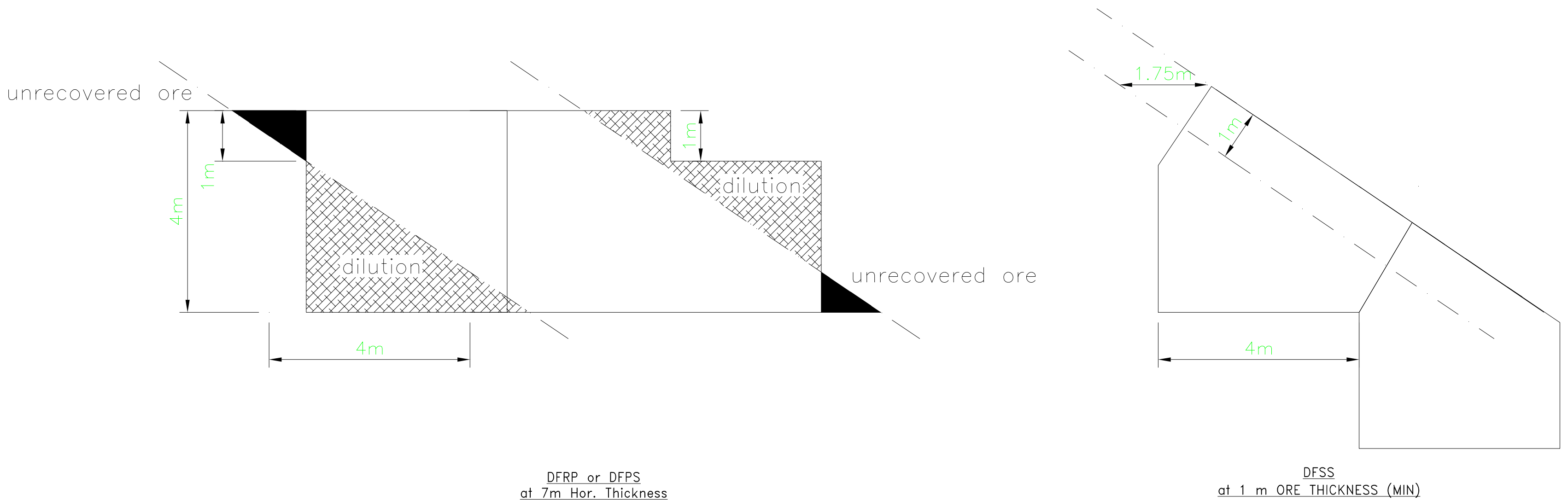
1.5.2 Dilution and Recovery Factors

Dilution and recovery factors were estimated for the various mining thicknesses based on the average orebody dip of 34°. It is assumed that the footwall drift will follow the ore footwall as a marker, keeping the ore/waste contact at a distance of 1-m from the back (3-m of waste beneath 1-m of ore on a 4-m wall, matching the cut-off criteria proportions). Similarly, the ore will be mined to the hanging wall until 1-m of ore is left in the face.

For the narrowest ore, a shanty-back drift is assumed. A recovery of 95% was selected for the narrowest ore, which is a judgment factor that considers the narrow width of the stope and the overlapping mining shapes from the shanty-back profiles.

Both situations are depicted in Figure 1-2.

Figure 1-2: Dilution and Recovery Factor Assumptions



A recovery factor of 80% was assumed for barrier pillar extraction, reflecting the difficult geometries that will be encountered.

For all mining methods, losses were assumed for impacting broken ore in the paste backfill floor. A 0.1 m layer of broken ore is assumed to be lost.

Paste backfill dilution from over-mucking was also assumed (an average of 0.05 m from all exposed backfill walls and 0.15 m from all backfill floors).

Table 1-8 shows the dilution and recovery factors by horizontal ore width.

Table 1-8: Stope Dilution and Recovery Factors by Horizontal Ore Width

Stope Horizontal Thickness (m)	Dilution % (waste/ore)	Recovery %
0.5	727.2	95.0
1	328.5	95.0
2	129.2	95.0
3	62.7	95.0
4	45.2	90.7
5	35.4	92.6
6	29.1	93.8
7	24.7	94.4
8	21.5	94.8
9	19.0	95.2
10	17.0	95.5
11	15.4	95.7
12	14.1	95.9
13	13.0	96.1
14	12.0	96.2
15	11.2	96.3
16	10.5	96.4
17	9.9	96.5
18	9.3	96.6
19	8.8	96.7
20	8.4	96.7
21	7.9	96.8
22	7.6	96.9
23	7.2	96.9
24	6.9	96.9
25	6.7	97.0
26	6.4	97.0
27	6.2	97.1
28	5.9	97.1

All waste dilution was added at a grade of 0.2% Zn, 0.048 % Cu, 0.03 % Pb, 7.67 g/t Ag and 0.052 g/t, the background grade derived from the resource estimate. Backfill dilution was added without grade.

1.6 Mining Equipment

Mobile trackless diesel-powered equipment has been assumed for the mine. The underground equipment list is shown in Table 1-9. All drill equipment is electric-hydraulic.

Table 1-9: Underground Equipment List

Unit Type	Model	Units Required
Jumbo drill	Axera 7-240	2
Rock bolting machine	Robolt 5-126	1
Large LHD	Toro 1400	1
Medium LHD	Toro 007	2
Small LHD	EJC 145	1
Haul truck	Toro 50	2
Grader	Cat 120H	1
Bulldozer	Cat D4	1
Shotcrete machine	Triple 4ce	1
Shotecrete supply	Triple 4ce	1
ANFO loader	Triple 4ce	1
Electrical	Triple 4ce	1
Mechanical	Triple 4ce	1
Construction	Triple 4ce	1
Supplies	Triple 4ce1	
Supervisor/mancarrier	Toyota landcruiser	3
Haul truck (ejector)	EJC30SX	1
Longhole drill	TBA	1

1.7 Mine Haulage

Ore and waste rock will be hauled from the muckpiles to remuck stations using LHD units (load-haul-dump, also known as scooptrams). At least two sizes of LHDs will be employed in the mine: 6.1 m^3 units to load trucks and support the development program and 4.6 m^3 units for stoping.

Ore will be loaded into 50 t diesel trucks by LHD units for haulage to the surface via the 1345 Portal. It will be dumped directly into the crusher bin at the mill.

Most waste will be retained in the mine for use as fill, placed in mining voids before being encapsulated in paste backfill. The smaller EJC30SX with ejector bucket will be used for this purpose. Waste hauled to the surface will be stockpiled on the existing temporary waste storage pad for eventual haulage to the tailings pond using surface equipment (Caterpillar 966 FEL and 730 articulated truck).

Both LHD sizes are capable of loading both sizes of truck.

1.8 Waste Disposal

A waste pad was constructed during the 2005 test-mining program at a distance of approximately 1 km from the mine towards the airstrip. Approximately 30,000 tonnes of waste and 5000 tonnes of ore were added to the waste pad during in 2005. The waste pad will be expanded as part of the exploration phase in 2006 to contain an additional

3900 tonnes of ore and 63,700 tonnes of waste from the 2006 exploration period and pre-production mine development.

The waste pad is a temporary facility. During the first few years of operations, the waste rock will be relocated to the tailings impoundment site for permanent disposal and the site will be reclaimed.

1.9 Mine Backfill

To promote overall mine stability, mining voids will be filled. Three types of backfill will be used: paste backfill with cement addition, loose waste generated by the lateral development, and DMS float product. Over the LOM, the following quantities of fill will be required:

- 2.18 Mt of paste backfill
- 0.14 Mt of loose waste
- 0.36 Mt of DMS float rock

DFPS stope herringbone panels will be filled immediately after excavation. A bulkhead will be placed at the entrance to the panel as close as possible to footwall drift to minimize the open span at the intersection. Loose waste and DMS float rock will not be placed in primary herringbone panels so that the fill is self-standing when exposed by the secondary panel mining. The secondary panels can be filled with any material provided only paste backfill is against the bulkhead.

DFSS stopes will be filled incrementally. An open strike length of 20 m is assumed for each pour. Any voids in the filled stopes will be filled as part of the filling for the next, upper lift.

Upon completion of mining, it is assumed that 95% of all mining voids will be filled.

Approximately 36% of all mine development will be filled, including portions of the main ramp and stope accesses. Ventilation development, including raises and vent access drifts, will be kept open for the LOM then filled and sealed as part of closure.

1.9.1 Paste Backfill

The paste backfill plant will be located in the industrial complex at the northwest corner, closest to the portals. The paste will be manufactured from unclassified tailings. Portland cement will be added to the paste to fully hydrate the water, causing the product to maintain its form as paste and add strength such that it can be exposed by adjacent mining.

There will be three paste strength requirements:

- Low strength fill in locations where the fill will never be exposed by adjacent future mining.
- Medium strength fill for areas that require self-standing strength fill walls after being exposed by adjacent mining.
- High strength fill for backfill that will be undercut by future mining.

Based on the results of the testwork, the following binder contents are assumed for the three applications: low strength - 1% binder; medium strength – 4% binder; high strength – 8% binder. A 7" slump is assumed for all paste.

Paste backfill will be passed to the mine through the main ramp via the 1345 Portal in 150 mm Schedule 40 or Schedule 80 steel pipe that will be rigidly mounted in the main ramp. The final 300 m of each paste line will be HDPE pipe that will be laid on the floor of the stope.

As paste backfill is quite viscous, the head gained by the vertical drop will not be adequate to deliver the paste backfill to the stopes. A positive displacement pump will be used to move the paste through the line.

The paste backfill distribution system is shown graphically on long sectional view on Drawing MO-X-032.

1.9.2 Loose Waste and DMS Float Rock

Most waste rock from development headings will be hauled into open stopes as fill using a 30 tonnes truck without leaving the mine. The same truck will be used to haul DMS float rock from surface to the mine.

The haulage truck has been sized to fit in a 4 m wide stope heading. It will be equipped with an ejector bucket to eliminate the need for backslashes for bucket dumping.

The mine will be equipped with a D4 dozer for use in final placement of either material prior to filling. Care will be taken to place the material such that it will not be exposed by future adjacent mining, as it will have no self-supporting strength. The material will be completely encapsulated by paste backfill.

The DMS rock may be added to the paste backfill in the plant to increase strength or conversely reduce binder content. This will require further testwork before implementing.

1.10 Mine Ventilation

YZC employed the services of Mine Ventilation Services Ltd. of Fresno, CA to prepare a ventilation model for the mine. The mine ventilation system will employ one main fresh air feed (the B1260 FAR) and three exhaust conduits (the L1300 RAR, the W1290 RAR and the 1345 portal). The ventilation system will be a “push” system, with one main fan atop the B1260 FAR. This installation will be equipped with a direct-fired propane mine-air heating system. A maximum ventilation flow of 211 m^3/s is anticipated for the mine.

The B1260 FAR will be located at 439918 E, 6810982 N and will extend from near the main ramp in the saddle zone at 1260 m to surface at 1391 m, a distance of 131 m. It will be equipped with a steel manway and platforms to act as an emergency egress.

The W1290 RAR will be located at 439959 E, 6810859 N and will extend from near the end of the W1290DD at elevation 1284 to surface at 1381, a distance of 91 m.

The L1300 RAR will be located at 439705 E, 6811219 N and will extend from near the end of the L1272DD at elevation 1300 to surface at 1400, a distance of 100 m.

All three bored raises will be driven vertically at 90° and will be 3-m in diameter. All will be lined with shotcrete to ensure long-term stability.

The mine plan and operating principals will attempt to minimize re-use of air flows; that is having mine workers receive air that has already been partially contaminated by equipment operating up-stream.

Air will enter the mine via the B1260 FAR and travel up and down the main ramp to the various stope levels. It will be drawn into each stope block via the stope access drift. Both the Lynx and Wolverine zones will be equipped with exhaust systems comprised of interconnected vertical raises and ventilation transfer drifts located in the hanging wall of the deposit. The Lynx system will ultimately exhaust via the L1300 RAR and the Wolverine system via the W1290 RAR.

Secondary fans (50 to 200 HP) and flexible vent tubing will provide ventilation for the individual headings.

Regulators installed in the raise and drift bulkheads will control airflow to the various headings.

After the initial stoping level is complete, a system of culverts encapsulated in the stope backfill will maintain flow to the exhaust raise network.

The ventilation system is shown graphically on a long sectional view for the early stages of mine development in Drawing MO-X-030 and the LOM system is shown on Drawing MO-X-031.

1.11 Mine Services

1.11.1 Power

Power will be used in the mine primarily by the drilling equipment, secondary ventilation fans, and larger de-watering pumps.

Power will be drawn from the main generators located adjacent to the processing building. The electrical cable will enter the mine via the 1345 Portal and be distributed throughout the mine in the development headings. Power cables will be suspended in the mine development headings. An overland cable from the surface generators at the mill will supply the main intake fan and the mine air heater. The power distribution system is shown in long sectional view on Drawing MO-X-029.

The estimated electrical load for the Wolverine underground has been compiled from data provided by YZC. The estimated load has been tabulated in Table 1-10. The primary intake air fan, which will be serviced directly from the surface distribution system, is excluded from the load list. The duty factors applied to the loads (factor to account for simultaneous operation of the various loads) are based upon typical values for other similar projects.

As per Table 1-10, the electrical load for the project will be approximately 1850 kVA connected and 874 kVA demand. Power factor correction equipment is not required initially, as the load is relatively small and the initial feeder lengths are relatively short (<400 m).

Table 1-10: Wolverine Electrical Load List

Description	Unit Power		Qty	Duty Factor	Connected kVA	Operating Demand		
	hp	kW				kW	kVAR	kVA
Mobile								
Jumbo Drill	145	108	2	33%	290	72	64	96
Rock Bolter	80	60	1	33%	80	20	18	27
Fixed								
Main Dewatering Pumps	100	75	2	50%	200	75	67	100
Secondary Fans	50	37	20	50%	1,000	373	330	498
Secondary Pumps	13	10	10	25%	130	24	23	33
Diamond Drills	50	37	2	70%	100	52	46	70
Lighting & Misc.	50	37	1	100%	50	37	33	50
Total					**1,850**	**653**	**581**	**874**

1.11.1.1 System Voltages

When starting a new project, many alternate voltages may be considered for service to the mine. The electrical system will be divided into two basic systems: distribution and utilization.

Although other voltages could be considered for site distribution, 4.16 kV will be used for the following reasons:

- The relatively small underground load (presently less than 1 MVA)
- 4.16 kV can be used for larger motors directly (i.e., no additional transformations required)
- Electrical energy is generated at 4.16 kV

The utilization system includes the electrical systems that provide electrical energy directly to the various electrical loads. Depending upon the rating of the load (kVA or horsepower), the utilization system voltages may be 4.16 kV, 600 V, or 120 V.

Motors rated larger than 200 hp will be 3 phase, 4000 volts and serviced from a 3-phase, 3-wire, 4160 V system. As per the defined load list, all motors underground will be <200 hp. This recommendation has been included in the event larger motors are required in future. Motors rated 3/4 hp to 200 hp inclusive will be 3-phase, 575 volt and serviced from a 3-phase, 3-wire, 600 V system.

The use of nominal 600 V systems is preferred over 480 V systems for the following reasons:

- 480 and 600 V systems basically utilize the same electrical components but 600 V systems result in 20% less current for a given load, which permits the use of smaller cables and potentially fewer components (i.e., lower installed cost)
- 600 V systems permit a 25% increase in feeder length while maintaining a similar voltage drop (i.e., better voltage regulation which provides better productivity).

Lighting, convenience receptacles and fractional horsepower motors will be single phase, derived from either 3-phase 4-wire 120/208 volt, or single-phase 3-wire 120/240 volt systems.

The utilization system voltages will be derived from the following transformers:

- 4.16 kV - 600 V
- various 600 V - 208 V or 600 V - 120/240 V units in various locations

1.11.1.2 Underground Distribution System

The underground distribution system is comprised of the 5 kV mine feeders and the underground electrical system(s).

The main items in this electrical system include the following:

- one 5-kV mine feeder initially routed through the ramp
- one future 5-kV mine feeder routed through the fresh air raise
- three 4.16 kV - 600V portable power centers to service the mining loads directly and indirectly
- 600 V portable distribution centers to service the mining loads directly

1.11.1.3 System Description

Initially one main feeder cable will connect from the surface switch/switchgear to the production levels via the ramp. This cable will be 5-kV rated, three conductor, copper, unshielded, #4/0 AWG, aluminum armoured, PVC jacketed Teck type.

In the future a second feeder cable will connect from the surface switch/switchgear to the production levels via the fresh air raise. This cable will be 5-kV rated, three conductor, copper, PVC jacketed mine shaft type complete with integral galvanized steel support wires. As the installed cable length will be less than 200 m the cable will be supported with the integral support wires thus avoiding intermediate support brackets and systems. Due to the cost of shaft cable installation, a larger cable size should be used in the fresh air raise to ensure additional future capacity without the requirement for cable replacement (i.e., adequate capacity in the initial main infrastructure).

Ultimately, two feeders would permit production, ramp and/or level development in alternate areas simultaneously. Having two feeders will also prevent a single electrical feeder failure from causing a complete mine electrical outage. Feeder cables routed through the mine workings will be supported with messenger wire and brackets rock bolted to the back and or walls along access ramps and drifts. The ampacity requirements for the feeders will be calculated in accordance with the Canadian Electrical Code.

Junction points (boxes) will be installed at each level entrance and/or at approximately 300 m intervals. Appropriate excavations will be provided to ensure safety, access and protection for these junction boxes.

Portable power centers will be used for servicing the mining equipment. The portable power centers will be connected to the 5-kV feeder at the nearest junction point (box) via #4/0 AWG 5-kV armoured cable. Each portable power centers will incorporate 750 kVA, 4.16 kV - 600 V dry type transformers, 600 V distribution panel, 600 V power receptacles and 120 V lighting panel. Power receptacles will be complete with pilot and ground fault monitoring equipment meeting the requirements of the latest edition of CSA M421 (Use of Electricity in Mines). The portable power centers will service mining equipment directly (when in close proximity) using the 600 V power receptacles or will service 600 V portable distribution centers when mining loads are more remote.

The portable power centers will have a central location to service multiple levels simultaneously. The portable power centers will be located within appropriate excavations to ensure safety, access and protection. Portable 600 Volt distribution centers will be located on each level to permit isolation of the respective level as required and to provide protection (over current, ground fault and pilot monitoring) of the various mobile mining equipment feeders. The portable 600 V distribution centers will be skid mounted.

1.11.2 Communications

Based upon the requirements discussed within this section, the Wolverine project will install an industrial Ethernet system to provide underground communication for the short and long term. The purpose of this system will be to provide voice communication from underground to surface or other personnel in the mine.

1.11.3 Compressed Air

Compressed air will be used for stoper and jackleg drilling, secondary pumping, ANFO loading and cleaning with blowpipes. It will also be used as emergency ventilation for the refuge stations and to deliver the stench gas in case of a mine fire.

Compressed air for the mine will be tapped off the main compressed air system for the mill.

Compressed air will be supplied in metal pipes suspended in the upper corners of the development and stope headings with other mine services lines. Nominally, a 200 mm diameter pipe will be used in the main ramp and 50 to 100 mm diameter pipes in secondary headings and stopes.

1.11.4 Mine Dewatering

Groundwater and drill water will be collected in the mine in ditches that will direct flow to numerous regularly placed secondary sumps. Solids will be settled in these sumps and the clean water will be pumped to main sumps for additional settling. The main sumps will be twinned with a dirty sump overflowing by gravity to a clean sump. Water from the clean sump will either be recycled for drill water or pumped to a surface settling pond in a dedicated dewatering pipeline suspended in the main ramp with the other mine service lines.

1.11.5 Supplies

Bulk mine supplies not requiring heated storage will be kept on the existing portal pads. This includes such items as steel or plastic pipes, bolting supplies, ventilation tubing, steel sets, shotcrete, hydraulic oil, and timber. Customized racks and overhead cover will be built as required to properly protect each commodity.

Some commodities, such as rockbolt resin, will require a minimal level of heating. An existing coverall on the lower portal pad will be used to store such items.

Smaller and costlier supplies, such as drill bits, equipment parts, and small tools will be kept in the main warehouse in the industrial complex.

The existing powder and cap magazines will continue to be used during operations. These will be relocated to new sites on the existing temporary winter road to keep them as remote as possible from other facilities, such as the camp and industrial area. The placement of these magazines has been governed by the British Table of Distances. On average, 24 hours of explosives will be stored in magazines located throughout the mine at any given time.

1.11.6 Emergency Egress

Travel to the mine will normally be in and out of the 1345 and 1360 Portals. FAR #1 will be equipped with regularly spaced landings and ladders, providing a secondary emergency means of egress.

Main access development in the mine will at times provide two paths of escape to these two mine exits. Alternate routes will be maintained as close as possible to the working face. For this purpose, and to control ventilation flow, regular connections will be made in the ore between sublevels.

1.11.7 Refuge Stations

Portable fiberglass refuge stations will be employed in the mine. Three units are anticipated during operations. Each will be connected to air and water and will be supplied with a fixed telephone line. The location of each unit will change regularly over the life of the mine to accommodate development crews operating in blind headings.

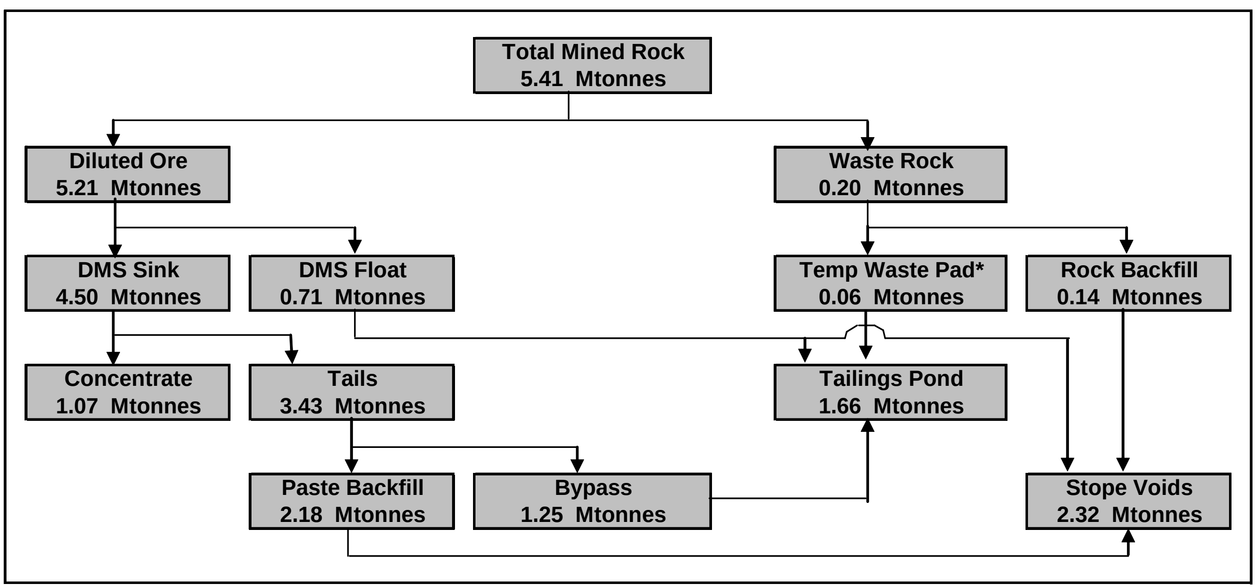
1.12 Materials Balance

A materials balance flowsheet is shown in Figure 1-3 for the life of the operation.

As shown, the total quantity of broken rock (ore plus waste) is 5.41 Mt over the LOM, which includes diluted ore and development waste rock.

Note that the tailings pond will be comprised of 0.06 Mt of development rock, 0.35 Mt of DMS floatation rock and 1.25 Mt of tailings from the processing plant.

Figure 1-3: Material Balance for the Operation



* includes 0.03 tonnes from the 2005 Test Mining Program

1.12.1 Development Forecast

The mine will be developed in three distinct phases: exploration, pre-production development, and operations. Mine development was done during the exploration phase by a contractor, and this will be the case for the pre-production development. The mine will be owner-operated.

1.12.1.1 Exploration

Test mining lasted from June to November 2005, during which time the following was accomplished:

- excavated the 1345 Portal
- the main ramp was driven approximately 250 m from 1345 to 1320 m elev; the 1320 stope access drift and 150 m development in, or adjacent to, the ore was driven
- obtained a bulk sample for metallurgical processing
- used the tailings from the metallurgical sample for paste backfill testwork
- developed dilution, recovery, productivity, and cost criteria for the FS mining methods
- tested and evaluated the stratigraphic options for the location of the mine development (footwall, hanging wall, in ore)
- drilled 58 ore definition holes from surface

1.12.1.2 Pre-Production Period

The pre-production period will last from October 2006 to July 2007, during which time the mine will be prepared for full operating status. This will include approximately 986 m of additional lateral development plus 228 m of raise development and include the following work:

- rehabilitate the existing workings, including: replacing the roadbed with segregated aggregate, shotcreting the main ramp walls, and encasing in concrete the existing steel sets at the top of the ramp
- establishing access to six active ore production faces on four mining horizons
- providing additional development for ventilation distribution and emergency egress, including: intake and exhaust raising to surface
- installing and commissioning several mining facilities and systems, including: paste backfill distribution, power distribution, communications, ventilation, compressed air, water supply and dewatering
- begin mining, commission the mill and backfill systems and ramp-up to full production

Table 1-11 shows a summary of the pre-production development schedule.

This quantity of development will allow the mine to begin the ramp-up to full production starting in January 2007. A five-month period is allowed for the transition of crews from contractor to owner’s crews.

Table 1-11: Pre-production Development Schedule, (m)

Heading Description	2006			2007							Total
	Oct	Nov	Dec	Jan	Feb	Mar	Apr	May	Jun	Jul	
Lateral											
Ramp	31	60	40	62	56	62	60	83	40		495
Stope Access	47	23			21	48	25		45	40	248
Diamond Drill Drift							4	124	38		166
Vent Drift									77		77
Total Lateral	**78**	**83**	**40**	**62**	**77**	**110**	**89**	**207**	**200**	**40**	**986**
Raising											
Main Intake/Exhaust									131	97	228
Level Exhaust											
Manway											
Total Lateral									**131**	**97**	

1.12.1.3 Operations

All development during operations phase, including the extension of the main ramp to depth will be driven by the production forecast and will be deferred as long as possible - driven just in time for production. This will be done to defer development costs and minimize exposure time to negate re-habilitation prior to or during use. Diamond drill drifts will be the exception, driven at least 6 months ahead of production from the areas to which they will provide drill access, to allow time to drill the definition drillholes and process their results well ahead of mining.

Each stope access crosscut will be collared and driven four rounds (approximately 15 m in length) as the main ramp is excavated, so that the face can be continued later without disrupting activities or services in the main ramp.

Table 1-12 shows a summary development schedule for the operations period of 2008 to 2018.

Table 1-12: Operating Development Schedule, (m)

Description	2008	2009	2010	2011	2012	2013	2014	2015	2016	Total
Lateral										
Ramp	168	251	98	198	145	126	193	282	25	1,486
Stope Access	89	81	269	137	192	234	212	235	206	1,655
Diamond Drill Drift	233	166	406		149	315				1,269
Vent Drift	227	118	45	111	105	72	76	133	80	967
Total Lateral	**717**	**616**	**818**	**446**	**591**	**747**	**481**	**650**	**311**	**5,377**
Raising										
Main Intake/Exhaust	100									100
Level Exhaust	48	32	32	32	64	16	32	48	16	320
Manway	126	42	42	42	42	63	21	84	21	483
Total Lateral	**274**	**74**	**74**	**74**	**106**	**79**	**53**	**132**	**37**	**903**

Note that there is no development in the final two years of the mine's life, 2017 and 2018, during which time the barrier pillars will be extracted in the saddle zone from the main ramp.

1.13 References

Barton, N., Lien, R., Lunde, J. 1974. Engineering Design of Tunnel Support. Rock Mechanics. Vol. 4, No. 4.

Bieniawski, Z. T. 1976. Rock Mass Classification in Rock Engineering. In Exploration for Rock Engineering, Proceeding of the Symposium, (ed. Z. T. Bieniawski). Balkema.

Bieniawski, Z. T. 1989. Engineering Rock Mass Classifications. John Wiley and Sons Inc., Toronto, Canada.

Grimstad, E., Barton, N. and Loset, F. 1993. Updating the Q-system for NMT. Proceedings of International Symposium on Sprayed Concrete, Fagernes, Oslo, Norwegian Concrete Association.

Drawings

Appendix A