2.5 Mine Plan

2.5.1 Overview

The current Wolverine reserve is estimated at 3.5 Mt grading 12.43 % zinc (Zn), 1.44% lead (Pb), 1.37% copper (Cu), 337 g/t silver (Ag) and 1.59 g/t gold (Au). It is expected that this reserve will be increased to approximately 5 Mt at a similar grade for the Feasibility Study (FS) due to the following considerations:

- The indicated resource will be expanded through the conversion of inferred resources due to definition drilling.
- A change of mining methods and the incorporation of a dense media separation plant (DMS) will permit mining at narrower widths than previously assumed, allowing for the inclusion of additional resources in the mine plan and reserve estimate.

The Wolverine Mine will operate at a daily production rate of approximately 1500 t/d. This number represents an extraction of 1250 t/d from the resource plus 250 t/d of dilution, primarily from the hangingwall of the orebody. Accordingly, the mine life is estimated at 12 years.

While the massive sulphide ore is “medium” to “very strong” rock, most host rock is generally “weak” in both the hangingwall and footwall of the deposits. Two exhalite zones are exceptions, the EXCP (a calcite unit) and the EXMT (a magnetic unit referred to as the “iron formation”). Most long-term development will either be located in one of these exhalite units or in the ore itself. The existing 250 m of ramp has been located in the EXCP unit, which has been found to be suited to this purpose. The EXCP generally occurs 40-50 m into the hangingwall of the ore.

There will be two portal entries for the mine: the existing one at 1345 m elevation and a new portal at 1360 m elevation, which will be located on the same bench as the industrial complex (Figure 2.1-2). The existing 1345 Portal will be used as an exhaust outlet for the mine and will house one of the main exhaust fans. The 1360 Portal will be the primary entrance and exit for all men, equipment and supplies.

The mine will be accessed by a 4.8 m wide by 5 m high ramp via the 130 Portal. The ramp will be driven downgrade at – 15%. It will be located central to the two orebodies in a thin zone referred to as the “hump zone”.

Stope access drifts will be driven at 20 m vertical increments from the main ramp. These will generally be 4 m wide by 3.5 m high and driven initially at -15%. As the stopes will be mined in the up-dip direction, subsequent elevations will be achieved by breasting the back of the stope access. The ultimate maximum grade of the stope access will be +15%.

Two mining methods will be employed: Drift and Fill (DF) and Uphole Slashing (US). DF mining will include access by a stope drift located in the ore adjacent to the footwall contact for wider stopes and in the footwall waste rock directly beneath the ore for narrower stopes. Poor roadbeds are anticipated in the stopes because of the softness of the footwall waste rock and the backfill. This will be rectified by leaving a 0.5 m layer of broken ore on the floor, which will be recovered just prior to backfilling the void.

Ore and waste haulage will be done with 40-50 t diesel haulage trucks that will be loaded by large scooptrams (6.1 m^3).

The mine will employ paste backfill as the primary fill. The paste plant will be located at the north end of the industrial complex, closest to 1360 Portal entrance. Mine development waste rock will also be used as loose unconsolidated fill in applications where it will not be exposed by future adjacent mining.

Approximately 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 two raises and the exhaust flowing out of the mine via an exhaust raise and the existing 1345 Portal. 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 1360 Portal. A leaky feeder system will be employed for mine radio communications.

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

Mining will be done using diesel-powered mobile equipment. Drilling will be accomplished using electric hydraulic drill jumbos. A rockbolting machine will be used for bolting. Large (6.1 m^3) and small (2.7 m^3) will be used to accommodate varying mining widths. The haulage trucks will be 40-50 t units.

Over the Life of Mine (LOM), a total of 6.4 Mt of diluted ore will be mined plus 1.94 Mt of development waste. The final locations of the various products is as follows:

- 1.4 Mt of concentrate to smelters
- 4.01 Mt of tailings (2.7 Mt) and waste rock (1.3 Mt) used as fill in the mine
- 2.93 Mt of tailings (1.25 Mt) and waste rock (1.68 Mt) permanently placed in the tailings pond

2.5.2 Project Development

The mine will be developed in three phases:

1. test mining as part of an advanced exploration project, which is ongoing
2. preproduction development
3. operations

2.5.2.1 Test Mining

Test mining is projected to last from June to November 2005, during which time the following will be accomplished:

- excavate the 1345 Portal
- drive the main ramp approximately 250 m from 1345-1320 m elevation, the 1320 stope access drift and 150 m development in or adjacent to the ore
- obtain a bulk sample for metallurgical processing
- use the tailings from the metallurgical sample for paste backfill testwork
- develop dilution, recovery, productivity, and cost criteria for the FS mining methods

- test and evaluate the stratigraphic options for the location of the mine development (footwall, hangingwall, in ore)

2.5.2.2 Pre-Production Period

The preproduction period will last from May 2006 to September 2007, during which time the mine will be prepared for full operating status. This will include approximately 1300 m of additional lateral development and include the following work:

- collar the new 1360 Portal for mine access and haulage directly to the mill
- establish access to three mining horizons, including the extension of the main ramp and the excavation of the stope access crosscuts
- provide additional development for ventilation distribution and emergency egress, including raising to surface
- install and commission 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

2.5.2.3 Operations

Mine operations will start with access to three adjacent production horizons at the top of the mine accessing both Lynx and Wolverine orebodies. The mine will pass from the "pre-production phase" to "operations phase" when a sustained production rate of at least 800 t/d has been achieved.

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 to defer costs and minimize exposure time to negate re-habilitation prior or during use. Diamond drill drifts will be the exception, driven at least six 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.

A full description of the operations phase is contained in Section 3.3.

2.5.3 The Orebody

2.5.3.1 General Description

The orebody is an inclined bedded deposit comprised of two zones, the Lynx and Wolverine. Both are tabular, dipping at 34-38º to the northwest with a strike of approximately 1000 m. Each varies in thickness from 20 m true thickness to less than 1 m.

A simplified visual representation of the orebody is two eggs side-by-side on a skillet that are connected, one representing each zone. The connected egg whites represent a zone of thinner ore between the Lynx and Wolverine zones called the "hump zone". As in the analogy, both deposits tend to thin at higher elevations and on the lateral extents of the deposit. While there is some indication that the orebody thins at depth, the down-dip extension is largely untested and open.

Horizontal sections cut through the center of the deposit can be represented by an elongated barbell; the bulbous "weights" representing the Lynx and Wolverine zones and the barbell representing the thinner hump zone.

Cleavage planes appear to be primarily parallel to the plane of the ore. Faults and joints tend to be vertical and oriented perpendicular to strike (parallel to the dip) of the deposit.

For a more detailed description of the deposit and property geology, refer to Section 2.3.

2.5.3.2 Geological Resources

The most current resource estimate, shown in Table 2.5-1, was prepared by Nilsson Mine Services (Nilsson) in 2000.

Table 2.5-1 Wolverine Indicated Resource Estimate (Nilsson, 2000)

	Indicated	Inferred
Mt	4.941	0.498
Zn,%	13.0	13.61
Pb,%	1.58	1.70
Cu,%	1.43	1.36
Ag,g/t	379.4	365.3
Au,g/t	1.76	1.51

"Indicated" resources were based on a linear distance of 25 m from the nearest diamond drill hole intercept. "Inferred" resources were based on a linear distance of 50 m.

2.5.3.3 Mining Reserve

The current ore reserve estimate was also prepared by Nilsson in 2000, as shown in Table 2.5-2.

Table 2.5-2 Wolverine Probable Reserve Estimation (Nilsson 2000)

	Total Probable Reserves
Mt	3.469
Zn, %	12.43
Pb, %	1.44
Cu, %	1.37
Ag, g/t	336.6
Au, g/t	1.59

The Nilsson reserve is based on a longitudinal overhand cut-and-fill technique proposed by BGC Engineering (2000). Using this mining method, the weak hanging wall is fully exposed. A minimum mining thickness of 4 m was assumed and an overall average dilution of 17% was estimated.

Based on a review of this information and underground experience during the 2005 test mining program, YZC makes the following observations with regard to these estimates:

1. The cut-and-fill mining method selected by BGC (year) will generate more dilution than was included in the reserve estimate.

2. Alternate mining methods proposed herein (DF and US, as described in Section 2.2) may be more appropriate for the orebody.
3. These alternate methods may allow the mining of narrower portions of the orebody than that selected by BGC.
4. The existing estimates do not include the results of two campaigns of surface drilling in the years 2004 and 2005, which will convert much inferred resources to indicated resources.

Considering these factors, it is anticipated that the reserve will be increased in the FS (which is being developed at present). A total mine yield of 5.0 Mt is assumed.

2.5.4 Production Rate

The project work to-date has assumed a milling rate of 1250 t/d.

A simple method of arriving at an indicative mining rate is Taylor's rule, an experienced appreciation of the constraints imposed by the ore body geometry and project economics, as follows:

$$\text{Daily Production Rate} = 0.014*(\text{reserves})^{0.75}$$

For a 5 Mt orebody, Taylor's rule yields a nominal mining rate of 1480 t/d. The projected production rate for the mine is now set at 1500 t/d of diluted ore, based on a new appreciation of the increased proportion of external dilution observed during the underground test mining program. The new nominal production rate simply extracts the original resource at the same rate of 1250 t/d and adds on average an extra 250 t/d for additional dilution.

The mine's capability to achieve and sustain this rate will be determined in the bankable feasibility study. A mine life of approximately 12 years is assumed based on an annual (diluted) steady-state production rate of 584,000 t/y, including allowances for project ramp-up and wind-down.

2.5.5 Geotechnical Considerations

2.5.5.1 Ground Conditions

The intact strength of the various rock types was estimated by using standard ISRM procedures and by using a point-load tester. A summary of the rock strengths as defined by BGC (2000) is shown in Table 2.5-3.

Table 2.5-3 Rock Strength – Field Test Runs

Rock Unit	ISRM Rating (Strength Range)	Comments
Hanging Wall	R1 to R3 (1 MPa to 50 MPa)	Very weak to medium rock. Generally weak.
Iron Formation	R3 – R4 (50 MPa to 100 MPa)	Medium to Very Strong. Generally Strong.
Ore	R3 to R5 (50 MPa to 250 MPa)	Medium to Very Strong rock.
Footwall	R1 to R3 (1 MPa to 50 MPa)	Very Weak to Medium rock. Generally Weak.

The rock mass rating for the hanging wall and footwall are poor and in the ore zone it is considered fair to good.

2.5.5.2 Ground Support

Drift and stope backs will be supported with a combination of rockbolts, weld mesh, and shotcrete as required. Ground support will be applied to the exposed rock walls and back immediately after mucking and prior to any other cycle activity.

Rockbolts will in general be resin-grouted #6 rebar for the back and 33 mm splitsets for the walls. Rock bolts will be installed on 1 m centers in the back and 2 m centers on the walls. The nominal lengths of the bolts will be 1.8 m; however the minimum length of rebar will be one-third of the heading's span. In very poor ground it may be necessary to apply a 25 mm thick layer of shotcrete on the hanging wall, covering bolts and screens.

2.5.5.3 Geotechnical Program

The underground mining program for all phases of mine development will have a thorough and structured geotechnical program to ensure adequacy of support mechanisms and aid YZC in understanding the rock mass and its response to mining. This will include visual observation, instrumentation, rock testing, and the regular usage of professional consulting services.

2.5.5.3.1 Visual Observation

While it sounds simple and intuitive, visual observation is often the best and most comprehensive way to monitor changes in ground conditions in a mine. The key to establishing a good system is the reliable recording of the observations made. A binder will be kept in the engineering office with daily inspection sheets filed chronologically. The engineers, supervisors, and surveyors will document any observations made by recording and where possible by photographing the specific location and visual observation made. This may include loading of rockbolts plates, loading of screen, fresh loose rock on the ground, groundwater inflow, or the formation of cracks in the ground or shotcrete. Cracks will be further monitored by being spray painted so that further movement and spalling is obvious.

2.5.5.3.2 Instrumentation

A suitable instrument for monitoring ground movement or convergence of a drift is a tape extensometer. Tape extensometer is a simple, portable instrument used to accurately measure changes in distance between two points. Measurements are made in any direction – vertical to horizontal – between two reference points. The changes in distance between these two points over period of time can be monitored with accuracy, reliability and repeatability. Normally, five permanent anchor points will be installed around the perimeter of a drift profile and well marked. The initial readings of point to point will be taken and recorded. These will then be measured on a regular basis to mark any change in the initial readings to monitor drift convergence.

2.5.5.3.3 Rock Testing

The primary rock test that will be done during the test mining program is point load testing to predict the range of unconfined compressive strength (UCS) of each rock type for use in rock mass classification. A tester will be located at site for use on core and rock samples as required. Samples will be tested perpendicular to the primary cleavage planes such that the rock strength rather than the shear strength of the cleavage plane. For correlation of the point load test results, representative rock types will also be tested at a commercial laboratory.

2.5.6 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 dimension (2.7-6.1 m^3).

The estimated mine development requirements are shown in Table 2.5-4 for each development type, broken out by exploration phase, pre-production development period, and Life-of-Mine (LOM). See Figure 2.5-1 for a plan view of the mine layout, showing the extent of each phase. Figure 2.5.-2 shows a simplified projection of the mine development on a typical geologic section.

Figure 2.5-1 Life of Mine Development, Plan View (Vol. 2)

Figure 2.5-2 Projection of Mine Development on Typical Geologic Section (Vol. 2)

Table 2.5-4 Mine Development Requirements by Phase

Heading Type	Dimension, m			Nominal Grade	Development Requirements, m			
	Width	Height	Length		Expl	PrePro	Opera-tions	LOM
Main Ramp	4.8	5.0	-	-15%	268	460	1,763	2,492
Stope Access	4.0	3.5	-	-15%	45	135	792	972
Diamond Drill Drift	4.0	3.5	-	2%		350	3,010	3,360
Ventilation Development	4.0	3.5	-	2%	59	130	408	597
Diamond Drill Cut-out	4.0	3.5	4.0	2%	24	118	370	512
Sump	3.0	3.0	6.0	-15%	12	18	70	100
Electrical Bay	3.0	3.0	3.0	2%	3	3	24	30
Safety Bay	1.5	1.8	1.5	2%	14	23	89	125
Remuck Station	4.0	3.5	16.0	2%	16	48	282	346
Truckloading Backslash	4.8	6.5	5.0		5	15	88	108
				Totals	446	1,299	6,895	8,640

Stope headings including stope drifts (in ore or waste), stoping panels, and in-stope development for ventilation or in-ore access between mining levels are not included in the these totals.

2.5.6.1 Main Access Ramp

The mine will be accessed by the 1360 Portal, which will be collared next to the industrial complex (Figure 2.1-2). The existing 1345 Portal will be used as a fresh air feed to the mine. Both portals are named by their approximate elevations, representing meters above sea-level.

The mine will be accessed by a ramp driven at -15% grade located centrally between the Lynx and Wolverine deposits in a zone of weaker mineralization (sub grade ore) called the “hump zone”. At present, 250 m of ramp has been developed during the 2005 test mining program via the 1345 Portal. The ramp has been sized at 4.8 m wide by 5 m high to accommodate a future production fleet of 40-50 t diesel haulage trucks.

While the footwall is the usual location for a main access ramp, the footwall rock at Wolverine is schistose and weak. However, there are two competent units in the hangingwall of the deposits,– a calcareous exhalite (the EXCP) at a nominal horizontal distance of 40 m from the orebody and an iron formation (EXMT) at a nominal horizontal distance of at least 85 m. The existing 250 m of ramp driven for the 2005 test mining program was strategically placed in the EXCP unit, which is the preferred unit. However, in areas where the EXCP unit cannot be used, the ramp will be located in the EXMT or directly beneath the massive sulphides.

The orebodies remain open and untested at depth and the mine is bounded by the property boundary. The down-dip extension is owned by TeckCominco. For this reason, the main ramp is located in the footwall at depth.

2.5.6.2 Stope Access Drifts

Stope access crosscuts will be located at 20 m vertical increments each providing access to five x 4 m high mining lifts. They will be driven a nominal length of 45 m at 4.3 m wide by 4.0 m high to accommodate 6.1 m^3 scooptrams. The orientation will be perpendicular to the stope strike to minimize the length of the drive and maximize stability, as it will also be perpendicular to the bedding and cleavage planes of the hangingwall 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.

The gradient of both the main ramp and the stope access drifts will be maintained at -15% to minimize development requirements. This is not a usual practice; normally both headings are flattened for an intersection. However, it is not unique; Falconbridge’s Kidd Creek Mines in Timmins, Ontario, for example, maintains an average grade of -17% through all intersections.

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 driven later without disrupting activities or services in the main ramp.

2.5.6.3 Diamond Drill Drifts

Diamond drill drifts will be provided at approximately 100 m vertical intervals for ore definition prior to mining. Diamond drill drifts will be driven in a shanty-back profile at minimum widths for drilling (nominally 3.5 m wide) with the back of the drift abutting the footwall of a competent EXMT unit. Drill drifts will be driven the extent of the ore body to provide complete coverage for definition drilling.

2.5.6.4 Miscellaneous Development

Diamond drill cut-outs will be driven on the diamond drill drifts and on the main ramps. On the diamond drill drifts, their spacing will be determined by the drillhole density required for ore definition. In main ramps, a 3 m x 3 m drill station will be located at the center of each curve at the extent of the ramp's profile.

Sumps will be driven approximately every 300 m on the main ramp. Safety bays will be located every 30 m along the main ramps. The presence of a diamond drill cut-out or other development will negate the need for a dedicated safety bay.

Remuck stations will be provided approximately every 300 m in the main ramp and diamond drill drifts. Stope access crosscuts will be started as the main ramp is excavated then serve as remucks or sumps until required for stope access.

Truck loading backslashes will be provided in the main ramp at all stope access intersections.

Passing lanes will be provided in the main ramp at regular intervals (nominally every 300 m), by widening out the ramp to 8 m such that a large mobile unit can pass a second stationary one.

2.5.6.5 Raising

Raising will be done for ventilation and emergency egress from the mine. Because of the weakness of the host rock, all raises will be raisebored or blind-shaft drilled from surface. To prevent degradation of the raise walls, they will be lined with concrete or shotcrete.

2.5.7 Mining Methods

2.5.7.1 Summary

The stopes will be mined in 4 m high horizontal lifts from footwall to hangingwall, and paste backfill with loose waste from the development program will be used to fill the mined void. The mining direction will be up-dip. When one lift is mined and filled, the next will be mined at an elevation 4 m higher and the backfill of the previous lift will form the floor of the stope.

A simplistic plan view of each lift can be represented by an elongated barbell, bulging out for both the Lynx and Wolverine orebodies, with a thinner portion of ore connecting them (the hump zone). The extremities of both orebodies appear to taper in ore width. Figure 2.5-3 shows the selected mining methods applied to a typical stope lift.

Figure 2.5-3 Selected Mining Methods Applied to a Typical Stope Lift (Vol. 2)

The stope access crosscut will be located at the center between the orebodies as shown. A footwall stope drift will be driven from the stope access crosscut along the footwall contact parallel to the strike in both directions through the two ore bodies to the economic extremities of the two ore zones.

There are two basic mining methods that will be employed: drift and fill (DF) and uphole slashing (US). Drift and Fill is further subdivided by whether the footwall stope drift is in ore or waste. Mining method selection will be determined by vertical ore thickness, as shown in Table 2.5-5

Table 2.5-5 Selection of Mining Method by Ore Thickness

Vertical Ore Thickness (m)	Mining Method
>8	Drift and Fill – footwall drift in ore
4 to 8	Drift and Fill – footwall drift in waste
>4	Uphole Slashing

The primary reasons for selecting these mining methods are as follows:

- A high percentage extraction of the deposit can be achieved due to lack of permanent pillars in ore and ability to mine thinner ores.
- Most of the mining backs will be in ore, providing a competent back for most stope headings.
- The poor ground of the hangingwall will have minimal exposure, controlling external dilution and enhancing safety for the workers.
- High productivity can be maintained.

2.5.7.2 Reference Terms

All terms used to describe different components of a stope have been included in the glossary. Figure 2.5-4 is a pictorial aid for understanding these terms. A typical stope is shown on plan and cross sectional views.

Figure 2.5-4 Mining Reference Terms (Vol. 2)

The naming convention shown on the figure is explained as follows:

Stopes will be named for the orebody (“L” for Lynx, “W” for Wolverine or “H” for Hump) followed by the elevation of the lowest sill in meters above sea level. The stope in the figure is W1220, where “W” represents the Wolverine orebody and “1220” represents the floor elevation of the first stoping lift.

Lifts will be named for the stope followed by the lift number, counting from the lowest to the highest. In the figure, the active lift is W1220.4, signifying the fourth lift of W1220 Stope located from elevations 1232-1236 m. In the figure, Lifts W1220.1-W1220.3 have already been mined and filled and W1220.4 is being mined atop the backfill. Stopes will normally extend 28 m vertically and contain seven 4 m high lifts.

As shown, W1220 Stope would be located above W1192 Stope and below W1248 Stope. The uppermost lifts (6 and 7) of W1192 Stope are shown on the figure and have not been mined yet. The lowest lift of W1248 Stope is also shown, with active mining.

W1220 Stope would be on the same elevation and share a stope access with L1220 and H1220 Stopes.

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

2.5.7.3 Drift and Fill Stoping

For wider zones of ore (>8 m vertical thickness), the footwall stope drift will be driven 6 m wide in ore along the footwall contact of the ore. Stoping panels will then be excavated at 6 m wide from the footwall stope drift in a “herringbone” fashion at an angle of approximately 45º to the footwall stope drift. These panels will be driven to and expose the argillitic hangingwall contact.

The panels will be extracted using a primary and secondary sequence. The primary panels will be mined first with solid ore backs and walls over the undercut of the previous lift. These will then be tight-filled with waste rock and paste backfill. The bulkheads will be placed as close to the footwall stope drift as possible to maximize the amount of contact between backfill and the exposed back. The secondary panels will then be mined between the backfilled primary stopes and therefore have 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.

The hangingwall is primarily composed of very poor graphitic argillite and excessive dilution is anticipated at the ends of the herringbone panels once the hangingwall is exposed. It is likely 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 hangingwall is anticipated. Mucking will continue until dilution is excessive, rendering the muckpile uneconomic, at which point the panel will be closed for backfilling.

Panels will be filled 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 bulkheads designed such that they can be placed and sealed rapidly and recovered after the pour for later use. This process will also allow more complete filling of each individual panel than a muckberm and fill fence system, enhancing the stability of the stope.

For ore 4-8 m vertically thick, the method will be modified by locating the footwall stope drift completely in the waste rock of the footwall directly beneath the ore. The width of the footwall stope drift will be reduced to 4 m and the profile will be changed to shanty-back beneath the ore. Primary and secondary herringbone stoping panels will then be excavated in the same fashion but will also be reduced in width to 4 m.

2.5.7.4 Uphole Slashing (US)

For ore less than 4 m vertically thick, the stope drift will be located directly beneath the ore, but will be reduced in width to 3 m, the minimal profile for mobile equipment. The ore above the footwall stope drift will be drilled and blasted using uphole slash holes drilled by a longhole drill. Slashing will start at the furthest extent of the drift and will retreat back to the drift and fill area. Blasting will be done in increments of 4 m at a time. The mined void will not be filled and is expected to cave. As a result, it may be necessary to occasionally leave short strike lengths of ore as pillars to stabilize the rock mass. If required, paste backfill could be poured into the void from the next upper footwall stope drift when the next lift is accessed and mined.

General Discussion

Mining method will be determined on the basis of ore thickness. To aid in the selection, testholes will be drilled regularly in the backs and walls of the footwall stope drifts as they are driven to locate the hangingwall contact and determine ore thickness. This will assure integrity of the stope back and help set the transition points from one stoping method to another. Assaying the cuttings from the testholes will not be required, as the ore/waste hangingwall contact will be defined by the cuttings colour and penetration rate.

For any given lift, the footwall stope drifts will start in the thin ore of the hump 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. Ultimately, it will be recovered, or partially recovered by US.

The ore will 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.

It is anticipated that to sustain the production rate of 1250 t/d (undiluted), three to four stope rounds will be required daily from the footwall stope drift and herringbone panels of the DF stopes plus 11 m of strike length slashed in the US stopes.

The mine will operate numerous lifts simultaneously; as many as four will be active at the same time. Ramp connections will be made between active stoping lifts in ore along the footwall to provide ventilation connections and emergency egress.

Roadbed will be an ongoing concern of the mine. Most of the stope will have a solid rock floor; however, the rock type is a soft rhyolite schist that is anticipated to be easily gouged by the mine vehicles. Portions of the stope, in particular the herringbone panels, will have paste backfill in the floor. Paste backfill is a very poor mucking floor, as the tires of the equipment, particularly the scooptrams, tend to bite into the hardened paste and churn it. To remedy both mucking floors, a 0.5 m, layer of ore will be left on the floor until the completion of mining. It will be recovered from mining voids just prior to filling.

2.5.8 Haulage

Ore and waste rock will be hauled from the muckpiles to remuck stations using scooptrams. At least two sizes of scooptram will be employed in the mine: 6.1 m^3 units will be the primary scooptram, used to muck ore and waste from most headings and to load the haulage trucks. Smaller 2.7 m^3 scooptrams will be used to muck out any heading that is less than 4 m wide.

Ore and waste rock will be loaded into 40-50 t diesel trucks and hauled to surface. Ore will be hauled out of the 1360 Portal and dumped directly into a coarse ore bin at the crushing plant. Waste rock will be stockpiled on the existing pad for eventual haulage to the tailings pond using surface equipment. Some will be returned.

2.5.9 Backfill

To promote overall mine stability, most mined voids will be filled. Two types of backfill will be used, paste backfill and loose waste generated by the lateral development.

The DF stopes and their accesses will be filled as completely as possible. Approximately 25% of the primary stoping panels will be filled with loose waste. The remaining 75% of the void will be filled with paste backfill, encapsulating the waste rock so that the backfill walls will be self-supporting when exposed by secondary mining. Secondary stopes and stope access drifts will be filled as tightly as possible with loose waste (it is assumed that 70% of the voids can be filled). The remaining 30% will then be filled with paste backfill.

US stopes and their accompanying stope drifts in the waste footwall will be left unfilled as much as possible. In some instances, it will be necessary to check a caving stope back with fill. In such instances, paste backfill will be pumped into the cave void through an inclined borehole, either on the stope lift at the brow or from the next, upper stope lift. It is assumed that 30% of all US stopes will be filled.

Mine development will not be backfilled unless required for regional stability. It is assumed that 20% of all mine development will be filled.

The paste backfill plant will be located in the industrial complex at the north end, closest to the 1360 Portal. The paste will be manufactured from unclassified tailings. 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. Precise recipes will be developed through testwork to determine product strength versus set time and binder addition. At present, an average cement content of 3% is assumed.

Paste backfill will be passed to the mine through the main ramp via the 1360 Portal, Paste backfill will be delivered throughout the mine in steel pipe that will be rigidly mounted in the main ramp. 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.

2.5.10 Materials Balance

A materials balance flowsheet is shown in Figure 2.5-5 for the life of the operation. The original total mined rock and the final location of each material have been shown in bold. As shown, the total quantity of broken rock (ore plus waste) is 8.34 Mt over the LOM. The final location of these products is 1.4 Mt concentrate that will be trucked to smelters, 2.93 Mt stored in the tailings pond, and 4.01 Mt stored in stope voids as fill.

Note that the tailings pond will be comprised of 0.63 Mt of development rock, 1.05 Mt of DMS flotation rock, and 1.25 Mt of ground tailings from the processing plant.

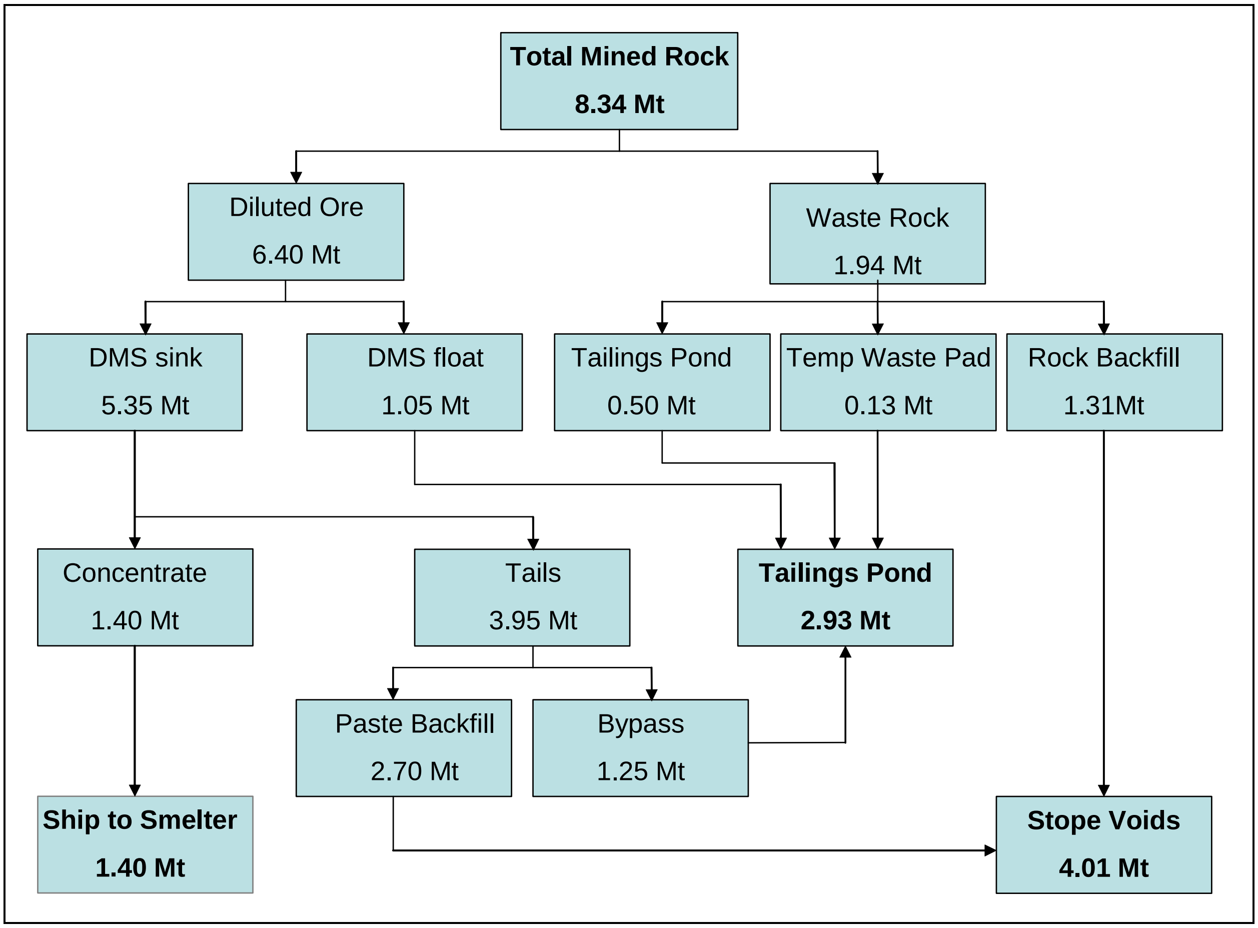

Figure 2.5-5 Material Balance for the Operation

2.5.11 Mine Services

2.5.11.1 Ventilation

A total ventilation flow of 165 m^3/s is anticipated for the mine based on the equipment list shown in Table 2.5-6. An upcast ventilation system is proposed with two fresh air raises (the FAR #1 and FAR #2) and one return air raise (RAR #1).

FAR #1 will be collared and will extend vertically from near the main ramp in the hump zone at 1216 m to surface at 1400 m, a distance of 184 m. FAR #2 will be collared and will be inclined at approximately 68º to a deeper portion of the main ramp at a lower elevation of 1132 m, a distance of approximately 300 m.

RAR #1 will be collared and will extend vertically from near the main ramp at 1310 m elevation to surface at approximately 1395 m elevation, a distance of 85 m.

The raises will be shotcreted or concrete lined. Both fresh air raises will be equipped with main fan installations and direct-fired propane mine air heating installations. FAR #1 will have ladders and platforms installed to provide the mine with a means of emergency egress.

Air will then flow up the ramp to RAR #1 and will flow out of both the 1345 and 1360 Portals, neither of which will be equipped with exhaust fans. An exhaust system of short drifts and raises will be established in the footwall of the main ramp to provide an exhaust conduit that parallels the fresh-air supply in the main ramp.

Ventilation for the individual headings will be provided by use of secondary fans (25-75 hp) and flexible vent tubing.

Ramp connections between stope drifts will be regularly provided throughout the mine for ventilation flow and emergency egress. These headings will generally be driven along the footwall in ore.

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. However, a strict policy of fresh in – exhaust out without re-using the flows is unavoidable. Regular monitoring of flow volumes and air quality will be done by mine supervision and technical staff to ensure that all flows meet or exceed quality standards.

Airflow to the various heading will be controlled by the use of regulators installed in the raise and drift bulkheads.

2.5.11.2 Power and Communications

Power will be used in the mine primarily by the drilling equipment, secondary ventilation fans, and larger de-watering pumps. The estimated connected load is 2.2 MW with an average load of 1.4 MW. These are fairly low in comparison to most operations due to the lack of facilities, crushing or conveyances in the mine.

Power will be drawn from the main generators located adjacent to the processing building. The electrical cable will enter the mine via the 1360 Portal and be distributed throughout the mine in the development headings. Power cables will be suspended in the mine development headings. The two main intake fans and the mine air heaters will be supplied by overland cable from the surface generators at the mill.

The mine will be equipped with a leaky feeder system for radio contact. Individual and vehicle radios will be used. A fixed phone system will also be installed throughout the mine with phones placed in all refuge stations and at regular intervals along the main ramp.

2.5.11.3 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 50-100 mm diameter pipe in secondary headings and stopes.

2.5.11.4 Mine Dewatering

Groundwater and drill water will be collected in the mine by use of 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 pipe line suspended in the main ramp with the other mine service lines.

2.5.11.5 Mine Supplies

Bulk mine supplies not required heated storage will be kept on the existing portal pads (Figure 2.1-2). This includes such items as steel or plastic pipe, 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. A small quonset hut or similar structure will be built on the lower portal pad for 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.

2.5.11.6 Explosives

The existing powder and cap magazines will continue to be used during operations. These will be relocated to a new location on the existing temporary winter road to make their locations 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.

ANFO (ammonium nitrate fuel oil) will be the base explosive for the mine with dynamite and wrapped emulsion used in secondary quantities as projected in Table 2.5-6.

Table 2.5-6 Projection of Daily Explosives Usage

Explosive Type	% of Total	kg/d
ANFO	50%	750
Emulsion	25%	375
Dynamite	15%	225
Total		1,350

2.5.11.7 Emergency Egress

Travel to the mine will normally be in and out of the 1360 Portal. 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.

2.5.11.8 Refuge Stations

Portable fiberglass refuge stations will be employed in the mine. Three units are anticipated during operations. Each will be hooked 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, in particular to accommodate development crews operating in blind headings.

Dedicated lunch rooms will not be provided in the mine, and the portable refuge stations are not designed to accommodate both functions.

2.5.12 Mine Equipment

The mine will be serviced by diesel-powered mobile equipment. A preliminary list is shown in Table 2-5-7.

Table 2.5-7 List of Projected Mine Mobile Equipment

Unit Type	Model	Number
Jumbo Drill	Axera 6-240	2
Rockbolting Machine	Robolt 5-126	1
Scissor Lift	Getman A64	1
Large Scooptram	Toro 1400	3
Small Scooptram	Toro 006	1
Haul Truck (underground)	Toro 40	2
Grader	Caterpillar 12H	1
Powder Truck	Getman A64	1
Bulldozer	Cat D5	1
Kubota	Mine Master	3
Toyota	Land Cruiser	3
Mobile Shotcrete Machine	Getman A64	1
Crane Truck	Getman A64	1