Independent Technical Report on the Wolverine Project – Finlayson District, Yukon

Report No.: 0652630101-REP-R0004-01

28 February 2007

Report to:

YUKON ZINC CORPORATION

Independent Technical Report on the Wolverine Project – Finlayson District, Yukon

EFFECTIVE DATE: 28 FEBRUARY 2007

Prepared by	Marten Regan	Date	28 February 2007
Reviewed by	Peter Wells, AScT BComm	Date	28 February 2007
Authorized by	Rick Alexander, P.Eng.	Date	28 February 2007

WARDROP

Suite 800, 555 West Hastings Street, Vancouver, British Columbia V6B 1M1
Phone: 604-408-3788 Fax: 604-408-3722 E-mail: vancouver@wardrop.com

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

GLOSSARY

UNITS OF MEASURE

Annum (year)	a
Centimetre	cm
Cubic centimetre	cm^3
Cubic metre	m^3
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Degree	°
Degrees Celsius	°C
Foot	ft
Gram	g
Grams per tonne	g/t
Greater than	>
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilogram	kg
Kilometre	km
Kilovolt	kV
Kilovolt-ampere	kVA
Kilowatt	kW
Kilowatt hour	kWh
Less than	<
Litre	L
Litres per minute	L/min
Megawatt	MW
Metre	m
Metres per minute	m/min
Metres per second	m/sec
Micrometre (micron)	µm
Millimetre	mm
Million	M
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Ounce	oz
Parts per million	ppm
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Quarter	Qtr.
Second (time)	sec

Specific gravity	SG
Square kilometre	km^2
Square metre	m^2
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Volt	V
Week	wk
Year (annum)	a

ABBREVIATIONS AND ACRONYMS

ACME Analytical Laboratories Ltd.	ACME
ALS Chemex Laboratories	ALS Chemex
Atna Resources Ltd.	Atna
Butterfield Mineral Consultants Ltd.	BMC
Capital Cost Allowance	CCA
Coefficient of Variation	CV
Dense Media Separation	DMS
Drift and Fill with Primary and Secondary Panels	DFPS
Drift and Fill with Retreat Panels	DFRP
Drift and Fill with Side Slash	DFSS
Environmental Assessment	EA
Finlayson Lake District	FLD
Fresh Air Raise	FAR
Graphitic Argillite	ARGR
Hatch Associates	Hatch
High-Density Polyethylene	HDPE
Klohn Crippen Berger	Klohn
Large Corporation Capital Tax	LCT tax
Load Haul Dump	LHD
Locked-Cycle Test	LCT
Maintenance and Repair Contract	MARC
Mine Ventilation Services, Inc.	MVS
Net Smelter Return	NSR
Process Research Associates Ltd.	PRA
Procon Mining and Tunnelling Ltd	Procon
Quality Control/Quality Assurance	QA/QC
Return Air Raise	RAR
Rock Mass Rating	RMR
Rock Quality Designation	RQD
Snowden Mining Industry Consultants Ltd.	Snowden
Standard of Deviation	SD
Total Core Recovery	TCR
Underhill Geomatics Ltd.	Underhill
Voice Over Internet Protocol	VOIP
Wardrop Engineering Ltd.	Wardrop
X-ray Diffraction	XRF
Yukon Zinc Corporation	YZC

1.0 SUMMARY

This technical report summarizes the results of the Optimized Feasibility Study of the Wolverine Project for Yukon Zinc Corporation (YZC) completed by Wardrop Engineering Inc. (Wardrop) in January 2007.

A previous feasibility study, carried out by Hatch of Vancouver was issued in May 2006. In 2005, an advanced exploration program was conducted to provide access for test mining. This program consisted of developing a portal and 450 m of decline into the ore zone. The underground is been kept dewatered by a care and maintenance crew living at the existing camp on site. Personnel, materials, and supplies are brought to site by air using the 950 m long airstrip near the proposed process facilities.

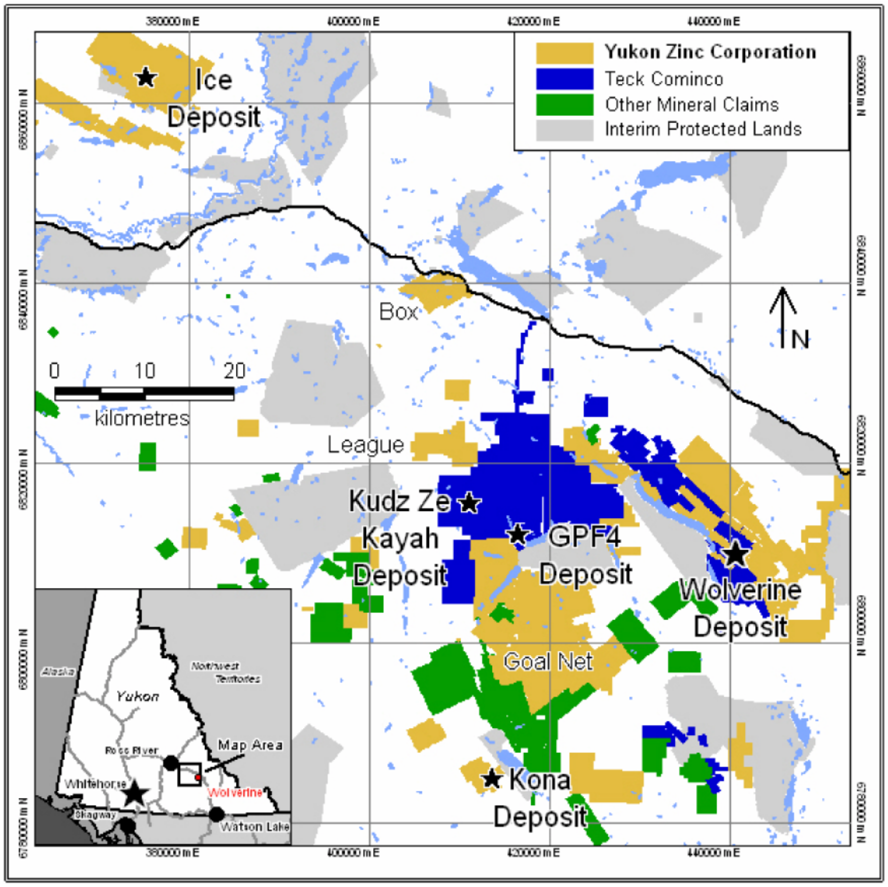
1.1 PROPERTY DESCRIPTION

This section was written by G. Giroux, P.Eng. and C. Pearson, P.Geo., and extracted from the Hatch Independent Technical Report posted to SEDAR on 20 June 2006. The Wolverine property is located 190 km northwest of Watson Lake in the Yukon (Figure 1.1). The mine will employ approximately 214 employees for a total duration of approximately 12 years.

The Wolverine property consists of 1064 quartz-mining claims located in the Watson Lake Mining District, Yukon (Figure 1.2). The property is accessed using helicopter or fixed-wing aircraft at present. The original winter road used for the 1996 drill program has only limited passage during the winter months, although it can be re-established along its original trace, as was done in the winters of 2005 and 2006.

The Wolverine deposit is a polymetallic volcanogenic massive sulphide (VMS) deposit, hosted in siliciclastic and volcanogenic rocks of the lower Mississippian Wolverine Lake Group within the Finlayson Lake District (FLD). The FLD is a known region of base metal mineralization in the Yukon Territory, which hosts several VMS-style deposits.

Figure 1.1 General Location Map



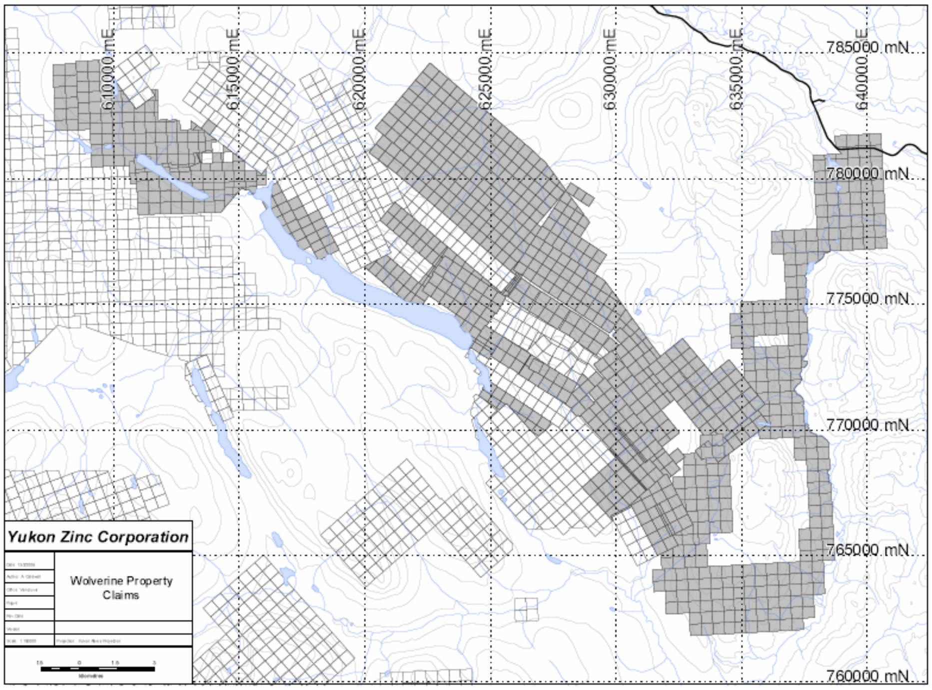
1.2 OWNERSHIP

YZC has written this section. YZC owns 100% of the claims within the areas that mine site infrastructure and access road location are proposed, except for the Money claims for which it has an option agreement that provides for access.

Atna Resources Ltd (Atna) retains a royalty interest on net precious metals revenues only. The Precious Metals Royalty provides for Atna to receive 4% of net proceeds from the sale of silver and gold when the silver price is greater than US$5.00 and less than US$7.50/oz. The Precious Metals Royalty increases to 10% of the net proceeds from sale of silver and gold when the silver price is over US$7.50/oz. No royalty is payable to Atna on zinc, copper, and lead.

A portion of the Wolverine Project lands is also subject to two small royalties to the original claim holders. The Kink 3 claim, which covers approximately half the Wolverine Deposit, is subject to a 1% net smelter return (NSR) royalty payable to Nordac Resources Ltd (now Strategic Metals). Half of the NSR royalty may be purchased for $500,000. The Nordac Royalty is 1.0% of NSR, reducing to 0.5% after cumulative payments of $500,000 have been made.

Figure 1.2 Property Claim Map



Note: Wolverine property claims are in dark grey.

The Foot 1-20 and Pak claims are subject to a 0.5% NSR royalty payable to Equity Engineering. It may be purchased at any time for $500,000. Most of the Wolverine Deposit and the Sable Zone are located on these Foot claims.

1.3 Resources and Reserves

This section was written by G. Giroux, P.Eng., and extracted from the Hatch Independent Technical Report posted to SEDAR on 30 June 2006. A. Polk, P.Eng. prepared the estimate of ore reserves for the Wolverine orebody shown in Table 1.2, overleaf. The Wolverine massive sulphide deposit is a weakly undulating tabular body striking northwest and dipping shallowly to the northeast. There are two significantly thickened stratiform sulphide zones, the Lynx and Wolverine zones, separated by a deformed package of Wolverine stratigraphy with dominantly replacement and stringer sulphide mineralization called the Saddle zone.

Table 1.1 shows the mineral resource.

Table 1.1 Mineral Resource Estimate (US$80/t)

Resource Category	Tonnes	Zn %	Cu %	Pb %	Ag g/t	Au g/t
Measured	493,000	12.44	1.18	1.48	298.8	1.50
Indicated	3,968,000	12.10	1.16	1.58	361.8	1.72
Total	4,461,000	12.14	1.16	1.58	354.8	1.70
Inferred	1,693,000	12.16	1.23	1.74	385.4	1.71

Table 1.2 Ore Reserves Estimate (Diluted)

Category	Tonnes	Zn %	Cu %	Pb %	Ag g/t	Au g/t
Proven	564,000	10.31	0.96	1.24	246.9	1.24
Probable	4,588,000	9.59	0.91	1.26	286.2	1.37
Total	5,152,000	9.66	0.91	1.26	281.8	1.36

1.4 MINING

A. Polk, P.Eng. wrote this section. Mechanized drift and fill mining will be employed as the stoping method in the Wolverine Mine. Electric hydraulic drill jumbos mounted on mobile diesel carriers will be utilized as the primary production drills, advancing stopes and selectively mining the shallow dipping ore. Load haul dump units (LHDs) will load and haul the broken ore from the stope headings and into low profile 30 and 50 tonne capacity haulage trucks for transport to surface. Electric hydraulic bolting machines, likewise mounted on diesel carriers, will be utilized to install rock support in the form of split sets, re-bar, and wire mesh.

Stoping blocks will be 20 m in height and each will require an access from the centrally located decline. Each stoping block will be comprised of 5 x 4 m high lifts. Accesses will initially be down grade from the decline to the lowest lift of a stope. Once a lift has been mined and completely backfilled, the access will be back-slashed to gain an additional 4 m of elevation and the next lift started.

Cemented paste fill created from the process will be pumped underground in a piped delivery system and into voids after they are mined. Low head bulkheads will be installed at the entrance to a drift panel that needs to be filled. These will typically be constructed with split sets, cables, wire mesh, and geotextile material. The bulkheads will occasionally need to be shotcreted to achieve a seal and prevent fill spillage.

In 2005 an advanced exploration program was conducted to investigate the rock conditions, confirm the applicability of the mining method, and produce a bulk sample. The 1345 (elevation) Portal and approximately 450 m of tunnel were developed to provide access and perform test mining. At the end of 2005, the decline was allowed to fill with water over the winter season.

In 2006, the access decline was dewatered and several areas of the tunnel required extensive rehabilitation. During the summer of 2006, most of the existing decline was shotcreted and additional support was added and is now deemed to be sufficiently supported for long term access into the mine. As a consequence of the extensive amount of rehabilitation that was required in 2006, a wet mix shotcrete delivery system will be purchased and utilized, particularly in waste development tunnelling.

A production rate of 1,700 t/d of diluted ore is planned for the Wolverine Mine. An additional 150 t/d of waste will also be created by operations, with some of the waste being hauled to surface stockpile as dictated by operational needs. It is planned that all of the waste brought to surface will eventually be moved back underground and permanently stored in voids as a small portion of the fill.

1.5 METALLURGY AND PROCESSING

John Fox, P.Eng. of Laurion Consulting, wrote this section. Several months of metallurgical testwork was carried out in the last half of 2006. This work was done to confirm results obtained in the metallurgical campaign conducted in year 2000. The knowledge gained from all of the testwork was then used to prepare the design criteria and produce the flowsheets for the optimized feasibility study. Additional dense media separation testwork was conducted to confirm the application and benefit of the dense media separation (DMS) process.

The process plant equipment was selected to accommodate a run-of-mine (ROM) production rate of approximately 1,700 t/d diluted ore. ROM production will be crushed and then upgraded using a dense media separation plant to produce 1,400 t/d of mill feed.

Wardrop calculated the post-DMS mass balance; the mill feed is projected to be 4,238,149 tonnes grading 11.70% Zn, 1.10% Cu, 1.52% Pb, 340.86 g/t Ag, and 1.64 g/t Au.

Two staged grinding will be employed followed by differential flotation and rougher concentrate regrinding to produce three separate concentrates: copper, lead, and zinc. The copper and lead concentrates include payable levels of gold and silver. Thickening and filtration will be used to produce saleable concentrates, which will be trucked to Stewart, BC, for shipment to overseas smelters.

1.6 PERMITTING

YZC wrote this section. Exploration activities at the project site have been and continue to be governed by two licenses, a Mining Land Use Permit, and a Type B Water Licence.

The Environmental Assessment (EA) review was completed in September 2006. The approval documents signed by the departments of Energy, Mines, and Resources and the Executive Council Office concluded that: "… taking into account the implementation of the mitigation measures and follow-up, the project is not likely to cause significant adverse environmental effects." A Quartz Mining Licence was issued in December 2006.

Revised documentation for a Type A Water Licence, required for water use and waste deposition during construction activities and operations, was submitted to the Yukon Water Board in mid January 2007. YZC anticipates issuance of the Type A Water Licence in late summer 2007.

1.7 ENVIRONMENTAL

Wardrop wrote this section. Environmental management issues associated with YZC's Wolverine Mine and mill are primarily associated with water management during operations and groundwater at closure. Wolverine Creek is fed by groundwater from the proposed mining area, and Go Creek will receive treated effluent from the tailings management facility. Baseline environmental studies therefore focused on groundwater and surface water environments. Aquatic data were collected in 1995, 1996, 1997, 2001, 2005, and 2006. Wolverine, Go, and Money creeks, as well as Little Wolverine and Wolverine lakes, were the focus of the characterization studies, but the program was expanded at times to include creeks along the road route and regional sites. Components characterized include hydrology (interpretation of evaporation and snowmelt data, regional and local hydrometric data, flow frequency analysis, and peak and low flow analysis), water and sediment quality (total suspended solids, pH, conductivity, alkalinity, sulphate, metals, and nutrients), periphyton, benthos, zooplankton, fish, and fish habitat.

1.8 CONSTRUCTION

Wardrop and YZC wrote this section. The current development plan provides for commencement of construction of the all season road and mobilization of equipment in the first half 2007, and excavation of plant site in the second quarter, followed by foundations and mill steel in third quarter. Completion of the mill and facilities and commissioning of the mill are scheduled for fourth quarter of 2008. Full production is planned for first quarter 2009.

An EPCM contracting approach has been assumed for the Project. A maximum workforce of about 200 is anticipated during construction.

1.9 PROJECT ECONOMICS / CAPITAL COST

Wardrop prepared an economic model and carried out a financial evaluation of the project. In accordance with industry standards, the project was evaluated on a 100% equity financed basis and no project debt financing is assumed although leasing of certain equipment has been incorporated. This has the effect of reducing capital costs and reducing operating costs.

Wardrop prepared the total capital cost for the design and construction. The capital costs are estimated to a -5% to a +15% accuracy level and included the mine, the mill, and all associated infrastructure. The operating cost also prepared by Wardrop was based on the updated process flowsheets, design criteria, and reflects current costs as 4th quarter 2006.

The total capital costs for the project including contingency and owners cost is Cdn$207.6 million and based on all new equipment. The unit operating cost is approximately Cdn$95.58/t mined. The estimated capital costs and operating costs incorporated into the financial analysis are summarized in Tables 1.3 and 1.4.

Table 1.3 Capital Cost Summary, (Million Cdn$)

	Cost	Percentage
Project direct cost	138.943	75.81
Project indirect cost	44.343	24.19
Total, direct+indirect	183.281	100.00
Contingency (% to total direct+indirect)	24.375	13.30
Total Project Costs	207.662	

1.10 OPERATING COST

Wardrop wrote this section. The total operating cost is calculated to be $95.58/t of ore mined based on a 621,308 t/a mine rate and using power generation on site (see Table 1.4).

Table 1.4 Operating Costs – 621,308 t/a

Item	Staff, Supplies, and Operation ($)	Maintenance ($)	Power ($)	Total ($/t mined)
Mine	36.99	4.82	5.19	47.00
Mill	19.54	3.63	9.72	32.89
G&A	15.25	-	-	15.25
Tailings	0.06	0.08	0.30	0.44
Total	71.84	8.09	15.21	95.58

1.11 ECONOMIC ANALYSIS

Wardrop wrote this section. The economic model prepared was based on the following assumptions:

- mine construction to start in 2007 with commissioning in late 2008
- moving average exchange rate
- moving average metal prices
- NSRs.

The model was prepared on a pre-tax and pre-finance basis as the tax regime is unknown at this time. There were four metal price scenarios developed and used for the evaluation and these are:

Case 1 – Model with 3 Year Moving Average Prices
Case 2 – Model with Combined 2 and 3 Year Moving Average Prices
Case 3 – Model with 2 Year Moving Average Prices
Case 4 – Scenario Model with Current Prices for Comparison Purposes Only.

Table 1.5 provides a summary of the economic analysis and shows the results giving the IRR and NPV with a discount rate of 8% for each case. Table 1.6 provides the metal prices used for each of the cases used in the analysis. Revenue was determined by grade and metal price, adjusted for marketing, transportation costs and mining recovery. A series of sensitivity analyses were prepared and included in Section 23.8 of this report. Table 1.7 provides a summary of payable metal in relation to the recovered metal in the concentrates.

Table 1.5 Economic Analysis Summary, (Cdn$)

	Pre-tax IRR (%)	Pre-tax NPV 8% ($ million)	Payback Period	3 Year Full Production Cumulative Pre-tax Cash Flow	Average Annual Cash Flow For First 3 Years Pre-tax Full Production
Case 1	18.9	104.8	3.9	172.5	57.50
Case 2	22.6	134.3	3.2	204.9	61.47
Case 3	26.3	184.2	3.0	217.7	65.31
Case 4	56.8	571.7	1.5	439.3	13.18

Table 1.6 Metal Prices, (US$)

	Zinc (US$/lb)	Copper (US$/lb)	Lead (US$/lb)	Silver (US$/oz)	Gold (US$/oz)
Case 1	0.87	1.85	0.48	8.54	486.85
Case 2	0.87 and 1.07	1.85	0.48 and 0.52	8.54 and 9.48	486.85 and 526.65
Case 3	1.07	1.85	0.52	9.48	526.50
Case 4	1.84	1.85	0.76	12.69	626.01

Table 1.7 Payable Metal by Concentrate

	Units	Metal Recovered to Concentrates (LOM)	Payable Metal (LOM)
Zinc	(t)	461,364	376,979
Silver	(oz)	38,597,810	33,179,153
Copper	(t)	42,261	35,403
Gold	(oz)	150,949	119,311
Lead	(t)	45,892	25,661

For this level of study, the contingency was based on the confidence of the design to a maximum of -5% to +15%.

Revenue was determined by grade and metal price, and adjusted for marketing, transportation cost, and mining recovery.

1.12 MARKETING

Wardrop wrote this section. The most likely markets for all three concentrates are Asian, specifically Japan, Korea, and China. Shipment to these markets will be made through the Port of Stewart, BC, which is being used regularly for exports of Eskay Creek and Huckleberry mines.

1.13 CONCLUSIONS

Wardrop wrote this section. Technically and economically, the Wolverine Project is a viable project.

2.0 INTRODUCTION AND TERMS OF REFERENCE

2.1 INTRODUCTION

Yukon Zinc Corporation (YZC) commissioned an optimized feasibility study (OFS) for the wolverine project in August 2006. An earlier feasibility study on the project was completed in May 2006. Among other changes, the OFS is based on an increased mill throughput rate from previous 1,250 t/d to 1,400 t/d.

The purpose of the OFS was to provide the YZC management team the necessary information to make a decision with respect to the development of the Wolverine project. All previous concepts were reviewed followed by a new scope definition, capital and operating costs developed and an overall economic evaluation was carried out.

2.2 TERMS OF REFERENCE

Wardrop was retained by YZC to produce a complete OFS of the Wolverine project that is compliant with National Instrument 43-101 (NI 43-101).

In preparation of this report, Rick Alexander, P.Eng., visited the property during the third week of November 2006. Additional information was obtained and discussions conducted regarding the nature of the deposit and general site information.

The persons taking responsibility for certain sections of this report, and the extent of their responsibility for each section for the purposes of the NI 43-101 are shown below.

Responsible Person	Independent QP	Company	Primary Areas of Responsibility	Relevant Section
Rick Alexander, P.Eng.	Yes	Wardrop	Study Compilation, Construction, Capital & Operating Cost Estimates & Financial Analysis	- 1.9, 1.11, 1.12, 1.13, 2.2, 18, 19.1, & 23.10 - Summarized from inputs of other responsible persons: 1.8 1,10, 2.1, 3 to 15, 17.1 to 17.8, 20.1, 20.3, 23.3, 23.7, 23.8, 23.9 & 23.11
Allan Polk, P.Eng.	Yes	Snowden Mining Industry Consultants, Ltd.	Mining, Reserve Estimates & Recoverability	- 1.4 17.9, 19.3, 20.2, 23.1, & 23.11 - With Gary Giroux: 1.3 - Information relating to areas of responsibility in Section 1.10, 23.7, & 23.8
Cliff Pearson, P.Geo.	Yes	Pearson Geological Services Inc.	Resource Estimates, Geology, Mineralization, Exploration, Drilling, Sampling & other topics covered in relevant sections	- With Gary Giroux: 1.1, 10 to 15, 17.1 to 17.8, 19.2 & 20.1
Gary Giroux, P.Eng.		Giroux Mining Consultants Ltd.	Resource Estimate, Geology, Mineralization, Exploration, Drilling, Sampling & other topics covered in relevant sections	- With Cliff Pearson: 1.1, 10 to 15, 17.1 to 17.8, 19.2, & 20.1 - With Al Polk 1.3
John Fox, P.Eng.	Yes	Laurion Consulting	Metallurgy & Processing	- 1.5 16, & 23.2

Table continues…

Responsible Person	Independent QP	Company	Primary Areas of Responsibility	Relevant Section
Robert Lo, P.Eng.	Yes	Klohn Crippen Berger Ltd.	Drainage, Tailings Disposal & Water Reclaim	- Information relating to areas of responsibility in Sections 23.7 & 23.8
Arvind Dalpatram, P.Eng.	Yes	Klohn Crippen Berger Ltd.	Drainage, Tailings Disposal & Water Reclaim	- Information relating to areas of responsibility in Sections 23.7 & 23.8
T. Dietrich, P.Eng.	Yes	Dietrich Consulting Inc.	Mining Costs for Underground Electrical & Communications Systems	- Information relating to areas of responsibility in Section 23.7
YZC	No	-	Title/Ownership of Property, Permitting, Markets & Marketing, Transportation Costs, Taxes & Royalties	- 1.2, 1.6, 23.4 & 23.6 - Information relating to areas of responsibility in Sections 23.7, 23.8 - With Wardrop: 1.8, 2.1, 20.3, 23.8, &23.9
Ken Deter, P.Eng.	Yes	Wardrop	Metallurgy Overview	- 19.4
David Tyson, R.P.Bio.	Yes	Wardrop	Environmental	- 1.7 & 23.5

3.0 RELIANCE ON OTHER EXPERTS

3.1 RELIANCE

Wardrop has also relied on others for information as well as QPs in this report. Information from third-party sources is referenced. Wardrop has used information from these reports under the assumption that they were prepared by technically competent persons. Data provided to Wardrop is listed in Section 21, References.

3.2 PREVIOUS TECHNICAL REPORT

Information on the items set out in Sections 4 to 15 and 17.1 to 17.8 of this report can be found in the previous Wolverine Mineral Property Resource Estimation Independent Technical Report posted to SEDAR on 15 Mar 2006. Sections 1.1 and 1.3 of this report can be found in the Hatch Independent Technical Report posted to SEDAR on 30 June 2006. To Wardrop's knowledge, there has not been any material change in the information since that date.

3.3 ENVIRONMENTAL AND LEGAL

After a systematic review and assessment by Wardrop of all project interactions with valued ecological, social, and cultural components of the Wolverine project environment, it has been determined that the project will have no significant adverse environmental effects. The detailed design and project development process includes measures for ongoing assessment and action to ensure that YZC can meet its commitment to environmental protection. Adaptive management, including monitoring and refinement of mitigation measures, in consultation with the Yukon Territorial Government (YTG), the Kaska Dena and other interests as appropriate, will ensure project success.

3.4 MARKETING

Butterfield Mineral Consultants Ltd. & H.M. Hamilton Associates did a marketing study, which provided current smelter terms for zinc, copper, and zinc concentrates possible buyers of the Wolverine concentrates, and shipping options and costs. This marketing study has been relied on in the preparation of Section 23.3 of this report.

4.0 PROPERTY DESCRIPTION AND LOCATION

Refer to "Wolverine Mineral Property Resource Estimation," posted to SEDAR on 15 March 2006. This report was prepared by Mr. Cliff Pearson, P.Geo. and Mr. Gary Giroux, P.Eng.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Refer to "Wolverine Mineral Property Resource Estimation," posted to SEDAR on 15 March 2006. This report was prepared by Mr. Cliff Pearson, P.Geo. and Mr. Gary Giroux, P.Eng.

6.0 HISTORY

Refer to "Wolverine Mineral Property Resource Estimation," posted to SEDAR on 15 March 2006. This report was prepared by Mr. Cliff Pearson, P.Geo. and Mr. Gary Giroux, P.Eng.

7.0 GEOLOGICAL SETTING

Refer to "Wolverine Mineral Property Resource Estimation," posted to SEDAR on 15 March 2006. This report was prepared by Mr. Cliff Pearson, P.Geo. and Mr. Gary Giroux, P.Eng.

8.0 DEPOSIT TYPES

Refer to "Wolverine Mineral Property Resource Estimation," posted to SEDAR on 15 March 2006. This report was prepared by Mr. Cliff Pearson, P.Geo. and Mr. Gary Giroux, P.Eng.

9.0 MINERALIZATION

Refer to "Wolverine Mineral Property Resource Estimation," posted to SEDAR on 15 March 2006. This report was prepared by Mr. Cliff Pearson, P.Geo. and Mr. Gary Giroux, P.Eng.

10.0 EXPLORATION

The following is extracted from the Wolverine Mineral Property Resource Estimation Independent Technical Report (2006) prepared by C. Pearson, P. Geo, and G. Giroux, P.Eng. (posted on SEDAR on 15 March 2006).

In 2005, a total of 59 drill holes from a definition drilling program were cored through the Wolverine Deposit for a total of 11,730 m. There were no new geological or structural anomalies, as well as no internal misses in the massive sulphide mineralization, which reinforced the continuity of massive sulphide mineralization. The drill collar locations are presented in Appendix I of the 15 March 2006 Feasibility Study prepared by Hatch.

Another key aspect of the 2005 exploration activities on the Wolverine Deposit was the underground test mining for the Advanced Exploration Program. A total of 450 m of underground development was successfully completed including drifting for approximately 110 m through massive sulphide mineralization. Based on test mining, it appears that ground conditions should not be problematic for YZC, as development continued, and that the location and tenor of the massive sulphide mineralization was comparable to the results of the nearby drill holes.

The drill hole collars, underground workings, and surface facilities constructed during the 2005 advanced exploration program were surveyed by Underhill Geomatics Ltd. (Underhill) and Procon Mining and Tunnelling Ltd. (Procon).

11.0 DRILLING

11.1 DRILLING

The following is extracted from the Wolverine Mineral Property Resource Estimation Independent Technical Report (2006), prepared by C. Pearson, P.Geo. and G. Giroux, P.Eng. (posted to SEDAR on 15 March 2006).

11.2 CORING SUMMARY

An initial drill program of 15 BQ and NQ2 drill holes were completed on the Wolverine Deposit in 1995 totalling 3,684.9 m; which were part of a larger 24 drill hole, regional program that totalled 6,440.9 m. This was followed by 12,012.1 m of drilling in 1996 in 36 drill holes and 5,558.9 m of drilling in 1997 in 24 drill holes. The objectives of the drilling were to delineate the massive sulphide mineralization discovered in 1995. To test for the extension of mineralization elsewhere along the favourable Wolverine trend, an additional 12 drill holes were completed in 1996 totalling 4,913.3 m and an additional 17 drill holes were completed in 1997 totalling 6,341.0 m. In 2000, 7 NQ drill holes were completed totalling 957.8 m, which was primarily on the Lynx Zone in order to collect additional data on the Lynx Zone for the Pre-Feasibility Study known at the time as the Finlayson Project. Drilling in 2004 included large diameter drill holes (HQ) across the Wolverine deposit as part of a multi-purpose exploration program that collected data for definition drilling of the massive sulphide mineralization and geotechnical data on the rock-types, as well as the collection of new samples for metallurgical analysis and dense media testing. A total of 8 drill holes were successfully completed totalling 1,758.2 m. Definition drilling continued in 2005 with the successful completion of 59 NQ and NQ2 drill holes totalling 11,730.1 m. A summary table of all drill collar locations is presented in Appendix I of Wardrop's Optimized Feasibility Study and a summary of all assay composite statistics is presented in Appendix II and III of Hatch's Feasibility Study.

11.3 DRILLING PROCEDURE

The following drilling procedure is in part reconstructed from field observations of drill core by Jason Dunning, P.Geo., Vice-President Exploration who supervises the exploration activities on the Wolverine deposit since April 2003. Mr. Dunning concluded that drilling was conducted to industry standards with core recovery being generally acceptable, notably excellent in the banded iron formations and semi-massive to massive sulphide mineralization and notably poor in some sections of hanging wall argillite.

Though diamond drilling employed various core sizes from 1995 to 2005, extraction of core was performed using a standard diesel/hydraulic, wireline equipped core drilling rig. Drill core was recovered using a 10 ft long core barrel and placed sequentially in core boxes. Wooden block footage markers recorded the depth at the end of each 10 ft run by the Drillers. Drill core was then placed in 5 ft long wooden core boxes/trays, which were then covered and transported to the Core Logging Shack for the Geologist to examine.

11.4 COLLAR SURVEYS

From 1995 to 2000, Underhill of Whitehorse, Yukon, a professional survey firm, registered to practice surveying in the Yukon, had conducted drill collar location surveys of the Wolverine deposit area using standard accepted surveying methods. The Wolverine deposit drill hole collar locations had been surveyed (UTM northing, UTM easting, and elevation) in NAD 27, Zone 9. Procon completed a similar survey of the 2005 drill collar locations, as well as numerous 1995 to 2004 drill collar locations during 2005; which when validated and verified by Underhill who came to the property in October 2005 to act as a third-party auditor of Procon's survey work. There were no discrepancies noted between Underhill and Procon data from the drill collar surveys.

11.5 DOWN HOLE SURVEYS

Drilling activities on the Wolverine deposit were comprised drill holes ranging from less than 150 m to several drill holes that were greater than 350 m drill holes; which were principally measured for deviation using a Sperry Sun (magnetic-photographic) single shot instrument prior to 2004. In 2004 and 2005, YZC purchased a Reflex EZ Shot (digital-magnetic). A typical spacing for these down hole survey measurements was 50 m; however, on longer, deeper drill holes, survey spacing was tightened to approximately 30 m.

12.0 SAMPLING METHOD AND QUALITY CONTROL MEASURES

The following is extracted from the Wolverine Mineral Property Resource Estimation Independent Technical Report (2006), prepared by C. Pearson, P.Geo. and G. Giroux, P.Eng. (posted to SEDAR on 15 March 2006).

The drill core was logged at site by a team of Geologists trained using a common logging system. This logging system uses nomenclature inherited from the Wolverine Joint Venture with data being entered into the computer on the Lagger data entry system. Data collected includes, all drill collar location data, geology, geotechnical observations, structure, and sample location information. The Geologists validate the data for accuracy on a weekly basis in order that new drill sections and level plan maps can be created in order to facilitate the ongoing interpretation of the geological model for the Wolverine deposit.

Assay and geochemical samples are marked by the geologist using coloured lumber crayons with the assay tag number from ALS Chemex Laboratories (ALS Chemex) not only written on the drill core, but also stapled to the core box. It should be noted that sample intervals were chosen to reflect either abrupt geological contacts or significant visually observed changes in grade, but where more evenly mineralized, a more uniform sample interval was preferred. The drill core is then placed in a sampling rack in the saw room and cut by the core technician. One half of the drill core is sawn and placed inside a 6-mil poly sample bag along with the corresponding sample tag from ALS Chemex; which possesses a bar-code. The other half of the drill core is then placed back into the core box for permanent record. The sample bag is then sealed using a zap-strap to prevent tampering and placed sequentially on the layout table in the saw room. Samples are then sequentially placed into pre-labelled rice sacks, which are then sealed by metal tie-straps. It should be noted that each drill hole is shipped to the laboratory individually with the rice bags colour-coded per individual drill hole. Bag numbers and sample locations within each bag are also tracked. All rice sacks containing assay samples are transported by YZC's charter company Alkan Air, who flies from the Wolverine deposit to Whitehorse, Yukon, where Alkan contacts Air North, transports the rice sacks from Whitehorse to ALS Chemex in North Vancouver, British Columbia.

13.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following is extracted from the Wolverine Mineral Property Resource Estimation Independent Technical Report (2006), prepared by C. Pearson, P.Geo. and G. Giroux, P.Eng. (posted to SEDAR on 15 March 2006).

13.1 ANALYTICAL PROCEDURES

On reaching ALS Chemex, the samples are logged into an internal tracking system and the sample is weighed with the weight recorded. The sample is then dried and crushed in order to pulverize approximately 250 g that pass through a 75 µm sieve. All of the crushed reject material is then weighed and stored, noting that ALS Chemex ensures through quality control that at least 85% of the pulp passes through a 75 µm sieve. The pulp is then taken for analysis using techniques identified for high-sulphide samples. Gold and silver are analyzed using the Au-GRA21 and Ag-GRA21 packages respectively; which are both 30 g fire assay with a gravimetric finish. It should be noted that gold has a detection range of 0.05 to 1,000 ppm and silver has a detection range of 5 to 10,000 ppm. Base metals analyzes are done with the ME-ICP61a procedure; which is a 25 element ICP package with a perchloric, nitric, and hydrofluoric acid decomposition that has an ICP-AES method. Detections limits have a wide range because of the 25 elements. Over limits for copper (Cu-OG46), lead (Pb-OG46), zinc (Zn-OG46), arsenic (As-OG46) and antimony (Sb-AA08) are in place with copper, lead, zinc, and arsenic done by an ore-grade, aqua regia digestion with an instrument finish. Detection ranges for copper are 0.01% to 50%, 0.01% to 30% for lead, 0.01% to 30% for zinc, and 0.01% to 30% for arsenic. A second over-limit for zinc is in place, as there are +50% zinc assays known to occur in the Wolverine deposit. Antimony over-limit analyzes are done by a hydrochloric acid and potassium chlorate digestion and AA analysis with an upper detection limit of 0.01%. Mercury is not part of the ME-ICP61a package, so mercury is analyzed by Hg-CV41 that is an aqua regia digestion and cold-vapour atomic absorption (AA) analysis with a detection range of 0.01 to 100 ppm. Should the detection limits be exceeded, re-analysis is done using the Hg-CV42 technique, which is an aqua regia digestion and cold-vapour AA analysis with a detection range of 0.1% to 10%. Selenium is done by X-ray diffraction (XRF) pressed pellet analysis with a detection range of 2 to 10,000 ppm. Specific gravity is measured on the pulverized sample using a pycnometer using the OA-GRA08b technique.

13.2 QUALITY CONTROL AND QUALITY ASSURANCE

13.2.1 STANDARDS

A total of 30 STD#1, 26 STD#2, and 20 STD#3 samples, totalling 76 standard samples that were submitted for analysis together with the drill cores samples from the Wolverine deposit in 2005; noting the approximate insertion of one standard per 20 samples submitted. Summary statistics of copper, lead, zinc, silver, and gold are presented in Table 13.1. In the opinion of the authors, an acceptable number of standard samples were submitted and the standards demonstrate an acceptable level of analytical accuracy at ALS Chemex. It should be noted that it would have been more appropriate to use standards prepared from the Wolverine deposit; however, such standards can be created as material becomes available from underground development.

Table 13.1 Statistics for Standards

	STD 1					STD 2					STD 3				
	Cu	Pb	Zn	Ag	Au	Cu	Pb	Zn	Ag	Au	Cu	Pb	Zn	Ag	Au
Average	24,163	12,377	216,500	98	26	25,630	32,083	240,100	183	4.4	37,747	28,121	216,300	218	6.4
Median	24,250	12,300	216,500	99	2.2	25,600	31,900	241,000	184	4.8	39,800	29,500	215,000	210	5.6
SD	589	336	50,000	5	1.7	346	855	50,000	9	1.3	6,094	5,767	73,000	75	3.9
Minimum	22,300	11,850	20,800	88	2	25,000	30,200	227,000	163	2.2	24,000	12,200	207,000	101	2.3
Maximum	25,100	13,300	232,000	109	11	26,400	33,400	248,000	195	6	47,200	35,300	241,000	482	21

13.2.2 BLANKS

A total of 71 blanks were submitted for analysis together with the drill core samples of the Wolverine deposit in 2005; noting the approximate insertion of one blank per twenty samples submitted. In the opinion of the authors, an acceptable number of blanks were submitted and the blanks demonstrate an acceptable level of analytical accuracy at ALS Chemex.

13.2.3 DUPLICATES

A total of 59 duplicate samples were submitted for analysis together with the drill core samples from the Wolverine deposit; noting the approximate insertion of one duplicate sample per 20 samples submitted. Correlation plots (X-Y) were constructed for copper, lead, zinc, silver, and gold with regression lines that were fit to the data points. If outliers were present, the outliers were identified by the magnitude of the sample pair variances. Correlation between sample pairs for copper, lead, zinc, and silver is high with outliers representing small percentage of the population. Gold displayed a lower correlation coefficient suggesting a higher variance. However, despite of the scatter sometimes associated with gold, the authors consider that an acceptable number of duplicate samples were submitted and that the duplicates demonstrate an acceptable level of analytical accuracy at ALS Chemex.

13.2.4 INTERNAL QUALITY CONTROL AT ALS CHEMEX

At ALS Chemex, an internal quality control/quality assurance (QA/QC) report is sent to YZC along with the original data from each geochemical submission to the laboratory. This QA/QC report includes data from the analysis of certified and in-house standards, blanks, and duplicates done in order to maintain quality control for the laboratory. The recording technician and the Chief Technician verify all results prior to data being sent to YZC.

13.2.5 ANALYTICAL CHECK LABORATORY

A total of 105 samples from five drill holes were submitted for analysis at an independent, analytical check laboratory. After selecting the samples from the database, a request was processed at ALS Chemex to ship pulp splits to ACME Analytical Laboratories Ltd. (ACME) of Vancouver, British Columbia. The pulps were shipped by truck from North Vancouver to ACME with analytical techniques pre-selected in order to parallel as close as possible the analytical techniques at ALS Chemex. The analytical techniques at ACME were chosen from a QA/QC procedural report (Cook, 2005) written for YZC. Correlation plots (X-Y) were constructed for WV05-144, WV05-158, WV05-165, WV05-168, and WV05-174 with regression lines that were fit to the data points. If outliers were present, the outliers were identified by the magnitude of the sample pair variances with R^2-values presented in Table 13.2. Correlation between sample pairs for copper, lead, zinc, and silver is high with outliers representing small percentage of the population. Gold displayed a lower correlation coefficient in some cases suggesting a higher variance. However, despite the scatter sometimes associated with gold, the authors consider that an acceptable number of duplicate drill holes were submitted and that the duplicates demonstrate an acceptable level of analytical accuracy between ALS Chemex and ACME.

Table 13.2 R^2-values for ALS Chemex vs. ACME Comparison for Metal Values

Element	Drill Hole				
	WV05-144	WV05-158	WV05-165	WV05-168	WV05-174
Cu	0.9984	0.9935	0.9982	0.9993	0.9997
Zn	0.9986	0.9863	0.9973	0.9988	0.9942
Pb	1	0.9981	0.9958	0.9988	0.9994
Ag	0.9985	0.9861	0.9984	0.9737	0.9974
Au	0.8844	0.7764	0.991	0.9859	0.976

14.0 DATA VERIFICATION

The following is extracted from the Wolverine Mineral Property Resource Estimation Independent Technical Report (2006), prepared by C. Pearson, P.Geo. and G. Giroux, P.Eng. (posted to SEDAR on 15 March 2006).

14.1 TECHNICAL REVIEW BY INDEPENDENT QUALIFIED PERSON

C. Pearson, P.Geo., an Independent Qualified Person, visited the Wolverine deposit from 12 to 13 October 2005. He also toured the facilities of ALS Chemex in North Vancouver on 27 October 2005 to review sample preparation and assaying procedures for the property drill core samples. The visit to the Wolverine deposit enabled him to examine drill site layout; drill hole collar locations; drill down hole survey data; drilling procedures; drill core handling procedures; geotechnical and geological core logging; assay sample collection and preparation methodologies; sample transportation and security from site to ALS Chemex; and drill core storage facilities at the Wolverine deposit. In addition, the underground development was critically examined looking specifically for ground conditions, geological structures, and massive sulphide zone continuity and grade. Discussions with YZC personnel at the Wolverine deposit and in the Vancouver office provided a review of data handling procedures, and time was spent in the office physically examining drill hole surveys, drill core logs, plans and sections. The ALS Chemex tour permitted a close examination of assaying and sample prep techniques.

14.2 DIAMOND DRILLING, HOLE SURVEYS, AND HOLE PLOTS

C. Pearson, P.Geo. noted that surface diamond drilling has been the main method of sampling for the Wolverine deposit. This work has taken place over an 11-year period (1995-2005). Recent underground development has accessed the deposit and traced it over a 110 m strike length, allowing sampling of the in situ massive sulphides. An extensive network of drilling access roads and drill setup sites has been developed and maintained over the 11-year period and now provides sufficient drilling platforms to define the deposit at approximately a 25 m drill hole spacing. Drill hole collar locations are marked and labelled, but most drill casing rods have been removed. All holes are cemented through the massive sulphide horizon and thus cannot be easily re-established. All drill hole collar locations are surveyed, check surveyed, and defined in relation to the property grid. All drill holes were down-hole surveyed at 30 m (minimum) to 50 m (maximum) intervals. Recent drilling (2004-2005) has used the Reflex digital down-hole survey instrument. Earlier drilling programs used the Sperry Sun down-hole surveying system in conjunction with acid-etch dip tests (hydrofluoric acid in test tubes). The random checks of both Reflex and Sperry Sun survey records in the YZC office suggest that these survey data are reliable and accurate. The survey data is entered into a drill hole plotting program that locates the holes in three dimensions and provides the requisite Northing, Easting and Elevation to locate the mineralized intersections. Drill holes appear to deviate in a regular and predictable fashion and this consistency aids in the layout of new drill hole locations. A small number of drill hole plots show abrupt azimuth changes. These isolated, yet sharp, deviations are questionable and might be explicable by magnetic variation, but do not have significant influence overall in the author's opinion. No surface drill holes have been intersected in the underground development to date, therefore no direct check can be made of calculated versus actual drill hole location. Drill core dimension is directly recorded on the individual core logs, along with core recoveries. Most drill holes have been cored to an NQ-size; however, there are a small percentage of drill holes cored with NQ2-, HQ- or BQ-sizes. Core recovery in the mineralized zone is generally good to excellent, at over 90%, but is usually much poorer in the hanging wall and footwall rocks. The Wolverine deposit geology office and core shed have restricted access and are locked when not in use, which therefore provides a reasonable degree of security for non-public drill core, samples, and data.

14.3 ON-SITE CORE HANDLING AND SAMPLE COLLECTION

C. Pearson, P.Geo. noted that drill core is transported by truck from the active drill site to the core logging facility on a daily basis by either drilling contractor or YZC's employee; noting that the core box lids are securely nailed in place. A core geological technician sorts, opens and labels the boxes; which are then logged for geotechnical data such as rock quality designation (RQD), total core recovery (TCR) and rock mass rating (RMR) before the core is disturbed by the geologist who records structural and geological data such as rock type, alteration and sulphide mineralization. Considerable effort is made to capture geotechnical data for use in mine planning. RQD is recorded as the percentage of undisturbed core pieces greater than 10 cm in length. TCR refers to 'total core recovery' as the percentage of core recovered in each 3 m run. In addition, other parameters such as joint spacing and density, joint roughness, water inflow, weathering, core angles and rock strength are recorded. The RMR calculation defined for recent drilling uses all these parameters to estimate rock strength for underground development. Mineralized drill core sample locations and lengths are clearly marked by the geologist, and the core is passed along to the core technician for splitting or sawing. One half of the sampled core is returned to the box, with the other half being tagged and bagged securely for transport to the assay lab. Before sampling, all drill core is color photographed to provide a permanent record of the undisturbed core. Drill core sample intervals are selected by the geologist to conform to established parameters such as maximum/minimum sample lengths, rock lithologies and mineralization type. The sampled core, with its defining bar-coded sample tag, is placed in rugged, 6-mil plastic bags. These bags are securely sealed with a Zap-strap and placed into pre-labelled rice sacks for transport. Geotechnical and geological logging data (including sample data) is entered into a computer program for interpretation and permanent storage. Processed core boxes are stored onsite in either storage core racks or cross-piled on pallets. There are two designated storage areas at the Wolverine deposit that contain an estimated 50,000 m of drill core. Most of this core is currently in very good to good condition and well labelled. Some 42,000 m of this core is racked, with the remaining 8,000 m on pallets. Approximately 80% the core is from Wolverine deposit drilling, with the remainder representing regional exploration work.

14.4 Sample Transportation and Security

C. Pearson, P.Geo. noted the secured plastic sample bags are placed sequentially in rice sacks, which are then sealed with metal ties and labelled for transport to ALS Chemex or other laboratory. Each drill hole is shipped to the assay lab separately, with the rice sacks colour-coded by drill hole. These sample sacks are accumulated and moved by company truck to the nearby property airstrip, where Alkan Air picks them up and flies them to Whitehorse, Yukon Territory. On arrival in Whitehorse, the samples are stored in the fenced and locked Alkan Air compound until delivered by Alkan to the scheduled Air North flight to Vancouver. Air North delivers the samples directly to ALS Chemex or other laboratory. When the samples arrive at ALS Chemex, they are promptly received and processed as described in Section 13.1. This sample transport system is well established, efficient, and reasonably secure.

14.5 Assaying Procedures

A tour was conducted of ALS Chemex in North Vancouver on 27 October 2005 by Mr. Cliff Pearson, P.Geo. in order to determine how the Wolverine deposit core samples are processed and stored. Upon arrival to the laboratory facilities, each sample is weighed and bar code checked for tracking purposes. The sample is then dried and crushed, and a 250 g sub-sample is split off and pulverized to a standard of 85% passing 75 µm. The remaining sample material is stored as crushed reject for later use if required. The 250 g pulverized sample (the pulp) is coded for tracking and continues along the system for various analyses. Gold and silver are analyzed by 30 g fire assay with gravimetric finish. Base metals (copper, lead, and zinc) are assayed by aqua-regia digestion with AAS finish. Selenium is analyzed by XRF pressed pellet. Specific gravity is determined by the pycnometer method. ALS Chemex store the analyzed pulps for later use if required by YZC.

14.6 Quality Assurance and Quality Control Procedures

YZC has in place a rigorous QA/QC program for sampling and assay control. This QA/QC program has been applied to the recent drilling campaigns of 2004-2005. Previous drill programs did use external QA/QC programs, as well as utilizing internal checks and standards currently being applied by YZC (S. Cook, 2005). In this report, it was noted that analytical quality was monitored and controlled by the insertion of at least one reference sample per analytical batch of twenty samples and the insertion of at least one duplicate and one procedure blank per analytical batch of forty samples. It may also be noted that the successive drilling campaigns have spatially overlapped and biyearly results for both zone location and grade have been consistent, yielding no internal misses in the Wolverine deposit.

As drill core sampling progresses, YZC geological staff inserts blanks, field duplicates and control standard samples in a random, yet consistent manner. In each group of 20 samples, there is one blank, one field duplicate, one prep duplicate, and one property control standard sample. Blank samples come from an essentially barren rhyolite unit found approximately 5 km north of the Wolverine deposit. Control standard samples are comprised of massive sulphide material collected from Myra Falls polymetallic VMS mine, British Columbia, which has a similar mineralogical content. In future infill definition and underground drilling programs, it will be useful to collect a series of Wolverine deposit specific standard samples for assay QA/QC; noting that the internal standards must cover a range of ore grades for both base and precious metals. It should also be noted that ALS Chemex have their own internal and complementary QA/QC program in place, running in parallel with YZC's blind QA/QC program. Ongoing data from this program is readily available to Jason Dunning, Vice President Exploration for YZC and is tracked closely in conjunction with the internal blind QA/QC program. A second check assay program was conducted at ACME; noting that cross checks revealed a strong reproducibility between assay samples from either of the two laboratories.

14.7 DATA HANDLING AND SECURITY

A computer program is used to record the drill hole data, and drill hole logs are printed as needed. These logs contain the requisite geological, geotechnical, structural and sampling data for plotting and interpretation. The property Project Geologist routinely audits the database to validate it, and if necessary, the drill core that is stored on site is re-examined to confirm the data. Assay data from ALS Chemex or other laboratory is provided directly and securely only to the Vice-President of Exploration. This data is transmitted by e-mail and an assay certificate is mailed. It can also be accessed by Jason Dunning, Vice President Exploration, directly at ALS Chemex through their secure Webtrieve program with password only access. This assay data is entered into a master database at the YZC corporate office and provided to the property Project Geologist to be integrated into the drill hole logs and site database.

14.8 UNDERGROUND VISIT

The underground workings from the Advanced Exploration Program at the Wolverine deposit were examined by C. Pearson, P.Geo. on 12 October 2005. The deposit presented itself approximately where expected from the drill holes and was traced from some 40 m at the time of his visit. Massive sulphide mineralization was physically continuous over that distance, generally over 3 m thick and disrupted only by small-scale faulting. Grade continuity could not be ascertained onsite except through visual inspection; however, the massive sulphide mineralization at the ramp face and along the walls varied in content from copper rich to zinc rich, over short distances. An underground sampling program undertaken by the advanced exploration program geologists may shed more light on metal grade distribution and continuity in the massive sulphides. Overall, the massive sulphide unit appears to be very competent. The hanging wall and footwall units were also competent in proximity to the massive sulphide mineralization, compared to drill core indicators.

14.9 Conclusions of Data Verification

The Wolverine massive sulphide deposit has been defined to date by some 154 surface drill holes in a number of drilling campaigns over eleven years. These drill programs have been competently and professionally planned and operated. Drill site access is good; the sites are well maintained and the holes well labelled. Hole collars are surveyed and tied to the property survey grid. Drill core is efficiently and professionally handled, logged, sampled, assayed, and stored. Most core is still available on site and is in good condition.

Data accumulation and handling is well done and secure. Excellent QA/QC programs are in place and an integral part of the process. ALS Chemex provides a high quality assaying service and utilize extensive QA/QC programs internally to ensure reliability. The underground development recently completed has intersected the Wolverine deposit and traced it for some 110 m. This work has demonstrated physical continuity to the zone over that distance. Grade continuity and distribution should be examined in detail to test these parameters against the geological model developed from the surface drilling programs.

15.0 ADJACENT PROPERTIES

15.1 ADJACENT PROPERTIES

The following is extracted from the Wolverine Mineral Property Resource Estimation Independent Technical Report (2006), prepared by C. Pearson, P.Geo. and G. Giroux, P.Eng. (posted to SEDAR on 15 March 2006).

15.2 INTRODUCTION

The following summary information pertaining to the Kudz Ze Kayah, Ice, and Kona deposits is taken from fully disclosed information from various references. The authors have no previous or current working relationship with Pacific Ridge and hold no common shares, options, or warrants of Teck-Cominco, YZC, and/or Pacific Ridge. It should be noted that the authors have not been able to verify the information contained in this section and that the information is not necessarily indicative of the mineralization of the Wolverine deposit that is the subject of this technical report. It should be noted that the mineralization associated with the Kudz Ze Kayah, Ice, and/or Kona deposits is not part of the mineralizing system that formed the Wolverine deposit. All resources and reserves are from public sources, as pertains to operation of each of the deposits discussed below.

15.3 KUDZ ZE KAYAH AND GP4F

The Kudz Ze Kayah Deposit (previously known as ABM) is located within the Pelly Mountains of the southeast Yukon approximately 200 kilometres northwest of Watson Lake, Yukon. Sulphide mineralization underlies Quartz Mining claims Tag 17, 18, 22, and 23 (Schultz, 1996; Schultz, 2001). In 1993, the first mineral claims were staked after Cominco personnel identified a quality geophysical drill target, mineralized float, anomalous soil geochemistry, and geology permissive for VMS mineralization. The catalyst to the effort was the discovery of a well-mineralized magnetite-bearing banded massive sulphide float specimen by Cominco prospector/geologist A.B. (Bruce) Mawer, which returned values of 0.84% Cu, 8.44% Pb, 9.49% Zn, 597 g/t Ag, and 4.80 g/t Au. The target was drill tested in April 1994 and intersected 22.5 m of mineralization in two zones having a combined grade of 0.5% Cu, 2.8% Pb, 10.0% Zn, 278 g/t Ag, and 2.9 g/t Au. In total 52 diamond drill holes totalling 8,485 m were completed in 1994 along with ground control surveys, ground and regional airborne geophysical surveys, claim staking, detailed mapping in the vicinity of the deposit, regional and detailed exploration geochemistry and baseline environmental sampling (Schultz, 2001). An additional 133 diamond drill holes totalling 16,178 m were completed in 1995 at the Kudz Ze Kayah deposit and on nearby regional targets including GP4F. A total of 17 diamond drill holes totalling

3,566 m were completed in 1997; seven holes tested the down dip/plunge and fault offset margins of the Kudz Ze Kayah deposit; two holes tested regional targets in the area; and, nine holes were completed in the Fault Creek area. Drilling has identified 13,720,000 tonnes grading 6.0% Zn, 1.6% Pb, 0.90% Cu, 139.2 g/t Ag, and 1.38 g/t Au in the Kudz Ze Kayah Deposit and 1,500,000 tonnes grading 6.4% Zn, 3.1% Pb, 0.1% Cu, 90.0 g/t Ag, and 2.0 g/t Au in the GP4F Deposit (Schultz, 2001).

15.4 ICE DEPOSIT

The Ice property hosts VMS mineralization where primary copper minerals are found in massive sulphide horizons and stockwork zones as chalcopyrite with pyrite and occasional bornite. Secondary copper minerals occur above or peripheral to the primary mineralization and were formed by either in situ oxidation or precipitation following leaching and groundwater transport (Moore et al., 2003). A total of 121 diamond drill holes (10,584 m) have been completed on the Ice property, 87 of which were drilled in 1997. All but six of the drill holes were drilled in a 600 m x 400 m area. The outlying drill holes tested soil geochemical anomalies. These holes failed to intersect significant mineralization and the anomalies were attributed to glacial or groundwater dispersion. The best assay results were obtained near the centre of the drill area in a 350 m long, approximately 50 m wide zone consisting predominantly of primary massive sulphide mineralization. Copper intersections within this zone include: 5.20% over 20.56 m (drill hole IC96-34), 8.56% over 5.92 m (ID97-11), 3.57% over 28.55 m (ID97-13), 4.09% over 7.55 m (ID97-20), 4.31% over 19.75 m (ID97-36), 3.47% over 25.09 m (IC97-46), 3.20% over 28.51 m (IC97-57), 3.79% over 5.69 m (IC97-70) and 4.18% over 8.97 m (IC97-84). This high-grade core is surrounded by a broad halo containing thick intersections of lower-grade mineralization. Copper grades in the halo typically range from 1.5% to 3% in massive sulphide, 0.5% to 1.2% in stockwork sulphide and 0.2% to 1.5% in secondary mineralization. The massive sulphide mineralization usually contains potentially significant gold (0.2 to 0.8 g/t), silver (2 to 20 g/t), cobalt (0.02 to 0.08%) and zinc (0.2 to 0.6%). Multi-element analyses indicate that no significant detrimental elements are present in the mineralization (Moore et al., 2003). Diamond drilling at the Ice property has defined an Indicated mineral resource of 4,562,000 tonnes grading 1.48% Cu, including about 3.4 Mt of near-surface mineralization at the same grade. The indicated resource is overlain by a significant amount of additional near surface oxide copper mineralization for which there is insufficient data to estimate a resource (Moore et al., 2003).

15.5 KONA DEPOSIT (FYRE LAKE PROPERTY)

Pacific Ridge Exploration Ltd.'s 100% owned Fyre Lake property is located in southeast Yukon, and is host to a "Besshi-type" copper-cobalt-gold VMS deposit. The property is composed of 169 contiguous Quartz Mining claims covering 35 km^2. The Fyre Lake property covers over nine kilometres of favourable host rocks with geochemical and geophysical targets indicative of VMS mineralization. During 1996 and 1997, the company focused its attention to delineating one target, the Kona deposit, through completion of 23,200 m of drilling in 115 holes (Blanchflower et al., 1997). The Kona Deposit consists of two parallel northwest trending zones of copper-cobalt-gold massive sulphide mineralization found in horizons with mineralized thicknesses varying from 8 m to 40 m over a length of 1,500 m and a width of 250 m. A NI 43-101 compliant report prepared by Minorex Consulting Ltd. in August 2002, the Kona VMS deposit is calculated to contain 15.4 Mt within which deposit 8.2 Mt grades 2.1% Cu, 0.11% Co and 0.73 g/t Au, utilizing a 1.0% Cu cutoff. Metallurgical studies prepared by Lakefield Research Limited in June 1997 indicate metal recoveries of 90% for copper and 70% for gold and cobalt. Cobalt is associated with pyrite and can be efficiently recovered as a separate product from copper-gold concentrates. Using prices of US$1.00 for copper, US$365/oz of gold and US$10/lb copper, an independently prepared scoping study by Kilborn Engineering Pacific Ltd. in August 1997, suggests economic viability for a 20 Mt reserve, with an open pit grade of 2.0% Cu, 0.7 g/t Au and 0.12% Co, and an underground grade of 3.0% Cu, 1.0 g/t Au and 0.12% Co. With a presently defined deposit of eight million tonnes, exploration potential is well demonstrated for the discovery of additional mineralization through drilling within a 20 Mt envelope. Significant exploration potential remains, over and above determining the ultimate size of the Kona massive sulphide deposit. A 4 km long magnetic anomaly located northeast of Kona and a three kilometre-long magnetic anomaly lying west of Kona are larger and more intense as compared to the magnetic feature reflecting the Kona mineralization. These anomalies represent priority drill targets for discovery of additional massive sulphide deposits.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 INTRODUCTION

This section was written by Mr. J. Fox, P.Eng.

The following sections describe the metallurgical test work campaigns carried out in the past along with the most recent test work done during the last half of 2006. Analysis of the various metallurgical tests including the most recent ones was carried out and the conclusions reached were then used to develop the design criteria and process equipment for the Optimized Feasibility Study.

16.2 METALLURGICAL TEST PROGRAMS

Prior to 2006 there were three major metallurgical campaigns conducted, each one for the purpose of developing a process flowsheet to selectively produce copper, lead and zinc concentrates. In all campaigns including the 2006 one, variables were adjusted in rougher kinetic and open circuit cleaner flotation tests. Locked cycle tests were then applied to the most promising flowsheets, grinds and reagent combinations. It is the locked cycle results that were almost entirely focused upon in the analysis of the metallurgical campaigns.

In 1997, Amtel carried out test work on a composite of diamond drill core obtained during the 1997 exploration program. The diamond drill core came from the Wolverine and Lynx zones of the ore body. A total of 38 flotation tests were conducted along with three locked cycle tests. The optimal results from one of the locked cycle tests used a primary grind of 80% passing 80 µm and regrinds of 80% passing 24, 26, and 25 µm for copper, lead, and zinc respectively. Predicted metallurgical results of recovery/grade (percent) for copper, lead, and zinc were 75/21.8, 40/23.2, and 87/51.0, respectively.

In early 2000, PRA carried out metallurgical testwork using diamond drill core increments obtained from the Wolverine, Lynx, and Saddle zones of the Wolverine deposit. Three composite samples were produced for this series of testwork. Bond ball mill work index carried out on the three composites gave results of 9.5, 9.8, and 9.5.

Three locked cycle tests with primary grinds, P_{80} of 73, 57, and 53 µm were tested. It was found that with the 57 µm primary grind there was an improvement in the lead grade of the final cleaner concentrate The regrind size was not indicated in these tests however a time of regrind was recorded and it appeared to be similar to the size found in the earlier Amtel work. Metallurgical results from the PRA 2000 testwork showed recovery/grade percent for copper, lead, and zinc of 79.9/21.3, 46.0/22.3, and 89.4/54.2 respectively.

In 2004 and 2005, SGS Lakefield carried out testwork to evaluate Heavy Media Separation for removing waste dilution from the ore. It was demonstrated that upgrading by Heavy Media Separation worked well with waste rejected with little metal loss.

In 2005, following the Heavy Media Separation, SGS Lakefield carried out a metallurgical test campaign. Composite samples for this work consisted of diluted and undiluted Wolverine and Lynx mineralization. The flotation testwork was based on the lines of investigation followed by Amtel with no regard given to the PRA 2000 testwork. SGS however concluded that a pre-flotation circuit is required prior to copper rougher flotation, to enhance lead flotation recovery.

It was found that the Wolverine and the Lynx zones had similar mineral assemblages but the Lynx zone had slightly finer grained. Bond ball mill work index of the Lynx mineralization was 8.0 kWh/t whereas the Wolverine mineralization was 10.6 kWh/t. The process flow sheet, grind and reagent scheme developed by Amtel in 1997 was used for the basis of this campaign and was found suitable for both Wolverine and Lynx mineralization.

The SGS metallurgical program was carried out on six composites, consisting of individual zones as well as the two zones combined. A projected metallurgical balance for a locked cycle test using an overall composite from both mineralized zones was provided by SGS. The primary grind was 80% passing 75 µm and the regrinds of the rougher concentrates were 21, 16 and 18 µm for copper, lead, and zinc respectively. Recoveries and grades realized in this test were 76.7/23.8, 47.1/42.0 and 79.5/54.7 for copper, lead, and zinc respectively.

In 2006, PRA carried out another metallurgical test program. The main purpose of this program was to resolve discrepancies in results obtained at different laboratories in the past and then optimize the ore processing procedures for maximum recovery of metals.

Two composites were prepared for the testwork. One was from bulk samples taken from the Lynx ore zone underground after the decline was constructed. This sample (No. 1) had a slightly higher grade than the average of the ore body. The second composite (No. 3) was prepared from a variety of drill hole intercepts left over from the SGS testwork of 2005. This composite had a lower grade than the average of the ore body.

The SGS 2005 test programme determined an optimized grind in the P_{80} passing 60 µm range rather than 75 µm. However, even with this finer grind, the results obtained by SGS in 2005 could not be matched by PRA. Testwork then continued in an effort to duplicate results obtained in the PRA 2000 test campaign.

Numerous flotation tests were carried out to evaluate procedures followed by three locked cycle tests on composite No. 1 and one locked cycle on composite No. 3. The last locked cycle test done was that on composite No. 3, the one with lower than average head grade. Resulting recovery and grade percentages for the test on composite No. 3 were 85.1/.18.0 for copper, 24.0/5.9 for lead, and 91.2/54.3 for zinc. It is of interest to note the high recovery of zinc even though the head grade sample was lower than average.

The PRA testwork demonstrated that a finer primary grind in the range of 80% passing 48 µm to 55 µm was required to obtain overall improved recovery. This may be due to the bulk of the work being carried out on the finer grained Lynx zone. It was also determined that the pre-floatation circuit ahead of copper flotation was not required.

As a result of the various tests carried out by PRA in 2006, and confirming the earlier testwork done in 2000, a more conservative selection of recovery and grades was selected for the Optimized Feasibility Study: zinc recovery to be 89% with a grade of 54%, lead recovery to be 46% with a concentrate grade of 22% and copper recovery to be 80% with a concentrate grade of 21%. In addition, 23% of the gold and 13% of the silver will be recovered in the lead concentrate and a further 33% of the gold and 60% of the silver will be recovered in the copper concentrate.

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

This section was extracted from the Wolverine Mineral Property Resource Estimation Independent Technical Report (2006), prepared by C. Pearson, P.Geo. and G. Giroux, P.Eng. (posted to SEDAR on 15 March 2006).

17.1 BACKGROUND

The following section summarises the resource estimation completed in November and December 2005 by Gary Giroux on the Wolverine deposit. Orebody modelling was performed by Jason Dunning, Vice-President Exploration for YZC in conjunction with independent consultant Cliff Pearson, noting that Cliff Pearson, P.Geo., also performed an audit of YZC databases. Gary H. Giroux, MASc., P.Eng performed Geostatistics, variography, and block model estimation. Gilles Dessureau, project geologist for YZC, who interpreted the geological framework and the nature of the sulphide mineralization, did geological interpretation of the Wolverine deposit.

The mineral resource estimate prepared by Gary H. Giroux are of Measured, Indicated, and Inferred categories. All computer modeling of the Wolverine deposit was performed using Surpac Minex 3D software with data processing, variography, and geostatistics done in Techbase 3D software.

The geological framework of the Wolverine deposit that formed the 3D model was examined with emphasis placed on the review and distribution of lithology, mineralization types, and structural features in the immediate deposit area. This review of drill section and plan maps, drill core logs, and selected drill core boxes/trays resulted in a refined geological model that provided the basis for geological framework. The general alignment of the semi-massive to massive sulphide mineralization was found to be reasonable in terms of grade distribution of the model. Although there were not any internal misses or negative drill holes, which did not pierce the semi-massive to massive sulphide mineralization, the model limits along the flanks of the deposit were stopped at either the halfway point between a positive or a negative drill hole or at a distance of 25 m away from a positive drill hole.

17.2 DATABASE VERIFICATION

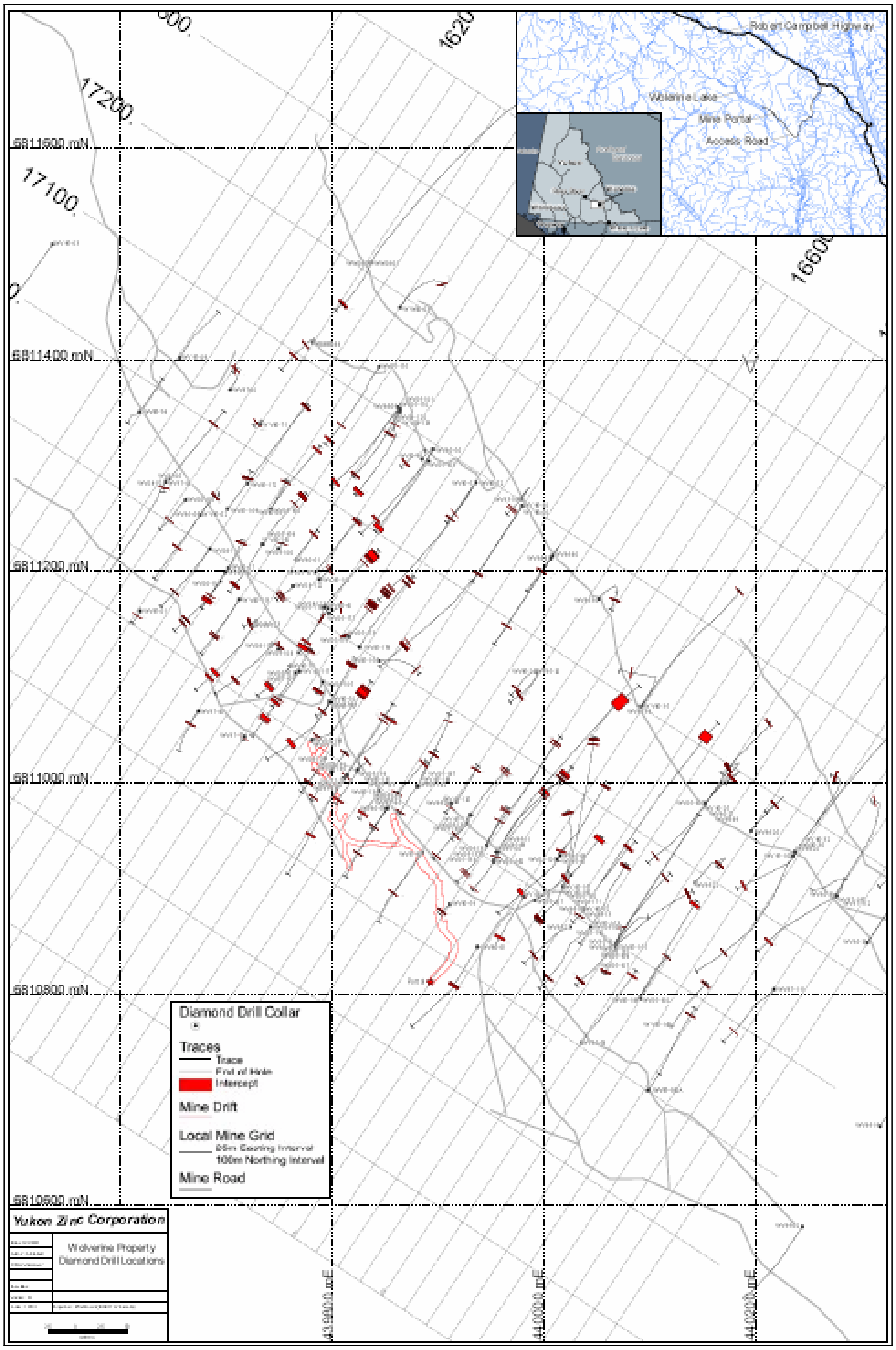
On 24 October 2005, YZC supplied Gary Giroux with data files for the Wolverine deposit in the Yukon Territory. The data files contained 154 drill holes with downhole surveys (either Sperry Sun or Reflex EZ-Shot) located on a local grid established in 1995; however, it is important to note that only 154 of the drill holes pierced the massive sulphide mineralization, which was used in the new mineral resource estimation. It is also important to note that there were no internal misses within the massive sulphide unit. Underhill or Procon has surveyed all drill holes between 1995 and 2000. In October 2005, YZC contracted Underhill to conduct an independent validation of the drill collar locations surveyed by Procon in 2005. No significant deviances were noted.

YZC provided the topography data for the Wolverine deposit, which allowed the creation of a 3D shell to represent local topography. Figure 17.1 shows all 159 drill hole collars and their traces with contours and the local grid; noting that only 154 drill holes out of 159 in the database pierced the massive sulphide mineralization of the Wolverine deposit. It was these 154 drill holes that were used in the creation of the 3D solid model for the mineralization.

It should be noted that there is only one database administrator for YZC, Mr. Andrew Caldwell who maintains the integrity of the data on the YZC server in the Vancouver office, noting that data was transmitted from the field by Mr. Gilles Dessureau, Project Geologist, YZC, for the Wolverine deposit to Mr. Caldwell for filing on the network server. No original data can be changed on the server unless authorized by Mr. Caldwell; meaning that any requirement for data requires copies to be distributed to relevant parties with no change to the integrity of the original data. For the purposes of the Surpac 3D model, a complete copy was provided to Jason Dunning, Vice President Exploration, YZC who was the only administrator for the work relating to the mineral resource, Surpac 3D model. The database and block model were backed up on the network server on a daily basis. A copy of the mineral resource database was provided to Mr. Gary Giroux for variography and geostatistics, but this manipulation did not affect the original database.

Principal validation and subsequent interpretation of the geological framework was done on grid north facing vertical drill sections. Locally, plan views were used for cross checks and control purposes. The Wolverine deposit outline was interpreted initially on 12.5 m section spacing, which was then cut into a 6.25 m section spacing and revised systematically. The tight section spacing allowed for the more accurate modeling of upper zones of semi-massive to massive sulphide mineralization in the immediate hanging wall of the main body of sulphide mineralization known as the Wolverine, Saddle, and Lynx zones. Overall, four (4) solid models were created: (1) Main Zone (Wolverine, Saddle, Lynx), (2) Upper Lynx 1, (3) Upper Lynx 2, and (4) Upper Wolverine. Each of these four zones were coded individually and then meshed into separate entities for block modeling purposes.

Figure 17.1 Drill Hole Distribution Plot for Wolverine Deposit



17.3 DATA ANALYSIS

17.3.1 ASSAY DATA

YZC provided survey, assay, geotechnical and geological data for 154 diamond drill holes having 1,008 down hole surveys and 5,378 assay intervals (all in a data base). The assay data contained some “<” values and a few “>” values that were adjusted using the rule of one half of the detection limit for the “<” values and the detection limit for the “>” values. Statistics for copper, zinc, silver, gold, and SG are shown in Appendix II and III of the Pearson and Giroux’s resource estimation report. The massive sulphide zone was interpreted from drill hole logs and cross sections by YZC geologists. The intersections for each hole that penetrated this zone were provided. Based on this interpretation, all assays were tagged as MZONE if within the mineralized zone and WASTE if outside. The statistics for each variable within each zone are provided in Table 17.1.

Table 17.1 Statistics for Assays within Mineralized Zone and Waste

	Au (g/t)	Ag (g/t)	Zn %	Pb %	Cu%
Mineralized Zone					
Number	997	997	997	997	997
Mean	1.64	345.4	11.83	1.48	1.24
SD	1.88	388.1	7.22	1.43	2
Minimum	0.003	0.5	0.01	0.01	0.001
Maximum	16.63	5067	50.9	9.26	23.1
CV	1.14	1.12	0.61	0.97	1.61
Waste					
Number	4234	4234	4234	4234	4234
Mean	0.075	11.78	0.33	0.04	0.09
SD	0.327	62.68	1.4	0.19	0.5
Minimum	0.001	0.1	0.01	0.01	0.001
Maximum	6.86	1999	34.5	6.53	17.3
CV	4.36	5.32	4.22	4.56	5.46

Each of the five major economic metals was evaluated within both the mineralized zone and waste to determine if capping was required and if so at what level. Lognormal cumulative probability plots were used to examine the distribution of grades for each variable. Plots for gold, silver, zinc, lead, and copper were produced for both the mineralized zone and waste and are included in Pearson and Giroux’s resource estimation report.

17.3.2 MINERALIZED ZONE

For each variable within the mineralized zone, multiple overlapping grade populations were identified, with the upper most in each case, determined to represent erratic high grades. An effective capping procedure is to use the threshold of two standard deviations above the mean of the second highest population as a capping level. The capping level and number of samples capped for each variable is summarized in Table 17.2.

Table 17.2 Summary of Capping Levels used in Mineralized Zone and Waste

Variable	Cap Level	Number Capped
Mineralized Zone		
Zn	35%	3
Ag	2,177 g/t	6
Au	12.1 g/t	3
Cu	12.40%	5
Pb	7.40%	3
Waste		
Zn	8.20%	25
Ag	6.37 g/t	7
Au	3.1 g/t	12
Cu	2.60%	22
Pb	2	7

17.3.3 WASTE

A similar approach was used outside the mineralized zone in material considered waste at this time. For variables in waste, the upper two populations were considered erratic and capped. The threshold used was two standard deviations above the mean of population 3. Table 17.2 presented a summary of capping levels and number of samples adjusted.

17.3.4 COMPOSITES

For the identified massive sulphide mineralized zone uniform 2 m down hole composites were produced that honoured the mineralized zone boundaries. Small intervals at the boundaries were combined with the adjacent sample if less than 1 m. As a result, the composites file represented a uniform support of 2 ± 1 m samples. Table 17.3 shows the statistics for each of the five variables within both, the mineralized zone and waste.

Table 17.3 Statistics for 2 m Composites within Mineralized zone and Waste

	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	Cu (%)
Mineralized Zone					
Number	475	475	475	475	475
Mean	1.65	340	11.93	1.53	1.11
SD	1.65	310.6	5.56	1.18	1.24
Minimum	0.024	2.43	0.004	0.01	0.001
Maximum	10.28	2177	32.95	7.01	8.83
CV	1	0.91	0.47	0.77	1.12
Waste					
Number	3568	3568	3568	3568	3568
Mean	0.052	7.67	0.2	0.03	0.048
SD	0.164	33.35	0.57	0.07	0.177
Minimum	0.001	0.1	0.01	0.01	0.001
Maximum	3.1	621.9	6.45	1.21	2.49
CV	3.16	4.34	2.88	2.45	3.7

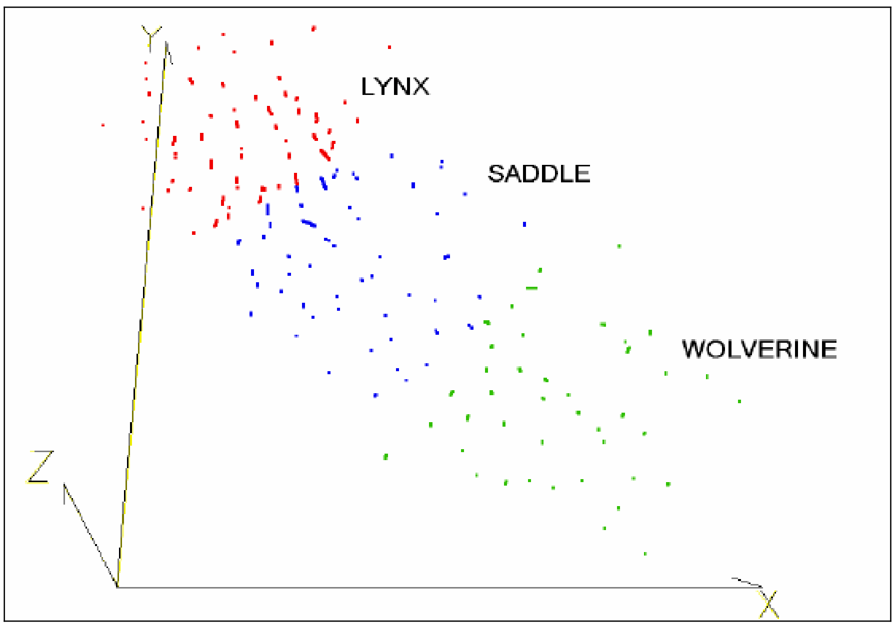
A correlation coefficient matrix (see Table 17.4) for both the mineralized zone and waste shows excellent correlation between gold-silver, gold-lead, and silver-lead. Reasonable correlation is shown between gold-zinc, silver-zinc, and lead-zinc.

Table 17.4 Correlation Coefficient Matrix for 2 m Composites

	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	Cu (%)
Mineralized Zone					
Au (g/t)	1	-	-	-	-
Ag (g/t)	0.8692	1	-	-	-
Zn (%)	0.2771	0.2808	1	-	-
Pb (%)	0.6906	0.7671	0.3695	1	-
Cu (%)	-0.1661	-0.105	0.1443	-0.2445	1
Waste Zone					
Au (g/t)	1	-	-	-	-
Ag (g/t)	0.7554	1	-	-	-
Zn (%)	0.4742	0.4622	1	-	-
Pb (%)	0.5	0.5486	0.6671	1	-
Cu (%)	0.253	0.2961	0.363	0.2067	1

The 2 m composites were subdivided into three zones; Lynx, Saddle, and Wolverine, based on their locations; which is presented in Figure 17.2.

Figure 17.2 Isometric Plot showing 2 m Composites Colour Coded by Zone



Statistics for grades within each of these zones (see Table 17.5) shows very similar zinc and lead averages for Lynx and Wolverine with elevated gold and silver in Wolverine and higher copper in Lynx. The Saddle zone has lower zinc, silver, gold, and lead than either the Lynx or Wolverine but higher copper values. A series of cumulative distribution plots for each variable showing the three separate zones (Pearson and Giroux's resource estimation report) shows similar patterns.

Table 17.5 Statistics for 2 m Composites within Mineralized Zone sorted by Zone

	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	Cu (%)
Lynx Zone					
Number	195	195	195	195	195
Mean	1.45	336.6	12.93	1.66	1.13
SD	1.34	267	5.68	1.12	1.27
Minimum	0.024	2.43	0.046	0.01	0.001
Maximum	7.46	2031.6	32.95	5.41	8.83
CV	0.92	0.79	0.44	0.68	1.13
Saddle Zone					
Number	140	140	140	140	140
Mean	1.34	273.1	9.75	1.24	1.31
SD	1.48	318.7	5.36	1.18	1.45
Minimum	0.03	5	0.04	0.02	0.004
Maximum	8.53	2172	25.1	7.01	7.73
CV	1.11	1.17	0.55	0.95	1.1
Wolverine Zone					
Number	140	140	140	140	140
Mean	2.25	411.8	12.71	1.66	0.88
SD	2.02	343.6	5	1.23	0.88
Minimum	0.07	16.7	2.22	0.08	0.005
Maximum	10.28	1973.8	25.94	6.81	5.37
CV	0.9	0.83	0.39	0.74	0.99

17.4 VARIOGRAPHY

The statistics and the cumulative frequency plots for each zone show minor differences but the same overall form and tenor, especially for the two most significant economic minerals zinc and silver. This combined with the much smaller data sets (if subdivided into three domains) led to combining the three zones for variography. Pairwise relative semivariograms were produced for each variable along the three principal directions of information, namely: along strike at azimuth 130 to 145, down dip at azimuth. 50 to 55 dip -30 to -40 and across dip at azimuth 230 to 235 dip -50 to -55. In all cases, spherical nested models were fit to the data. Geometric anisotropy was demonstrated in all cases. Table 17.6 presents the parameters for each of the variables.

In general nugget to sill ratios are good ranging from a low of 8% for silver to a high of 22% in lead within the mineralized zone and a low of 20% in silver to a high of 31% in gold within waste. Low nugget to sill ratios indicates low sampling variability.

Table 17.6 Summary of Semivariogram Parameters

Variable	Direction	Nugget Effect C_0	Short Structure C_1	Long Structure C_2	Short Rang A_1 (m)	Long Range A_2 (m)
Mineralized Zone						
Zn	Az. 140 Dip -20	0.05	0.23	0.09	60	160
	Az. 50 Dip -40	0.05	0.23	0.09	45	60
	Az. 230 Dip -50	0.05	0.23	0.09	10	50
Ag	Az. 145 Dip 0	0.05	0.4	0.15	60	180
	Az. 55 Dip -40	0.05	0.4	0.15	30	40
	Az. 235 Dip -50	0.05	0.4	0.15	10	20
Au	Az. 145 Dip -10	0.1	0.2	0.26	50	160
	Az. 55 Dip -40	0.1	0.2	0.26	20	130
	Az. 235 Dip -50	0.1	0.2	0.26	5	15
Pb	Az. 145 Dip 0	0.15	0.33	0.21	50	160
	Az. 55 Dip -40	0.15	0.33	0.21	25	100
	Az. 235 Dip -50	0.15	0.33	0.21	10	40
Cu	Az. 140 Dip -30	0.1	0.5	0.2	50	150
	Az. 50 Dip -40	0.1	0.5	0.2	25	50
	Az. 230 Dip -50	1.1	0.5	0.2	5	20
Waste Zone						
Zn	Omni Directional	0.2	0.4	0.2	10	60
Ag	Omni Directional	0.2	0.4	0.4	15	60
Au	Omni Directional	0.2	0.2	0.25	22	50
Pb	Omni Directional	0.1	0.12	0.14	15	45
Cu	Omni Directional	0.2	0.2	0.38	10	50

17.5 BLOCK MODEL

A block model composed of blocks 3 m x 3 m x 2 m in dimension was superimposed on the geologic 3D solid, as presented in Figure 17.3. The block model was rotated along the strike of the mineralized zone.

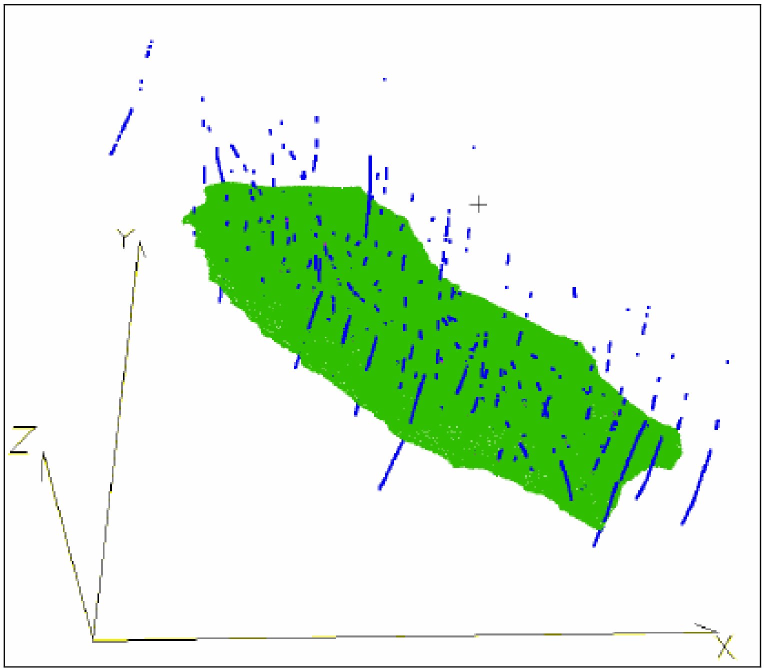
Model Origin

Mid point of Top Lower–left Block

X coordinates:	439379.479	Column Size:	3 m 345 columns
Y coordinates:	6810991.650	Row Size:	3 m 250 rows
Top Z coordinates:	1560.0	Level Size:	2 m 290 levels

X axis rotated 30° clockwise

Figure 17.3 Showing Blocks within the Mineralized Zone and Drill Hole Composites



17.6 BULK DENSITY

A total of 2,100 specific gravity determinations were made for the Wolverine deposit in both mineralized and waste samples. ALS Chemex made measurements on half core samples submitted for assay. The procedure used at ALS Chemex is as follows. The rock or core section (up to 6 kg) weighed dry on a balance for method OA-GRA08 or is covered in a paraffin wax coat in the case of OA-GRA08a and is weighed on a balance. The sample is then weighed while it is suspended in water. From the data, the specific gravity is calculated.

OA-GRA08: $$\text{Specific Gravity} = \frac{\text{Weight of Sample (g)}}{\text{Weight in Air (g)} - \text{Weight in Water (g)}}$$ or

OA-GRA08a: $$SG = \frac{A}{B\text{-}C - [(B\text{-}A)/Dwax]}$$

Where A =Weight of sample in air
B =Weight of waxed sample in air
C =Weight of waxed sample suspended in water
D =Density of wax

Each specific gravity value was assigned a location and interpolated into the block model using inverse distance squared. Table 17.7 shows a comparison of statistics from the specific gravity assays to estimated blocks.

Table 17.7 Statistics for Specific Gravity in both Mineralized and Waste Zones

	Mineralized Zone		Waste Zone	
	SG Assays	SG Blocks	SG Assays	SG Blocks
Number	956	127,583	1,151	109,798
Mean	3.97	3.96	2.85	2.87
SD	0.56	0.33	0.22	0.15
Minimum	2.26	2.82	2.08	2.29
Maximum	5	4.73	4.71	4.23
CV	0.14	0.08	0.08	0.05

17.7 Block Model Interpretation

Ordinary kriging was used to interpolate grades into the block model. For any block with a proportion of mineralized solid greater than zero, kriging was undertaken. The kriging process was completed in four passes using expanding search ellipses. For the first pass, kriging was attempted if a minimum of four composites were found within a search ellipse equal to one quarter the semivariogram range for the variable being estimated. A further restriction of limiting the (across structure) direction to a maximum of 5 m was placed on this pass to ensure at least two drill holes were found. For blocks not estimated, a second pass was completed with search ellipse dimensions equal to half the semivariogram range. For blocks still not estimated, a third pass was completed using the entire range of the semivariogram. Finally, a fourth pass using roughly two times the semivariogram range for zinc, was used to fill in un-estimated blocks within the solid. In all passes, the minimum required composites to estimate a block was four while the maximum used was 16. If more than 16 composites were found within the search ellipse the closest 16 to the block centroid were used. Table 17.8 shows the parameters used and number of blocks estimated during each pass.

The block model was evaluated using plans and cross sections to check validity. A comparison of average grades is presented in Table 17.9 for 2 m composites compared to block values.

Table 17.8 Search Parameters for Kriging

Pass	Number Estimated	Major Axis	Semi. Maj. Axis	Minor Axis	Major Axis Dist. (m)	Semi. Major Axis Dist. (m)	Minor Axis Dist. (m)
Zinc							
1	16,006	Az. 140 Dip -20	Az. 50 Dip -40	Az. 230 Dip -50	40	15	5
2	67,078	Az. 140 Dip -20	Az. 50 Dip -40	Az. 230 Dip -50	80	30	25
3	60,017	Az. 140 Dip -20	Az. 50 Dip -40	Az. 230 Dip -50	160	60	50
4	28,534	Az. 140 Dip -20	Az. 50 Dip -40	Az. 230 Dip -50	320	120	40
Silver							
1	14,347	Az. 145 Dip 0	Az. 55 Dip -40	Az. 235 Dip -50	45	10	5
2	65,492	Az. 145 Dip 0	Az. 55 Dip -40	Az. 235 Dip -50	90	20	10
3	70	Az. 145 Dip 0	Az. 55 Dip -40	Az. 235 Dip -50	180	40	20
4	22,665	Az. 145 Dip 0	Az. 55 Dip -40	Az. 235 Dip -50	320	120	40
Copper							
1	11,155	Az. 140 Dip -30	Az. 50 Dip -40	Az. 230 Dip -50	37.5	12.5	5
2	47,810	Az. 140 Dip -30	Az. 50 Dip -40	Az. 230 Dip -50	75	25	10
3	65,581	Az. 140 Dip -30	Az. 50 Dip -40	Az. 230 Dip -50	150	50	20
4	48,081	Az. 140 Dip -30	Az. 50 Dip -40	Az. 230 Dip-50	230	120	40
Gold							
1	41,650	Az. 145 Dip -10	Az. 55 Dip -40	Az. 235 Dip -50	40	35.5	5
2	70,963	Az. 145 Dip -10	Az. 55 Dip -40	Az. 235 Dip -50	80	65	7.5
3	54,360	Az. 145 Dip -10	Az. 55 Dip -40	Az. 235 Dip -50	160	130	15
4	5,154	Az. 145 Dip -10	Az. 55 Dip -50	Az. 235 Dip -50	320	120	40
Lead							
1	34,734	Az. 145 Dip 0	Az. 55 Dip -40	Az. 235 Dip -50	40	32.5	5
2	70,963	Az. 145 Dip 0	Az. 55 Dip -40	Az. 235 Dip -50	80	65	7.5
3	54,360	Az. 145 Dip 0	Az. 55 Dip -40	Az. 235 Dip -50	160	130	15
4	5,154	Az. 145 Dip -40	Az. 55 Dip -40	Az. 235 Dip -50	320	120	40

Table 17.9 Statistics for 2 m Composites within Mineralized Zone compared with Kriged Block Values

	Comp. Zn (%)	Block Zn (%)	Comp. Ag (g/t)	Block Ag (g/t)	Comp. Aug (g/t)	Block Aug (g/t)	Comp. Cu%	Block Cu%	Comp. Pb %	Block Pb %
Number	475	171,635	475	171,635	475	171,635	475	171,635	475	171,635
Mean	11.93	12.06	340	374.7	1.65	1.73	1.11	1.19	1.53	1.65
SD	5.56	3.09	310.6	197.3	1.65	1.05	1.24	0.8	0.01	0.68
Minimum	0.04	1.39	2.43	13.82	0.024	0.08	0.001	0.005	0.01	0.09
Maximum	32.95	29.66	2177	1882.7	10.28	8.85	8.83	7.41	7.01	6.13
CV	0.47	0.26	0.91	0.53	1.00	0.61	1.12	0.67	0.77	0.42

17.8 CLASSIFICATION

Based on the study herein reported, delineated mineralization of the Wolverine deposit is classified as a resource according to the following definition from NI 43-101:

"In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by

CIM Council on 20 August 2000, as those definitions may be amended from time to time by the Canadian Institute of Mining, Metallurgy, and Petroleum."

"A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge."

The terms Measured, Indicated, and Inferred are defined in NI 43-101 as follows:

"A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity."

"An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed."

"An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes."

The geologic continuity of the Wolverine deposit is well established through diamond drilling and underground development. YZC geologists have yet to miss the zone when planning in fill drill holes. The grade continuity can be quantified by semivariogram analysis. The classification into measured/indicated/inferred was based on the search ellipse that was used to estimate grades into the blocks.

- Measured – Both Zn and Ag were estimated in pass 1 using a search ellipse with dimensions equal to ¼ the semivariogram range
- Indicated – Zn or Ag were estimated in pass 2 using a search ellipse with dimensions equal to ½ the semivariogram ranges
- Inferred A – Zn or Ag estimated in pass 3 using a search ellipse with dimensions equal to the full range of the semivariogram
- Inferred B – Both Zn and Ag estimated in pass 4 with a search ellipse with dimensions equal to twice the range of the zinc semivariogram

17.9 MINERAL RESERVE ESTIMATES

A. Polk, P.Eng., wrote this section.

17.9.1 STATEMENT OF ORE RESERVES

Table 17.10 provides the estimated ore reserves for the Wolverine Mine.

Table 17.10 Ore Reserves Estimate

Category	Tonnes	Zn %	Cu %	Pb %	Ag g/t	Au g/t
Proven	564,000	10.31	0.96	1.24	246.9	1.24
Probable	4,588,000	9.59	0.91	1.26	286.2	1.37
Total	5,152,000	9.66	0.91	1.26	281.8	1.36

To determine the proportion of the resource which would prove to be economically viable, a preliminary estimate of the value of the mineralised rock in the form of a net smelter calculation was performed. This calculation utilized the results from metallurgical test work, reasonable expectations for the value of metals during the life of mine, and estimated treatment charges from third party smelters. The following metal prices (US$) and resultant NSR calculation[1] has been utilized:

Zn $0.60/lb, Cu $1.10/lb, Pb $0.30/lb, Au $400/oz, Ag $7/oz

NSR (US$) = (9.894 x Zn%) + (13.07 x Cu%) + (1.297 x Pb%) + (0.1435 x Ag g/t) + (6.657 x Au g/t)

Using this calculation, and the average grade of the resource, it is determined that the average value of the mineralised resource is US$200/t, or approximately Cdn$230. Previous studies performed upon the Wolverine deposit suggest that when the NSR of diluted material fell below Cdn$80 then the material would likely be uneconomic.

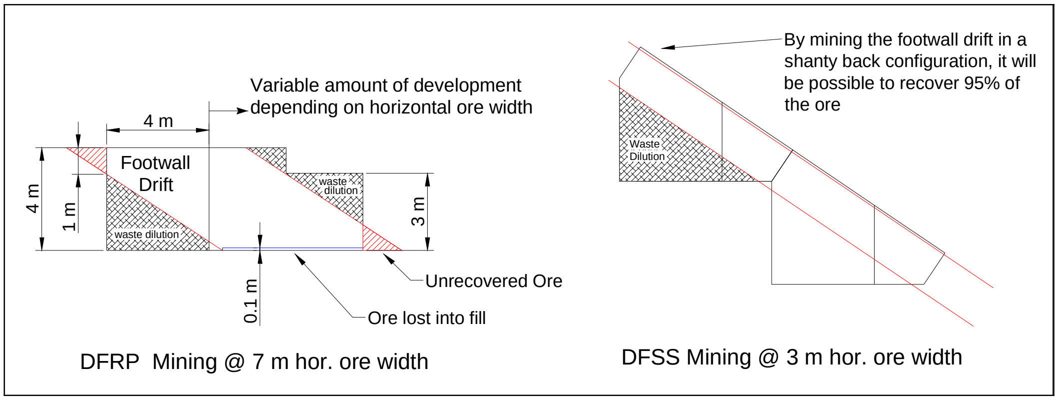
17.9.2 RESERVE ESTIMATION METHODOLOGY

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 cutoff criteria proportions). Similarly, the ore will be mined to the hangingwall until 1 m of ore is left in the face. Figure 17.4 shows a sectional view of this.

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, also shown in Figure 17.4.

[1] The NSR calculations are found in Pearson and Giroux's report.

Figure 17.4 Dilution and Recovery Factor Assumptions



Where mining will occur above other mined and filled voids, a 0.1 m layer of broken ore is assumed lost into fill.

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

Table 17.11 presents a summary of the dilution and recovery criteria used in the creation of the mining reserve.

Table 17.11 Summary of Dilution and Recovery Criteria by Ore Thickness

Ore Hor. Thickness (m)	Dilution % (waste/ore)	Recovery (%)	Mining Method	Backfill Dilution (%)
0.5**	774.5	95.0	DFSS	1.0
1	350.3	95.0	DFSS	1.0
2	138.2	95.0	DFSS	1.0
3	67.5	95.0	DFSS	1.0
4	51.6	90.1	DFSS	1.0
5	40.6	91.7	DFRP	2.0
6	33.4	92.8	DFRP	2.0
7	28.4	93.5	DFRP	2.0
8	24.7	94.1	DFPS	2.5
9	21.9	94.5	DFPS	2.5
10	19.6	94.9	DFPS	2.5
11	17.8	95.1	DFPS	2.5
12	16.2	95.4	DFPS	2.5
13	15.0	95.6	DFPS	2.5
14	13.9	95.8	DFPS	2.5
15	12.9	95.9	DFPS	2.5
16	12.1	96.0	DFPS	2.5
17	11.4	96.1	DFPS	2.5
18	10.7	96.3	DFPS	2.5
19	10.2	96.3	DFPS	2.5
20	9.6	96.4	DFPS	2.5
21	9.2	96.5	DFPS	2.5
22	8.8	96.6	DFPS	2.5
23	8.4	96.6	DFPS	2.5
24	8.0	96.7	DFPS	2.5
25	7.7	96.7	DFPS	2.5
26	7.4	96.8	DFPS	2.5
27	7.1	96.8	DFPS	2.5
28	6.9	96.9	DFPS	2.5

Note: ** 0.5 m wide ore is deemed to be uneconomic

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 Au, the background grade derived from the resource estimate. Backfill dilution was added without grade.

17.9.3 *Creation of Stope Shapes*

Plan view slices of the geological model were produced in 10 m increments over the entire deposit. These were then utilized to create bounding stope strings under the following criteria:

- strings were created to bound areas of the thinnest (< 4 m), thin (> 4 m and < 7 m) and thick (> 7 m) mineralization
- areas with typical mineralized horizontal ore width < 1 were not included in the shapes
- additional strings were created to bound areas of resource that would contain stope accesses or decline excavations
- strings were created at the planned starting elevation of stopes for the appropriate zone, either the Wolverine or Lynx side of the decline
- the strings were created sufficiently large to be able to surround any mineralization from the planned starting elevation to the planned completion elevation for any stope, i.e., 20 m.

These strings where then extruded vertically upwards by 20 m to create three dimensional digital wireframes that outlined all sufficiently thick mineralized material within the geological block model and which reflected typical mineralized widths, probable mining method, and proximity to the stope accesses and/or decline. Each wireframe was given a unique name based on the starting elevation, probable mining method, and approximate sectional location. An example is: 1110_0000_BSS, which starts on elevation 1110, around section 0000E, is proximal to the ramp (B = Barrier Pillar), and which is expected to be mined by the BFSS mining method.

All of these wireframes were analysed with respect to the geological block model in Surpac™ and a complete listing of all contained resources tabulated in each, including volume, tonnage, grade, and resource category.

17.10 Application of Dilution, Recovery, and Cutoff

Once all of the contained mineralised resources were estimated for each of the stope blocks, the approximate length of each stope was measured from the plan view strings. This length and the volume of contained resources from Surpac™ were used to estimate the average horizontal mineralized width within any single stope block so that the dilution and recovery criteria presented in Table 17.11 could be applied.

A cutoff of any diluted resources with a resultant diluted grade of < 5 % Zn was applied in the calculation of the mineable reserve. Most of this material was associated with excessively diluted very thin ore in three locations:

- at the extremities of the ore zone
- around the decline, in the thinnest ore of the Saddle Zone
- in the thinnest hangingwall lenses.

The formula for NSR from previous study work (see Hatch's feasibility study, May 2006) was used to estimate that typical ROM material would have a value of approximately Cdn$190 after smelter charges. It was estimated that after dilution, any resources with a NSR value of less than half of the average value (Cdn$95) should not be included within the mining reserve. Therefore, a cutoff of 5% Zn was chosen.

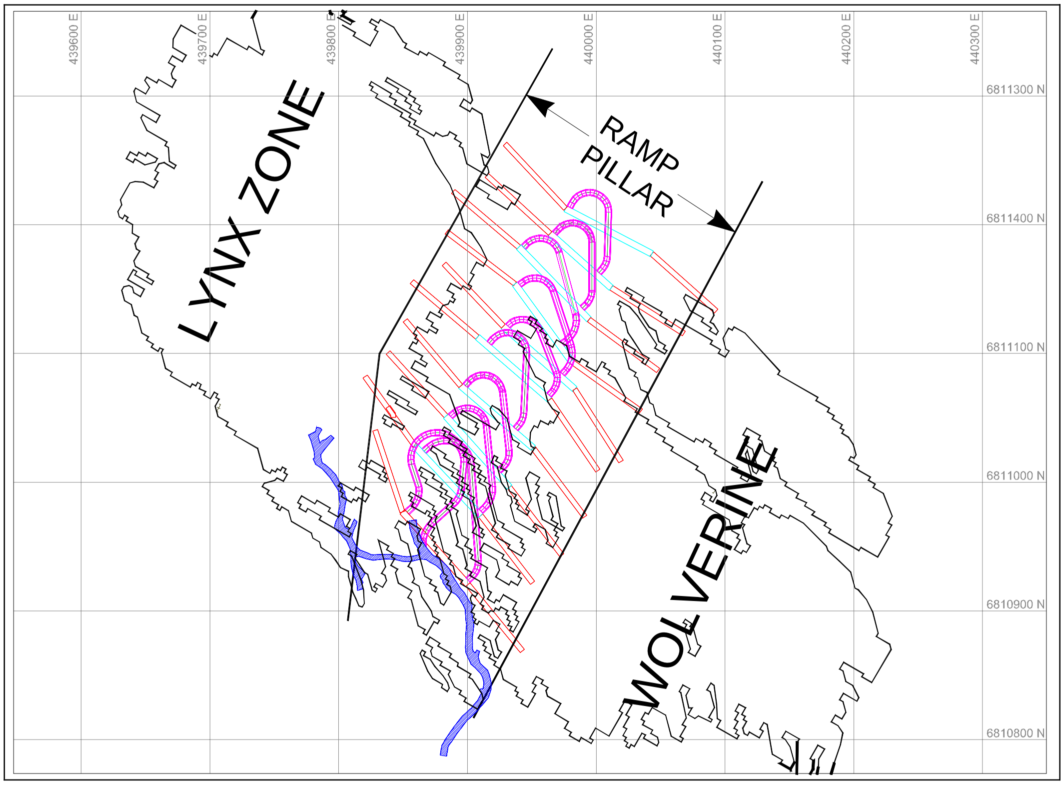
Inferred or Uncategorized resources were also excluded from the material which was ultimately tabulated as a reserve. However, some low grade, Inferred or Uncategorized resources will need to be mined due to the nature of the mining method, and the complex geometry of the boundaries between resource definitions. As a result, a factor of additional stoping metres, 3.5% has been added to the development schedule and treated as waste. An associated increase in costs as well as equipment and labour requirements has been applied. To estimate this, all production from stopes is roughly converted to equivalent metres of development advance and 3.5% of the stoping metres are added to the development schedule.

Lastly, an overall recovery was applied to reflect stope blocks that were in close proximity to either the decline, or to the stope accesses. It was assumed that a recovery of 80% of the resources within these areas would be possible; hence 20% of the Measured and Indicated resources around the decline and stope accesses are effectively left as permanent pillars.

No other permanent pillars are designed for the mine plan, however, it should be noted that a considerable amount of Inferred and Uncategorized resources are included within the overall boundaries of the stoping blocks. It is expected that some of this material will need to be mined, and that it will also be possible to leave behind some of this material as pillars should the resources prove to be uneconomic. In the unlikely event that all of the Inferred and Uncategorized resources were eventually found to be uneconomic it would be impossible to leave all of this material behind as pillars and more than 3.5% of all rounds within the stoping plan would need to be handled as waste. The boundary between the Measured and Indicated resources and other resources is a very complex shape and is shown in Figure 17.5 in the context of the ramp location as well as the stope accesses. Particularly in the area around the stope accesses, where the boundary is convoluted, it is very difficult to determine if additional waste will need to be mined, or if in fact the inferred resources will prove to be mineable ore. This is highlighted as both a risk and opportunity within the mining plan. Snowden notes that only Measured and Indicated Resources have been converted to reserves. In areas where the decline passes through

Inferred resources, the material has been treated as waste and is assumed to be hauled to surface for disposal.

Figure 17.5 Boundary of Measured and Indicated Resources



In discussions with YZC personnel and Dr. Khosrow Aref (geotechnical consultant), it was deemed appropriate to assume that no permanent pillars would be required in the stoping areas due to the placement of tight cemented paste fill. However, Snowden recommends that numerical modelling is undertaken to confirm this assumption as soon as possible and before the start of the final mine design.

18.0 OTHER RELEVANT DATA AND INFORMATION

Wardrop is not aware of any additional information or explanations beyond those contained in this report required to make this document understandable and not misleading.

19.0 INTERPRETATIONS AND CONCLUSIONS

19.1 PROJECT ECONOMICS

This section was written by Wardrop. A total of four cases were examined as part of the economic evaluation process. A summary of each case is shown below:

Case 1 – Model with 3 Year Moving Average Prices

Case 2 – Model with Combined 2 and 3 Year Moving Average Prices

Case 3 – Model with 2 Year Moving Average Prices

Case 4 – Scenario Model with Current Prices for Comparison Purposes Only.

Table 19.1 shows NPV at 8% discount and payback period among other information for each of the four cases.

Table 19.2 shows the metal prices used for each case. Table 19.3 shows a summary of payable metal in relation to the recovered metal in the concentrates.

Table 19.1 Economic Analysis Summary, (Cdn$)

	Pre-Tax IRR (%)	Pre-Tax NPV 8% ($ million)	Payback Period	3 Year Full Production Cumulative Pre-tax Cash Flow	Average Annual Cash Flow For First 3 Years Pre-tax Full Production
Case 1	18.9	104.8	3.9	172.5	57.50
Case 2	22.6	134.3	3.2	204.9	61.47
Case 3	26.3	184.2	3.0	217.7	65.31
Case 4	56.8	571.7	1.5	439.3	13.18

Table 19.2 Metal Prices, (US$)

	Zinc (US$/lb)	Copper (US$/lb)	Lead (US$/lb)	Silver (US$/oz)	Gold (US$/oz)
Case 1	0.87	1.85	0.48	8.54	486.85
Case 2	0.87 and 1.07	1.85	0.48 and 0.52	8.54 and 9.48	486.85 and 526.65
Case 3	1.07	1.85	0.52	9.48	526.50
Case 4	1.84	1.85	0.76	12.69	626.01

Table 19.3 Payable Metal by Concentrate

	Units	Metal Recovered to Concentrates (LOM)	Payable Metal (LOM)
Zinc	(t)	461,364	376,979
Silver	(oz)	38,597,810	33,179,153
Copper	(t)	42,261	35,403
Gold	(oz)	150,949	119,311
Lead	(t)	45,892	25,661

It can be seen in the tables above that the project is economically viable even in Case 1, which uses the lowest metal prices. Case 1 has an internal rate of return (IRR) of 18.9%.

When the current metal prices were evaluated in Case 4, the IRR was 56.8% and the payback is very short at 1.5 years.

Wardrop's conclusion is that the project is not only economically viable but is technically viable as well.

19.2 GEOLOGY

This section was extracted from the Wolverine Mineral Property Resource Estimation Independent Technical report (2006), prepared by C. Pearson, P.Geo. and G. Giroux, P.Eng. (posted on SEDAR on 15 March 2006).

- Six, successive, successful, surface diamond drilling programs in 1995, 1996, 1997, 2000, 2004, and 2005 have expanded the deposit to the current Measured and Indicated mineral resource of 4.46 Mt grading 12.14% Zn, 1.16% Cu, 1.58% Pb, 354.8 g/t Ag , and 1.70 g/t Au (US$80) and an Inferred mineral resource 1.69 Mt containing 12.16% Zn, 385.1 g/t Ag, 1.23% Cu, 1.71 g/t Au and 1.74% Pb (US$80).
- The six successive, successful surface diamond-drilling programs also defined four zones of semi-massive to massive sulphide mineralization of consistent and potentially mineable dimension. The four zones are known as Main Zone, Upper Lynx1, Upper Lynx 2, and Upper Wolverine.
- There are 159 drill holes completed near the Wolverine deposit; however, only 154 drill holes successfully pierced at least one of the four zones of semi-massive to massive sulphide mineralization.
- Results from all the diamond drilling programs have refined the interpretation of the size and shape of the deposit, while demonstrating remarkable consistency of the calculated tones and grade of the mineral resource.
- Physical continuity of the sulphide sheet is demonstrated by the absence of internal barren or negative drill holes; however, the apparent minimal structural disruption of stratigraphy is also demonstrated by the absence of internal barren or negative drill holes, but, locally there is definite fault disruption of the semi-massive to massive sulphide mineralization that will require additional work to fully understand the full nature of the structural disruption.

19.3 MINING

This section was written by A. Polk, P.Eng. Underground operations at the Wolverine mine have been planned to achieve a daily production rate of 1850 tonnes of diluted ore per day. It has been assumed that 8% of days during a year will be down due to maintenance or unforeseen circumstances. An effective average production rate of 1,700 t/d will result over any year long period.

A small dynamic stockpile of ore will be used near the portal and crushing circuit on surface so that the daily differences in production can be managed. In the event that considerable process downtime occurs, a larger ore stockpile will be available so that mining can continue unabated.

The mining reserve of 5.2 Mt will provide for an operating mine life of 9.5 years, including 1.5 years of pre-production development activities. A portal at elevation 1345 and short section of decline was developed as part of an underground test mining program in 2005. This decline will be extended downwards to follow the shallow dip (~35°) of the orebody. The decline will be centrally located within the deposit and in the thinner ore of the Saddle Zone.

Drift and fill will be utilized as the mining method. The hangingwall and footwall rock conditions are very poor, and the method has been chosen for its selectivity and for the ability to manage the size of open spans in stope excavations. Three variations of the mining method are used so that stope productivity can be achieved and also so that the rock conditions in the hangingwall are managed with minimum dilution.

Paste back-fill will be the primary filling material for the mine, again in response to the poor ground conditions in the hangingwall of the orebody and the need to manage open spans. Loose waste rock from development headings and loose waste floated from the dense media separation (DMS) plant will also be placed in stope voids.

Mining for both development and production purposes will be undertaken with diesel powered mobile equipment. Drilling operations will be performed primarily with electric hydraulic drill jumbos mounted on diesel carriers. Rock-bolting operations will be performed with electric hydraulic units, capable of bolting from remote locations and with a combination of resin re-bar, split-sets, and wire mesh screen.

The very poor condition of the waste rock, which surrounds the orebody, has precipitated the need to install an efficient wet mix shotcreting system. A mixing unit with auger will be purchased and installed on surface. This will be used to mix bulk bagged premix shotcrete with water, which will then be dispensed into a transmixer unit for hauling the batch of shotcrete underground, where it will be sprayed onto the rock by a remotely operated spraying unit.

Load haul dump units (LHDs) will be used to load both waste and ore from development and stope headings into diesel haulage trucks for haulage either to surface or to stopes.

Two 50 tonne capacity haulage trucks will eventually be used as the primary haulage units. These will run continuously in the decline, taking both ore and waste to surface. An additional, smaller capacity 30 tonne haulage truck with an ejector style bucket will also be used. This unit will haul waste material into stope headings for dumping of loose fill or DMS reject material.

A maximum of 185 m^3/s of ventilating air will be required based on a projected equipment list for the mine. Two ventilation raises and the main decline will be used to create a ventilating system. An exhaust fan at the top of the exhaust raise will be the primary air mover for the mine. Direct-fired propane heaters will be installed at the top of the decline and the smaller secondary raise so that air temperatures are maintained above freezing throughout the seasons.

In-stope escapeways will also be developed during the course of mining to provide emergency egress, either to the stope above or below, should access be cutoff for whatever reason.

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

The underground mine design for the Wolverine project has been developed with information and data from several sources, including expert opinion in the areas of underground mine design, mine geotechnical design, ventilation systems design, and paste backfill systems design, as well as experience gained during an underground exploration and test mining program. During the process of the study, several stages of peer review and independent review were undertaken to confirm the validity of the design, parameters, and assumptions. The selected equipment and mining methods are consistent with the production rates required to meet the target feed rates to the ore processing facilities and over the life of the operation.

19.4 METALLURGY AND PROCESS

This section was written by Wardrop.

Metallurgical testwork programs have been carried out in a number of laboratories over the last decade. These programs were carried out on a number of samples and composites representing all zones of the deposit.

Most recent metallurgical testwork was conducted on different sets of samples in order to confirm the flowsheet procedure selected and to define metallurgical parameters, which are required for the design of the processing plant. The testwork included DMS tests and flotation tests. The outcomes of these tests can be summarised as follows:

Dense Media Separation Testwork showed that a clear separation between the dilution waste rock and valuable ore components was obtained with minimal losses of < 1% of each of the metals to the discard fraction at the media specific gravity of 2.85 g/cm^3.

Mineralogical examinations have indicated that the main minerals of interest are relatively fine-grained and will require a relatively fine primary grind followed by the regrinding of the rougher concentrates in order to achieve saleable grade metal concentrates. There are indications that the Lynx zone material could be finer-grained than ores from the other Wolverine areas and this is where most recent testwork samples have been taken.

The most recent testwork on fine grained ores showed that a primary grind P_{80} of 53 µm, and a secondary regrind P_{80} of about 20 µm, will be required to achieve a suitable recovery and concentrate grade for each product. The process equipment selected in the optimized feasibility study will achieve these grinds at the designed throughput rate.

Locked cycle test results indicated that a sequential flotation separation into a copper concentrate, lead concentrate and zinc concentrate gave acceptable results except for the small amount of lead concentrate produced.

A copper concentrate of 21% copper with a recovery of 80% will be attained, also recovering 33% of the gold and 60% of the silver.

The lead concentrate will have a grade of 22% lead with a recovery of 46%. The gold recovery into the lead concentrate will be 23% together with 13% of the silver.

The zinc concentrate will grade 54% zinc at a recovery of 89%.

The processing facilities have been selected based on proven technologies. Equipment sizing has been based on testwork and experience gained from other operating mines and equipment vendors' recommendations. The process plant will be capable of producing the designed concentrates from the mine at the indicated ore grades.

20.0 RECOMMENDATIONS

20.1 GEOLOGY

The following is extracted from the Wolverine Mineral Property Resource Estimation Independent Technical Report (2006), prepared by C. Pearson, P.Geo. and G. Giroux, P.Eng. (posted to SEDAR 15 March 2006).

- There should be consistent and ongoing site exploration; which is vital to the operation. The host stratigraphy of the Wolverine deposit remains very prospective, as do the areas along strike. Yukon Zinc has developed an excellent understanding of the local and regional geology and is well positioned for ongoing exploration both at site and regionally.
- Additional infill, definition drilling will be required to upgrade the Inferred mineral resources category to Measured and/or Indicated mineral resource categories. However, the majority of this infill, definition drilling can be accomplished from surface drill setups if absolutely necessary; noting that the timing of this infill, definition drilling can be flexible as underground development will not provide access in the near future. It should also be noted that the upgrading of Inferred mineral resources to either Measured or Indicated mineral resources should occur during the mine capital cost payback period.
- There should be timely and consistent ore infill, definition drilling programs; which is vital for mine planning and efficient production. Ore infill, definition drilling must be done well ahead of production development; noting that development headings dedicated to infill, definition drilling may be required.
- There should be detailed face, back and wall sampling of the semi-massive to massive sulphide mineralization every shift; noting that such a program undertaken in the 110 m of development in the massive sulphide mineralization advanced YZC's understanding of metal grade zonation. Timely, yet detailed mapping and sampling of on-going development will be vital to efficient mine production and grade reconciliation efforts.
- The extensive, high quality assay database available for the Wolverine deposit provides the framework for a detailed study of metal grade zonation throughout the deposit. This work should be ongoing and will provide useful information ahead of production.
- A structural study should be undertaken in order to more accurately model the faulting and/or folding associated with the Wolverine deposit; noting that an accurate structural model can result in either a positive, predictive mine plan that adds tonnage and removes mine development, as well as lowering development costs or a negative mine plan that removes tonnage and hinders mine development by raising development costs.

- A study on the distribution of the barren to weakly mineralized argillite units that are intercalated with the semi-massive to massive sulphide mineralization should be undertaken in 2006. This study should also determine the possible effect the argillite units have on dilution of grade potential of the Wolverine deposit. Unfortunately there is insufficient data because of the current drill density to precisely correlate individual argillite beds within any degree of certainty because of small scale structural disruption. The argillite units are effectively waste-rock that will be rejected by DMS; however, the argillite units have currently been incorporated into the composites as internal dilution.

20.2 MINING

This section was written by A. Polk, P.Eng. Before implementation of the mining plan for the Wolverine Project, there are three aspects which are required to be investigated as a critical portion of the underground mine design. These are as follows:

- Numerical modelling is required to assess the long term rock movements and stress increases around the infrastructure headings to ensure that major infrastructure features have been supported adequately and/or that more appropriate locations could be chosen to reduce future risk.
- Numerical modelling is required to assess the recoverability of sill pillars between stopes in both the Lynx and Wolverine mining areas. Longer term stress variation over the life of stopes must be assessed with respect to a variety of stoping geometries at varying horizontal ore width. This work is critical to assess the full recoverability of all sills in all geometries.
- Definition diamond drilling is required in the area of the planned infrastructure to ensure that appropriate and realistic designs are created for final construction. This is particularly important because it is planned for much of the access decline to follow the footwall contact of the ore. Sufficient orebody knowledge is critical for this to be accomplished. Estimates for the amount of diamond drilling required and the costs for the same have been included as part of the mining plan as well as the capitalised expenses before pre-production development begins.

Once the infrastructure design and mining plan has been solidified, the following additional items warrant further detailed investigation to optimize the mining plan:

- A review of the mining fleet should be performed with consideration of both maintenance and production requirements over the LOM. An optimised fleet size, both in terms of unit numbers as well as unit dimensions and capabilities, needs to be developed with input from suppliers, maintenance experts, and mining experts. Snowden believes that a maintenance and repair contract (MARC) should be considered in discussions with equipment suppliers given the remote location of the mine site.

- The paste fill system, both on surface in the thickening, storage, and pumping system, and underground in the distribution network and operating parameters, needs to be designed in a more thorough fashion. Various operating scenarios including such variables as the typical pour size and pour rate need to be investigated to optimize the paste system. The ability to inject a portion of crushed waste into the paste pumping system on surface would be very valuable for the operation and this should also be investigated.
- Rheology for the paste plant feed must be tested.

20.3 PROJECT OPPORTUNITIES

This section was written by Wardrop and YZC.

- The process has been designed for a primary grind of P_{80} = 53 µm. It appears that at least some of the deposit can be processed at a coarser grind of P_{80} = 75 µm and still achieve optimum metallurgical results while decreasing the grinding costs.
- The flotation cells selected in the feasibility are tank cells; however, column flotation cells in the regrind cleaners are potentially suitable and will result in lower capital and operational costs.
- Negotiations with a trucking firm for hauling concentrate and backhauling other consumables should result in decreased overall transportation costs.
- There is an opportunity to negotiate lower smelter costs for zinc than assumed in the optimized feasibility study. This is based on information that current smelter contracts for 2007 are lower than YZC has assumed.
- There is a possibility to recover saleable selenium from the tailings water treatment plant. Revenues of about $150,000 could be generated.
- An additional lead cleaner stage could increase the lead concentrate grade, though at a negative impact on recovery. However, this has not been tested yet in a laboratory.

21.0 REFERENCES

Documents used in preparation of this technical report are as follows:

Colpron, M. et al., 2001. Ancient Pacific Margin: An update on stratigraphic comparison of potential volcanogenic massive sulphide-hosting successions of Yukon-Tanana Terrane, northern British Columbia and Yukon. In Yukon Exploration and Geology, 2000, Exploration and Geological Services Division, Yukon, Indian and Northern Affairs, pp. 97-110.

Cook, S., 2005. Quality Assurance and Quality Control Protocols for the Collection, Preparation, and Analysis of Core Samples in Mineral Exploration. Unpublished Internal Yukon Zinc Corporation Report 63 p.

Dietrich, T., P.Eng., Dietrich Consulting Ltd., February 2006. Wolverine Project Feasibility, Underground Electrical.

Dietrich, T., P.Eng., Dietrich Consulting Ltd., February 2006. Wolverine Project Feasibility, Underground Communications.

Government of Yukon, 2005a. Draft Technical Guidelines: Guidelines and Conditions for Mine Site Reclamation and Closure for New Mines - Technical Issues.

Government of Yukon, 2005b. Draft Financial Assurance Guidelines for Mine Site Reclamation and Closure Security Instruments.

Government of Yukon, 2006a. Yukon Mine Site Reclamation and Closure Policy.

Government of Yukon, 2006b. Screening Report for Yukon Zinc Corporation's Wolverine Project. Development Assessment Branch.

Hairfield, C., Mine Ventilation Services, Inc., 22 November 2006. Ventilation Design Memo.

Harris Exploration Services, 8 November 2006. Mineralogical Examination of Test Products from Project 0605706.

Hatch Associates Ltd., 2006. Wolverine Project, Yukon Territory, Canada: 43-101 Independent Technical Report. Submitted to Yukon Zinc Corporation.

Hatch Associates Ltd., May 2006. Wolverine Project Detailed Feasibility Study.

Khosrow, Aref, PhD. P.Eng., 2 February 2006. Geomechanical Mine Design Assessment of the Wolverine Deposit.

Kwok, D. and Klein, B., 20 November 2006. DMS Testing on Yukon Zinc Samples, Institute of Mining Engineering, University of British Columbia.

Mine Systems Design, Inc., 19 April 2006. Paste Backfill – Testwork Report Wolverine Project.

Murphy, D.C. and Piercey, S.J., 2000. Syn-Mineralization Faults and their Re-activation, Finlayson Lake Massive Sulphide District, Yukon-Tanana Terrane, Southeastern Yukon. In Yukon Exploration and Geology, 1999, Exploration and Geological Services Division, Yukon, Indian and Northern Affairs, pp. 55-78.

Murphy, D.C. et al., in preparation. Mid-Paleozoic to Early-Mesozoic Tectonostratigraphic Evolution of Yukon-Tanana and Slide Mountain Terranes and Affiliated Overlap Assemblages, Finlayson Lake Massive Sulphide District, Southeastern Yukon. Ancient Pacific Margin Contribution.

Olafson, R. and Dessureau, G., Yukon Zinc Corporation, 17 November 2006. Personal Communication.

Pearson, C. and Giroux, G., 2006. Wolverine Mineral Property Resource Estimation – Independent Technical Report, p. 48.

Piercey, S.J. et al., 2003. Neodymium Isotope Geochemistry of Felsic Volcanic and Intrusive Rocks from the Yukon-Tanana Terrane in the Finlayson Lake Region Yukon, Canada. In, Canadian Journal of Earth Sciences, Vol. 40, 2003, pp.77-97.

Piercey, S.J., et al., 2001. Geological Characteristics of High-level Sub-volcanic Porphyritic Intrusions Associated with the Wolverine Zn-Pb-Cu Volcanic-hosted Massive Sulphide Deposit, Finlayson Lake District, Yukon, Canada. In Yukon Exploration and Geology, 2000, Exploration and Geological Services Division, Yukon, Indian and Northern Affairs, pp. 335-346.

Piercey, S.J., et al., 2002. Geochemistry and Tectonic Significance of Alkalic Mafic Magmatism in the Yukon-Tanana Terrane, Finlayson Lake region Yukon. In Canadian Journal of Earth Sciences, Vol. 39, 2002, pp.1729-1744.

Piercey, S.J., et al., in preparation. Paleozoic Magmatism and Crustal Recycling along the Ancient Pacific Margin of North America, Northern Cordillera. Ancient Pacific Margin Contribution.

Process Research Associates Ltd., 14 June 2000. Flotation Study on Samples from the Wolverine Deposit.

Snowden, January 2007. Mining Portion of Underground Feasibility Study.

Tucker, T.L., 1999. Summary Report on the Geological, Geochemical, Geophysical, Geotechnical, Metallurgical, Environmental, and Diamond Surveys on the

Wolverine Property, Yukon Territory, Canada. Submitted in partial fulfillment of a geological report required by the Vancouver Stock Exchange in accordance with Appendix 19B guidelines.

Tucker, T.L., et al., 1997. Wolverine Massive Sulphide Project, Yukon Territory, Canada—Westmin Resources Limited and Atna Resources Ltd, Vancouver, B.C. In Yukon Exploration and Geology, 1996, Exploration and Geological Services Division, Yukon, Indian and Northern Affairs, pp. 53-55.

Wallace, K., Mine Ventilation Services, Inc., 10 February 2006. Ventilation Design Memo – Revision 3.

Wardrop Engineering, January 2007. Optimized Feasibility Study

Yukon Government, 2006. Screening Report for Yukon Zinc Corporation's Wolverine Project. Development Assessment Branch.

Yukon Zinc Corporation, 2005. Wolverine Project, Yukon Territory: Environmental Impact Report. Submitted to the Development Assessment Branch.

Yukon Zinc Corporation, 2006a. Wolverine Project Reclamation and Closure Plant for Final Closure. Submitted to Yukon Energy, Mines and Resources and Yukon Water Board.

Yukon Zinc Corporation, 2006b. Wolverine Project EAR Response to Public and Regulatory Reviews. Submitted to the Development Assessment Branch.

Yukon Zinc Corporation, 2007. Wolverine Project Revised Documentation in Support of Water Use Application Q204-065.

22.0 CERTIFICATE OF QUALIFIED PERSONS

All Certificates are included overleaf.

CERTIFICATE OF QUALIFIED PERSON

Rick Alexander, P.Eng.
800 – 555 West Hastings St.
Vancouver, BC
rick.alexanderr@wardrop.com

I, Rick Alexander, P.Eng. of Vancouver, BC, do hereby certify that as an author of this Technical Report on the Wolverine Property, Finlayson District, Yukon, and dated 28 February 2007; hereby make the following statements:

- I am a Consultant with Wardrop Engineering Inc. with a business address at 800 – 555 West Hastings St., Vancouver, BC.
- I am a graduate of the University of Alberta, Edmonton, Alberta, Canada, (B.Sc., Mechanical Engineering, 1985).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (Licence #17101).
- I have practiced my profession for +20 years since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- I am responsible for supervising the preparation of Sections 1.9, 1.11, 1.12, 1.13, 2.2, 18, 19.1, & 23.10 of this technical report titled "Technical Report on the Wolverine Property, Finlayson District, Yukon," dated 28 February 2007. I visited the Property on November 20 and 21, 2006.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 28th day of February 2007 at Vancouver, British Columbia.

Original document, Revision 01 signed and sealed by Rick Alexander

Signature

WARDROP

CERTIFICATE OF QUALIFIED PERSON

Ken Deter, P.Eng.
800 – 555 West Hastings St.
Vancouver, BC
ken.deter@wardrop.com

I, Ken Deter, P.Eng. of Vancouver, BC, do hereby certify that as an author of this Technical Report on the Wolverine Property, Finlayson District, Yukon, and dated February 2007; hereby make the following statements:

- I am a Manager of Metallurgy with Wardrop Engineering Inc. with a business address at 800 – 555 West Hastings St., Vancouver, BC.
- I am a graduate of the University of Missouri, Rolla, MO, USA, (B.Sc., Metallurgical Engineering, 1970).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (Licence #148287).
- I have practiced my profession for +30 years since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- I am responsible for the preparation of Section 19.4 of this technical report titled "Technical Report on the Wolverine Property, Finlayson District, Yukon," dated 28 February 2007. I have not visited the Property.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 28th day of February 2007 at Vancouver, British Columbia.

Original document, Revision 01 signed and sealed by Ken Deter

____
Signature

CERTIFICATE OF DAVID TYSON, M.Sc., R.P. Bio. (BC)

400-286 Broadway
Winnipeg, MN
david.tyson@wardrop.com

I, David Tyson, RP Bio., of Winnipeg, MN, do hereby certify that as an author of this Technical Report on the Wolverine Property, Finlayson District, Yukon, and dated 28 February 2007; hereby make the following statements:

- I am a Senior Environmental Project Manager in the mining Division of Wardrop Engineering Inc. with a business address at 400 – 386 Broadway, Winnipeg, MN, R3C 4M8.
- I am a graduate of the University of Manitoba, Winnipeg, MN, (B.Sc. (Zoology), 1990, and M.Sc. (Zoology), 1996).
- I am a member in good standing of the College of Applied Biology, British Columbia, as a Registered Professional Biologist (Licence #1013).
- I have practiced my profession in the environmental sciences continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, I am not a qualified person for the purpose of NI 43-101.
- I am responsible for the preparation of Sections 1.7 and 23.5 of this technical report titled "Technical Report on the Wolverine Property, Finlayson District, Yukon," dated 28 February 2007. I have not visited the Property.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- I consent to the filing of this Technical Report with any stock exchange or other regulatory authority and any publication by them, including electronic publication I the public company files on their web sites accessible by the public, of this Technical Report.

Signed and dated this 28th day of February 2007 at Vancouver, British Columbia.

Original document, Revision 01 signed and sealed by David Tyson

Signature

CERTIFICATE OF QUALIFICATION

I, John R.W. Fox, of 1677 Deep Cove Road, North Vancouver, British Columbia, V7G 1S4 do hereby certify that:

1) I am a consulting metallurgical engineer with an office at 302-304 W. Cordova St., Vancouver, British Columbia V6B 1E8.

2) I am a graduate of the University of Leeds (UK) in 1971 with B.Sc. in Applied Minerals Sciences.

3) I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.

4) I have practised my profession continuously since 1971.

5) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.

6) In the report entitled "independent Technical Report on the Wolverine Project" dated 28 February 2007, for Yukon Zinc Corporation. I am responsible for the direction of the 2006 metallurgical test programme and review of earlier testwork data, and the development of preliminary flowsheets forming the basis of the plant design, as well as writing the sections in the report of metallurgy. I have not made a site visit.

7) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report.

8) I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101.

9) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public files on their websites accessible by the public.

Dated in Vancouver this 28th day of February, 2007.

Original document, Revision 01 signed and sealed by John R.W. Fox

Signature

Suite 550 – 1090 West Pender Street
Vancouver BC V6E 2N7
CANADA
Telephone +1-604-683-7645
Facsimile +1-604-683-7929
apolk@snowdengroup.ca
www.snowdengroup.com

Perth, Brisbane, Vancouver, Johannesburg, London

Allan Polk, P.Eng
Senior Consultant
Snowden Mining Industry Consultants Inc.

Certificate of Author

I, Allan Polk, P.Eng, (QP), do hereby certify that:

1. I am Senior Consultant of:

   Snowden Mining Industry Consultants Inc.
   Suite 550, 1090 West Pender Street
   Vancouver, BC, Canada,
   V6E 2N7

2. I graduated with a Bachelor of Science degree in Applied Science from Queens' University in 1990.
3. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia as well as the Association of Professional Engineers of the Yukon
4. I have worked as an engineer or under the direct tutelage of a professional engineer for 13 years since graduation.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I visited the Wolverine property on August 14th and 15th, 2006.
7. I am the author of the report titled, 'Mining Portion of Underground Feasibility Study, January 2007' for the Wolverine project and issued on behalf of Snowden Mining Industry Consultants Inc. (Snowden) to Yukon Zinc and Wardrop Engineering Ltd. Wardrop Engineering Ltd. has integrated the results of Snowden's mining study into a feasibility study document titled 'Optimised Feasibility Study, Wolverine Project' and dated January 2007 as well as a technical report describing the results of the study and titled 'Independent Technical Report on the Wolverine Project – Finlayson District, Yukon', dated 28th, February 2007.
8. I have previously been involved with the Wolverine project as an independent reviewer of the mining portion of a previous Feasibility Study, compiled by Hatch and titled 'Detailed Feasibility Study for the Wolverine Project' and dated April 2006.
9. I have read the Technical Report and Feasibility Study document and as of the date of the report, January, 2007, to the best of my knowledge, information and belief, the Technical Report contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 28th Day of February, 2007.

____
Signature of Qualified Person

Allan Polk, P.Eng (CP)
Name of Qualified Person

23.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

23.1 MINING OPERATION

This section was written by A. Polk, P.Eng.

23.1.1 SUMMARY OF MINING OPERATIONS

Underground operations at the Wolverine Mine have been planned to achieve a daily production rate of 1,850 t/d of diluted ore. It has been assumed that 8% of days during a year will be down due to maintenance or unforeseen circumstances and hence the typical yearly production rate at steady state is 620,000 tonnes of diluted ore. An effective average production rate of 1,700 t/d will result over any year long period. Development and production schedules have been created and costs estimated and applied to reflect achievement of this rate over the LOM. Hence, a small portion of surge capacity has been provided within the underground mine design, labour force, and equipment list.

As it is very typical to have daily fluctuations in the amount of ore which is hauled to surface, a small dynamic stockpile of ore will be used near the portal and crushing circuit on surface so that the daily differences in production can be managed. In the event that considerable process downtime occurs, a larger ore stockpile will be available so that mining can continue unabated.

The mining reserve of 5.2 Mt will provide for an operating mine life of 9.5 years, including 1.5 years of pre-production development activities at the beginning of the mine life as well as a final year of decreased mine production during which a retreat mining process will be undertaken and all voids filled.

A portal at elevation 1345 and short section of decline was developed as part of an underground test mining program in 2005. This decline will be extended downwards to follow the shallow dip (~35°) of the orebody. The decline will eventually be excavated downwards from the current lowest point at elevation 1315 to elevation 1100 at an average gradient of -12%. The existing portal and decline will be the main access into and out of the mine over the life of operations.

The decline will be centrally located within the deposit and in the thinner ore of the Saddle Zone. Where possible, the decline will follow the plunge of the ore, attempting to stay in the better ground conditions associated with the massive sulphide deposit. However, the geometry prohibits continuous ramping in ore so the ramp will need to spiral out into the hangingwall of the deposit. It is recognized that there is an inherent risk with placing the decline in the hangingwall of the deposit. As a result, all ore in close proximity to the decline has been defined as a barrier pillar, which will be mined in a retreat process at the end of the mine life. Information available thus far suggests that the footwall rock conditions are even worse and are prohibitive to development advance.

Drift and fill will be utilized as the mining method. The hangingwall and footwall rock conditions are very poor, and the method has been chosen for its selectivity and for the ability to manage the size of open spans in stope excavations. Three variations of the mining method are used so that stope productivity can be achieved and also so that the rock conditions in the hangingwall are managed with minimum dilution. Hence, the mining method variations are applied in response to the horizontal ore width.

Paste back-fill will be the primary filling material for the mine, again in response to the poor ground conditions in the hangingwall of the orebody and the need to manage open spans. A paste back-fill plant will be constructed on surface, near the 1345 portal, and a paste delivery line will be hung along the back of the access decline and into stopes. Loose waste rock from development headings and loose waste floated from the DMS plant will also be placed in stope voids, but this will be a small portion (~20%) of the total fill requirements.

Mining for both development and production purposes will be undertaken with diesel powered mobile equipment. Drilling operations will be performed primarily with electric hydraulic drill jumbos mounted on diesel carriers. Rock-bolting operations will be performed with electric hydraulic units, capable of bolting from remote locations and with a combination of resin re-bar, split-sets, and wire mesh screen.

The very poor condition of the waste rock which surrounds the orebody has precipitated the need to install an efficient wet mix shotcreting system. A mixing unit with auger will be purchased and installed on surface. This will be used to mix bulk bag pre-mixed shotcrete with water which will then be dispensed into a transmixer unit for hauling the batch of shotcrete underground, where it will be sprayed onto the rock by a remotely operated spraying unit.

LHDs will be used to load both waste and ore from development and stope headings into diesel haulage trucks for haulage either to surface or to stopes. LHDs used in stopes will be mid sized units with 3.9 m^3 capacity buckets. These units will primarily be used to move material from headings into remucks. A larger capacity LHD with a 4.8 m^3 bucket will be used to load material from a remuck into the haulage trucks for haulage to surface. An additional, smaller capacity unit with a 2.7 m^3 bucket will be purchased for use in face clean-up and tight geometry mucking situations.

Two 50 tonne capacity haulage trucks will eventually be used as the primary haulage units. These will run continuously in the decline, taking both ore and waste to surface, where it will be dumped into the process plant, or into a temporary waste dump. An additional, smaller capacity 30 tonne haulage truck with an ejector style bucket will also be used. This unit will haul waste material into stope headings for dumping of loose fill or DMS reject material. In the first three years of mine production, a single 50 tonne unit and a single 30 tonne unit will be required. A second 50 tonne capacity haulage truck will be purchased at the beginning of 2012, the fourth year of production.

A maximum of 185 m^3/sec of ventilating air will be required based on a projected equipment list for the mine. Two ventilation raises and the main decline will be used to create a ventilating system. Fresh air will travel down both the main decline and a smaller, secondary raise, which will contain a permanent escape manway. After use in the working areas, the air will be collected in a main exhaust raise for removal from the mine. An exhaust fan at the top of the exhaust raise will be the primary air mover for the mine. Direct-fired propane heaters will be installed at the top of the decline and the smaller secondary raise so that air temperatures are maintained above freezing throughout the seasons. A short additional section of raise will be required near the top of the decline to allow access into the mine without impeding the action of the burners during the winter months.

Stope escapeways will also be developed during the course of mining to provide emergency egress either to the stope above or below should access to the stope be cutoff for whatever reason. Two continuous lines of raises will be developed in the mine and these will be located near the hangingwall contact and in the thickest ore in both the Lynx and Wolverine zones.

Power to the mine will be supplied by the main generators at the industrial complex. Power will be delivered to underground operations in cable run down the main ramp via the 1345 Portal. Power will initially be provided to the underground operations at 4.16 kV power and this will then be transformed to 600 V power for typical use underground. An industrial Ethernet cable will be employed for mine radio and other communications.

Compressed air for underground operations will be a small requirement for the mine. A small compressor will be installed in the shop area, immediately adjacent to the 1345 Portal, with piping routed down the main decline and into working areas. Compressed air will primarily be used for de-watering, through the use of pneumatic pumps during the drilling phase of operations.

23.1.2 MINING METHOD AND VARIATIONS

Drift and fill is the mining method selected for the project. The primary reasons for selecting drift and fill are that a high proportion of the deposit can be extracted, as no permanent pillars are required and the selective nature of the method will allow mining in areas where the ore grade mineralization is thin. Additionally, the poor rock conditions of the immediate hangingwall are managed appropriately by utilizing small excavations and quick stope cycle times.

Mining will begin in the upper portions of the orebody, to minimize the time and costs required to develop sufficient stopes for full-scale production. However, mining within any stope begins at the bottom of a stope, with a 4 m lift being mined. Each stope will be comprised of five lifts, over 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. Because overall mining of the orebody has started at the top, the fifth and final lift in any stope will occur below the first lift of the stope above, exposing the backfill.

In all stopes, a footwall drift will be excavated ahead of stope mining. This drift will be excavated at a moderate size and will follow the footwall contact of the orebody, trying to keep the back of the drift in ore. Behind the advancing face, a portable diamond drill will be utilized to drill horizontal test-holes towards the hangingwall of the ore so that the horizontal width of the orebody can be determined accurately. More test holing will be required on the first lift in any stope. As each lift is mined and mapped, more knowledge is available and fewer test-holes will be required.

Once the width of the orebody has been established, one of three drift and fill variants will be chosen based on the orebody width as shown in Table 23.1.

Table 23.1 Selection of Mining Method by Ore Thickness

Horizontal Ore Thickness	Mining Method
< 4 m	Drift and fill with side slash (DFSS)
4 to 7 m	Drift and fill with retreat panels (DFRP)
> 7 m	Drift and fill with primary and secondary panels (DFPS)

Table 23.2 shows the overall percentage of ore-by-ore thickness. On any single stope lift, it is expected that some of each mining method variation will be used.

Table 23.2 Proportion of Mining Reserve by Ore Thickness

Horizontal Ore Thickness	Tonnes	% of Reserve	Average Horizontal Thickness (m)
> 7 m	3,819,807	74	12.2
4 to 7 m	775,441	15	5.9
< 4 m	556,211	11	2.9
Total	5,151,459	100	9.9

23.1.3 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. Primary stoping panels will then be excavated at 5 m widths leaving temporary 5 m pillars between. These will be developed from the footwall stope drift in a ‘herringbone’ fashion at an angle of approximately 60° to the footwall stope drift. These panels will be driven towards the hangingwall until the argillitic contact is reached.

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.

The hanging wall is primarily composed of very poor graphitic argillite (ARGR). To avoid excessive dilution at the end of the primary panels, it is expected that shotcreting of the last round will be required. This will be performed on an 'as required' basis and will be dependant on the rock conditions in the immediate hangingwall as exposed by the ends of the panels.

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. It is expected that paste bulkheads will be built with laced cables, wire mesh, and geotextile material that is attached to the walls with split-sets. In some cases, the bulkheads will need to be shotcreted. In some cases, posts will need to be installed to ensure that the backfill covers as much of the intersection span as possible. This process will also allow more complete backfilling of each individual panel than a muck berm and fill fence system, enhancing the stability of the stope by minimizing the open span.

This variant of drift and fill mining has considerable advantages as follows:

- The herringbone panels will intersect the hangingwall of the orebody at a steep angle, providing minimal span exposure of the weak rock, and ultimately minimizing dilution of waste while maximizing recovery of the valuable ore.
- On any single stope lift, many primary panels can be in production at one time, providing high productivity from few faces.
- After primary panels are completed, it will be possible to place loose waste fill at a moderate rate reflective of the rate at which it is created during development tunnel advance. This will allow much of the waste created underground to be placed in stope excavations directly and without re-handle.

However, DFPS mining has some disadvantages as follows:

- The resultant spans at the intersections of the primary panels and the footwall drift can become large and extra rock support in the form of longer rebar and/or cable-bolts will be required occasionally.
- Tight filling in the primary panels will become of critical importance to manage the spans at the intersections. If tight filling is not achieved then the mining of the

secondary panels in between the primary panels will create even larger spans, again precipitating the need for additional and longer rock support in the form of cables. Mismanagement of the span size and tight filling practices can ultimately remove any productivity gains from the DFPS method.

23.1.4 Drift and Fill with Retreat Panels (DFRP)

For ore areas with 4 m to 7 m of horizontal thickness, 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 five to seven days, the next adjacent panel will be mined, exposing the backfill of the previous panel along the wall. In this fashion, the stope will incrementally retreat towards the stope access.

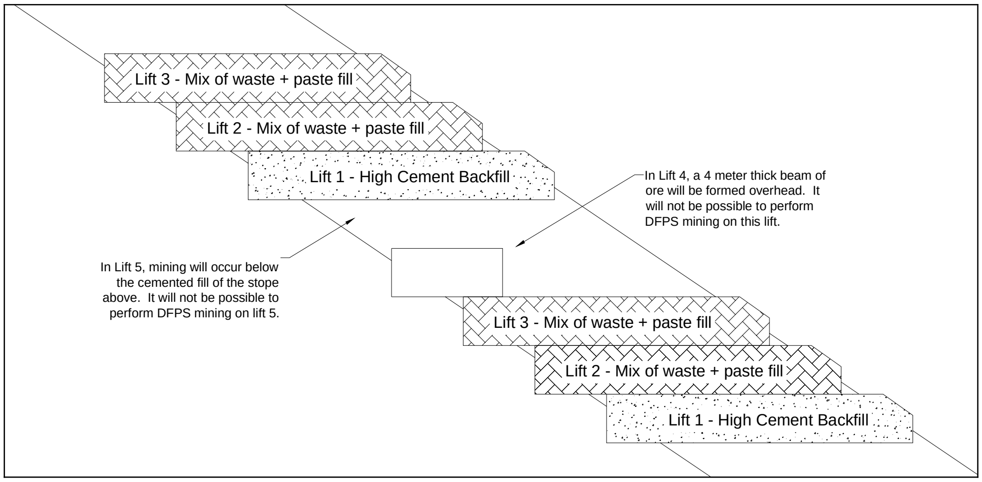
DFRP mining has a distinct advantage over DFPS mining in that excavation spans are controlled more easily. Effective intersections are not created until each panel is mined and the rock conditions can be assessed at the start of the panel to ensure that problems will not be encountered.

DFRP mining, and a variation of DFRP mining, Longitudinal DFRP, will also be used in some locations where the ore is wide, but where opening large spans is not possible. These situations are as follows:

- locations where the ore is wide but localised ground conditions at the footwall drift prohibit the creation of large intersection spans
- the top two lifts of any stope, where opening large spans underneath the final lift, or underneath the cemented fill of the stope is not recommended.

In the second highest lift of any stope, a 4 m thick ‘beam’ of ore will be present above the lift, and opening large spans in this case may cause the ore above to become unstable, depending on the presence of joints or structure within the beam. A more conservative approach to mining in this case is warranted due to the risk of failure. The situation is presented in Figure 23.1.

Figure 23.1 Use of Longitudinal DFRP in Wide Ore



Due to these reasons, the proportions of the ore reserve by mining method are presented in Table 23.3, a modification from the strict proportions of the deposit by horizontal ore width, which was presented in Table 23.2.

Table 23.3 Proportion of Mining Reserve by Mining Method

Method	Tonnes	% of Reserve	Average Required (t/d)
DFPS	2,291,884	44	814
DFRP / LDFRP)	2,303,364	45	831
DFSS	502,069	10	185
Ramp Development	54,142	1	18
Total	5,151,459	100	1,848

23.1.5 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 recoverable ore. Once the economic extent of the stope is reached, 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, but will be occasionally shotcreted to ensure that failure does not occur.

The width of the footwall drift and the length of the jumbo booms will determine the maximum width of additional slashing which can be accomplished in DFSS stopes. With 4 m steel utilized, a typical slash width of 2.5 m is expected.

The individual blasts will range in strike length exposure depending on the width of the slash and the geotechnical conditions of the immediate hangingwall. After a short length of slashing, a bulkhead will be placed in the footwall drift and the stope void will be filled as tightly as possible with paste backfill. It has been assumed that

12 m of hangingwall length will be achievable on a normal basis, with occasional shotcrete used to maximize this length without allowing caving of the hangingwall. As an operational philosophy, the method will be applied under the assumption that the hangingwall will not be allowed to unravel, producing potential operational issues on subsequent lifts. A conservative approach to this method will pay off in the end, and extensive slashing in an effort to recover a small portion of the resources in a short time period has not been planned and is not recommended.

23.1.6 Hanging Wall Lenses

There are occurrences of thin lenses in the hangingwall of both the Lynx and Wolverine zones; these vary in thickness from 1 m 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:

- In the cases where more than 5 m of waste lies between the hangingwall and footwall lenses, a short length of access tunnel will be excavated over to the hangingwall lens. A waste pillar will be left between the two lenses and the hangingwall lens will be mined by one of the three described methods, again dependent upon the width of the ore.
- In cases where there less than 5 m of waste 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.

23.1.7 Mine Access

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 23.4. Each development type is broken out by pre-production development period, and LOM. A complete LOM layout for the mine is shown in Dwgs. MO-X-001 through -004. A sample level plan showing two sides of the mine is provided in Dwg. MO-X-007. These drawings can be found in Wardrop's Optimized Feasibility Study dated January 2007.

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

These tables 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 Dwg. MO-X-005 of Wardrop's study.

Table 23.4 Mine Development Requirements

Heading Description	Dimension (m)		Development Period		
	Width	Height	Pre-pro	Operations	LOM
Lateral					
Ramp	5	5	743	1,345	2,088
Stope Access	4	4	382	979	1,361
In-stope Development**	4	4	737	n/a	737
Diamond Drill Drift	4	4	54	1,343	1,397
Vent Drift	4	4	277	261	538
Total Lateral	-	-	2,193	4175	6,121
Raising					
Main Intake/Exhaust	4	diameter	150	-	150
Level Exhaust	3	diameter	-	328	328
Main Escapeway	2	diameter	85	160	245
Stope Escapeway	1.8	1.8	74	522	596
Total Raising	-	-	309	1,010	1,319

Note: **Sufficient in-stope development is shown during the pre-production period to ensure that stope escapeways are accessed and in place before full-scale production is initiated. After that, in-stope development is part of the stoping plan and therefore not included as part of the development plan.

MAIN ACCESS RAMP

The mine will be accessed by a single ramp driven at an average gradient of -13.5% and located centrally between the Lynx and Wolverine deposits in and around a zone of thin mineralization called the 'saddle zone'. Intersections will be flatter to minimize maintenance upon the equipment, and straight sections of ramp will be steeper, up to a maximum gradient of -15%. The poor and schistose nature of the waste rock suggests that this should be the maximum gradient for reasonable trafficability. The ramp has been sized at 5 m wide x 5 m high, with an arched back and/or shanty back profile (arched in waste or thick ore, shanty in thin ore) to accommodate a future production fleet of 50 tonne diesel haulage trucks. All equipment and personnel within the mine will use the single main access ramp for daily access. A typical arched ramp profile and other heading profiles are shown in Dwg. MO-X-008.

During the 2005 test-mining program 250 m of ramp was developed via the 1345 Portal in the hanging wall of the deposit (see Dwg. MO-X-006). The ramp will continue in the hanging wall of the deposit for an additional 200 m to the 1290 m elevation. At this point, the ramp will enter the ore of the saddle zone. After this point, the ramp will be extended downwards in an alternating sequence of sections in ore, then waste, then ore, etc.

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. As a result of this, all the ore around the ramp and the start of the stope accesses is considered to be part of a pillar, which will be mined at the end of the mine life. The ramp was located in the thin and low grade ore of the Saddle Zone both because it is centrally located within the framework of the overall deposit and secondly because a relatively small portion of the total resource will need to be tied up within a pillar until the end of the mine life.

STOPE ACCESS DRIFTS

Every 10 vertical metres of ramp advance a stope access will be started, with every other access lying 20 vertical metres below another. Stope accesses are designed to be in ore and will be driven 4 m wide and with a shanty back which follows the hangingwall waste contact. Where the ore is sufficiently thick, an arched profile will be used, with a drift height of 4.4 m measured to the highest point of the arch. Typical stope access profiles are shown in Dwg. MO-X-008. Stope accesses will be driven downgrade at -15%, to allow minimization of the back slashing required to reach subsequent lifts. In both cases, sufficient height within the stope accesses must be maintained to allow clearance for the installation of 1.2 m diameter flexible ventilation tubing.

DIAMOND DRILL DRIFTS

Four diamond drill drifts will be excavated in the hangingwall above the deposit, two on each side of the main access ramp. Drill drifts will be located approximately 45 m to 50 m away from the hangingwall contact of the ore zone and are positioned to ideally drill off the ore zone to a definition level for production purposes.

VENTILATION ACCESS DRIFTS

Every 40 vertical metres of ramp advance (every other spiral), ventilation access drifts will be excavated to allow the development of the ventilation and escapeway systems. These will be developed at 4 m wide x 4.4 m high with an arched profile. Exhaust raise accesses will typically be developed at a negative gradient to contain ground water, which is created in the raises. Escapeway accesses will be typically developed at a positive gradient to ensure that water does not run into the escapeway system and is instead diverted. An overall description of the ventilation system is provided in Section 23.1.10.

MISCELLANEOUS DEVELOPMENT

Remucks – On every spiral of the main access ramp, a re-muck will be excavated to allow efficient use of the mine equipment. These will typically be driven at 5 m wide x 4 m high and will be 12 m to 15 m in length. Ideally, these will be located where the ramp is being developed in ore that is thick, both to take advantage of the extra revenue from ore development, as well as to allow the creation of intersections in better ground conditions.

Safety Bays – Every 30 m along the length of the main access ramp, a 1.5 m wide x 2.0 m high x 1.8 m deep safety bay will be excavated. These will typically be located on the inside of ramp corners.

An additional allowance of 5% of the length of the heading has been added to all ramp, vent access, and diamond drill drift development to account for extra-unplanned development advance.

Sumps – Small collection sumps will be excavated at the top of the stope accesses. These will typically be excavated at 3 m wide x 3 m high x 5 m in length. There will also be two main sumps in the mine, with excavation sizes typical of main ramp development, 5 m wide x 5 m high.

23.1.8 RAISING

Raises will be constructed for ventilation and emergency egress from the mine. Raise bores will be utilized for the excavation of main ventilation fresh air and exhaust routes, and conventional development raises will be excavated for the purposes of in-stope escapeways.

The main exhaust raise will be comprised of five sections of bored raises, which will be excavated over the life of the operation. The first section will be excavated during the pre-production period and will be a 4 m diameter x 110 m long raise at approximately -85°.

As the main access ramp is advanced downwards, new sections of raises will be excavated. Every 40 vertical metres along the length of the ramp, a new section of main exhaust raise will be excavated with a raise bore unit. These sections will be comprised of two, parallel, 3 m diameter raises at approximately -60°.

All main exhaust raises will be supported with fibre-reinforced shotcrete via the use of a remote raise climber.

The main mine escapeway will also be comprised of five sections of bored raise, similarly excavated over the life of the operation. The first section will be excavated during the pre-production period and will be a 2 m diameter x 85 m long raise at approximately -85°. Again, the subsequent sections of escapeway will be excavated as the main access ramp proceeds downward. Every 40 vertical metres along the length of the ramp, a new section of escapeway will be excavated. These sections will typically be 40 m long and at a dip of approximately -60° and will be 2 m in diameter.

Conventionally driven (stoper/jackleg) stope escapeways will also be excavated over the life of the mine. These will be excavated at 1.8 m wide x 1.8 m high and will be driven close to the hangingwall contact of the orebody and from stope to stope.

23.1.9 MINE EQUIPMENT

Mobile trackless diesel-powered equipment has been assumed for the mine. The underground equipment list is shown in Table 23.5. All of the main production and development drill equipment will be electric-hydraulic.

Table 23.5 Mobile Underground Equipment List

Unit Type	Model	Units Required
Jumbo drill	Axera 7-240	2
Rock bolting machine	Robolt 5-126	2
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	Normet	1
Shotcrete supply	Normet	1
ANFO loader	Triple 4ce	1
Electrical	Triple 4ce	1
Mechanical	Triple 4ce	1
Construction Scissor/Combi	Triple 4ce	1
Supplies – Nipping	Triple 4ce1	1
Supervisor/mancarrier	Toyota Landcruiser	3
Haul truck (ejector)	EJC30SX	1
Diamond drill	Hydracore Gopher	1
Pneumatic longhole drill	Not specified	1

Note: that the diamond drill and longhole drill are not diesel units.

The equipment list has been developed from first principle cycle time estimates and includes factors and/or adjustments as follows:

- An effective 50 minutes of each hour (83%) is always applied to reflect the mechanical availability of the equipment.
- Each 11-hour shift has an effective shift length of 9.5 hours to reflect breaks and travel time.
- Schedule requirements are based on an assumed yearly downtime of 8% of days and this is reflected in the equipment requirements.
- In some cases, such as for haulage, an efficiency factor has been applied to reflect realistic issues in underground mining, such as traffic on a single ramp. Other factors include:
  - second boom lower efficiency on the drill jumbos
  - fill and load factors for LHD and haulage trucks
  - cycle time estimates include additional time for jumbo and rock-bolter travel between headings, setup, scaling, and teardown
  - extra time for the installation of mesh
  - other miscellaneous factors.

HAULAGE EQUIPMENT

There are five separate haulage needs for the underground mobile equipment:

- Ore or waste will need to be loaded and hauled from underground re-mucks to the process plant or to a surface waste dump. In general, a 4.8 m^3 capacity (12.5 tonnes) LHD unit will be used for loading in this system. The unit will load either waste or ore from a remuck located along the main access ramp into 19.3 m^3 capacity (50 tonnes) diesel haulage trucks for hauling to surface. This is the main haulage need for the operation and it is planned for the units to be used exclusively along the main access ramp where sufficient ventilation flow has been designed. One TORO 1400 LHD and two TORO 50 Haulage trucks have been specified for this operational haulage need, with one of the TORO 50 haulage trucks being required at the beginning of the mine life, and a second truck being purchased after four years of production.
- Ore or waste will need to be loaded and hauled from an advancing development or stope heading to a re-muck located on the main access ramp. In general, 3.9 m^3 capacity (10 tonnes) LHD will be used for this purpose and stope accesses have been sized under the assumption that a unit of this size will be used rather than the larger LHD unit. Two TORO 007 LHD units have been specified for this operational need.
- Waste from an advancing development or stope heading will need to be hauled to a stope as part of the filling needs for that stope. Either of the 3.9 m^3 or 4.8 m^3 capacity LHD units will be used to load waste into a smaller capacity haulage truck, which is sized to fit into the stope areas. An 11.6 m^3 capacity (30 tonnes) EJC30SX unit equipped with an ejector system will be utilized to haul waste into stopes as a portion of the fill. Most of the waste created underground will be hauled directly into a stope as fill, rather than hauled to surface.
- It will be beneficial to maximize the portion of waste utilized within the stope as a portion of the fill. As a result, waste or DMS reject material from surface will need to be hauled back underground to a stope as part of the filling needs for that stope. On surface, a front end loader (within surface equipment requirements) will be used on a part time basis to load the 11.6 m^3 capacity haulage truck with waste for haulage back underground for placement within the stopes.
- A smaller, 2.5 m^3 capacity LHD will be used underground for face clean-up duties, particularly in the stopes, where the shanty geometry will prohibit the use of a larger LHD unit for recovery of the valuable ore. A single EJC 145 LHD has been specified for this purpose.

Both of the primary production LHD units are capable of loading both sizes of truck. However, only the smaller LHD units and smaller capacity truck will fit within the geometry of the stope accesses. Hence, ventilation systems have been sized for equipment on this basis.

Drilling Equipment

Electric Hydraulic drill jumbos with two booms will be utilized for all development and stoping advance in the mine. Two units capable of drilling 4.0 m long holes will be required for this operational need. For this purpose, Tamrock Axera 07-240 units have been specified.

Pneumatic jack-leg drills will also be used on an occasional basis for the drilling of small diameter, short holes for the development of safety bays and conventional escapeway raises.

Ground Support Equipment

Three different ground support systems will be employed at the operation: bolts with mesh, wet-mix shotcrete, and cable-bolts. Equipment capable of installing both systems will be utilized at the operation.

Rock-bolting Units

Two mechanized rock-bolting units will be utilized as the primary ground support installation units for the operation. Each unit will be equipped with a boom style drill unit capable of installing both 2.4 m long rebar with resin as well as 1.8 m long split sets. Each unit will also be capable of holding/installing mesh along with the specified bolts. The Tamrock Ro-Bolt 5-126 units have been specified for this purpose.

In addition to the mechanized rock-bolting units, a few pneumatic stoper drills will be purchased for the installation of rock bolts on an ‘as required’ basis. In particular, the stoper drills will be used to install longer length re-bar for use in larger span intersections as described in the geotechnical report. It is expected that 3.0 m long rebar with resin will be installed in intersections.

Shotcreting Equipment

A wet mix shotcreting will be utilized at the Wolverine operation. This system will involve three components, a mixing and dispensing unit on surface, a transmixing unit for delivering the shotcrete to an underground location, and a shooting unit for spraying the shotcrete on the rock surface.

On surface, a used, converted cement truck bin has been sourced. This unit will be fed with bulk bags of shotcrete, which will be purchased and delivered to site ready for hydration with a measured amount of water. Batches of approximately 3 m^3 will be mixed on surface with water and tested for quality before being dispensed into a unit capable of transferring the batch underground. The mixing bin will be equipped with an auger unit for dispensing.

The prepared batch of wet shotcrete will be dispensed into a transmixing unit for delivery underground. This unit will be a diesel unit purposely built for underground shotcrete delivery and has been specified by Normet. The transmixing unit will deliver the wet shotcrete to a location underground where a shooting unit awaits.

The shooting unit will spray the shotcrete on the rock surface using a remote controlled boom arm capable of reaching all planned excavations. This unit is equipped with a cement piston pump. A Normet unit has also been specified for this purpose. As specified by the Geotechnical report for the project, only high quality shotcrete with fibres will be utilized underground for rock support.

It is estimated that the shotcreting system will be capable of shooting two batches per shift underground if necessary (6 m^3). However, it is estimated that on average, somewhat less than one batch per day (2.5 m^3) will be required. Snowden recognizes that shotcreting capability must be available on both shifts due to the ground conditions and production demands, but if the system is fully operated on a two shift per day basis, then shotcrete use will be far more than required for ground support. Workforce equivalent to 1.5 shifts per day has been assigned to the task of shotcreting. In reality, some of this workforce will occasionally be assigned to other duties around the mine, and on night shift, it will occasionally be necessary to assign an additional employee to shotcreting duties. Snowden recognizes that time is of the essence for the placement of shotcrete in very poor ground conditions.

The shotcreting system will be purchased and in place for use during the pre-production development period so that the efficiencies of the system can be utilized by the contractor.

Cable-Bolting Equipment

A small longhole drill capable of drilling 10 m long upholes will be used to drill holes for the installation of cement grouted cable-bolts in larger span intersections or where the presence of wedges is indicated by structural mapping. It is expected that a pneumatic long-hole drill will be purchased for this occasional purpose.

A grout pump will also be used to facilitate the installation of the cable-bolts.

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 development, and loose DMS float product. Over the LOM, the following quantities of fill will be required:

- 2.18 Mt of paste backfill
- 0.40 Mt of loose waste rock
- 0.22 Mt of DMS float rock.

Once mining is completed, it has been planned to fill 95% of all underground voids for closure purposes.

LOOSE WASTE OR DMS REJECTS AS FILL

The amount of waste and/or DMS float rock, which can be placed in stope excavations, will be dictated by the mining sequence, the resultant mining geometry, and the production requirements of the operation. In cases where mining will occur beside another filled stope, it will be necessary for the entire wall to be composed of paste. As a result of this requirement, it will only be possible to place a minimal amount of loose fill (either waste or DMS rejects) in primary mining panels as described for DFPS. More waste can be placed in secondary panels, in DFRP panels, and in DFSS mining panels. The mining geometry and probable maximum amount of placed waste is shown in Figures 23.2 and 23.3. These figures suggest that a maximum of approximately 50% of the mining void could be filled with an estimated 540 tonnes of loose waste fill each day.

Figure 23.2 Waste Fill in DFPS Mining Panels

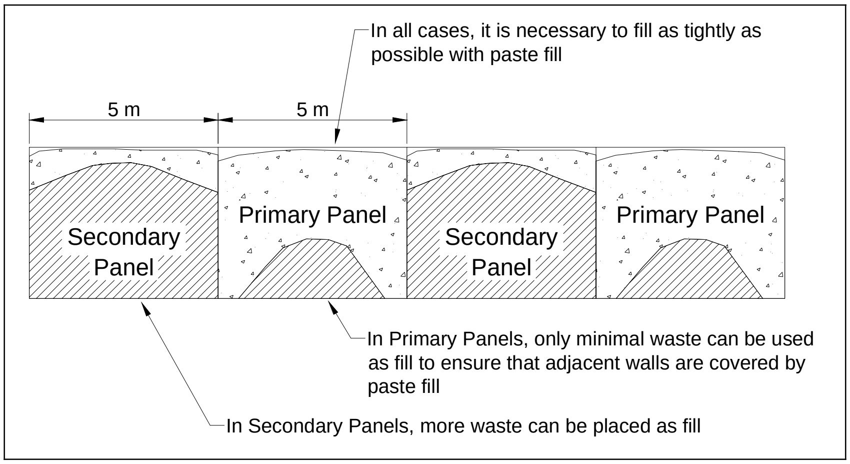
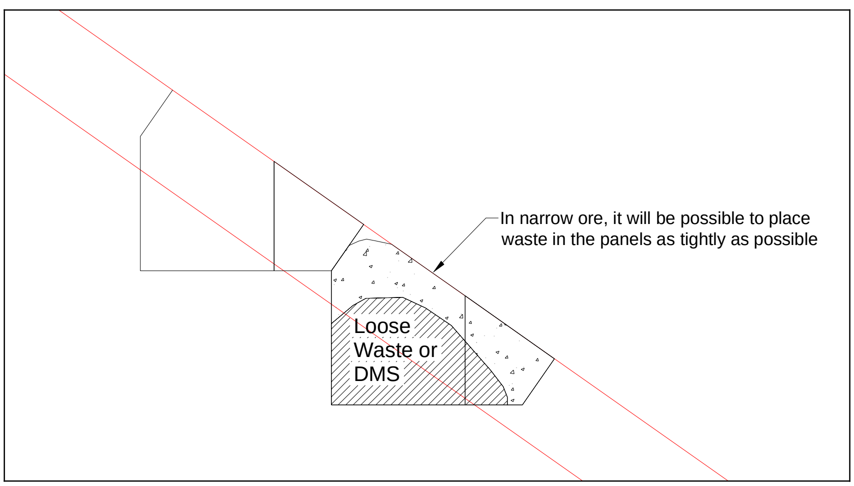


Figure 23.3 Waste Fill in DFSS Stoping Areas



Loose fill material cannot be utilized in the first lift of any stoping block, because mining in the fifth lift of the stope below will need to be performed only under competent engineered paste fill with a high proportion of cement. This again reduces the maximum amount of waste that can be placed in stope voids to approximately 40% of the total stope void. This does not consider production constraints.

Over the life of the mining operations, approximately 400,000 tonnes of waste will be created from development waste. This amounts to only 14% of the total stope voids and it is emphasized that the waste development schedule is very moderate over the life of the operation, with a considerable portion of the waste development being completed during pre-production. In the first two years of the project (including pre-production), 118,000 tonnes of waste will be created and all of this will be hauled to surface and stored as a temporary stockpile.

In DFPS stopes, there will be some opportunity to place waste on a basis that reflects the rate at which it is created by development. This opportunity exists because many primary panels will be completed and will not need to be filled until the secondary panel retreat reaches that location. Figure 23.2 shows the amount of fill that can be placed directly into primary panels. It is estimated that 10% of the total DFPS mining void will be available for immediate placement of waste and this represents 5% of the total mined void, or 55 t/d of placed waste.

There will also be some opportunity to place waste directly into DFRP and DFSS stopes, but this will be dictated by production pressure because panels will need to be filled so that production can resume in the stope. Waste will be created at approximately 150 t/d during the first years of production. Fifty-five tonnes of this material can be placed on an average basis in DFPS primary panels and it has been assumed that an additional 65 t/d of the waste (1 round every 3 to 4 days) can be placed directly into a stope, which is ready for backfilling. Therefore, it is estimated that 80% of the total waste can be hauled directly from a heading into a stope as loose backfill. This will be hauled with the smaller capacity (11.6 m^3) truck with an ejector bucket. The remainder (30 t/d) will need to be hauled to surface and then returned underground as allowed by the availability of stopes, which are ready for filling. Therefore, it is estimated that there is a daily haulage need to handle 180 tonnes of loose waste from development.

It is estimated that the single large capacity LHD and the single small capacity haulage truck are capable of hauling 1,200 t/d of waste so there is considerable extra haulage capacity within the waste haulage system. In reality, the smaller capacity haulage truck will haul waste or ore depending on the production needs of the mine.

In the last four years of the mine life, a negligible amount of waste will be created from development and during this time, the surface stockpile of waste will be depleted and an additional 220,000 tonnes of DMS reject material will be hauled underground as loose fill.

PASTE BACKFILL

The paste backfill plant will be located in the industrial complex at the northwest corner, closest to the portal. The paste will be manufactured from unclassified tailings. Portland cement will be added to the paste, 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 not 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.

Mine Systems Design of Kellogg, Idaho, performed laboratory testwork to assess the amenability of the tailings material to the formation of paste, and to estimate the amount of cement required to strengthen the paste sufficiently for mining needs. Their report is included in Appendix B.3 for reference of the Optimized Feasibility Study prepared by Wardrop.

Based on the results of the testwork, the following binder contents are assumed for the three applications: low strength = 1% cement; medium strength = 4% cement; high strength = 8% cement. The amount of binder required is also dependent upon the amount of time available for curing. According to the paste strength testwork and Mine System Design's experience 5% cement and a curing period of seven days will be required to create a material that is sufficiently strong to be used as a floor for men and equipment. This will not be required for the Wolverine operation, and it has therefore been assumed that 4% cement and a curing period of six days will be sufficient to create a wall of paste fill, against which mining will proceed with minimal difficulty. This criterion has been used for DFRP mining, for which the curing period becomes critical.

In all cases, it has been assumed that the surface paste system will be capable of generating sufficient pulp density so that a 7" slump can be achieved for the material at 74% solids by weight.

It is estimated that 865 tonnes of paste will need to be poured daily (430 m^3) to fill voids on a steady state basis so that mining can proceed. This is quite close to the typical pour size of voids so that every day a small bulkhead and piping will need to be installed and a pour location prepared. Every shift, two men will be assigned to filling needs for the operation, whether it is constructing bulkheads or monitoring the pouring of fill. It is expected that split sets and cables will be used to anchor wire mesh to the walls of the drift and geotextile will be used to help seal the bulkheads. Occasionally, shotcrete will be required to help seal the bulkheads.

Table 23.6 provides an estimate of average cement usage by mining type for the operation based on the geometry of the lifts and adjacent mining.

It is estimated that 865 tonnes of paste will need to be poured daily (430 m^3) to fill voids on a steady state basis so that mining can proceed. This is quite close to the

typical pour size of voids so that every day a small bulkhead and piping will need to be installed and a pour location prepared. Every shift, two men will be assigned to filling needs for the operation, whether it is constructing bulkheads or monitoring the pouring of fill. It is expected that split sets and cables will be used to anchor wire mesh to the walls of the drift and geotextile will be used to help seal the bulkheads. Occasionally, shotcrete will be required to help seal the bulkheads.

Table 23.6 Cement Usage by Lift and Mining Variant

Mining Method	Typical Pour Size Pour Volume (m^3)	Cement Strength Type (%)			Binder Content (%)
		High	Regular	Weak	
DFPS – Lift 1	355	100	-	-	8.0
DFRP – Lift 1	117	100	-	-	8.0
DFSS – Lift 1	480	-	100	-	4.0
DFPS – Lifts 2 – 5	355	-	33	67	2.0
DFRP – Lifts 2 – 5	450*	-	95	5	3.9
DFSS – Lifts 2 – 5	480	-	-	100	1.0
DFPS – Ave	355	20	26	54	3.2
DFRP – Ave	383	20	76	4	4.7
DFSS – Ave	480	-	20	80	1.6

Note: *Estimated volume size for DFRP mining includes longer panels associated with longitudinal drift-and-fill mining in cuts 4 and 5.

Paste backfill will be passed to the mine through the main ramp via the 1345 Portal in 150 mm Schedule 80 steel pipe that will be rigidly mounted in the main ramp. The final 300 m of each paste line will be high-density polyethylene (HDPE) pipe that will be installed in the back of the footwall drift. This HDPE pipe will typically be sacrificed during pouring operations.

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. A 300 hp pump has been sized on surface for this requirement.

The overall fill requirements have been discussed with industry experts in paste technology and it is generally agreed that the paste fill system can achieve the goals of the operation. However, Snowden recognizes that the filling requirements of the operation are complicated, and a considerable amount of extra flexibility will be required in the paste fill system. An important mitigating feature of the operation is that only 66% of the total tailings produced will be required to fill the underground voids. Thus, extra fill will be available during times when filling must catch up to production. Although the surface paste system will be capable of pouring all daily fill requirements in one shift, workforce has been applied to the task on both shifts in case this is required.

Snowden believes that sufficient workforce, equipment, and costs have been applied to the filling system but recommends that additional fill system design be completed before the implementation of the project. A more detailed design, which consolidates all of the operational needs of the system, is required.

23.1.10 MINE VENTILATION AND ESCAPEWAYS

VENTILATION SYSTEM

Snowden has designed the mine ventilation for the Wolverine operation to accommodate all phases of the mine life. A factor of 100 cfm (2.8 m^3/min) per brake horsepower and the equipment list provided in Table 23.5 has been used to determine the total ventilation requirements for the operation. Based on this, it is estimated that 185 m^3/sec of ventilating air will be required for the operation.

The main ventilation fan will be installed on surface on top of an exhaust raise, acting to suck air through the development tunnels. Mine Ventilation Services, Inc. (MVS) of Fresno, California, was engaged during the Hatch study to estimate an appropriate size of fan for ventilation requirements. They were also engaged during the current study, to re-estimate system pressures given the new system geometry. The results suggested that the originally specified fan would be sufficient for LOM ventilation requirements. The fan as specified and quoted is an Alphair 11200-AMF-6600 Full bladed fan with an 800 hp, 3 Phase motor at 4.16 kV operating voltage. A variable frequency drive will also be purchased for the fan, to maximize the efficiency of the power usage over the LOM and as the ventilation network changes. It has been assumed that a silencer will be mounted on this exhaust fan. MVS's reports are provided in Appendix B.4 of Wardrop's Optimized Feasibility Study for reference.

The main access ramp will act as the fresh air conduit, as fresh air will flow down the main ramp and back up to surface within the exhaust raise(s). The main components of the ventilation system are:

- An exhaust raise which is comprised of five legs of raise, developed as the decline advances downwards. The initial raise (1280 return air raise (RAR)) will be a 4 m diameter, bored raise from surface to a location at approximately elevation 1277, and will be excavated during the pre-production development period. Subsequent lengths of ventilation raise will be developed by using a raise bore and will be comprised of two parallel x 3 m diameter raises. Access to each of these raise sections will be from short sections of development tunnel just off the East side of the main access ramp. Regulators with doors allowing the access of men and equipment will be installed in between the ramp and ventilation raise to allow adjustment of the volumes of air, which flow from the ramp into the exhaust raise.
- The main access ramp will act as the main fresh air source for the mine, with air flowing down the ramp and into the exhaust raise through the regulators described above.
- An isolation door and short (40 m long) section of 4 m diameter raise bore will be required at the top of the main access ramp so that propane fired heaters can be integrated into the ventilation system and so that temperatures in the main ramp are maintained above freezing. This 40 m long ventilation raise (1340 Fresh Air Raise (FAR)) will be excavated during the pre-production development period. The propane system will be capable of heating the air to above freezing

temperatures on days of extreme cold temperatures. MVS provided estimates for the amount of propane required to heat the required mine ventilation from -75°C to +3°C.

- A second raise, which is also comprised of five legs of raise and also developed as the decline advances downwards. The topmost section of the raise (1300 FAR) will be excavated during the pre-production development period. This raise will act as a small supplemental fresh air source for the mine and will be equipped with ladders and landings for the purposes of egress from the mine in case of emergency. A second main ventilating fan and propane heating system will be installed at the top of this raise so that positive pressure (and hence assured fresh air) is maintained in this raise at all times. Accesses into each of these raise sections will be provided by short lengths of development tunnel just off the West side of the main access ramp. In the case of a mine fire, the accesses into this raise will always be available for safe refuge in fresh air. Regulators equipped with man-doors will be installed at each of the accesses into this raise to control the amount of air that flow out onto the ramp at each.
- Auxiliary ventilation will be required into each of the stopes and diamond drill drifts. It has been assumed that each stope and diamond drill drift will require the use of a 48" x 125 hp high pressure vane axial fan capable of delivering 25 m^3/sec or sufficient air for the following list of equipment to a maximum distance of 350 m from the main access ramp:
  - one 3.8 m^3 capacity LHD
  - one electric hydraulic drill jumbo
  - one electric hydraulic rock-bolting unit
  - one Toyota jeep.

During operations, it is more likely that auxiliary fans with different sized motors will be required, based on the distance that the ventilating air needs to be pushed. In all cases, it is expected that 48" (1.2 m) diameter ventilation tubing will be required through the stope accesses and part way into the stope headings.

Wardrop's Optimized Feasibility Study provides two drawings for a view of the ventilation system. Dwg. MO-X-009 shows the status of the mine development and ventilation system as of the end of the pre-production development period. Dwg. MO-X-011 shows the ventilation system and expected flows as of the end of Year 2012.

ESCAPEWAY SYSTEMS

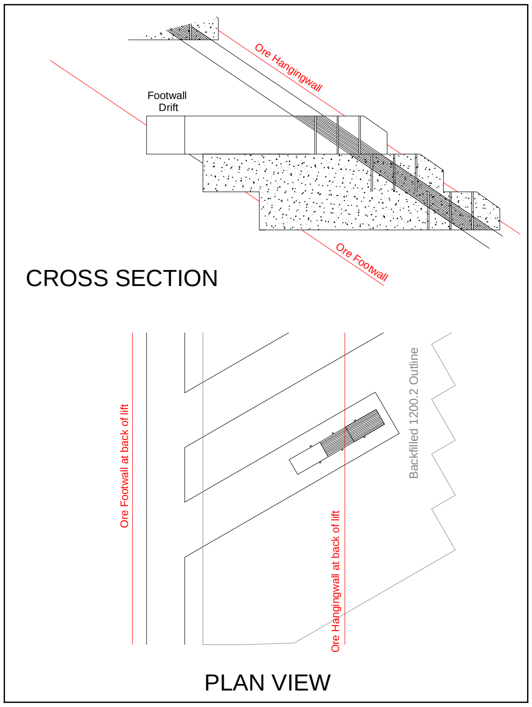
Two separate escapeway systems will be provided in the Wolverine Mine. One is an integral part of the ventilation system as described above and provides escapeway from the mine to surface in the event of an emergency. The second escapeway system is provided in the working stopes as described in the Yukon mines legislation.

The main escapeway system to surface will be permanently maintained as a positively pressurized fresh air system during the LOM. A fresh air fan and propane heating system will be installed at the top of this raise, with accesses into the raise every other spiral of the main access ramp. Regulators with man-doors will be installed at the accesses from the main ramp. Five accesses will be provided into this escapeway system along the length of the main ramp. In the event of a mine fire, personnel will be able to seek refuge in any of these accesses. It will be possible to seal the regulators should it be necessary and equipment for refuge stations will be provided in each.

The second escapeway system is intended to provide secondary egress from any of the stopes in the event of a cave-in or fire in the stope. These will be in the form of conventional raises developed in the thickest part of the ore with ladders installed. Escape will be possible either up to the stope above, or downwards to the stope below. Occasionally, this escape route will be disrupted by the mining process, when the last stages of filling on a stope level need to be completed. Occasional alterations will be required to the production schedule to ensure that if the escape route above a stope is disrupted, the escape route below that stope is maintained. It is estimated that each disruption will be for a period of approximately three months, so that at any one time in the life of the operation, one level of the escape route is impassable. It is for this reason that a more permanent escape route to surface is required adjacent to the main ramp.

A wood or steel framework will need to be constructed in the escapeway panel to connect the raise from below to the raise above a lift during the filling process. This framework will need to be sealed to prevent fill infiltration into the escapeway raise. It is important to make the construction only the necessary width of the raise itself and not to the width of the existing mining panel. During filling, the voids around the framework will be filled, allowing mining of the adjacent panels to occur while not damaging the framework so that the integrity of the raise is maintained. Figure 23.4 provides a potential framework geometry, which can be used to isolate the escapeway during filling.

Figure 23.4 Stope Escapeway Raise Framework



If necessary, the raise can be re-accessed at the current lift elevation through the adjacent secondary panels by mucking through a short section of fill and breaking into the raise.

23.1.11 MINE SERVICES

MINE POWER

Tom Dietrich, of Dietrich Consulting Ltd. was engaged by YZC to provide specification and an estimate of costs associated with the underground distribution system. Mr. Dietrich's report is provided in Appendix B.5 of Wardrop's Optimized Feasibility Study for reference. This has been used to develop transformer and line locations for the study.

Power used for underground operations will be provided from the surface generators located near the mill buildings. 4.16 kV power will be generated by the surface generators and delivered to the mine through two separated feeders. Both of the feeders will run overland to the top of the 1340 FAR where one of the feeders will be run underground (down the raise) for the delivery of power. The other feeder (surface feeder) will continue overland to the top of the main exhaust raise (1280 RAR) and then farther to the top of the secondary escape raise (1300 FAR).

The surface feeder will be used to provide 4.16 kV power for the equipment shown in Table 23.7.

Table 23.7 Mine Operations Surface Power System Requirements

Description	Unit Power (hp)	Unit Power (kW)	Steady-State Units	Operation (h/d)
Main Exhaust Fan	800	597	1	24
Secondary Escapeway Fan	75	56	1	24
Air Compressor	125	93	1	5
Total Connected Load	-	0.8	MW	-
Average Load	-	0.7	MW	-

The second feeder (underground feeder) will be run underground down the 1340 FAR and along the main access decline. Power used underground will be transformed downwards to 600 V as typical for the industry.

Table 23.8 shows the required 600 V power usage and will be supplied by the underground feeder.

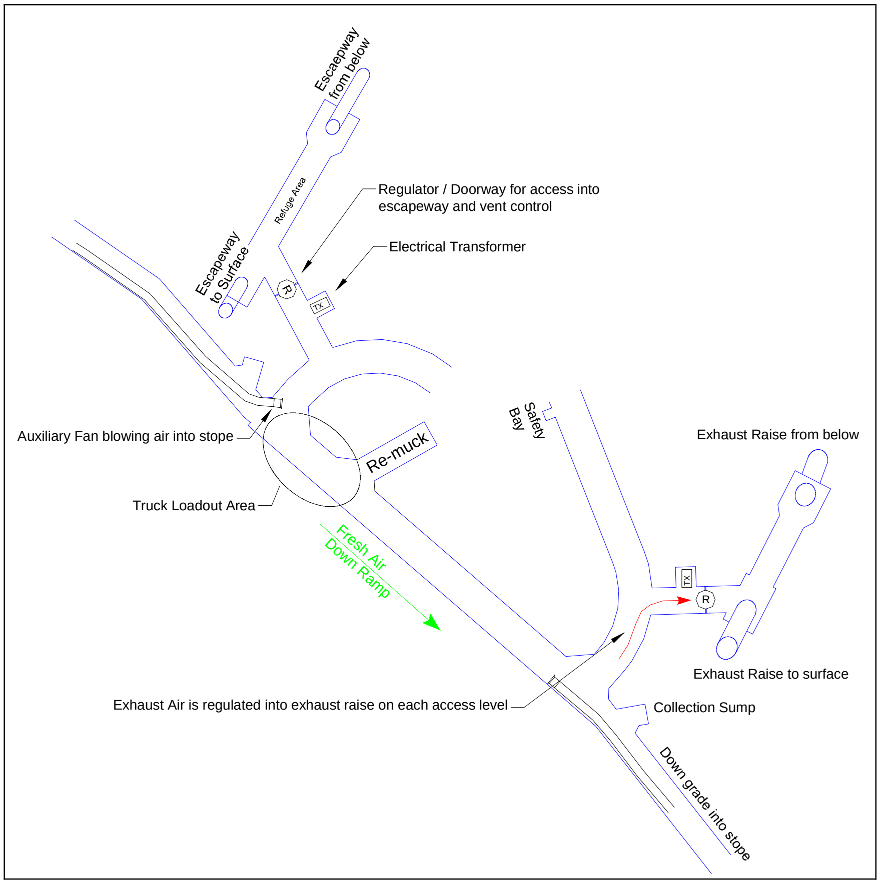
Two underground transformers (portable power centres) will initially be required within the distribution system to transform the feeder voltage (4.16 kV) to the useable system voltage (600 V). The useable system voltage can be delivered to equipment located approximately 300 m away from the power centres, and eventually the system will be stretched to its limits so that a third underground power centre will be purchased and installed underground.

Table 23.8 Mine Operations Underground Power System Requirements

Description	Unit Power (hp)	Unit Power (kW)	Steady-State Units	Operation (h/d)
Jumbo drill	145	108	2	14
Rock-bolter	80	60	2	14
Diamond drills	50	37	1	9
Auxiliary fans	125	93	10	17
Auxiliary fans	150	112	2	12
Main dewatering pumps	100	75	2	6
Secondary pumps	5	4	10	6
Total connected load	-	1.7	MW	-
Average load	-	1.0	MW	-

During the pre-production development period, the two initial power centres will be installed underground, one at the bottom of the secondary escapeway raise (1300 AR), and one at the bottom of the main exhaust raise (1280 RAR) in a small cut-out to the side of the access tunnel. Both will be located in front of the ventilation regulators required for the adjustment of ventilation flows. Figure 23.5 shows a typical spiral of the ramp with the proposed transformer locations.

Figure 23.5 Typical Ramp Spiral with Services



The portable power centres will be connected to the 4.16 kV feeder at the nearest junction point (box) via #4/0 AWG 5 kV armoured cable. Each portable power centre will incorporate 750 kVA, 4.16 kV to 600 V dry type transformers, 600 V distribution panels, 600 V power receptacles and 120 V lighting panels. 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 centres will service mining equipment directly (when in close proximity) using the 600 V power receptacles or will service 600 V portable distribution centres when mining loads are more remote. Both the power centres and/or portable distribution centres will be located in excavations sufficient to provide safety and access while also providing protection.

COMMUNICATIONS

An underground communications system will be used at the Wolverine Mine. This system will be provided using an industrial Ethernet cable and voice over internet protocol (VOIP) communications system. This system will be installed during the pre-production development period, and will be extended and maintained over the life of the operation. The system will be connected to surface for direct communication with various parties, including the offices and emergency response area in the event of an emergency.

COMPRESSED AIR

A limited amount of compressed air will be required underground on an average basis. The use of pneumatic face pumps for dewatering and the use of ANFO loaders for the charging of drill holes will be the primary requirements. There will be an occasional requirement to use stoper drills and/or jack-leg drills for the installation of longer rock support and for the drilling of safety bays and/or conventional raises.

It is estimated that on a typical basis, 400 cfm of compressed air delivered to underground operations at a minimum of 100 psi will be required.

Additionally, there will be occasional need for significantly increased compressed air underground when raise bore units are being used for the development of ventilation and secondary escapeway raises. During portions of this effort, at least 1,000 cfm of compressed air will be required.

A small compressor capable of delivering the typical compressed air supply will be purchased and installed on surface around the mill buildings. This compressor will be 125 hp and will be capable of providing 400 cfm to the underground operations. A 150 mm diameter, schedule 40 steel pipe will be run overland from this compressor to the top of the 1340 FAR, where it will then be run underground and along the main access ramp. From the main access ramp, 50 mm diameter schedule 40 steel pipe will be run into the stopes for compressed air usage there.

The underground compressed air system will also be attached to the main surface compressed air system for times when increased needs underground are encountered. This will be an occasional requirement and Snowden understands that sufficient capacity and flexibility will be available on surface for this to be possible.

MINE WATER SUPPLY

A system capable of delivering sufficient clean water (not potable) to the underground operation will be required on surface. Most of the water required underground will be used during drilling operations and for dust control on walls and muck-piles. It is estimated that a typical usage of 150 L/min will be required.

The mine underground water system will be connected to the surface clean water system, which will be capable of delivering supply water for the underground operations. A 100 mm diameter, schedule 40 steel pipe will be run overland from the surface clean water source to the top of the 1340 FAR, where it will be run underground along the main access ramp.

At stope access intersections, a 50 mm diameter schedule 40 steel pipe will be tied in and run into the stope headings for use there.

In addition to the surface connection, a clean water (not potable) system will be developed underground to try to take advantage of the existing ground water, which would otherwise need to be pumped. Diamond drill holes will be drilled to intersect ground water bearing structures. These diamond drill holes will then be plumbed into the mine water system with an integrated in-line pump.

MINE DEWATERING

Most of the water used underground will ultimately need to be pumped to surface along with ground water that seeps from the rock mass into the excavations. It is estimated that an average of 550 L/min of water will need to be pumped from the mine. The water will need to be pumped in three phases from the source (within stopes and advancing excavations) to surface. The three phases are:

- Water will be pumped from the source to small collection sumps using pneumatic face pumps. These low head pumps are used in the dirtiest water conditions at the face to make sure that drilling operations can proceed.
- Water will be pumped from the small collection sumps to one of two main sumps. Several of these small collection sumps will be located in close proximity to the main access ramp to ensure that water is kept off the ramp. Any additional water that is created around the ramp will run by gravity to these collection sumps. Low horsepower (~3 hp) sludge pumps will pump the water collected in these sumps to one of two main sumps in the mine. Some of the solids contained with the water will be collected in these sumps, which will be periodically maintained by cleaning with an LHD.
- Water will be pumped from the main sumps to surface in a staged pumping system with the bottom sump pumping into the top sump and then the top sump pumping to surface. The bottom sump will be a dual sump system with a dirty and a clean water side with facility to clean out the collected slimes on a regular basis without affecting the operation of the pump. The bottom sump will be located at approximately the 1130 elevation with an installed 75 hp electric pump capable of pumping 130 m of elevation up to the upper main sump at elevation 1260 elevation. The upper sump will have an installed 75 hp electric pump capable of pumping 100 m of elevation up to the surface at elevation 1360, for ultimate handling on surface.

Costs for the installation and supplies of a 75 mm inside diameter HDPE pipe are included in the cost of declining.

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 protect properly each commodity.

Some consumables, such as rock-bolt 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 British Table of Distances has governed the placement of these magazines. On average, 24 hours of explosives will be stored in magazines located throughout the mine at any given time. It is not expected that permanent explosives or cap magazines will be excavated underground.

Refuge Stations

As described above, five refuge stations will be provided for the operation as part of the escapeway system, one at each of the vent accesses into the main mine escapeway. Each of these will be provided with lighting, compressed air, seating, first aid equipment, communications, and potable water. All of the refuge station accesses will have regulator style barricades at the entrance, which will be capable of sealing in the event of an emergency.

Central Blasting

A central blasting system will be installed along the length of the access decline and into the stoping areas. At the end of each shift, development and stope blasts will be hooked to this central blasting system and initiated from a location near the portal. Each underground shift will be of 11 hours duration, so that there will be a 1 hour period between shifts for blast gases to clear.

23.1.12 General Shift and Assumptions

The following general assumptions have been used to determine development and production rates for the mine.

- total shift length........................11 hours
- breaks + travel ....1.5 hours
- active task time ....9.5 hours
- effective hour ....50 minutes
- effective days per year.............336 (assumed 8% down for maintenance).

23.1.13 Stope Productivity Assumptions

Stope productivity is a very important aspect of the mining plan for the Wolverine operation. This has lead to requirements for development advance and workforce associated with most of the activities around the mine. A series of stope productivity assessment were performed to develop typical productivity rates that would be applied for the three different mining method variants. The productivity assumptions critical to this assessment are as follows:

- headings in ore can generally advance at one round per day
- in DFPS mining, a total of three primary mining panels can be in production at any one time
- in DFSS or DFRP mining, only one heading can be in production at any one time
- in DFRP mining, the next adjacent mining panel can start mining six days after the completion of filling.

Given these assumptions, Snowden concludes that typical stopes will be capable of producing approximately 300 t/d of diluted ore, depending on the average ore width for the stope and different ratios of mining method variants. This includes the time for filling. An additional 2.5 to 3 months of time is added to the start of the stope life to account for time it will take to develop the footwall drift to the middle of the stope, access the stope escapeway location, and excavate the stope escapeway up to the level above for secondary egress.

Snowden also notes that the stope productivity estimate of 300 t/d is an aggressive goal.

During exploration and test mining activities during 2005, it was determined that 2 m/d of tunnel advance could be achieved in waste headings and that a heading size of 5 m x 5 m could be achieved. These waste development assumptions were used to create the final development and production schedule.

23.1.14 PRE-PRODUCTION PERIOD

A pre-production period will be required during which time services will be installed and production stopes are established in preparation of full-scale steady state production. It is expected that a mining contractor will be engaged to perform this work with use of the contractor's equipment, personnel, and supervision. Procon Mining and Tunnelling (Procon), of Vancouver, British Columbia, has been involved with the project since inception and has provided an estimate of costs and time associated with the pre-production activities.

The pre-production period will last from July 2007 to the end of September 2008, during which time the mine will be prepared for full operating status. This will include approximately 2,200 m of lateral development plus 310 m of raise development and will include the following work:

- rehabilitate the existing workings, including: replacing the roadbed with segregated aggregate, and rehabilitating any of the existing development should it be required
- install a sealing door near the top of the decline for ventilation control purposes

- install a temporary heating facility in front of the auxiliary ventilation system to prevent underground freezing during pre-production activities only
- establish access to seven active ore production faces on five mining horizons
- advance three in-stope footwall drifts and excavate two in-stope escape-ways in preparation for production
- provide additional development for ventilation distribution and emergency egress, including: intake, exhaust, and egress raising from surface
- install and commission several mining facilities and systems including power distribution, communications, ventilation, emergency egress, compressed air, water supply and dewatering
- install a paste-fill line into the mine
- begin mining and commissioning of the mill systems and ramp-up to 80% of full production
- perform a surface diamond drilling program aimed at defining the ore in the area of the decline development and improving the confidence of resources in the upper portions of the deposit.

Table 23.9 provides a schedule of development activities required during pre-production, which will prepare the mine for full-scale production.

Table 23.9 Pre-production Development Schedule

	2007						2008									
	Jul	Aug	Sep	Oct	Nov	Dec	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Total
Ramping	-	42	50	50	38	97	42	52	53	91	58	32	44	84	10	743
Stope Access	-	-	-	-	10	10	60	-	30	20	58	78	61	15	40	382
Vent Drift	10	5	-	15	28	-	10	58	18	-	-	15	40	31	47	262
In-Stope Development	-	-	-	-	-	-	-	-	-	-	-	-	150	225	362	737
Diamond Drill Drift	-	-	-	-	-	-	-	-	-	-	-	-	-	14	40	54
Total Jumbo Development	10	47	50	65	76	107	112	110	101	111	116	125	295	355	459	2,193
Vent Raises	-	-	-	-	-	-	-	-	40	110	-	-	-	-	-	150
Main Escapeways	-	-	-	-	-	-	-	-	-	85	-	-	-	-	-	85
In-Stope Escapeways	-	-	-	-	-	-	-	-	-	-	-	-	-	-	74	74
Total Raising	-	-	-	-	-	-	-	-	40	195	-	-	-	-	74	309

It is important that the surface diamond-drilling program begin before the start of underground development so that information from the drilling can be incorporated into the design of the decline. It has been assumed that the surface drilling program is started in March 2007 and that 6,325 m of drilling is performed before December 2007.

LOM Operations and Production Schedules

Tables 23.10 to 23.13 provide schedules of operations and production activities that occur over the LOM.

Some items of note within the production schedules are as follows:

- Mine development is completed as of year 2013. This allows the diamond drill drifts and ventilation system to be in place for all stopes, even though some do not being production until 2015.
- Stoping on levels 1320 and 1340 is tied to stoping on level 1300 and occurs afterwards.
- All mining in year 2017 is tied to the barrier pillar retreat around the access decline. In that final year, 80% of the Measured and Indicated resources (approximately 200,000 tonnes) around the decline and stope accesses will be mined in a retreat fashion. Snowden estimates that an additional 600,000 tonnes of Inferred ore will remain in place around the decline and stope accesses as permanent pillars.
- An estimated 100,000 tonnes of Inferred resources will be mined within the decline or stope accesses and hauled to surface. It is unknown if this will eventually prove to be economic. If not, this material will be hauled back underground and placed in stopes as loose fill. No revenue has been assumed from this material.

Table 23.10 LOM Development Schedule

	Pre-Prod	4Q 2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	Total
Ramping	743	60	313	270	297	265	140	-	-	-	-	2,088
Stope Accesses	382	60	192	208	121	197	201	-	-	-	-	1,361
Vent Accesses	277	27	34	64	92	44	-	-	-	-	-	538
Drill Drifts	54	269	381		300	393	-	-	-	-	-	1,397
In-Stope Development	737	-	-	-	-	-	-	-	-	-	-	737
All Lateral Development	2,193	416	920	542	810	899	341	-	-	-	-	6,121
Main Vent Raising*	150	80	-	86	82	-	80	-	-	-	-	478
Main Escapeway Raising*	85	40	-	40	40	-	40	-	-	-	-	245
In-Stope Escapeway Raises	74	56	56	28	56	112	74	56	56	-	-	568
All Raising	309	176	56	154	178	112	194	56	56	-	-	1,291

Note: * All main vent raises and main escapeways that will be developed with a raise bore unit have been capitalized. Costs associated with the development of these items have been included in sustaining capital.

Table 23.11 Stope Production Schedule

Stope/Level	Pre-Prod	4Q 2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	Total
1340	-	-	-	-	1,784	-	-	-	-	-	-	1,784
1320	-	-	-	-	68,913	-	-	-	-	-	-	68,913
1300	15,236	33,856	134,320	134,320	9,394	-	-	-	-	-	19,912	347,038
1280	15,236	25,869	102,631	102,631	95,258	13,482	-	-	-	-	34,665	389,771
1270	10,102	25,923	1,565	-	-	-	-	-	-	-	19,387	56,977
1260	4,968	16,102	99,370	100,397	88,397	13,803	-	-	-	-	34,817	357,854
1250	4,968	15,629	76,422	-	-	-	-	-	-	-	6,250	103,269
1240	-	8,182	98,068	102,169	72,924	23,299	-	-	-	-	13,409	318,051
1230	-	2,000	68,465	44,117	-	-	-	-	-	-	-	114,582
1220	-	-	26,957	92,081	106,338	56,763	-	-	-	-	7,209	289,348
1210	-	-	-	37,389	104,359	43,301	-	-	-	-	9,201	194,250
1200	-	-	-	-	52,295	106,793	106,793	18,201	-	-	-	284,083
1190	-	-	-	-	16,134	100,041	79,165	-	-	-	11,958	207,298
1180	-	-	-	-	-	95,517	106,555	106,555	11,700	-	2,357	322,683
1170	-	-	-	-	-	68,075	94,817	37,469	9,999	-	20,423	230,783
1160	-	-	-	-	-	58,041	103,702	103,702	22,584	-	4,326	292,354
1150	-	-	-	-	-	38,153	63,497	63,913	50,097	25,000	4,573	245,233
1140	-	-	-	-	-	-	55,000	79,947	39,890	25,000	2,457	202,294
1130	-	-	-	-	-	-	-	25,000	108,535	105,000	1,714	240,249
1120	-	-	-	-	-	-	-	-	91,872	51,260	-	143,132
1110	-	-	-	-	-	-	-	75,000	107,665	91,086	-	273,751
1100	-	-	-	-	-	-	-	-	69,819	35,379	-	105,198
1090	-	-	-	-	-	-	-	-	-	6,744	-	6,744
1080	-	-	-	-	-	-	10,000	108,398	108,397	67,707	-	294,502
Total	50,510	127,560	607,798	613,103	615,797	617,268	619,529	618,185	620,558	407,176	192,658	5,090,142

Table 23.12 Ore Production Schedule – All Sources

Item / Activity	Pre-Prod	4Q 2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	Total
Development ore	34,249	345	5,491	7,060	3,250	2,566	686	-	-	-	-	53,648
Raising ore	876	730	752	395	763	1,474	873	773	689	-	-	7,325
Stoping ore	50,510	127,560	607,798	613,103	615,797	617,268	619,529	618,185	620,558	407,176	192,658	5,090,142
All diluted ore mined	85,636	128,636	614,041	620,558	619,847	621,349	621,088	618,958	621,247	407,176	192,658	5,151,193
Included dilution	23,373	32,315	151,298	148,078	152,332	135,489	130,672	116,456	122,865	80,509	46,738	1,140,126
% Zn	8.11	8.80	9.20	9.29	9.46	10.18	10.20	9.85	9.91	9.81	9.23	9.66
% Cu	1.13	0.91	0.87	0.91	0.83	0.90	0.92	0.96	0.96	0.93	0.89	0.91
% Pb	0.96	1.24	1.38	1.39	1.33	1.20	1.14	1.13	1.25	1.28	1.24	1.26
Ag (g/t)	223.9	299.1	318.2	296.7	284.2	265.0	256.8	246.3	292.9	302.7	293.5	281.8
Au (g/t)	1.2	1.6	1.7	1.5	1.4	1.3	1.2	1.1	1.2	1.3	1.6	1.4

Table 23.13 Miscellaneous Activities Schedule

Item / Activity	Pre-Prod	4Q 2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	Total
Inferred ore mined in development	10,648	3,482	20,908	11,548	19,037	16,706	19,782	-	-	-	-	102,111
Development waste mined	59,792	20,653	34,471	18,927	31,697	38,621	4,914	-	-	-	-	209,076
Raising waste mined	6,327	2,032	-	2,157	2,073	-	2,032	-	-	-	-	14,621
Incidental in-stope waste mined	1,768	4,465	21,273	21,459	21,553	21,604	21,684	21,636	21,720	14,251	6,743	178,155
All waste mined	67,887	27,150	55,744	42,542	55,323	60,226	28,630	21,636	21,720	14,251	6,743	401,852
Waste hauled to surface	67,887	27,150	29,051	8,781	14,496	14,069	5,726	4,327	4,344	2,850	-	178,682
Waste hauled from heading to stope	-	-	26,692	33,761	40,828	46,156	22,904	17,309	17,376	11,401	6,743	223,170
Waste or INF or DMS hauled back U/G	-	-	20,285	28,371	21,876	16,464	39,857	44,958	45,072	29,865	116,930	363,678
Paste backfill poured underground	-	50,095	282,289	269,591	268,298	268,216	272,058	269,834	270,129	193,868	149,430	2,293,808
Shotcrete usage (m^3)	1,085	282	750	730	798	749	602	352	357	224	107	6,036
Diamond drilling (m)	7,728	2,116	8,395	8,395	8,395	8,395	8,395	8,395	8,395	-	-	68,609

23.2 RECOVERABILITY

This Section was written by Mr. J. Fox.

For information on recovery, refer to Section 16 Mineral Processing and Metallurgical Testing.

23.3 MARKETS

This section was written by Wardrop.

Three marketing studies carried out by YZC were used as reference:

- Butterfield Mineral Consultants Ltd., Dec. 2006. Concentrate Marketing Update
- Butterfield Mineral Consultants Ltd., Dec. 2005 (rev. Jan. 2006. Wolverine Project, Concentrate Marketing
- McKnight Consulting, Dec. 2006. Comments on the Butterfield Report.

These reports are included in Appendix F of the Optimized Feasibility Study prepared by Wardrop.

23.4 CONTRACTS

This section was written by YZC.

No contracts are in place at present time.

23.5 ENVIRONMENTAL CONSIDERATIONS

This section was written by Wardrop.

Environmental management issues associated with the Yukon Zinc Corporation's (YZC) Wolverine Mine and mill are primarily associated with groundwater, aquatic resources, and wildlife. Wolverine Creek is groundwater fed from the proposed mining area while Go Creek will receive treated effluent from the tailings management facility. Baseline environmental studies therefore focused on ground- and surface water environments. Aquatic data were collected in 1995, 1996, 1997, 2001, 2005, and 2006. Wolverine, Go, and Money creeks as well as Little Wolverine and Wolverine lakes were the focus of the characterization studies but the program at times was expanded to include waterbodies along the road route and regional sites. Components characterized include hydrology (interpretation of evaporation and snowmelt data, regional and local hydrometric data, flow frequency analysis, peak and low flow analysis), water and sediment quality (total suspended solids, pH, conductivity, alkalinity, sulphate, metals, nutrients), periphyton, benthos, zooplankton, fish and fish habitat (including baseline metals in tissue analysis).

23.5.1 HYDROLOGY/HYDROGEOLOGY

The mine, mill, and support infrastructure will be constructed in a previously undisturbed headwater area of two small streams, Wolverine and Go creeks. Wolverine Creek drains into Little Wolverine Lake and ultimately to Frances Lake. Go Creek drains through Money Creek into Frances Lake. Surface water flows have been characterized through local data collection and regional climate and hydrometric data. Generally, flows peak occur during the late-May freshet and decline steadily until freeze-up.

Baseline hydraulic conductivity in the mine area was characterized using inflatable packer testing apparatus at two exploration borehole locations. Groundwater quality was characterized through analyses for pH, conductivity, alkalinity, sulphate, metals, and nutrients. Water quality results from Wolverine Creek indicate a groundwater influx into the creek from the mineralized area of the Wolverine deposit. This is indicated through elevated metal concentrations observed during baseline assessments.

23.5.2 WASTE ROCK/TAILINGS/PASTE BACKFILL CHARACTERIZATION

The waste rock and tailings are potentially acid generating. Waste rock will be disposed of underground and tailings will be disposed of both underground and underwater in a tailings management facility (TMF). Underground tailings disposal will be in the form of a paste backfill.

In order to simulate and predict the effects of paste backfill on water quality at closure, a water quality prediction model was developed which used the weathering characteristics of the six major rock types, ore materials and paste backfill established in kinetic tests with humidity cells. The weathering characteristics included rates of sulphide oxidation, neutralization potential consumptions and metal release. Based on the water quality predictions, metal concentrations may be elevated in the underground mine water and may require treatment/management. The water quality prediction is based on ‘worst-case’ assumption regarding weathering rates and mine surface areas under natural field conditions. As such, predictions made are therefore considered to be conservative. Continued monitoring of humidity cells and waters from the mine will be conducted during operations to further refine predictions of mine water quality at closure.

23.5.3 ENVIRONMENTAL MANAGEMENT

YZC’s Environmental Management practices for the prevention of adverse impacts are detailed below.

WATER MANAGEMENT

Process water and any poor quality surface water will report to the Tailings Management Facility (TMF). Water will be reclaimed from the TMF for ore processing. Water from the TMF will report to the treatment plant. Water treated in the plant will be temporarily stored in a retention pond prior to release into Go Creek. Plant effluent not achieving discharge criteria will be returned to the TMF.

The water treatment plant will use high-density sludge (HDS) technology. The resulting sludge will be disposed of in the TMF. Bench-scale water treatment plant test-work is ongoing. YZC has committed to additional water treatment processes to supplement the HDS plant. Carbon columns and/or a patented biological selenium reduction process (bioreactor) will be used, as necessary, to ensure effluent meets licence discharge standards.

WASTE ROCK/TAILINGS MANAGEMENT

The ore body is a sulphide bearing rock with acid generating potential. All waste rock will be utilized underground as loose, unconsolidated fill. Potentially acid generating waste rock brought above ground will be temporarily stored on a lined, engineered pad. Any leachate from the storage pile will be collected in a sump and treated to license discharge standards. The majority of tailings will be used as paste backfill in the underground workings while the remainder will be permanently disposed of in the TMF. The TMF will be constructed in a natural depression in on the northeast valley slope of Go Creek and will be lined with a 20 mil geomembrane. At the end of mine life the tailings dam will store 0.88 Mm^3 of material. The closure spillway will have a 10,000 year return flood period event capacity. A leachate analysis of the proposed dam borrow materials concluded that leachate issues are not expected with the material. Geotechnical investigations were undertaken to confirm the stability of the location and the TMF has been designed to a 10,000 year return period with respect to seismic activity.

SEWAGE AND GREY WATER

A pre-packaged wastewater treatment plant will be located at the industrial complex. All grey and black water will be collected in an in-ground concrete surge tank and directed to the sewage treatment plant. Treated wastewater and digested sludge will be disposed of in the TMF.

WILDLIFE

Contact with, and effects on wildlife will be managed through the Wildlife Protection Plan. The Plan will require the approval of the Government of Yukon. Elements of the plan include the management of point source attractants, exclusion from the operations through fencing, firearm bans, limiting access to the roads, monitoring and personnel training.

The route for the access road was selected to minimize effects on the Finlayson caribou herd. The road will be gated to control public access.

Further details are available in the following documents previously submitted during the Wolverine Project environmental assessment and permitting phases:

- Environmental Assessment Report – October 2005
- Environmental Assessment Report – Response to Public and Regulatory Reviews – February 2006
- Environmental Assessment Report – Addendum Report – March 2006
- Environmental Assessment Report – Memorandums to Executive Council Office – May and June 2006
- General Site Plan (version 2006-02), November 2006
- Reclamation and Closure Plan for Final Closure, Advanced Exploration Phase – October 2006
- Phase 2 All Weather Access Road Plan (Yukon Engineering Services) – January 2007-02-21
- Revised Documentation in Support of Water Use Application Q204-065 (January 2007) including:
  - Tailings Facility Design And Construction (Klohn Crippen Berger)
  - Underground Mine Water quality at Closure (AMEC Earth and Environmental)
  - Water Quality and Water Balance Models (Lorax Environmental Services Ltd.)
  - Mine Development (Snowden Mining Industry Consultants Ltd.)
  - Mill Operation and Processing (Wardrop)
  - Hazardous Materials and Spill Contingency Plans (Lorax)
  - Decommissioning and Monitoring Plans (Lorax).

23.5.4 CLOSURE

The Responsible Authorities viewed the decommissioning and closure plans submitted with the Environmental Assessment Report (EAR) as conceptually and economically feasible. The development of a detailed decommissioning and reclamation plan is a regulatory requirement under the Quartz Mining License (QML) and the Water Use License (WUL).

Poor quality groundwater as a result of contact with the exposed mine walls will be intercepted prior to seepage into Wolverine Creek by a bio-pass passive groundwater remediation system down gradient of the mine workings. After closure, when the mine has flooded and anoxic conditions have developed, groundwater quality is expected to improve and return to near pre-mining conditions.

Treatment of TMF water will continue until discharge attains a compliant level. This is estimated by YZC to be three years following the cessation of tailings discharge into the TMF. Tailings will be contained beneath a 1.0 m neutral cap of DMS material and 0.5 m of water. Diversion ditches will be decommissioned in order to maintain a permanent water cover over the tailings.

Final decommissioning and closure will be split into two phases and is anticipated to take place over four years. The first phase, decommissioning, will commence following the last year of operation. Activities will include:

- Overall Environmental Site Assessment – to identify and clean-up any potential spills
- Mine Workings – install hydraulic plugs, seal openings, and allow workings to flood
- Portal Area – remove office and maintenance buildings, and decommission sumps
- decommission and reclaim waste rock pad
- decommission and reclaim any unused roads and trails
- recontour and revegetate borrow pits.

The second phase will last an anticipated three years and will consist of remediation of sites identified in the first phase, monitoring and maintenance of reclamation measures, treatment of TMF water, and final closure and reclamation of remaining site infrastructure. Once TMF water quality has achieved discharge guidelines, the treatment plant will be decommissioned. All remaining site infrastructure will be decommissioned and removed and the site reclaimed. At the final stage, the access road will be decommissioned and reclaimed.

23.5.5 Regulatory Requirements

In November 2004, YZC triggered a screening for the Wolverine Project under the *Yukon Environmental Assessment Act* through the submission of a Type A WUL application to the Yukon Water Board (YWB) and a QML application to the department of Energy, Mines and Resources. On 21 September 2006, the Development Assessment Branch of the Yukon Executive Council Office in consultation with the Responsible Authorities (RAs) issued a Screening Report that concluded that, *"…with the implementation of mitigation measures, the project was not likely to cause significant adverse environmental effects…*". The licensing agencies will incorporate the mitigation measures identified in the Screening Report into any licences. At the request of YZC, the drafting of the QML and WUL were decoupled, with the QML (QML-0006) subsequently issued on 5 December 2006. The QML is valid for 15 years. The QML authorizes YZC to conduct initial development work, including:

- construction of the access road
- construction of the industrial complex
- construction of the borrow pits
- construction of the camp

- underground pre-production development work
- spill contingency
- wildlife and heritage protection activities.

The Type A WUL application and revised documentation has been submitted to the YWB and is currently under review to determine whether the application is complete. Once the YWB has determined that it has a complete application, a public review process will proceed. The review process will require a public hearing. However, if no party expresses an interest in attending the hearing by the intent date, the YWB may cancel the hearing. The earliest anticipated date for the issuance of the WUL is September 2007.

YZC commitments and mitigation measures identified in the Screening Report, including the development and implementation of a Follow-up Program, have been incorporated into the QML and will be incorporated into WUL. Monitoring Program components include surface hydrology, surface water quality, hydrogeology, wildlife, and reclamation and temporary closure. As well, the project will be subject to the effluent quality and Environmental Effects Monitoring requirements prescribed in the Metal Mine Effluent Regulations of the *Fisheries Act*.

Additional permits involving industrial operations will also be required but these are administrative in nature and any impact assessment/mitigation matters that may be associated with these have been addressed in the EAR.

23.5.6 FINANCIAL ASSURANCE

As a condition of the QML, YZC is required to furnish and maintain a total $1,780,000 million in financial security with the Government of the Yukon. The schedule of payment is as follows:

- $195,000 within 30 days of the QML Effective Date
- $585,000 no later than 1 April 2007, unless there are delays in the starting of the access road construction
- $1,000,000 no later than 1 June 2007, unless there are delays in starting the construction of the industrial complex.

This total may be adjusted depending on the outstanding environmental liabilities. The Yukon Government currently holds an additional $755,546 for the advanced exploration phase of the project.

The YWB may require financial security in the Type A WUL. The application, amount, and schedule of payment are at the discretion of the YWB and will be outlined, if applicable, in the WUL.

23.5.7 SOCIO-ECONOMIC CONSIDERATIONS

The Environmental Assessment Report guidelines require a consideration of project effects on socio-economic conditions. Socio-economic conditions include employment opportunities, contract and business opportunities, community health, traffic interruption/safety, and maintenance of subsistence practices and traditional way of life.

An assessment of project effects on socio-economic conditions was included in the Environmental Assessment Report, focusing on effects in the communities of Ross River and Watson Lake. Whitehorse and Faro were predicted to be only peripherally affected.

COMMUNITY EFFECTS

The Environmental Assessment Report concluded that the project is not expected to result in significant adverse effects on the social and economic conditions of Ross River and Watson Lake. The direct and indirect employment opportunities offered by the project will provide positive social and economic benefits to community residents that could continue after closure, as skills could be transferred to other resource development projects. As well, since the project will be based on a fly-in/fly-out rotational shift operation, potential disruption to community life from large spontaneous influxes of workers is avoided.

In August 2005, YZC signed a Socio-economic Participation Agreement (SEPA) with the Ross River Dena Council. Though the SEPA is a privileged document between YZC and the Council, YZC has made public some aspects of the agreement. Commitments by YZC include hiring a Kaska liaison officer and priority hiring for Ross River Dena citizens and other Kaska individuals. There are also provisions to address opportunities for Kaska businesses to supply goods and services to the project. Additional provisions address social issues such as a commitment to maintain an alcohol- and drug-free camp, provide opportunities for training and scholarships, and community level support in the form of interest free loans to the Ross River Dena Council.

The RAs concluded that because the SEPA has been concluded and authorized, there should not be significant adverse socio-economic effects of the project on the community of Ross River and other communities where Kaska residents reside. The RAs also concluded that the anticipated enhanced employment is expected to benefit the social and economic conditions in the communities of Ross River and Watson Lake.

EFFECTS DUE TO ENVIRONMENTAL CHANGES

Consumptive and non-consumptive use of resources is prevalent within the project area with subsistence hunting and trapping being socially and economically important for the Kaska. Hunting, trapping, fishing, and outfitting activities could potentially be affected by the project. Mitigation measures designed to protect water, fish, and wildlife were outlined in the Environmental Assessment Report and the Screening Report. In the Screening Report, the RAs concluded the mitigation measures were robust enough to protect water, fish, and wildlife within the scope of the project area such that "…*the prevalent economic use of the area for subsistence and commercial harvesting will be protected and the social benefits of these activities will continue, and there will not be a likely significant adverse environmental effect*."

HERITAGE RESOURCES

Though there are known culturally significant historical resources within the scope of the project, the RAs agreed with YZC that no known physical and cultural values will be directly affected by this project.

23.6 TAXES

This section was written by YZC.

23.6.1 ATNA ROYALTY

The project is 100% owned by YZC subject to a sliding scale royalty payable to Atna on that portion of the NSR attributed to payable silver and gold, that is, net of the estimated precious metals share of smelter treatment and refining charges and concentrate transport costs. The royalty rate is zero at silver prices less than US$5/oz, 4%, if silver prices are between US$5.00 and US$7.50/oz, and 10% at silver price over US$7.50/oz (see Table 23.14). At the base case silver price of $7.00/oz, the royalty rate is 4%.

Table 23.14 Atna Royalty Rate

Silver Price	Nominal Royalty Rate
< US$5.00/oz	Nil
US$5.00/oz to $7.50/oz	4.0%
> US$7.50	10%

23.6.2 OTHER ROYALTIES

A royalty of 0.5% of NSR is payable to Equity Engineering covering the entire deposit area, to a maximum of $500,000. A second royalty is payable to Nordac Resources (now Strategic Metals) on a single claim that covers approximately one-half of the Wolverine deposit – the eastern lobe of the deposit. The Nordac Royalty rate is 1.0% of NSRs attributable to ores from that claim, reducing to 0.5% after cumulative payments of $500,000 have been made. YZC anticipates that both these royalties will be acquired prior to project commencement.

23.6.3 INCOME TAXES

The economic analysis has been done on an after-tax basis. Income taxes as applied to Canadian mining operations are generally levied at three levels: Federal and Provincial income taxes and Provincial or Territorial mining taxes. Each level of tax requires different calculations and may require separate tax depreciation pools (called Capital Cost Allowances in Canada). In the Yukon Territory, the territorial income tax rate is applied to the taxable income as calculated for Federal purposes.

The Wolverine Project, as with other new mining projects, attracts no corporate income taxes until mine development capital has been fully repaid. This is due to the 100% write-off of development capital under Canadian tax rules. Mining taxes however will be payable to the Yukon government.

23.6.4 LARGE CORPORATION CAPITAL TAX

In Canada, there are capital taxes applied to large corporations, called the Large Corporation Capital Tax (LCT tax), defined as those concerns with taxable capital greater than $50 million. Taxable capital essentially consists of retained earnings, shareholder equity, loans and advances, contingent and inventory reserves.

The LCT tax is being phased out by 2007. The rate for 2006 is 0.0625% for 2006 on taxable capital in excess of $50 million. The 4% Federal income surtax (see below) can be credited against the amount of LCT tax payable.

23.6.5 FEDERAL AND YUKON INCOME TAX, AND YUKON MINING TAX RATES

The current Federal corporate income tax rate for 2007 and subsequent years is 21% under a mandated phased tax reduction. The Yukon income tax rate is 15%. Also, a sliding scale Yukon mining royalty is applied to taxable mining income (as defined). This mining royalty starts at 3% and increases by 1% for each $5 million of taxable mining income.

There is also a Federal surtax of 4% of taxable income that can be credited against LCT tax payable. Changes in the taxation system will also phase out the previous "resource allowance" deduction while phasing in deductibility of crown mineral royalties (i.e., Yukon mineral royalties).

Table 23.15 Tax Rates

Tax	% of Federal Taxable Income (FTI)
Federal	21% of FTI
Yukon	15% of FTI
Yukon Mining Royalty	3 to 15+% sliding scale based on taxable income, as defined for royalty purposes

23.6.6 FEDERAL INCOME TAX CALCULATION

Table 23.16 outlines the general procedures for the calculation of Federal and Yukon income taxes.

Table 23.16 Summary of Federal/Yukon Income Tax Calculation Procedure

	Net Smelter Return*
Less	Mine operating costs Royalties Mining taxes payable Capital cost allowances - Class 41a 100% for new mine - Class 41b Interest expense Canadian Development Expense (100%) Canadian Exploration Expense (30% DB)
=	Taxable income before losses
+/-	Tax losses applied
=	Federal Taxable Income - Federal Income Tax: 21% of Federal Taxable Income - Federal Surtax: 4% of Basic Tax Payable (deductible from any LCT tax payable) - Yukon Tax Payable: 155 of Federal Taxable Income

Note: ***gross revenues less smelter deductions and concentrate transportation costs

23.6.7 CAPITAL COST ALLOWANCE

Depreciation of fixed assets for tax purposes is accomplished by allocating the cost of the assets to the appropriate class of tax pool, depending on the type of equipment purchased. Tax pools are depreciated on a declining balance basis at a percentage rate that is assigned to the type of pool. Tax depreciation is called capital cost allowance (CCA). The CCA may have little relationship to the accounting depreciation for the same asset. The CCA in the first year of service for a new asset is 50% of the normal rate.

23.6.8 CLASS 41a CCA

Most new mining and processing equipment purchased for a new mine (or significant expansion of existing mining operation) falls into Class 41a CCA. This class has a maximum write-off rate of 100% and can be used to create a tax loss if this is advantageous. A lesser write-off rate is also allowed.

Any remaining tax pools after mine closure can be written-off.

23.6.9 CLASS 41b CCA

Replacement equipment during mining operations will generally be included as Class 41b assets.

The maximum CCA rate is 25%.

23.6.10 RESOURCE ALLOWANCE

The former 25% resource allowance deduction is phased out as of 2007 and replaced by the deductibility of mining taxes.

23.6.11 CANADIAN EXPLORATION EXPENSE (CEE)

This includes expenses used to determine the existence, location, or quality of a mineral resource. A portion of new mine development capital will fall into CEE. This will generally include mine development capital that is not equipment or buildings such as stripping, site clearing, and shaft sinking. Up to 100% of the available CEE can deducted for tax purposes however, CEE cannot be used to create or extend a tax loss.

23.6.12 CANADIAN DEVELOPMENT EXPENSE (CDE)

The category includes the costs of property acquisition expenditures and certain mine development costs incurred after mine start-up. CDE can be deducted for tax purposes at a rate of up to 30% declining balance and can create a tax loss if this is advantageous.

23.6.13 NON-CAPITAL TAX LOSSES

Prior year non-capital (operating) tax losses can be carried forward for up to seven years and back for three (3) years.

23.6.14 INVESTMENT TAX CREDIT

A 10% non-refundable investment tax credit is earned for certain “pre-production mining expenditures,” defined as pre-production exploration expense and mine pre-production development expenses. This Study has assumed that this essentially means all non-depreciable pre-production capital costs are included. The tax credit is assumed to reduce the CEE pool when it is claimed.

23.6.15 YZC CORPORATION – CURRENT TAX POOLS

YZC’s opening Federal tax pools have been used to reduce future income taxes payable. The pool amounts were estimated by YZC as of as of 31 December 2005.

Table 23.17 Opening Income Tax Pools

Tax Category	Cdn$ 000s
CCA	-
CEE	17,428
CDE	7,093
Tax Losses (expiring after 2007)	1,901

23.6.16 YUKON MINING TAXES

Yukon mines are taxed under the Quartz Mining Act, legislation that originally dates from the 1800s. Under this legislation, mining taxes in the Yukon are based on gross mining profit (NSR less operating costs) but the taxable basis differs from the Federal calculations. Specifically, the annual depreciation for Yukon purposes is calculated as 15% on a straight-line basis, royalties and interest expense are non-deductible; however, Federal and Yukon corporate income taxes paid are deductible.

23.6.17 DEPRECIATION

Yukon mining taxes make an allowance for annual depreciation of the plant, machinery, equipment, and building of the mine. The allowance may not exceed 15% of the value of such assets at the commencement of the year (i.e., 15% straight line).

23.6.18 EXPLORATION AND DEVELOPMENT EXPENSES

Expenses incurred in sinking new shafts, creating new mine openings, workings, or excavations and stripping, trenching or drilling in respect of the property on which the mine is situated are allowable deductions. Any such expenses incurred in respect of any other land in the Yukon Territory belonging to the mine operator are also deductible. If exploration and pre-production costs are not deducted against the income of another mine of the operator in the Yukon in the year incurred, they cannot be deducted in a later year against the income of the mine.

23.6.19 PROCESSING ALLOWANCE

The “processing allowance” is a special allowance to reflect for the cost of mineral extraction between the mine mouth and the production of saleable product, i.e., metal concentrates. The availability of this deduction is subject to “ministerial discretion.” Based on discussions with the Yukon Government this study assumes a basic 15% annual deduction for processing assets limited to no less than 15% of this income before this deduction and no greater than 65% of Income before the deduction. This is similar to some other provincial jurisdictions. No processing allowance was assumed due to the discretionary nature of this deduction.

23.6.20 MINING TAX RATE

A sliding scale tax rate is applied to the taxable income for Yukon mining tax purposes (see Table 23.18). The mining tax payable for a year is due in October of the following year.

Table 23.18 Yukon Mining Royalty Rate Schedule

Annual Taxable Basis (for mining royalty)	Marginal Mining Tax Rate
Under $1 million	3%
$1 to $5 million	5%
$5 to 10 million	6%
$10 to $15 million	7%
On the excess above $15 million	1% increase in rate for each $5 million

23.7 CAPITAL AND OPERATING COSTS

This section was written by Wardrop.

23.7.1 CAPITAL COST

Refer to Table 23.19. Wardrop has based the estimate on the process flowsheets, layouts, and equipment list prepared for the Optimized Feasibility Study. The estimate is based on the 4th Qtr. 2006 costs. Wardrop has relied on the third party estimate for some sections in the capital and operating cost estimates.

Table 23.19 Capital Cost Breakdown Summary

Direct Cost	(Cdn$ millions)	Indirect Cost	(Cdn$ millions)
Site preparation and roads	23.1	Engineering	8.5
Mill and process	49.6	Construction management	8.6
Power generation	0.9	Construction Indirects	11.1
Tailings and water supply and reclaim	9.0	Materials and inventory	3.2
Service services facilities	15.5	Duties and freight	4.5
Permanent camp	5.7	Commissioning	0.8
Mining	35.1	Sub-total	36.7
Sub-total	138.9		
Total Cost			175.6
Owner's Cost			7.6
Contingency			24.4
Total Project Cost			207.6

The capital cost estimate is prepared such that the estimate will be used for controlling the EPCM phase of the project. Accuracy level of the estimate is -5% to +15%. The estimate does not include cost allowance for any undefineable risks.

Other contributors to the capital cost estimate are:

- YZC...Access road
- A. Polk, P.Eng., Snowden ....Mining and mining equipment
- Klohn Crippen .... Tailings dam and ditches

QUANTITIES

All quantities developed are based on general arrangement drawings, process design criteria, process flow diagrams, and equipment lists. Design allowances are applied to bulk materials based on discussions between the respective discipline and the estimator. Details on the respective discipline quantities are as described in the following sections.

BULK EARTHWORKS

Bulk earthworks quantities are based on rough grading designs done using Autodesk Land Development Desktop Civil Package. Topsoil and rock excavation is based on the geotechnical information available at the time of the study. Structural fill is costed based on aggregates being produced at site utilizing a portable crushing and screening plant; the price of the aggregate plant is included in the capital cost estimate included in line items. Earthwork quantities do not include an allowance for bulking or compacting materials; these allowances are included in the unit prices.

CONCRETE

Concrete quantities are based on “neat” line quantities from engineering designs and sketches with a 10% allowance included for overpour and wastage.

STRUCTURAL STEEL

Steel quantities are based on quantities developed from engineering design and sketches with 5% allowance made for growth and wastage. Pre-Engineered Buildings are based on layouts and section, and vendors’ budget quotations.

MECHANICAL

Major equipment specifications were prepared and issued to qualified vendors for budgetary quotations. Quantities for all platework and metal liners for tanks, launders, pump boxes, and chutes have been calculated based on detailed quantity take-offs developed from design drawings and sketches. A 10% allowance has been included for growth and wastage.

PIPING

Piping quantities and costs are based on a percentage of the total mechanical costs for each area.

ELECTRICAL

Electrical quantities are included as part of the electrical engineer data program, which includes total electrical costs for each piece of equipment, including cables.

An equipment list was generated using single line diagrams, project drawings, and sketches based on the project WBS structure. The equipment list was used to estimate plant loading and generation requirements. The site plan and equipment list were used to locate electrical buildings while minimizing cable runs. A single line diagram was developed that indicated all major electrical equipment including generators, transformers, feeders, power distribution centers (4 kV and 480 V), motor control centres (4 kV and 480 V), requirements for VFD by process, and rectiformers.

On-site work will consist of connecting the incoming transformer feed, MCC, and outgoing motor feeders.

Equipment specifications were prepared and quotes were provided by suppliers for all major equipment. Factoring and in-house pricing was used for smaller items, as required.

INSTRUMENTATION

Instrumentation quantities/costs are based on a percentage of mechanical equipment

ASSUMPTIONS

The project will be executed as described in the Optimized Feasibility Study dated 12 January 2007 in Section 9, 12, and 13, and in Appendices C, D, G, and H.

EXCLUSIONS AND INCLUSIONS

The items excluded from the capital cost estimate are as follow:

- working or deferred capital (included in economic model)
- financing costs
- refundable taxes and duties
- land acquisition
- currency fluctuations
- lost time due to severe weather conditions
- lost time due to force majeure

- additional costs for accelerated or decelerated deliveries of equipment, materials, and services resulting from a change in project schedule
- soil and subsurface conditions
- mine reclamation costs (included in economic model)
- mine closure costs (included in economic model)
- cost escalation post 4th Qtr. 2006
- community relations and permits (included in working capital)
- Owner's risk and opportunity assessment
- possible salvage values (included in economic model).

CONTINGENCY

A contingency of 13.3% is included in the total capital cost.

23.7.2 OPERATING COSTS

The May 2006, Hatch Associates' (Hatch) feasibility study process operating cost estimate was reviewed and an updated estimate was prepared on the basis of new test results and current costs.

The process operating cost estimate was prepared from data provided by suppliers and reconciled with a mining operation of similar size. Process operating supply costs are based on new test results and budgetary prices from vendors of consumables and reagents. Based on current study, the process operating cost per tonne, including general and administration (G&A) costs, is Cdn$43.39. The mill has been sized to process 1,702 t/d as DMS feed followed by rod milling at 1,400 t/d capacity with an availability of 92%.

Table 23.20 shows a summary the updated operating costs. A detailed operating cost estimate is provided in Appendix I of the Optimized Feasibility Study.

Table 23.20 Updated Operating Costs Summary (Cdn$)

Description	Manpower	Annual Cost ($)	Unit Cost ($/t ore)
Staff			
G&A	25	2,055,858	3.31
Operating Staff	10	1,006,135	1.62
Operating Labour	30	2,050,716	3.30
Maintenance Labour	17	1,131,792	1.82
Met and Assay Lab	8	564,390	0.91
Tailings Plant	4	258,354	0.42
Sub-total Staff	94	7,067,245	11.38
G&A Expenses			
Expenses	-	7,419,740	11.94
Sub-total Costs	-	7,419,740	11.94

Table continues…

Description	Manpower	Annual Cost ($)	Unit Cost ($/t ore)
Supplies (Process Plant)			
Operating Supplies	-	5,207,522	8.38
Maintenance Supplies	-	952,000	1.53
Power Supply	-	6,036,383	9.72
Sub-total Supplies	-	12,195,905	19.63
Supplies (Tailings+Water)			
Operating Supplies	-	40,099	0.06
Maintenance Supplies	-	50,000	0.08
Power Supply	-	184,261	0.30
Sub-total Supplies	-	274,360	0.44
Total	94	26,957,260	43.39

Basis of Estimate

G&A costs include the personnel costs for management and administrative support functions and loss control. The Owner's costs include insurance, head office expenses, external assays, legal services, recruitment, camp catering and maintenance, and personnel rotation transportation costs. Processing costs include the costs of direct process operations, labour, consumables (including reagents), maintenance supplies, and diesel power generation.

Hourly and staff wages are based on current wages from operating mines in northern British Columbia and the Yukon. Labour wage rates were calculated reflecting different crew schedules and hours of work. An average loaded salary factor of 25% has been used to cover the costs of benefits, CPP, WCB, EI, life and long-term disabilities, pension plan, and statutory holidays.

The proposed schedule at the plant is as follows:

- senior management
  - 10 hour shifts per day
  - 4 days on / 3 days off
- process plant
  - two 12 hour shifts
  - 14 days on / 14 days off
  - two crew basis to provide 24 hour coverage
  - personnel will work approximately 2,190 h/a
  - 50 weeks with two weeks vacation period.

Process operating supply costs are based on budgetary prices from vendors for the key consumables, including crusher mill liners, and reagents. Reagent consumption rates are based on the test results from Process Research Associates, Ltd. (PRA) and from past experience. Process maintenance supply costs are factored from equipment costs. Prices for fuel and lube oil are used for power operating cost calculations. The consumption rates and prices for liners and balls in crushing and grinding sections are based on the Grinding Index and Work Index, and industry standards.

The transportation services include crew shift rotations, non-Yukon staff flights, management transport to Vancouver, other flights, and bus services. Transportation services were estimated to cost $800,000 annually.

Camp costs include catering, housekeeping, and maintenance services for the camp.

Genset leasing is based on a minimal down payment with remainder equally written off over five years (5 x $2,100,000).

The power supply estimate is based on the use of a diesel power generating facility. The operating load estimate has been developed based on process equipment sized from the feasibility study. The fuel consumption rate with 95% efficiency was estimated to be 216 gal/kWh. Fuel and electrical energy costs were based on a price of $0.89/L and $0.23/kWh, respectively.

23.8 ECONOMIC ANALYSIS

Wardrop prepared an economic model for the project based on the following assumptions:

- mine construction start is 2007 with commissioning in the Year 2008
- moving average exchange rates
- moving average metal prices
- net smelter terms.

The model was prepared pre-tax and pre finance basis as the tax regime is unknown at this stage.

23.8.1 NET PRESENT VALUE AND IRR SUMMARY

The study included the development of a spreadsheet that can be used to evaluate the sensitivity of the project to various inputs. The ability to vary these inputs allows the impact of updated field results to be examined quickly and the effects on the project determined.

Capital, initial or sustaining, was considered in the year that it was spent. The mining sustaining capital was calculated based on the expected lives of each of the various pieces of equipment. For this level of study, the contingency was based on the confidence of -5% to +15%.

Revenue was determined by grade and metal price, and adjusted for marketing, transportation cost, and mining recovery.

A range of values for the price of metals was selected for the life of the project and economics determined for different cases. Marketing costs were estimated at 2% of the value of the metal in concentrate. The net revenue was calculated from the gross value less the marketing, transportation costs, NSR.

Four pre-tax IRR and NPV cases were examined and results as follows:

Case 1 – Model with 3 Year Moving Average Prices

In this case, the project has a pre-tax IRR of 18.90% and an NPV of Cdn$104.8 million at an 8% discount rate.

Case 2 – Model with Combined 2 and 3 Year Moving Average Prices

In this case, the two-year average metal price was used for the first two years and followed by three-year moving average metal prices for the remaining years of the project.

In this case, the project has a pre-tax IRR of 22.60% and an NPV of Cdn$134.3 million at an 8% discount rate.

Case 3 – Model with 2 Year Moving Average Prices

In this case, the project has a pre-tax IRR of 26.30% and an NPV of Cdn$184.2 million at an 8% discount rate.

Case 4 – Scenario Model with Current Prices for Comparison Purposes Only

In this case, the project has a pre-tax IRR of 56.80% and an NPV of Cdn$571.7 million at an 8% discount rate.

Table 25.3 shows a summary of the IRR and NPV at an 8% discount rate, and Table 25.4 the metal prices for the different cases.

Table 23.21 Economic Analysis Summary, (Cdn$)

	Pre-tax IRR (%)	Pre-tax NPV 8% ($ million)	Payback Period	3 Year Full Production Cumulative Pre-tax Cash Flow	Average Annual Cash Flow For First 3 Years Pre-tax Full Production
Case 1	18.9	104.8	3.9	172.5	57.50
Case 2	22.6	134.3	3.2	204.9	61.47
Case 3	26.3	184.2	3.0	217.7	65.31
Case 4	56.8	571.7	1.5	439.3	13.18

Table 23.22 Metal Prices, (US$)

	Zinc (US$/lb)	Copper (US$/lb)	Lead (US$/lb)	Silver (US$/oz)	Gold (US$/oz)
Case 1	0.87	1.85	0.48	8.54	486.85
Case 2	0.87 and 1.07	1.85	0.48 and 0.52	8.54 and 9.48	486.85 and 526.65
Case 3	1.07	1.85	0.52	9.48	526.50
Case 4	1.84	1.85	0.76	12.69	626.01

23.8.2 SENSITIVITY ANALYSIS

Sensitivities to metal price, exchange rate, capital cost, and operating cost on the IRR and NPV were considered. Figures 23.6 to 23.9 show the sensitivity trends. The most significant variable influencing the model was revenue.

Figure 23.6 IRR Sensitivity Analysis

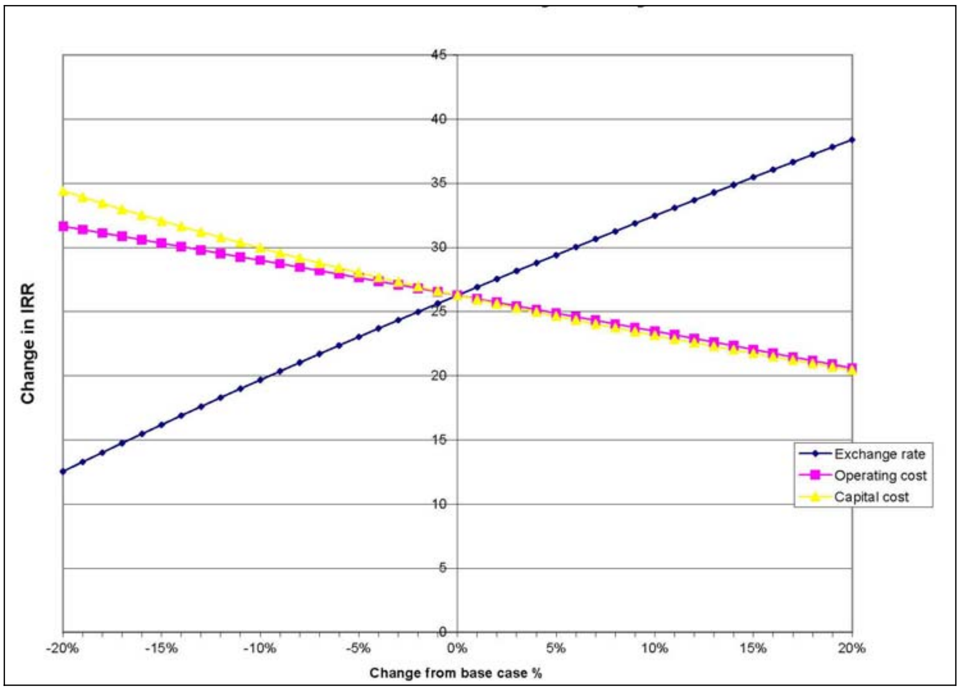


Figure 23.7 Metal Prices – NPV Sensitivity Analysis at 8%

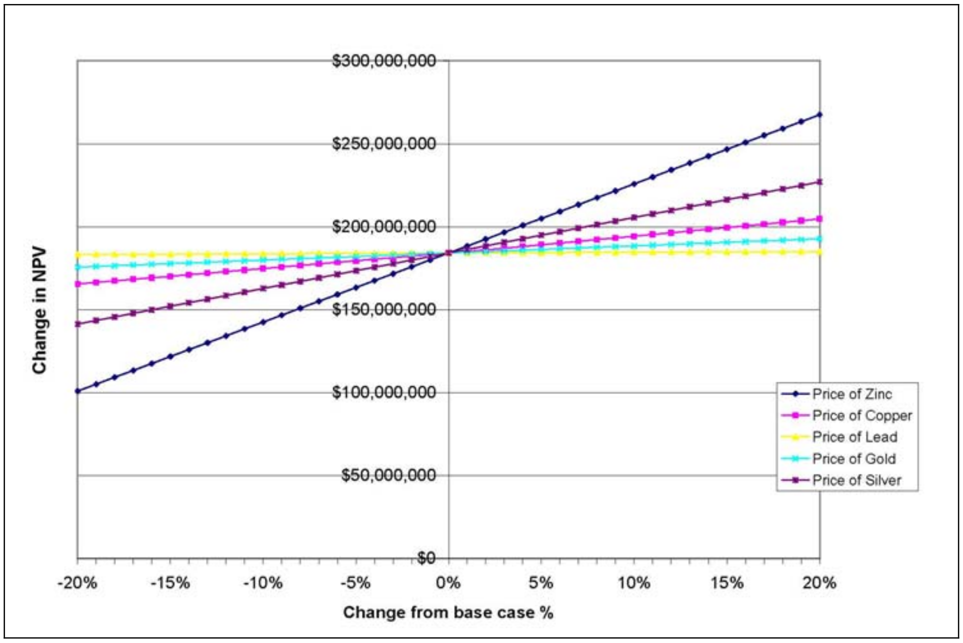


Figure 23.8 NPV Sensitivity Analysis at 8%

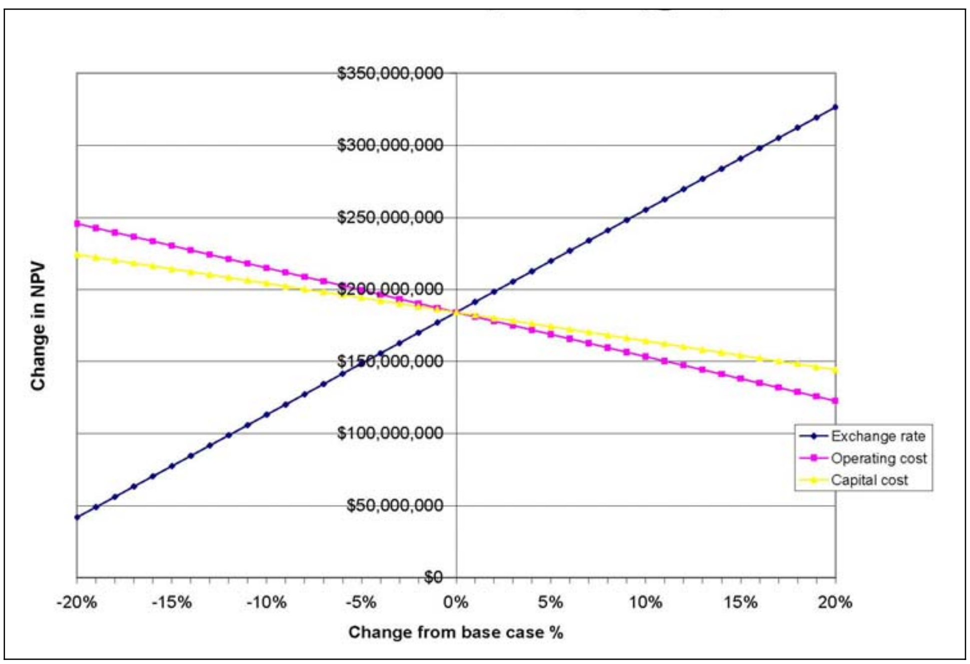
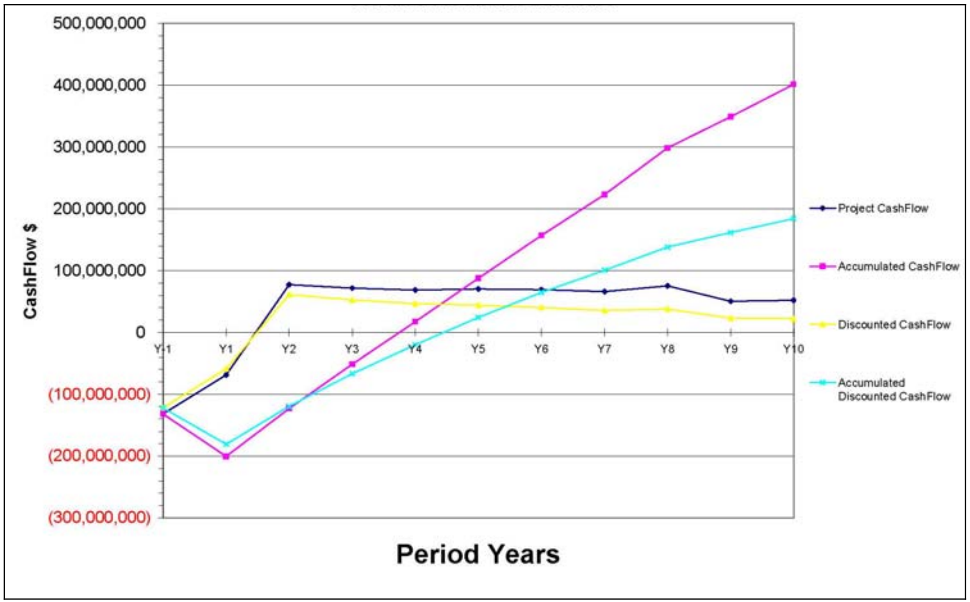


Figure 23.9 Pre-tax and Pre-finance Base Case



The exchange rate is important, as the price of metals is stated in United States dollars. The recent strength in the Canadian dollar relative to the United States dollar highlights the impact of this factor. One aspect of the exchange rate impact is the cost of capital. As the Canadian dollar increases in strength, the cost of initial capital for some of the major mining equipment would also change. The estimated effect is based on the approximate exposure of capital costs based on the United States dollar. The exchange rate sensitivity also shows that the model is more sensitive as the Canadian dollar strengthens against the United States dollar.

Appendix J of the Wardrop's Optimized Feasibility Study contains full details of the economic model.

23.9 Concentrate Shipments

This section was written by Wardrop with input from YZC. For the basis of this study, concentrates will be shipped to Asian markets through the Port of Steward. A site visit to the port facilities by YZC confirmed the facility is adequate for its requirements. It should be noted that at the request of the Mayor of Stewart, BC, a meeting was held and attended by city officials, politicians, contractors and various mining companies to discuss and identify future concentrate haulage requirements to be transported and stored in Stewart for export to strategically located smelting companies in Asia. As a result, it is expected the highways leading to and from Stewart will be maintained and upgraded to accommodate additional transportation requirements.

23.10 PAYBACK

This section was written by Wardrop. For a summary of payback period in each of the four cases evaluated, refer to Table 23.8.

23.11 MINE LIFE

This section was written by Wardrop. The Wolverine deposit is capable of generating a daily production rate of 1,850 t/d, with a steady state long-term production rate of 1,700 t/d. The mine design will allow for a mine life of approximately 9.5 years with a moderate amount of upside potential for extension.

24.0 DATE AND SIGNATURE PAGE

The undersigned prepared this technical report titled "Independent Technical Report on the Wolverine Project – Finlayson District, Yukon," and dated 28 February 2007. The format and content of this report are intended to conform to form 43-101F1 of National Instrument 43-101 (NI 43-101) of the Canadian Securities Administrators.

Original document, Revision 01
signed and sealed by Rick Alexander

Rick Alexander, P.Eng.

Original document, Revision 01
signed and sealed by Ken Deter

Ken Deter, P.Eng.

Original document, Revision 01
signed and sealed by David Tyson

David Tyson, R.P. Bio.

Original document, Revision 01
signed and sealed by Allan Polk

Allan Polk, P.Eng.

Original document, Revision 01
signed and sealed by John Fox

John Fox, P.Eng.