

Report to:



**Technical Report on the Wolverine
Mine**

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Report to:



TECHNICAL REPORT ON THE WOLVERINE MINE

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GLOSSARY

UNITS OF MEASURE

Above mean sea level.....	amsl
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes.....	Bt
Billion years ago.....	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm ³
Cubic feet per minute	cfm
Cubic feet per second	ft ³ /s
Cubic foot.....	ft ³
Cubic inch	in ³
Cubic metre.....	m ³
Cubic yard.....	yd ³
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius.....	°C
Diameter	Ø
Dollar (American)	US\$
Dollar (Canadian).....	CAD
Dry metric ton.....	dmt
Foot.....	ft

Gallon	gal
Gallons per minute (US).....	gpm
Gigajoule.....	GJ
Gigapascal.....	GPa
Gigawatt.....	GW
Gram.....	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than.....	>
Hectare (10,000 m ²).....	ha
Hertz	Hz
Horsepower.....	hp
Hour	h
Hours per day	h/d
Hours per week.....	h/wk
Hours per year	h/a
Inch.....	"
Kilo (thousand).....	k
Kilogram.....	kg
Kilograms per cubic metre	kg/m ³
Kilograms per hour.....	kg/h
Kilograms per square metre.....	kg/m ²
Kilometre.....	km
Kilometres per hour.....	km/h
Kilopascal.....	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts.....	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/m
Megabytes per second.....	Mb/s
Megapascal.....	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre.....	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne).....	t

Microns	μm
Milligram.....	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre.....	mm
Million.....	M
Million bank cubic metres.....	Mbm ³
Million bank cubic metres per annum.....	Mbm ³ /a
Million tonnes	Mt
Minute (plane angle)	'
Minute (time).....	min
Month.....	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa-s
Parts per million	ppm
Parts per billion	ppb
Percent.....	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time).....	s
Specific gravity	SG
Square centimetre.....	cm ²
Square foot	ft ²
Square inch.....	in ²
Square kilometre	km ²
Square metre	m ²
Thousand tonnes	kt
Three Dimensional.....	3D
Three Dimensional Model	3DM
Tonne (1,000 kg).....	t
Tonnes per day	t/d
Tonnes per hour.....	t/h
Tonnes per year.....	t/a
Tonnes seconds per hour metre cubed	ts/hm ³
Volt.....	V
Week.....	wk
Weight/weight	w/w
Wet metric ton.....	wmt
Year (annum)	a

ABBREVIATIONS AND ACRONYMS

Aboriginal Liaison Office	ALO
acid rock drainage	ARD
ACME Analytical Laboratories Ltd.	ACME
ALS Chemex Laboratories.....	ALS Chemex
antimony	Sb
Arctic Construction Inc.	Arctic
arsenic	As
Atna Resources Ltd.	Atna
Atomic Absorption Spectrophotometer	AAS
Boliden Limited	Boliden
cadmium	Cd
Canadian Institute of Mining	CIM
Canadian Standard Association.....	CSA
carbon.....	C
closed circuit television	CCTV
cobalt	Co
concentrate load-out	CLO
copper.....	Cu
dense media separation.....	DMS
digital terrain models.....	DTM
drift and fill with primary and secondary panels	DFPS
drift and fill with retreat panels	DFRP
drift and fill with side slash	DFSS
earnings before interest, tax, depreciation, and amortization.....	EBITDA
EBA Engineering Consultants Ltd.....	EBA
Energy and Metal Consensus Forecast	EMCF
Environmental Effects Monitoring	EEM
Expatriate Resources Ltd.....	Expatriate
feldspar-phyric	FP
Finlayson District	FD
Game Management Area.....	GMA
Game Management Zone	GMZ
general and administrative.....	G&A
gold.....	Au
gridded seam model	GSM
Ground Control Management Plan	GCMP
Ground Strength Index.....	GSI
health, safety and environmental	HS&E
high-density polyethylene	HDPE
human resources	HR
inductively coupled plasma	ICP
inductively coupled plasma-atomic emission spectroscopy	ICP-AES
instrument flight rules.....	IFR
iron.....	Fe
Job Hazard Analysis	JHA

lead.....	Pb
life-of-mine	LOM
linear low density polyethylene	LLDP
liquid natural gas.....	LNG
load-haul-dump.....	LHD
magnesium	Mg
mercury.....	Hg
metal leaching.....	ML
Metal Mining Effluent Regulations	MMER
mid-ocean ridge basalt.....	MORB
Minorex Consulting Ltd.	Minorex
MRDI Canada	MDRI
National Instrument 43-101.....	NI 43-101
net present value	NPV
net smelter return.....	NSR
nickel.....	Ni
North American Datum	NAD
Occupational Safety and Health Association	OSHA
ordinary kriging	OK
personal protective equipment.....	PPE
potentially acid generating	PAG
PricewaterhouseCoopers.....	PwC
Process Research Associates	PRA
Procon Mining and Tunnelling Ltd.	Procon
qualified person	QP
quality control/quality assurance	QC/QA
Quartz Mining Act	QMA
quartz mining license	QML
recoverable metal value.....	RMV
rock mass rating	RMR
rock quality designation	RQD
Rocky Mountain Geochemical	Rocky Mountain
Ross River Dena Council.....	RRDC
run-of-mine	ROM
sedimentary exhalative	SEDEX
selenium	Se
silver	Ag
Slide Mountain Terrane.....	SMT
stirred media detritor.....	SMD
sulphur	S
tailings management facility.....	TMF
Teck Cominco Ltd.	Teck Cominco
Teck Resources Ltd.	Teck Resources
threshold limit value	TLV
Traditional Knowledge	TK
Transportation Association of Canada.....	TAC

Underhill Geomatics Ltd.....	Underhill
Universal Transverse Mercator.....	UTM
variable frequency drive.....	VFD
visual meteorological flight.....	VFR
volcanic- and sediment-hosted massive sulphide.....	VSHMS
volcanic-hosted massive sulphide	VHMS
volcanogenic massive sulphide	VMS
Wardrop, a Tetra Tech Company	Tetra Tech
water use license	WUL
Westmin Resources Ltd.....	Westmin
Wolverine Property	the Property
x-ray diffraction	XRF
Yukon Engineering Services Inc.	YES
Yukon Zinc Corp.	Yukon Zinc
Yukon-Tanana pericratonic terrane	YTT
zinc equivalent	ZnEq
zinc	Zn

1.0 SUMMARY

1.1 INTRODUCTION

The Wolverine Mine, owned and operated by Yukon Zinc Corp. (Yukon Zinc) is a high-grade, zinc (Zn)-silver (Ag)-copper (Cu)-lead (Pb)-gold (Au) underground mine, with on-site milling capabilities of 1,700 t/d to produce zinc, copper and lead concentrates. The current life-of-mine (LOM) is nine years, based on a 3.8 Mt minable reserve.

Yukon Zinc completed major site construction at Wolverine Mine throughout 2009 and 2010. Mill commissioning commenced in 2011, and production is increasing to design capacity.

1.2 PROPERTY LOCATION AND DESCRIPTION

Yukon Zinc's Wolverine Property (the Property) is located in south-central Yukon, Canada, 171 km north-northwest of Watson Lake, and 282 km northeast of the city of Whitehorse. The mine site is in the Campbell Range, at the easternmost limit of the Pelly Mountains. The area features rolling, glacially-scoured mountains; elevations on the Property range approximately between 1,200 and 1,400 m. Glacial till covers the majority of the lower lying valleys and there is significant infilling by post-glacial sediments.

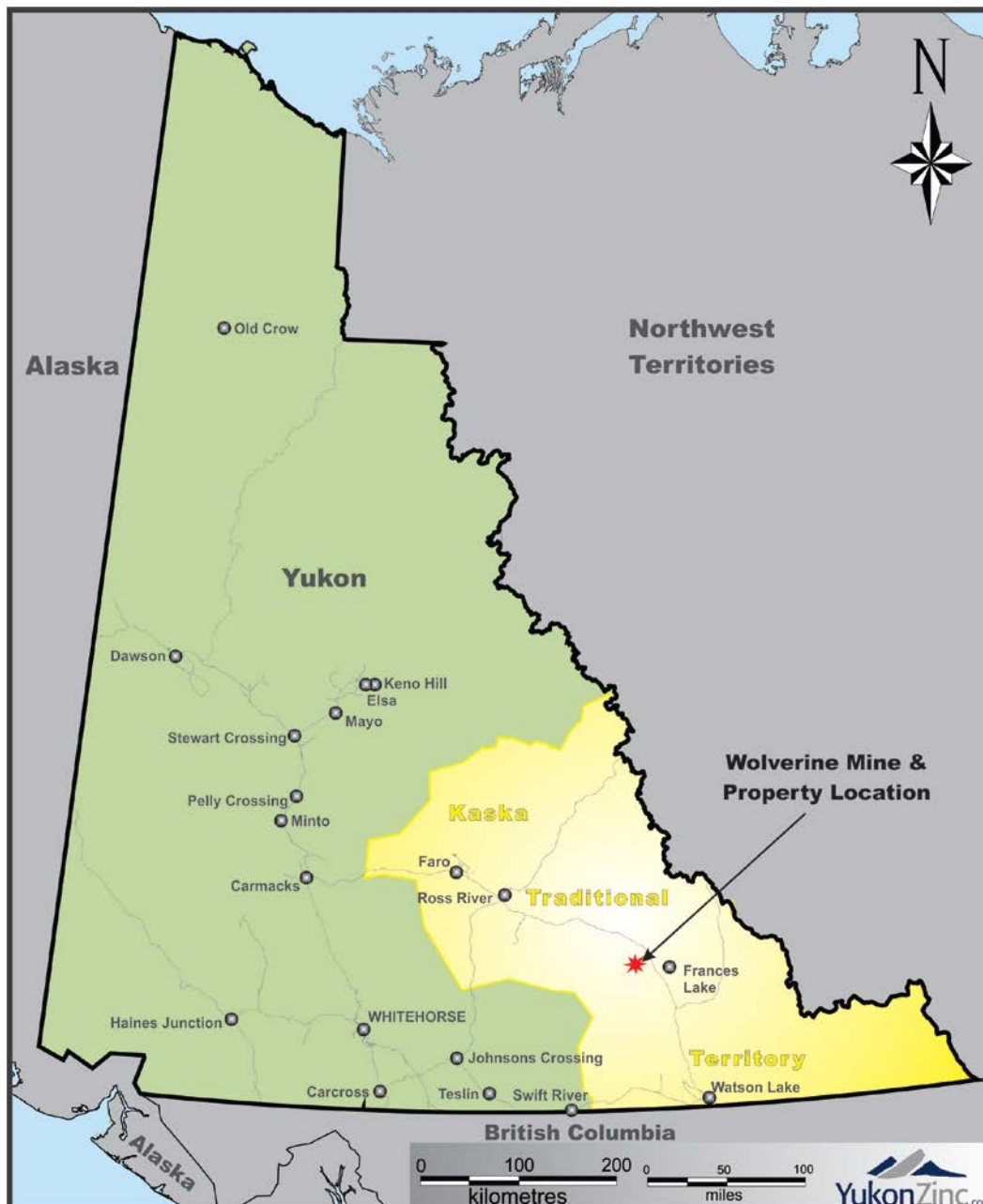
The mine site can be accessed by air, or via a 29 km access road. Figure 1.1 indicates the location of the Property and the Wolverine Mine within the Kaska Traditional Territory.

The Property consists of 1,033 quartz mining claims located in the Watson Lake Mining District, Yukon. The Wolverine Mine is 100% owned by Yukon Zinc.

The Wolverine Mine is 190 km by road north of Watson Lake and 180 km south of Ross River via the Robert Campbell Highway. The Property encompasses 167 km².

Other properties owned by Yukon Zinc in Yukon include Ref, Box, League, Goal Net, Blue Line, Shot, Winger, Shutout, Hat Trick, Red Line, Logan, Swift, Light, Money and the Ice Property which hosts the Ice deposit. The Ice deposit is located on the northern part of the FD, about 60 km east of Ross River. Up and downstrike of the Wolverine Mine are stratigraphic continuities of the mine's host rocks, these zones are called Fisher, Sable and Puck zones which are described in Section 23.0 and are part of the Wolverine contiguous claim group.

Figure 1.1 Property Location Map



1.3 RESOURCE ESTIMATE

Wardrop, a Tetra Tech Company (Tetra Tech) received drillhole data from Yukon Zinc, which included geological logs, assay and density results from 203 holes.

A total of 5,954 assay and 2,133 density results were imported into the drillhole database.

Tetra Tech estimated that the Wolverine deposit contains 4.14 Mt of Indicated Resources averaging 12.91% Zn, 374 g/t Ag, 1.3% Cu, 1.83 g/t Au, and 1.7% Pb, at a US\$150 recoverable metal value (RMV) cut-off. In addition, Tetra Tech also estimated that the Wolverine deposit contains 1.635 Mt of Inferred Resources averaging 11.8% Zn, 368 g/t Ag, 1.2% Cu, 1.56 g/t Au, and 1.6% Pb. The mineral resources are reported in Table 1.1.

Table 1.1 Mineral Resources at US\$150 RMV Cut-off

Classification	Cut-off (US\$)	Tonnes ('000 t)	Volume (m ³)	Zn (%)	Ag (g/t)	Cu (%)	Au (g/t)	Pb (%)
Indicated	150	4.140	1,045	12.9	374	1.3	1.83	1.7
Inferred	150	1.635	413	11.8	368	1.2	1.57	1.6
Contained Metals				Zn ('000 t)	Ag ('000 g)	Cu ('000 t)	Au ('000 g)	Pb ('000 t)
Indicated	-	-	-	534	1,548,360	54	7,576	70
Inferred	-	-	-	193	601,680	20	2,567	26

1.4 RESERVES AND MINING

The mining reserve is 3.8 Mt and will provide an operating mine life of approximately nine years. There are three main zones of mineralization. From southeast to northwest these are the Wolverine deposit, the Saddle deposit and the Lynx deposit. The Wolverine and the Lynx deposits carry the majority of the mineralization. A main access ramp from surface (elevation 1345 masl) provides access to the mine for personnel and materials. The ramp is located partially within the Saddle deposit, which is central to the Lynx and Wolverine deposits.

The Wolverine Mine has a mine plan daily production rate of 1,700 t of diluted ore. It is assumed that the mine will operate 24 h/d, 365 d/a. An ore stockpile on surface will be used to handle any surge in production which exceeds the mill feed rate.

Table 1.2 shows the in situ material that is contained within stope wireframes.

Table 1.2 In Situ Material Contained within Stope Wireframe Shapes

Deposit	Tonnes	Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)
Wolverine	1,784,867	389.09	2.12	1.15	1.43	11.66
Lynx	2,226,925	314.44	1.38	1.27	1.67	12.72
Saddle	164,245	427.84	1.90	1.86	1.68	11.14
Total	4,176,037	350.80	1.72	1.24	1.57	12.20

Wolverine mineralization already mined has been depleted from future reserves as shown in Table 1.3

Table 1.3 Material Mined from Stopes

Deposit	Total Tonnes Removed from Reserve
Wolverine	63,228
Lynx	114,043
Saddle	7,071
Total	184,342

Once the recovery and dilution factors have been applied to the in situ material and the Table 1.3 material has been depleted, the remaining material is considered to be reserve. Table 1.4 shows the minable reserve for the Wolverine Mine.

Table 1.4 Ore Reserve Estimate

Volume (m ³)	Tonnes	Category	Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)
1,003,062	3,751,452	Probable	327.29	1.60	1.17	1.47	11.55

A geomechanical investigation and evaluation of the rock mass quality and ground support design was completed by mining consultancy firm Rockland Ltd. in 2010. Their report was used as a basis for the comprehensive underground Ground Control Management Plan (GCMP) for the Wolverine Mine describing the risk assessment process, roles and responsibilities, ground support design, communication plan, training, incident investigations, monitoring, mapping, auditing and document controls policies. The report summarizes that the rock mass quality for the ore was “fair” while the footwall and hanging wall rock mass quality is “very poor” with numerous shallow dipping clay filled faults.

Drift and fill is the mining method used at the Wolverine site. This method has been chosen due to the potentially poor ground in the footwall and hanging wall, the flat lying dip of the deposit, varying horizontal thickness of the deposit and the selectivity required to achieve acceptable amounts of waste dilution.

Paste fill is the primary material used to backfill the stoping areas. The paste fill is fed from the backfill plant on surface and piped into the stopes. Waste development rock is used to supplement the paste fill when it is operationally and logistically convenient.

All lateral mining is done with diesel powered mobile equipment. Drilling is done with electric hydraulic drill jumbos. Rock bolting operations are performed by separate electric hydraulic units. Diesel powered load-haul-dump trucks (LHDs) are used to load both waste and ore material. Diesel powered 30 t haul trucks are used to transport both ore and waste to surface from the remucks underground. Raise excavations are mined with pneumatic hand held equipment. LHDs are used to load

both waste and ore from development and stope headings into the haulage trucks for haulage to surface.

A maximum of 326,000 cfm of ventilating air is required based on a projected equipment list for the mine. Two ventilation raises provide fresh air to the underground workings and the main ramp is used to exhaust the air to surface. Mine air heaters heat the mine air during the colder months to maintain a temperature above freezing.

The main diesel generators at the mine site supply power to the mine. Power is delivered from surface to the underground operations via cables in the main ramp.

The underground mine design for the Wolverine Mine has been developed with information and data from several sources, including expert opinion in the areas of mine geotechnical design and ventilation systems design. During the process of the study, several stages of peer review were undertaken to confirm the validity of the designs, parameters and assumptions.

The mine plan is consistent with the production rates required to meet the proposed production rates to the processing plant over the life of the operation.

1.5 METALLURGICAL TESTING AND PROCESSING

Tetra Tech conducted a review of the existing plant construction by way of a site visit in September 2011 and an audit to review the plant data generated to date, and advised regarding the overall plant operations. Tetra Tech was also provided with a report giving the results of a metallurgical investigation conducted during 2010 at the Central South University in China. The test work program was conducted using the unit processes and parameters which had formed the basis of the Process Research Associates (PRA) test work and which were used in the plant design.

A review of this test report indicated that the results obtained by the Central South University for concentrate grades and metal recoveries were generally consistent with the results which had been obtained by PRA, although the sample material used by the Central South University was from one part of the mine only. The test work procedure used by the Central South University did not include a pre-flotation step, and the results showed that there was no advantage in changing the flotation procedure from the existing sequential copper, then lead, then zinc flotation stages, to a bulk copper/lead flotation stage followed by the zinc flotation stage.

In 2011, the Wolverine Mine was in the ramping up phase, progressing towards the underground ore production target of 1,700 t/d. During this time, the process plant was operated on a campaign basis treating the available stockpiled production until depletion. Ore throughput rates of up to 1,100 t/d were treated during these periods, which enabled mill staff to identify and correct certain problem areas. The design throughput rate of 1,700 t/d had not yet been attained at the time of the plant visit. While the plant has achieved and is operating at the design primary grind size, the

regrind particle size had not yet been achieved. An ongoing problem is the performance of the pre-flotation circuit. While initially considered to be a scavenging unit for the removal of carbonaceous material with minimal amounts of valuable metal, this has been producing a significant concentrate mass together with associated valuable metal components, particularly silver and gold. Since this pre-flotation concentrate is discarded with the final plant tailings, the economic value of this loss is significant. Consideration is presently being given to retrofitting a dense media separation (DMS) circuit to take the load off the pre-flotation circuit which would then be allowed to function as per the original plant design. Additional test work is on-going to characterize the reasons and enable mine management to obtain a better understanding of the cause of the high losses associated with this circuit. This test work will also include a study of the tailings storage water that is recycled for re-use in the plant, which may contain residual concentrations of flotation reagents that affect the operation of the pre-flotation circuit.

At the time of the plant visit for the audit, the metallurgical performance had not achieved the individual metal target values for the periods when the plant was operational, although the concentrate grade and recovery values were steadily improving with each additional operating campaign. It is anticipated that an improved ore supply resulting in more consistent running times will lead to steady state operations which will contribute to the grade and recovery values stabilizing at the budget values.

1.6 INFRASTRUCTURE

The facilities and infrastructure for the Wolverine Mine include the following:

- site access road
- airstrip
- fuel storage and dispensing
- power generation and distribution
- process buildings
- assay laboratory
- wet shotcrete plant
- waste rock storage pads
- truck shop
- mining office complex
- administration, first aid and mine rescue buildings, and dry
- camp.

The road is a private, single lane road with passing bays, has restricted access and is operated under radio control. The road is used year round with minimal load restrictions.

The on-site gravel airstrip is 1,340 m long with 280 m of runway at the south end of the airstrip required to provide for an emergency stop way to allow an aircraft to land on the airstrip with a full passenger load. The airstrip is intended for restricted use only by day during good visual meteorological flight (VFR) conditions

Diesel fuel is supplied from six diesel fuel storage tanks (75,000 L capacity each) and an 8,500 L gasoline tank. Storage is based on two weeks of reserve in the event of road problems and/or use restrictions. A fuel truck transports diesel fuel and lubricant to the various mobile equipment as required.

The Wolverine Mine is served with on-site power generation. The total connected plant load is estimated to be 10,870 kW. The load is served by eight diesel generator sets, each rated 1,200 rpm, 1.26 MW continuous, 1.45 MW prime power generating at 4,160 V for a total installed generating capacity of 10.08 MW continuous, 11.60 MW prime power.

The process facilities consist of a crusher building, mill building and concentrate load-out building. Feed conveyors connect the crusher building to the mill and transport crushed ore to the rod and ball mills. Ancillary facilities include a wet shotcrete plant, assay laboratory, mining office complex, truck shop and camp.

Camp infrastructure at the Wolverine Mine consists of six 41-man dormitories, a kitchen, recreation hall, administration office, environment and safety offices, first aid office, mine rescue station and dry facilities. Additional support infrastructure includes a maintenance shed, firewater tank, potable water treatment plant, communication station, and sewage treatment plant.

1.7 ENVIRONMENTAL CONSIDERATIONS

Following submission by Yukon Zinc of the environmental assessment in October 2005, an Environmental Assessment Screening Report was issued by the Yukon Government in in 2006. It stated that "...taking into account the implementation of the mitigation measures and follow-up, the project is not likely to cause significant adverse environmental effects.". Based on this conclusion, the EA was approved and project was allowed to move forward to the permit issuance phase. The quartz mining license (QML) QML-0006 was issued December 5, 2006 by the Yukon Department of Energy, Mines and Resources. Water use license (WUL) QZ04-065 was issued October 4, 2007 by the Yukon Water Board. These two licenses outline the major operating and closure requirements for the Wolverine Mine. Various minor permits, including permits issued by Yukon Environment, Yukon Workers' Compensation Health and Safety Board, and the Canadian Nuclear Safety Commission outline operating conditions for activities at the Wolverine Mine not regulated by the QML and WUL.

The Wolverine Mine has specific obligations to protect wildlife and heritage resources, to manage waste generated by the mine, to manage hazardous materials, and to manage the quality and quantity of any liquid effluent discharges. These obligations and the associated management measures are detailed in the applicable environmental protection and operating plans. Specific matters related to water, waste rock, and tailings management are discussed in Section 20.

The closure and reclamation plan for the mine has been approved by the Yukon Government Department of Energy, Mines and Resources. The plan details the measures to be taken during both a temporary suspension of operations and permanent closure of the Wolverine Mine.

The Wolverine Mine is located within the traditional territory of the Kaska Nation. Consultation on the Wolverine Mine with the Kaska Nation began in 1995 in the form of meetings through advisory committees, technical working groups, and public open houses. In 1997-98, an Aboriginal Liaison Officer (ALO) was appointed in Ross River to liaise between the Kaska Nation and Yukon Zinc. The position was reinstated during 2000-01 by Expatriate Resources Ltd. (now named Yukon Zinc Corp.) and again in 2005 by Yukon Zinc, and is a currently active role.

1.8 CAPITAL AND OPERATING COSTS

1.8.1 CAPITAL COST

The total additional capital cost required for the Wolverine LOM is estimated at \$63.0 million and is distributed as per the following:

- capitalized mine development costs consisting of ramps, fresh air raises, level accesses, remucks and sumps – \$18.1 million
- capital purchases/expenditures including a DMS plant, tailings dam raise, warehouse, water treatment plant, process control improvements and generator upgrades required to execute full extraction of the Wolverine LOM reserves – \$19.2 million
- sustaining capital for the LOM – \$23.0 million
- closure and reclamation costs – \$2.6 million.

1.8.2 OPERATING COST

The total operating cost for the Wolverine LOM is estimated at \$662.0 million and is calculated based on a mine/mill annual run rate of 1,700 t/d under the following area costs:

- The mining unit cost was determined to be \$87.81/t mined and delivered to the mill.
- The mill process cost was estimated to be \$58.36/t milled.

- The general and administrative (G&A) unit cost is estimated at \$30.63/t milled.

1.9 ECONOMIC EVALUATION

The following financial results were established for the Wolverine Mine:

- Net smelter return (NSR) revenue was determined by metal grades and metal prices and adjusted for smelting, refining, transportation and marketing costs. The LOM NSR revenue was determined to be approximately \$1.3 billion
- Pre-tax cash flow (earnings before interest, tax, depreciation and amortization (EBITDA)), determined as net revenue less operating, capital and reclamation costs, was estimated as \$547 million for the LOM. The pre-tax net present value (NPV) of the Wolverine Mine is \$444 million at a discount rate of 8%.
- The Wolverine Mine is in production in 2012 and as such does not have a negative cash flow, therefore, no IRR or payback is realized.
- Post-tax cash flow was estimated as \$509 million for the LOM. The post-tax NPV of the Wolverine Mine is \$414 million at a real discount rate of 8%.
- The LOM average metal prices for three cases (base +10%, base, base -10%) were identified with their corresponding NPVs at discount rates of 8%, 5% and 0%, for both pre-tax and post-tax, and are presented in Section 22.0.

1.10 INTERPRETATIONS AND CONCLUSIONS

The Wolverine Mine is currently in production and is expected to produce a robust cash flow throughout LOM.

It was determined by Tetra Tech that the Wolverine Mine is economically viable. Tetra Tech's conclusion is that the Wolverine Mine is technically viable as well.

The Wolverine Mine possesses an economically minable resource with the potential for an increase in resource with additional diamond drilling. The current mine plan is consistent with the production rates to meet the required feed to the processing plant over the life of the operation.

1.11 RECOMMENDATIONS

Tetra Tech has made several recommendations to improve mine design, productivity, processing, recoveries and economic results. The recommendations are as follows:

- A detailed underground drill program to help define ore zones.

- Advance diamond drilling efforts ahead of the mining horizons being developed. This would improve the resources reserve potential by promoting material into higher categories.
- Relocate the main access ramp. This design improvement would eliminate repeatedly crossing the footwall contact with the main ramp infrastructure.
- Tetra Tech recommends a trade-off study be performed to examine the value of creating two separate ramps from the existing central main ramp.
- The heading sizes should be increased (when in mineralization and where geotechnically feasible).
- Perimeter control blasting techniques present a strong opportunity at the Wolverine site.
- It is recommended to use one manufacturer for mining equipment.
- Strategic placement of a low-cost web based closed circuit television (CCTV) system during the fill cycle to allow observation of the fill pour.
- Numerical modeling of the mining sequence is recommended.
- The primary ventilation system is powered by a single fan. It is recommended to use two smaller fans instead of one large fan.

The following are process related recommendations:

- On-going metallurgical test work by mine staff and outside consultants is recommended to assist in obtaining a detailed understanding of the ore characteristics and responses.
- Campaign production during the start-up phase should be targeted to confirm that the plant can treat 1,700 t/d on a continuous basis.
- The pre-flotation circuit requires characterization.
- Installation of a DMS circuit is recommended which has the potential to improve process results.
- A review of the regrind product sizes and circuit configurations is required.
- The reclaim water system requires an overall review regarding potential deleterious side effects of process water to the responses for the three circuits.

Each of these above matters is presently under review by the mine staff.

2.0 INTRODUCTION

Tetra Tech has prepared this technical report on the Wolverine Mine in general accordance with the guidelines provided in National Instrument 43-101 (NI 43-101) Standards of Disclosure for Mineral Projects.

Tetra Tech, all qualified persons (QPs), and authors of this technical report, are independent of the issuer, JinDuiCheng Canada Resources Corporation Limited, its directors, senior management, and advisers.

This report was prepared by Tetra Tech at the request of Mr. Ray Mah, Chief Operating Officer of Yukon Zinc, a Vancouver, BC-based company with a corporate office located at 701-475 Howe Street, Vancouver, BC V6C 2B3.

This report is based upon information Yukon Zinc provided to Tetra Tech, as well as observations and information that were obtained by Tetra Tech during the site visit to the Property.

Yukon Zinc has a 100% outright interest in the Property.

The following technical experts and QPs visited the Wolverine Mine on September 27, 2011 for three days:

- Andre De Ruijter, P.Eng.
- Robert Morrison, Ph.D., MAusIMM (CP), P.Geo.
- Roy Levesque, P.Eng.

A summary of QPs responsible for each section of this report is provided in Table 2.1

Table 2.1 Summary of QPs

Report Section	Company	QP
1.0 Summary	All	Sign off by Section
2.0 Introduction	Tetra Tech	Andre De Ruijter, P.Eng.
3.0 Reliance on Other Experts	Tetra Tech	Andre De Ruijter, P.Eng.
4.0 Property Description and Location	Tetra Tech	Robert Morrison, P.Geo.
5.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography	Tetra Tech	Robert Morrison, P.Geo.
6.0 History	Tetra Tech	Robert Morrison, P.Geo.
7.0 Geological Setting and Mineralization	Tetra Tech	Robert Morrison, P.Geo.

table continues...

Report Section		Company	QP
8.0	Deposit Types	Tetra Tech	Robert Morrison, P.Geo.
9.0	Exploration	Tetra Tech	Robert Morrison, P.Geo.
10.0	Drilling	Tetra Tech	Robert Morrison, P.Geo.
11.0	Sample Preparation, Analyses, and Security	Tetra Tech	Robert Morrison, P.Geo.
12.0	Data Verification	Tetra Tech	Robert Morrison, P.Geo.
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	Andre De Ruijter, P.Eng.
14.0	Mineral Resource Estimates	Tetra Tech	Robert Morrison, P.Geo.
15.0	Mineral Reserve Estimates	Tetra Tech	Roy Levesque, P.Eng.
16.0	Mining Methods	Tetra Tech	Roy Levesque, P.Eng.
17.0	Recovery Methods	Tetra Tech	Andre De Ruijter, P.Eng.
18.0	Infrastructure	Tetra Tech	Andre De Ruijter, P.Eng. Roy Levesque, P.Eng. Monica Danon-Schaffer, Ph.D., P.Eng.
19.0	Market Studies and Contracts	Tetra Tech	Philip Bridson, P.Eng.
20.0	Environmental Studies	Tetra Tech	Monica Danon-Schaffer, Ph.D., P.Eng.
21.0	Capital and Operating Costs	Tetra Tech	Philip Bridson, P.Eng.
22.0	Economic Analysis	Tetra Tech	Philip Bridson, P.Eng.
23.0	Adjacent Properties	Tetra Tech	Robert Morrison, P.Geo.
24.0	Other Relevant Data and Information	All	Sign off by Section
25.0	Interpretations and Conclusions	All	Sign off by Section
26.0	Recommendations	All	Sign off by Section
27.0	References	All	Sign off by Section

3.0 RELIANCE ON OTHER EXPERTS

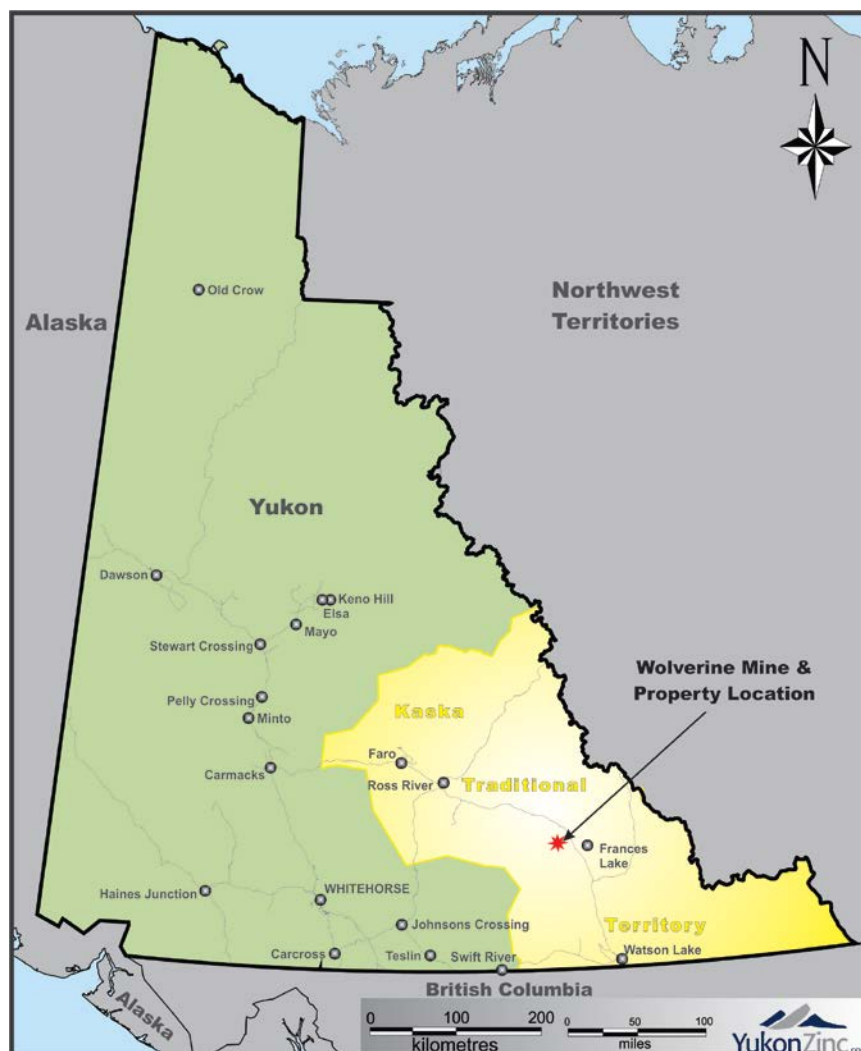
The QPs who prepared this report relied upon information provided by the following experts who are not QPs:

- Mr. Malcolm Swallow, of Swallow Services Ltd., has been relied on for advice on matters relating to mining engineering and mine design at Wolverine Mine, and other mine related items.
- Mr. Ray Mah, Chief Operating Officer of Yukon Zinc, has been relied on for advice on matters relating to markets and contracts.
- Mr. Stuart McDonald, Chief Financial Officer of Yukon Zinc, has been relied on for advice on matters relating to taxes.
- Ms. Jill Moore, Geologist of Yukon Zinc, has been relied on for advice on matters relating to Yukon Zinc's other properties.
- Mr. Tim Johnston, of PricewaterhouseCoopers (PwC), has been relied on for advice on matters relating to taxes in the economic modelling.

4.0 PROPERTY DESCRIPTION AND LOCATION

Yukon Zinc's Wolverine Property is located 190 km north-northwest of Watson Lake in south central Yukon. Figure 4.1 presents the location of the Property and Yukon Zinc's Wolverine Mine within the Kaska Traditional Territory. The Wolverine Mine is 190 km by road north of Watson Lake and 180 km south of Ross River via the Robert Campbell Highway. The Property encompasses 167 km² and is accessible either via a 29 km access road west from the Robert Campbell Highway or a 1,340 m long all-season gravel airstrip.

Figure 4.1 Yukon Map and Wolverine Mine Location

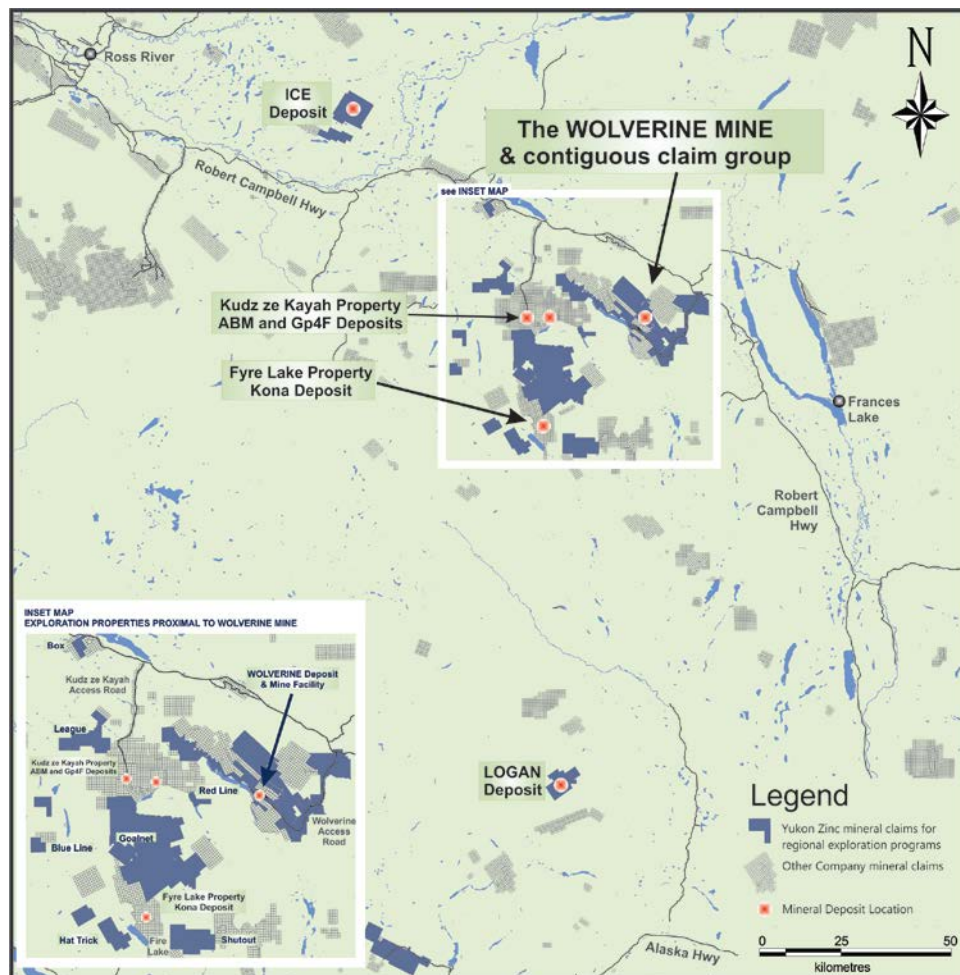


The Property consists of 1,033 contiguous quartz mining claims located in the Watson Lake Mining District, Yukon (Figure 4.3). The Wolverine Mine is 100% owned by Yukon Zinc.

Other properties owned by Yukon Zinc in Yukon include Ref, Box, League, Goal Net, Blue Line, Shot, Winger, Shutout, Hat Trick, Red Line, Logan, Swift, Light, Money, and the Ice Property which hosts the Ice deposit. The Ice deposit is located on the northern part of the FD, about 60 km east of Ross River. Up and downstrike of the Wolverine Mine are stratigraphic continuities of the mine's host rocks, these zones are called Fisher, Sable and Puck zones which are described in Section 23.0 and are part of the Wolverine contiguous claim group.

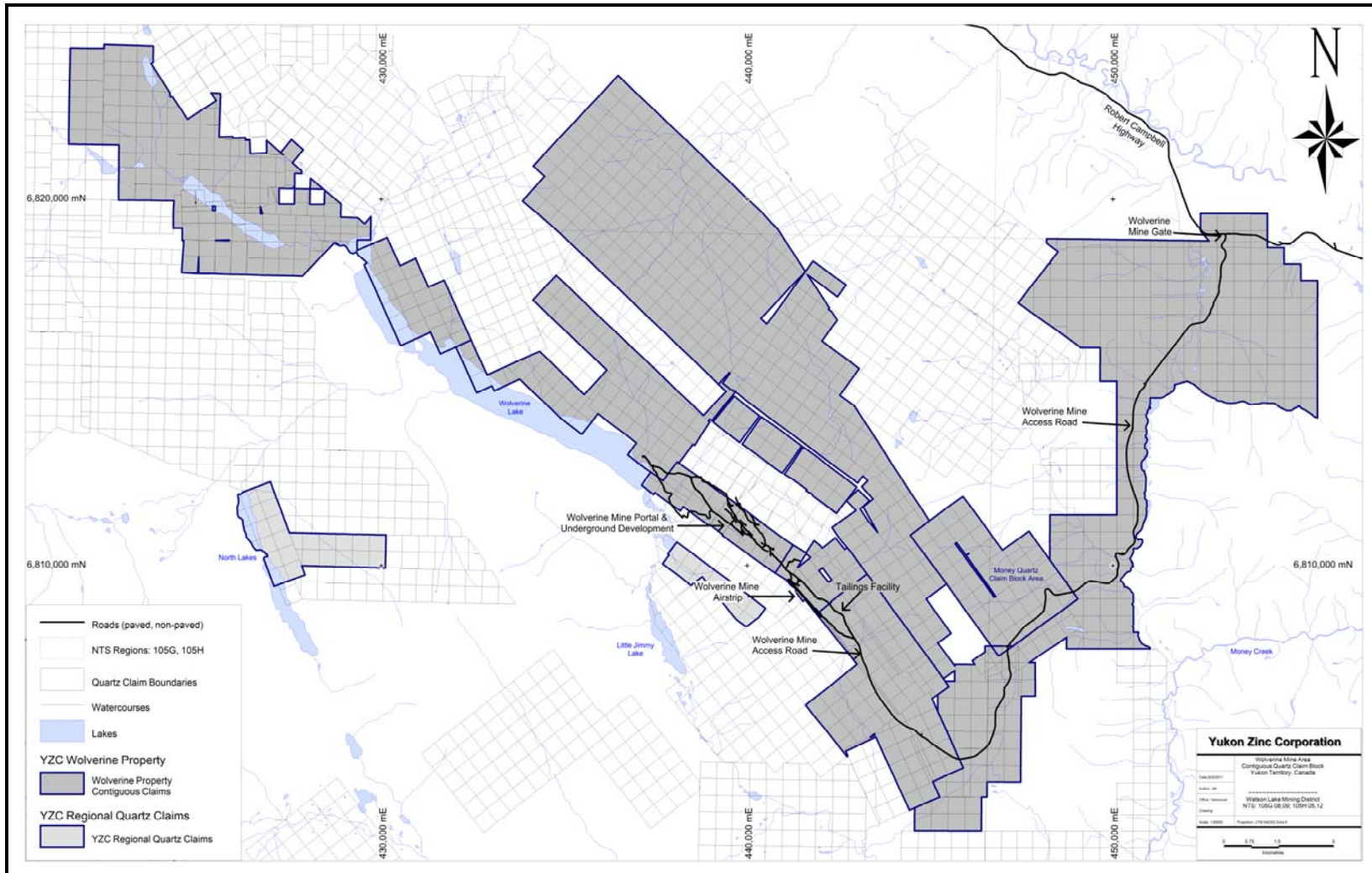
Other significant properties in the region are the Kudze Kayah Property, owned by Teck Resources Ltd., which hosts the ABM and GP4F deposits. The Fyre Lake Property, which hosts the Kona deposit, is owned by Pacific Ridge Exploration Ltd. (Figure 4.2).

Figure 4.2 Regional Deposit Location Map



Source: Yukon Zinc Corp.

Figure 4.3 Wolverine Property Claim Map



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY AND INFRASTRUCTURE

Access to the mine site is via a 29 km access road, or via air. A description of the mine access road, as well as the mine airstrip is provided in Section 18.0. Mine site infrastructure is also outlined in Section 18.0.

5.2 CLIMATE

Climate data collection has been ongoing at the Wolverine Mine since 2004. A HOBO® Onset weather station is installed on the south side of the airstrip. The weather station currently has the following data collection features:

- temperature
- relative humidity
- tripod mounted rain gauge
- solar radiation in and out
- wind speed and direction
- barometric pressure.

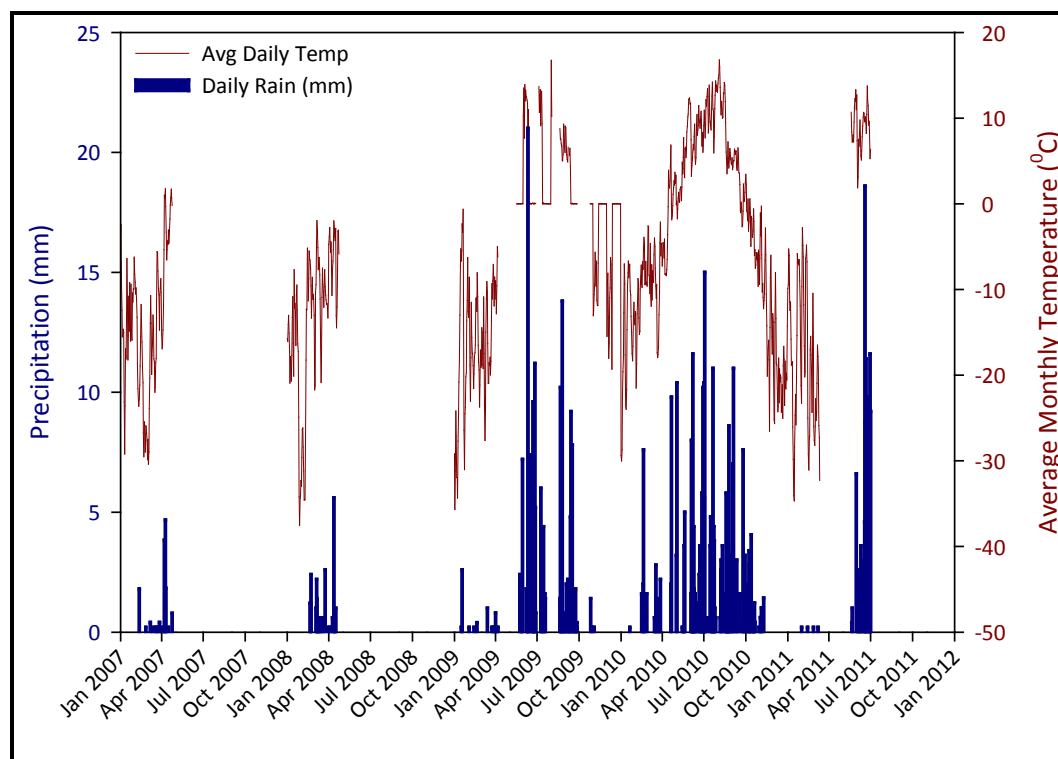
5.2.1 TEMPERATURE AND PRECIPITATION

Temperature and precipitation collected from the weather station are summarized in Figure 5.1. Gaps in the daily temperature data indicate a time when the station was not functioning; whereas gaps in the daily rainfall values can indicate either a gap in the data or that no precipitation was recorded that day (the HOBO® weather station does not record precipitation that falls as snow). The mean annual temperature is around -3°C. Mean monthly temperatures are below freezing from January through April and October through December and above freezing May through September. The minimum temperatures are approximately -30 to -40°C and the maximum temperatures are around 20 to 30°C.

Precipitation at the Wolverine Mine is on the order of ~500 mm per year, based on precipitation collected from the weather station (~300 mm) and snow pack

measurements (~200 mm). Estimates made during the Type A Water Use Application based on regional data had predicted precipitation values of ~570 mm.

Figure 5.1 Average Daily Temperature and Precipitation Collected from January 2007 through July 2011



5.3 LOCAL RESOURCES

Wildlife resources in the Wolverine Mine area and surrounding region include the Finlayson Caribou Herd, moose, black bear, grizzly bear, wolf, fox, coyote, wolverine, marten, mink, river otter, beaver, several raptors, ptarmigan, various waterfowl, and a variety of other forest birds. The lakes and small ponds/wetlands provide breeding and migratory habitats for waterfowl and other aquatic birds. The Finlayson Lake/River area and the east slope of the Pelly Mountains are part of the Tintina Trench migration corridor and are used extensively by waterfowl and other water birds, including trumpeter swans and sandhill cranes, on their north-south migrations.

Activities such as hunting, trapping and non-consumptive uses are not permitted in the immediate mine area. Descriptions of the various wildlife uses in the general mine area are provided below.

5.3.1 *HUNTING*

Target species for hunting include: moose, caribou, black bear, grizzly bear, mountain sheep, wolves, wood bison and occasionally mountain goats. Mountain goats are rare in Yukon and harvest of this species is closely regulated.

One of the most significant wildlife resources in the Finlayson area is the Finlayson Caribou Herd. This woodland caribou herd is highly valued as a subsistence base for the Kaska by resident sport hunters and Yukon guiding industry. Finlayson region moose are also an important resource for the same user groups, including the Kaska. The Finlayson region contains some of the highest recorded moose densities in Yukon. The Yukon Department of Environment regulates hunting in the territory and has established Game Management Areas (GMAs) for the management of wildlife species. GMAs are a combination of Game Management Zone (GMZ) and Game Management Subzone. The Wolverine Mine falls within GMZ 10 (subzone 1007, 1008, 1009) and GMZ 11 (subzone 1116, 1117, 1121 and 1122).

Teslin Outfitters Ltd. holds the commercial guiding rights for the Finlayson Lake area (Outfitter Concession #20). The primary game in Outfitter Concession #20 are stone and Fannin sheep, grizzly bear, caribou, moose, wolves, wolverine and game birds (ptarmigan and grouse); guiding typically occurs to the west of the Wolverine Mine area.

5.3.2 *TRAPPING*

Yukon is home to fourteen species of furbearing mammals that are trapped for their fur: beaver, coyote, fisher, coloured fox, Arctic fox, lynx, marten, mink, muskrat, otter, squirrel, weasel, wolf and wolverine. Yukon Department of Environment has established trapping areas to regulate the harvest of fur-bearing species, and keeps track of the annual fur harvest by monitoring trapping licences, export permits, fur dealer and taxidermist records and sealing certificates. Individual harvest information is confidential and is not normally released without the written permission of the registered trapping concessions holder. T. and F. Charlie (Concession 250) hold the trapping rights in the immediate vicinity of the Wolverine Mine, and there are group trapping rights (Concession 405 held by Ross River Dena Council) at the north end of the site access road.

5.3.3 *NON-CONSUMPTIVE USES*

The region surrounding the Wolverine Mine supports little formal recreation or tourism. Watson Lake to the south and Whitehorse to the east provide more formal recreational and tourism opportunities. Locally, Frances Lake is the main recreational attraction. Two recreational operations are located on Frances Lake: Frances Lake Campground and Frances Lake Wilderness Lodge. The Frances Lake Wilderness Lodge is located on the south end of Frances Lake between its two arms. Activities include daily guided excursions, canoeing, hiking, wildlife viewing,

photography, fishing and aurora viewing (in September). The lodge is also open for winter activities such as cross-country skiing, snow-shoeing and ice fishing. The Wolverine Mine is outside of these activity areas.

5.4 PHYSIOGRAPHY

The Wolverine Mine is in the Campbell Range, at the easternmost limit of the Pelly Mountains and abuts the broad Yukon Plateau to the north and east. The area consists of rolling, glacially scoured mountains with no significant peaks. Elevations on the property range approximately between 1,200 and 1,400 m. The main valleys are wide and U-shaped. Glacial till covers the majority of the lower lying valleys and there is significant infilling by post-glacial sediments.

Vegetation in the Wolverine Mine area ranges from open forest on lower elevation areas to dwarf shrub, herb, grass and lichen ecosystems at high elevations in the alpine. The majority of the mine site is within the subalpine zone, and the mine access road route is mainly in the boreal highland zone with some subalpine zone incursions. These zones are dominated by open forests of sub-alpine fir, black and white spruce, and lodgepole pine. Willow and scrub birch are common in riparian areas, areas with recent fire history, and steeper upland areas.

The aquatic ecosystems in the Wolverine Mine area are generally typified by cold and clean water. Wetlands abound and many small streams occur in the area. Waters in the upper drainage areas potentially affected by the Wolverine Mine (i.e. Go Creek, Wolverine Creek, small tributaries of Money Creek, Bunker Creek, and Putt Creek) all have limited to no fisheries potential. Go and Bunker Creeks flow into Money Creek, which flows east to Frances Lake in the Liard River drainage. Wolverine Creek flows north to Little Wolverine Lake, which in turn drains to Wolverine Lake and via Nougha Creek, discharges to the Finlayson River. Wolverine Lake and Little Wolverine Lake have been designated Conservation Waters.

The Wolverine Lake area lies within the limits of the McConnell Glaciation (youngest of the four glaciations in Yukon) and most of the geomorphic features in the area are related to this glaciation. McConnell glacial ice covered this area between 14,000 and 35,000 years ago. As the McConnell ice retreated and down-wasted, a complex network of ice tongues developed in valley bottoms. Morainal deposits are found at lower to mid-elevation and valley floors, and may contain a more complex assemblage of glacio-fluvial, colluvial and fluvial sediments (Mougeot 1996).

The main glacial soils in the vicinity of the tailings impoundment consist of up to 20 m of silty, sand and gravel, with cobbles. The area is underlain by bedrock strata generally paralleling the valley trend (i.e. striking in the direction of the valley). The bedrock consists of an interlayered sequence of volcanoclastic (rhyolite and quartz feldspar) and carbonaceous/argillic sediments, overlain with basalt. The iron

formation, which hosts the ore zone, trends northwest-southeast throughout the project area.

An explanation for each surficial material unit identified in terrain mapping is provided below. This information is based on field surveys conducted in 2005 by Axys Environmental Consulting Ltd.

5.4.1 *COLLUVIAL MATERIALS*

Approximately 24% of the Wolverine Mine area was identified as colluvium. This included sites with steep topography; especially where thin unconsolidated materials overlie steeply sloping bedrock (i.e. slopes in excess of 50%) as well as sites located in lower and toe slope positions. Colluvium is the result of mass wasting, dominated by the downward movement of materials due to gravity. This includes sites where the combination of porewater infiltration and gravitational forces induce rapid mass movement in certain sediments (i.e. slumping, debris flows) and bedrock (induced by frost shattering), the latter of which was identified in the Wolverine Mine area. Slow mass movements such as soil creep (i.e. slope wash) were also identified in the Wolverine Mine area.

Project area colluvial materials were generally unsorted, of variable-texture (silt loam to loam) and depending on the source area, often had a coarse fragment content greater than 50%. On upper slope positions, colluvium was generally less than 100 cm in thickness, while on mid and lower slope positions, colluvium was often thick enough to mask the underlying bedrock materials. For example, colluvial fans and cones were noted along lower valley positions.

Colluvium was dominated by soils of the Regosolic Order. The poor development of these soils is the result of the instability of the material. Of the Regosolic Order, only soils belonging to the Regosolic Great Group, which lacks an Ah horizon at least 10 cm thick, were mapped on colluvial materials. This included Orthic and Cumulic subgroups with Cumulic subgroups having buried mineral-organic layers and organic surface horizons of variable thicknesses. In general, most Regosols identified within the Wolverine Mine area were found on colluvium. Both Eutric and Dystric Great Groups of the Brunisolic order were also identified to a lesser extent. Although not found in this study, the potential for Turbic Cryosols also exists on north facing, poorly drained lower slopes underlain with permafrost (e.g. avalanche deposits).

5.4.2 *FLUVIAL MATERIALS*

Fluvial materials were most commonly found within the valley floors flanking contemporary streams, gently inclined slopes and fans at the base of slopes. These sediments, which accounted for approximately five percent of the study area, were generally well-sorted and consist of stratified gravel, sand and silt. Seepage within these materials is often not a concern due to the ease of pore water movement

through the sediment. Most of the contemporary stream channels are irregular in form and generate marginal fluvial plains with occasional back channels.

Cumulic Regosols and Humic Regosols are generally found on fluvial materials. These soils may have well-developed organic veneers and/or buried mineral-organic layers (as a result of periodic flooding) and organic surface horizons of variable thickness. Gleysolic soils are also common on fluvial materials as these areas are typically subjected to fluctuating water tables, period saturation and groundwater discharge. The field inventory reconnaissance did not provide a description of fluvial materials in the Wolverine Mine area.

5.4.3 *GLACIOFLUVIAL MATERIALS*

Glaciofluvial materials in the Wolverine Mine area were deposited by meltwater either in direct contact with the ice or beyond the ice margin as outwash. These deposits, which occupied about 2% of the Wolverine Mine area, varied in thickness from thin veneers to blankets of several meters. They were typically coarse-grained (gravel and sand), stratified and sorted. Glaciofluvial terraces generally formed the highest terraces preserved within the valleys; this feature was evident in the unnamed tributary of Finlayson River. Other glaciofluvial deposits were also found and described in this tributary as well as the Go Creek watershed.

Eutric Brunisols were the only soils identified on glaciofluvial deposits within the Wolverine Mine area. These soils tended to be well to rapidly drained and free of permafrost.

5.4.4 *LACUSTRINE MATERIALS*

Recent lacustrine sediments were noted along the shores of both Wolverine and Little Wolverine Lakes. These materials were limited in extent (less than one percent of the mine area) and typical of shoreline deposits subjected to periodic wave and ice action.

These sites are most-likely dominated by Regosolic, Organic and Gleysolic Soils.

5.4.5 *GLACIOLACUSTRINE MATERIALS*

Glaciolacustrine materials were only found in the Putt Creek valley and accounted for less than 1% of the Wolverine Mine area. These materials were less than 100 cm thick and underlain by glaciofluvial materials. They were also massive and dominated by silty loam soil textures; they lacked coarse fragments and were typically moderately well drained. Soils displaying these characteristics are often underlain by permafrost; the field inventory reconnaissance portion of this mine confirmed this potential by discovering thermokarst subsidence in the Putt Creek valley.

Only Eutric Brunisols were described and mapped on glaciolacustrine materials. The potential for Turbic Cryosols also exist in glaciolacustrine materials as evidenced by the presence of a thermokarst feature in the watershed of Putt Creek.

5.4.6 MORAINAL MATERIALS

Morainal materials had textures that varied from loamy sand to sandy clay loam and highly variable coarse fragment contents as described below.

- The moraine found in the watershed of Putt Creek was dominated by sandy loam textures with coarse fragment contents that varied from 25 to 50%.
- Morainal materials of Bunker Creek watershed were very coarse with textures varying from loamy to loamy sand and coarse fragment contents varying from 40 to 75%.
- High elevational moraines were dominated by coarse fragments (50 to 90%) and had varying textures (e.g. loamy sand, silt loam, loam, silty clay loam, sandy clay loam).
- The soil profiles of the morainal materials found in the Go Creek watershed were dominated by loamy and sandy clay loam textures and had highly variable coarse fragment contents (35 to 80%).

Overall, coarse-textured morainal materials were found extensively (approximately 64%) throughout the Wolverine Mine area, occurring on a variety of landscapes with thicknesses varying from being thick enough to mask the underlying bedrock topography to thin enough that it just caps it.

Soils belonging to the Brunisolic Order were the most commonly mapped soil on moraine. These soils have sufficient development to exclude them from the Regosolic Order but lack the degree or kind of horizon development specified for soils of the other orders. This group includes soils of various colours with both Ae (an upper surface horizon showing evidence of leaching) and weakly expressed B (a horizon under showing little evidence of clay, mineral and organic matter accumulations) horizons showing weak accumulations of aluminum or iron (Bfj) or an accumulation of clay (Btj). The most distinguishing characteristic of the Brunisolic Order is a brownish coloured Bm horizon of at least 5 cm.

Both Eutric and Dystric Great Groups of the Brunisolic order were classified and mapped within the Wolverine Mine area. These Great Groups are separated based on the pH of the uppermost 25 cm of soil. Dystric Brunisols have a pH less than 5.5 in the uppermost 25 cm while Eutric Brunisols have a pH of 5.5 or greater in the uppermost 25 cm. The occurrence of Eutric and Dystric Great Groups reflects the bedrock geology of the region as the pH of the bedrock strongly influences the pH of the surficial materials. Moreover, Dystric Brunisols are commonly associated with coarse igneous rocks at higher elevations whereas Eutric Brunisols are most often associated with sandy loam morainal materials. In this study, Dystric Brunisols on all

materials were only found on the slopes east of Go Creek and west of Bunker Creek. Orthic Eutric Brunisols was the most common soil found on morainal materials. This was evident in the Putt Creek watershed where both morainal materials and Orthic Eutric Brunisols dominated. Eluviated Eutric Brunisols were more common on well-drained, moraine especially in upper and crest slope positions.

Cryosolic (soils influenced by permafrost) soils have also developed within morainal materials, especially on mountain tops. Both Turbic and Static Cryosols were identified in the Wolverine Mine area. The profiles of Turbic Cryosols are cryoturbated. This was commonly observed in higher elevational morainal materials in the form of sorted and non-sorted nets, circles, polygons, and stripes. Soil exhibiting cryoturbation generally have permafrost within 2 m of the soil surface whereas Static Cryosols have permafrost within 1 m of the soil surface.

Soils of the Regosolic Order were also classified within morainal materials. These soils lack the well-developed profiles of the other soil orders. This poor development is the result of a number of factors including the youthfulness of the material and the impact of periglacial processes such as solifluction (slow gravitational downslope movement of saturated non-frozen overburden across a frozen or otherwise impermeable substrate). Within the Wolverine Mine area, Regosolic soils on morainal materials were only mapped on mountain tops. This includes soils belonging to the Regosolic and Humic Regosolic Great Groups. These Great Groups differ in that Humic Regosols have an Ah horizon of at least 10 cm thick.

Soils of the Gleysolic order were also occasionally mapped on morainal materials. These soils have features indicative of periodic or prolonged saturation by water; as such they are typified by reducing conditions. Gleysolic soils are usually associated with either a high groundwater table at some period of the year or temporary saturation above a relatively impermeable layer. Gleysolic soils often produce different vegetation communities than the surrounding soils due to the high moisture content and poor internal drainage. Both Orthic Regosols and Orthic Humic Regosols (Ah horizons of at least 10 cm) were mapped on morainal materials.

5.4.7 ORGANIC MATERIALS

Within the study area, organic accumulations were found in association with lacustrine, morainal and fluvial materials. Organic materials accounted for approximately 5% of the study and were most commonly associated with meadows, lower slopes and floodplains.

Soils belonging to the Organic order occur throughout the Wolverine Mine area. These soils are composed largely of organic materials and include soils commonly known as peat, muck, or bog soils. Organic soils generally occur in very poorly drained areas with long-term to permanent soil saturation. They are derived from the local vegetation and contain at least 30% organic matter by weight and classified based on the level of decomposition of the organic matter. The soil classes, based from least to most decomposed, are folic, fibric, mesic, and humic.

The four stages of decomposition are defined by the von Post scale of decomposition and the classification at the great group level is based on the properties of the second tier (~80 cm deep). Mesisols, which are composed primarily of partially decomposed organic materials, were most commonly found within the Wolverine Mine area. Fbrisols, composed largely of non-decomposed organic materials, were also identified and mapped. Organic Cryosols, which have permafrost within 1 m of the soil surface, were also described in the Wolverine Mine area.

6.0 HISTORY

Portions of this section have been taken from Pearson and Giroux (2006) and Moore (2009). Minor edits have been made for spelling, grammar and consistency.

6.1 EXPLORATION HISTORY PRE-2005

From Pearson and Giroux (2006):

Prior to being known as the FLD, the Wolverine Lake area was explored during the 1970s and early 1980s by a syndicate known as the Finlayson Joint Venture that was managed by Archer, Cathro & Associates (1981) Limited. Exploration activities identified several strong, multi-element soil geochemical anomalies that were directly associated with a vegetation kill zone that became known as the Fetish showing. This kill zone area was successfully cored by two small diameter drillholes in 1974. Both drillholes intersected low-grade copper and zinc sulphide mineralization; however, both drillholes were terminated above the favourable unit hosting the massive sulphide mineralization of the Wolverine deposit. Despite the presence of sulphide mineralization in the two drillholes, the quartz mining claims were subsequently allowed to lapse (Tucker 1999).

In 1993, Equity Engineering Limited (Equity) concluded that the area held promise for volcanogenic massive sulphide (VMS) mineralization, based on favourable stratigraphy and the presence of surficial geochemical anomalies. On behalf of Atna Resources Ltd. (Atna), Equity staked claims over the deposit area and carried out a field exploration program consisting of geological mapping, prospecting, and sampling. In 1995, Westmin Resources Ltd. (Westmin) optioned the Property from Atna (the Wolverine Joint Venture) and commenced a vigorous exploration program designed to evaluate the mineral potential of the more than 10 km of strike length of favourable volcanic stratigraphy. Exploration consisted of detailed geologic mapping and systematic grid soil sampling, which led to the definition of several multi-element geochemical anomalies. The strongest anomaly occurred in the area of the original Fetish showing.

Westmin commenced diamond drilling activities in August 1995 and the first hole into the Wolverine zone intersected massive sulphide mineralization in what is now known as the Wolverine zone. The fourth drillhole was down-dip of the first drillhole and intersected one of the highest grade intercepts on the property, 8.4 m of massive sulphide grading 7.63 g/t gold, 1358.3 g/t silver, 0.56% copper, 3.45% lead and 14.22% zinc. This discovery led to a decision by the Wolverine Joint Venture to conduct extensive drilling in the fall of 1995, as well as in both 1996 and 1997. After concluding the 1997 exploration program, 71 drillholes successfully defined the deposit and Westmin calculated an Inferred geological resource of

6,237,000 tonnes grading 12.66% zinc, 1.33% copper, 1.55% lead, 370.9 g/t silver and 1.76 g/t gold in January of 1998. The deposit was defined over a strike length of approximately 750 m and remained open down-dip to the northeast where it crossed onto the Wolverine claims owned by Teck Cominco Ltd. (Teck Cominco) (now Teck Resources Ltd.) (Teck Resources).

Also in 1996, Westmin commenced metallurgical testing on the Wolverine deposit and by late 1997, it had been determined that the concentrates from the Wolverine deposit contained unusually high levels of selenium; which at the time, was a deleterious contaminant that could significantly impact the marketability of the concentrates to smelters around the World. Further investigation of the metallurgical problem ended abruptly as a result of the takeover of Westmin by Boliden Limited (Boliden) in early 1998. Subsequent to the takeover of Westmin, Expatriate Resources Ltd. (Expatriate) (now Yukon Zinc Corp.) concluded a letter agreement in May 1998 for the purchase of Boliden's 60% interest in the Wolverine deposit. As a result, in June 1999 a new Wolverine Joint Venture (60% Expatriate and 40% Atna) announced that it was beginning an evaluation of alternative metallurgical methods for processing the concentrates from the Wolverine deposit.

In March of 2000, Expatriate announced that the Company had reached an agreement with Teck Cominco to purchase the Kudze Kayah Property and the Pelly Mountain Property, which includes the down-dip WOL claims adjacent to the Wolverine deposit. In May 2000, Expatriate commenced a drill program on the Pelly Mountain Property in order to expand the existing Wolverine Mine resource and further define the known mineralization. However, by October, the agreement with Teck Cominco had been terminated and Expatriate had to stop work on the prefeasibility study that was examining the joint development of Wolverine with Kudze Kayah Property deposits.

Exploration ceased on the Wolverine property until the summer of 2004 when Expatriate commenced a large-diameter drilling program to accomplish both infill on the known inferred geological resource and collection of samples for dense media separation (DMS) studies. In total, 1758.4 m were cored with $\frac{1}{4}$ of the massive sulphide drill core sampled for assay analysis at ALS Chemex Laboratories in North Vancouver, British Columbia and the remaining $\frac{3}{4}$ of the massive sulphide drill core sampled for DMS studies at SGS Lakefield in Lakefield, Ontario. Use of DMS technology is common in the metals, industrial minerals, and coal industries where there is significant density contrast between the materials being recovered and waste materials. The sample material from the pre-2004 drill core was mixed based on 30% hanging wall argillite, 50% massive sulphide, and 20% footwall argillite to form a blended sample. The results achieved a fifty percent weight rejection to the Float (waste) fraction. The loss of metals to the waste material was less than two percent of zinc, copper, and lead and approximately five percent for silver and gold. The DMS testing indicated that the samples tested were very amenable to heavy media separation; achieving a fifty percent weight rejection to the float fraction. In a full scale plant, the combined "sink" and "fines" products from the DMS plant feed would require further

processing by conventional grinding and flotation processes, while the nearly barren ‘float’ product from DMS would be discarded.

6.2 EXPLORATION HISTORY POST-2005

From Moore (2009):

In 2005, Expatriate changed its name to Yukon Zinc Corporation. A drilling program was completed finishing 59 drillholes totalling 11,712.5 m. These and previous drillholes were used to calculate a reserve and submitted as part of a NI 43-101 report(s) in 2006 and 2007. Underground development commenced, and totalled 450 m of underground advancement. This included 110 m of development within the massive sulfide mineralization. A bulk sample was submitted for metallurgical testwork. Yukon Zinc signed a Socio-economic Participation Agreement for Wolverine Project with Kaska First Nation, which included benefits such as employment, scholarships, training in all project phases and business opportunities.

Yukon Zinc had applied for a Quartz Mining License with the Yukon Territorial government and in 2006; the Quartz Mining License QML-006 was received for Wolverine Mine construction.

In 2007, Yukon Zinc completed the optimized feasibility study with Wardrop Engineering Inc. An access road from the Robert Campbell Highway to the airstrip and portal area was completed. Environmental baseline studies were finalized and major permitting was completed, the project acquired the Type A Water License and completed all major permit requirements for mine operations.

In 2009, Yukon Zinc contracted Kluane Drilling Company to diamond drill WV09-221 to WV09-249 drillholes. The drillholes were focused in the Wolverine deposit area, in the Lynx and Wolverine zones. The diamond drilling was conducted on claims: Foot A-12, YB71275; Kink 3, YA69009; Foot 10, YB45963; and Foot 12, YB51609.

6.3 HISTORICAL GEOLOGICAL RESOURCE ESTIMATION

From Pearson and Giroux (2006):

From 1995 to 2000, there have been six mineral resource estimations for the Wolverine deposit after successive, successful exploration campaigns that continued to delineate and define both massive sulphide and semi-massive sulphide mineralization.

There has been incremental growth in the historical geological mineral resources through continuing exploratory drilling on the Wolverine deposit from the Westmin ownership through to Yukon Zinc ownership. It should be noted that the metal grades have remained remarkably consistent through each subsequent increase in tonnage. Though the base case studies estimated from 1995 through to 2000 are

accurate historical geological mineral resource estimates for the Wolverine deposit and were considered relevant at the time, these historical estimations should be considered Inferred mineral resources, as they predate NI 43-101. However, from the detailed work these historical resources are of good quality and met the Industry standards of the day.

6.3.1 WESTMIN HISTORICAL RESOURCE ESTIMATIONS (1995-1996)

From Pearson and Giroux (2006):

Both the 1995 and 1996 internal Westmin historical geological Inferred mineral resource estimates were done using Medsystem (Minesight) software and are presented in Table 6.1. In 1996, the total drillholes in the Wolverine deposit that intersected solid sulphide mineralization rose to 49 with the size of the deposit expanding to 725 m by 400 m in plan view. The interpreted tabular zone of solid sulphide mineralization has an average dip of 25 to 32° to the north-northeast and striking 32° east of true north.

Table 6.1 Historical Mineral Resource Estimations (1995-1996)

Year	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
1995	3,052,400	1.87	350.2	1.25	1.43	12.99
1996	5,311,000	1.81	359.1	1.41	1.53	12.95

The geological control in creating the model came from detailed core logging by Westmin geologists who coded hanging wall, solid sulphide, and footwall rocks in order to differentiate them from each other. The composite assay results were done across the entire sulphide mineralization; however, it is important to note that no mineralization in the hanging wall and footwall rocks was included in the geological resource estimations. Only semi-massive and massive sulphide rock units were used; which allowed for a very accurate geological model to be created.

A decision was made to use a gridded seam model (GSM) in order to honour the 3-D locations of the drill intercepts and to allow for the cutting of section maps through the GSM. In GSM modeling, a 2-D horizontal grid, typically 10 m by 10 m, is overlain on the mineralization. The third dimension is defined as the difference in elevation of the top and bottom of the mineralization; which allow for the calculation of the thickness, either true or vertical, of the mineralization. Upon calculating the thickness of the mineralization, metal grades can be coded and each zone can be labelled. It should be noted that a boundary was placed on top of the GSM in order that the estimation of the geological resources could be constrained to an area not greater than 25 m outside of any drill hole pierce point. Interpolation of grade was done by inverse distance cubed with a 100 m by 100 m search, a strike of 100°, and a dip of 32°.

Westmin ran an internal quality control program in 1996 that included the submittal of 126, 250 g splits off the original rejects stored at ALS Chemex. These splits

from the original rejects were sent to Rocky Mountain Geochemical (Rocky Mountain) in Salt Lake City, Utah, USA. Comparisons between ALS Chemex and Rocky Mountain returned favourable results for both base and precious metals. Specific gravity measurements were all analytical measurements taken at ALS Chemex; which allowed for a length density weighted average of the composites to be calculated.

6.3.2 TURNER HISTORICAL RESOURCE ESTIMATION (1997)

The 1997 internal Westmin historical geological Inferred mineral resource estimation was done by Polygonal methodology. The overall parameters of the Westmin database used here were the same as employed in 1995 and 1996 except that the sulphide mineralization had now been intersected by 68 drillholes.

Turner (1997) described the first step in the geological Inferred mineral resource estimation was the calculation of the drillhole pierce point on the interpreted Wolverine Horizon (base of solid sulphide mineralization) for each intersection. These pierce points were calculated from basic trigonometric principles using the drill collar coordinates, the in-hole depth of the massive sulphide mineralization, and the down-hole survey results (Sperry Sun). The new outline of the solid sulphide mineralization was defined by ensuring that the outline reflected all the drill holes with sulphide intersections, as well as honouring the drill hole spacing. In certain areas, notably along the fringes of sulphide mineralization, positioning the outline was facilitated by simply bisecting the distance between drillholes that did and did not intersect sulphide mineralization.

Westmin constructed two polygonal models for the Inferred geological resource in 1997. The first polygonal model involved the connection of the centroids of the triangles that surrounded a single drill hole that producing a relatively simple square, pentagon, and/or hexagon. The second polygonal model used more complicated shapes that were constructed using the centroids of the surrounding triangles and midpoints of the sides of the triangles. Then for each model, the area of each polygon could be easily obtained in AutoCAD in square metres. As a validation exercise, Westmin compared the area from each of the two models by comparing the areas of the 68 polygons around the 68 drillholes and the variance was only 0.0002 m² (Turner 1997).

Tonnage for each polygon was calculated by multiplying the length-density weighted assay composite by the vertical thickness and by the area for each drill hole. Once a tonnage value was calculated for each polygon, all 68 polygon tonnages were summed and this then represented the tonnage for the Wolverine deposit. Tonnage values for each of the two polygonal methodologies employed by Westmin in 1997 are presented in Table 6.2 and Table 6.3.

Table 6.2 Historical Mineral Resource Estimation (Polygonal Method 1 – 1997)

	Tonnes	True Thickness	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
Upper Lens	499,602	1.9	1.76	407.8	0.76	2.22	11.73
Main Lens	5,297,238	3.9	1.77	370.5	1.26	1.57	12.68
Eagle Zone	81,014	0.98	1.38	328.83	0.4	2.12	8.03
SMSS	256,437	1.85	0.56	108.79	1.74	0.19	4.45
Total	6,134,292		1.71	362	1.23	1.57	12.19

Table 6.3 Historical Mineral Resource Estimation (Polygonal Method 2 – 1997)

	Tonnes	True Thickness	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
Upper Lens	496,602	1.9	1.78	412	0.76	2.24	11.84
Main Lens	5,263,948	3.88	1.74	366.5	1.27	1.56	12.65
Eagle Zone	84,615	1.01	1.38	328.83	0.4	2.12	8.03
SMSS	256,437	1.85	0.56	108.79	1.74	0.19	4.45
Total	6,100,712		1.69	358.9	1.23	1.57	12.18

The 1997 historical geological Inferred mineral resource was originally classed as Probable or Indicated Ore by Westmin (Turner 1997).

6.3.3 JURAS-TUCKER HISTORICAL RESOURCE ESTIMATION (1997-1998)

In 1998, Westmin initiated another historical geological resource to be calculated by MRDI Canada (MRDI) (now AMEC). MRDI determined the geological resource for the Main Lens of the Wolverine Deposit following the 1997 diamond drill program where it was noted that the Main Lens contains 95% of the geological resource in what appears to be a single tabular lens.

The 1998 geological resource estimate was based on 68 diamond drill holes that intersected sulphide mineralization. The technical approach employed by MRDI for the modeling the Main Lens was to approximate the shallow dipping, apparently tabular Wolverine massive sulphide body with a GSM. This process calculated the shape of the “seam” (massive sulphide unit) by creating two digital terrain models (DTM) using values for the ZTOP and ZBOT of the solid sulphide mineralization in the databases. The surfaces were then gridded creating ZTOP and ZBOT values for each block. These values were inputted into the GSM. Inverse Distance interpolation, controlled by geology code, calculated the GSM block grades and then populated into the GSM by level. Vertical thickness of the GSM seam was calculated for each block from the gridded seam model (Juras 1998; Tucker 1999).

The Westmin database, used in the modeling, included all assay, density and coded geological information. Individual density measurements from pulps were

measured at the time of assay at ALS Chemex. The data was loaded into Medsystem (Minesight) software, which was used to undertake the estimation of the resource. Each assay interval was coded with a density and geologic code. All of the assay intervals in each of the drill holes that possessed a similar geologic code were averaged into a single composite using both sample length and density as the weighting factors. The top and bottom of each composite; which is defined as the top and bottom of the respective seam or solid sulphide horizon; were stored as X, Y and Z coordinates in a model item called ZTOP and ZBOT. The GSM was checked by plotting plans and sections with drill hole assay and composite values, overlain by the relevant GSM blocks. Statistical checks of the model were calculated to confirm integrity of the model. Approximately 5% of the resource is in two upper lenses that were intersected by 13 and 4 drillholes respectively. The resources in these two lenses were also calculated using the 60 gridded seam model. This work was supervised by Terry Tucker for Westmin and is documented in an internal Westmin memorandum dated March 20, 1997. The historical geological resource published in early 1998 is summarized in the Table 6.4 (Juras 1998; Tucker 1999). MRDI grouped the Main Lens resource into three categories based on distance from a drill hole; however, it should be noted that MRDI did not classify the geological resources into the usual mineral resource categories. The MRDI applied definitions as stated in Mineral Resource/Reserve Classification: Categories, Definitions, and Guidelines, Canadian Institute of Mining (CIM) Bulletin September 1996, pp. 39-44. Tucker (1999) noted that MRDI grouped the geological resources of the Main Lens into measured (44%), indicated (48%) and inferred (8%) categories. The measured mineral resources are blocks within 25 m of a drill hole that contain mineralization, indicated mineral resources are those between 25 and 50 m of a drill hole containing mineralization, and inferred mineral resources are those more than 50 m from a drill hole containing mineralization. The small tonnage in the Inferred category indicates that there is generally less than 100 m separating drillholes in any direction. Tucker (1997) did not categorize resources in the Upper lenses. This author has classified the West Upper Lens 1 and 2 as in the Inferred geological resource category (Juras 1998; Tucker 1999).

Table 6.4 Historical Mineral Resource Estimation (1998)

	Tonnes	Resource Category	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
Main Lens*	2,600,727	Measured	1.79	370.7	1.33	1.54	12.56
	2,849,031	Indicated	1.81	378.4	1.34	1.54	12.64
Measured + Indicated	5,449,758		1.8	374.73	1.34	1.54	12.6
Main Lens*	444,691	Inferred	1.53	337.5	1.61	1.4	12.98
Upper Lens 1**	30,901	Inferred	3.11	818.9	0.45	5.36	14.63
Upper Lens 2**	312,012	Inferred	1.22	308.2	0.84	1.5	12.96
Inferred	787,604		1.47	344.78	1.26	1.59	13.04

Note: *Stephen Juras, MRDI Canada, February 16, 1998

**Terry Tucker, Westmin Resources Ltd., March 20, 1997

The Wolverine deposit is open down dip to the northeast, towards Teck Cominco's (now Teck Resources) Wolverine Claims; which were drilled in 2000 by Expatriate (predecessor to Yukon Zinc). Three drillholes were successfully completed with two intersecting significant solid sulphide mineralization along the down plunge extension of the Lynx zone. The third drillhole did, however, intersect the horizon that hosts the Wolverine deposit; however, it appears to have wandered from the plunge line. These three drillholes confirmed the continuity of the favourable geological environment down-plunge of the Wolverine deposit. No further work has been done since 2000 on the down-plunge geology of Wolverine.

The northwest and southeast edges of the Wolverine deposit are defined by five (5) wide spaced drillholes; noting that additional drillholes along the outside margin of the known mineral resources could yield either small, contiguous tonnages of massive sulphide mineralization or possible fault juxtaposed tonnages. These areas outside of the known resource provide project opportunity for deposit growth.

6.3.4 NILSSON HISTORICAL RESOURCE ESTIMATION (2000)

By 2000, the Wolverine deposit had advanced to the prefeasibility stage as part of the combined Finlayson Project; which at the time, was a joint development project with the nearby Kudze Kayah Property deposits, noting that the Finlayson Project was subsequently cancelled. As a result, the Wolverine deposit required a new resource model for the current development plan. A drillhole database was provided with an associated survey file. As well as containing assay and geochemistry data, the files contained geology and interpreted lithologic codes. Topography was provided on 5 m contour intervals. The object of the current exercise was to develop an update of the first phase of the Wolverine Lynx zone resource model in order to provide a basis for the mine plan. Key elements necessary for preliminary mine planning included the following: topographic surface; de-surveyed boreholes; assay statistics; composite data; composite statistics; zinc, copper, lead, gold, silver and SG distribution; hanging wall contours for massive sulphide mineralization; footwall contours for massive sulphide mineralization; and thickness isopachs for solid sulphide mineralization (Nilsson 2000).

A preliminary GSM was developed for all of the horizons using the composite data. Initial interpretation of the surfaces was done by gridding triangles of drillhole data. A first pass on all surfaces resulted in conflicting surfaces of relatively thin horizons. Restricting the interpolation of each horizon surface to the area of influence of data on that horizon improved the results but still resulted in conflicting surfaces. As a result, the Wolverine-Lynx Horizon only has been used in this mine plan pending rationalization of the other zone surfaces (Nilsson 2000).

The assay database was converted from Excel format to ASCII format, combined with the survey data and loaded into a Medsystem format binary file. The assay data were used to calculate composite intervals based upon the geology code. These composites are the length of the zone intercept. The density model used for

mine modeling was based upon inverse distance power weighting of the SG composite value (Nilsson 2000).

The hanging wall and footwall surfaces for the Lower Lynx and Wolverine zone were modeled by gridding the drillhole intercept data. These ZTOP and ZBOT surfaces were then used to calculate the gridded seam ZTOP-ZBOT variable THCK. The zinc, copper, lead, silver and gold grades were interpolated by the inverse distance power method using a 100 m search radius and power of three. The number of composites was limited to three and a quadrant search was used.

The geological resource estimate summary for the Wolverine deposit from 2000 is shown in Table 6.5. The total indicated and inferred resource is 5.4 million tonnes in a zone with an apparent average thickness of 4.68 m and a grade of 13.06% Zn, 1.43% Cu, 1.59% Pb, 378 g/t Ag and 1.74 g/t Au. The drill indicated resource within 50 m of a composite is 4.9 million tonnes 13.00% Zn, 1.43% Cu, 1.58% Pb, 379 g/t Ag and 1.76 g/t Au (Nilsson 2000). It should be noted, however, that the 2000 historical resource estimate did not take into account the hanging wall zones of massive sulphide mineralization, as calculated by Tucker (1997).

Table 6.5 Historical Mineral Resource Estimation (2000)

<i>Distance of Search</i>	<i>Tonnes</i>	<i>Category</i>	<i>Au (g/t)</i>	<i>Ag (g/t)</i>	<i>Cu (%)</i>	<i>Pb (%)</i>	<i>Zn (%)</i>
0-25 m	2,195,738	Indicated	1.798	372.1	1.409	1.574	13.05
25-50 m	2,745,209	Indicated	1.726	385.3	1.453	1.575	12.97
	4,940,947	Indicated	1.760	379.0	1.430	1.580	13.00
>50 m	498,300	Inferred	1.509	365.3	1.359	1.695	13.61

6.3.5 PEARSON – GIROUX RESOURCE ESTIMATE 2006

In October 2005, Yukon Zinc commissioned Cliff Pearson and Gary Giroux to complete a new mineral resource estimation and data compilation as part of an independent technical (NI 43-101) report for the Wolverine deposit. The resource estimate was based on 159 diamond drillholes using ordinary kriging as the grade interpolator.

Pearson and Giroux created 3D geological interpretations of the Lynx, Saddle and Wolverine zones, including additional hanging wall zones not considered in previous resource estimates. Geostatistics and variography were completed for zinc, copper, silver, gold and lead across the entire deposit.

The following is from the Pearson and Giroux (2006) report:

For any block with a proportion of mineralized solid greater than zero, kriging was attempted. 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 ¼ 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 ½ 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.

For their interpretation of resource classification, Measured was generally allocated to the cells successfully interpolated in the first search pass, Indicated was assigned to second search pass cells, and Inferred was assigned to cells interpolated with three or more search passes.

Table 6.6 tabulates the mineral resource inventory for Pearson and Giroux's estimate. Note that the cut-off is calculated as a function of NSR at US\$40.00.

Table 6.6 Summary of Wolverine Deposit Mineral Inventory in 2006 by Pearson and Giroux

Classification	Cut-off (US\$)	Tonnes (t)	Grade				
			Zn (%)	Ag (g/t)	Cu (%)	Au (g/t)	Pb (%)
Measured	40.00	509,000	12.17	292.27	1.15	1.47	1.46
Indicated	40.00	4,002,000	12.03	359.45	1.16	1.71	1.58
Measured +Indicated	40.00	4,511,000	12.05	351.86	1.15	1.65	1.57
Inferred	40.00	1,694,000	12.16	385.11	1.23	1.71	1.74

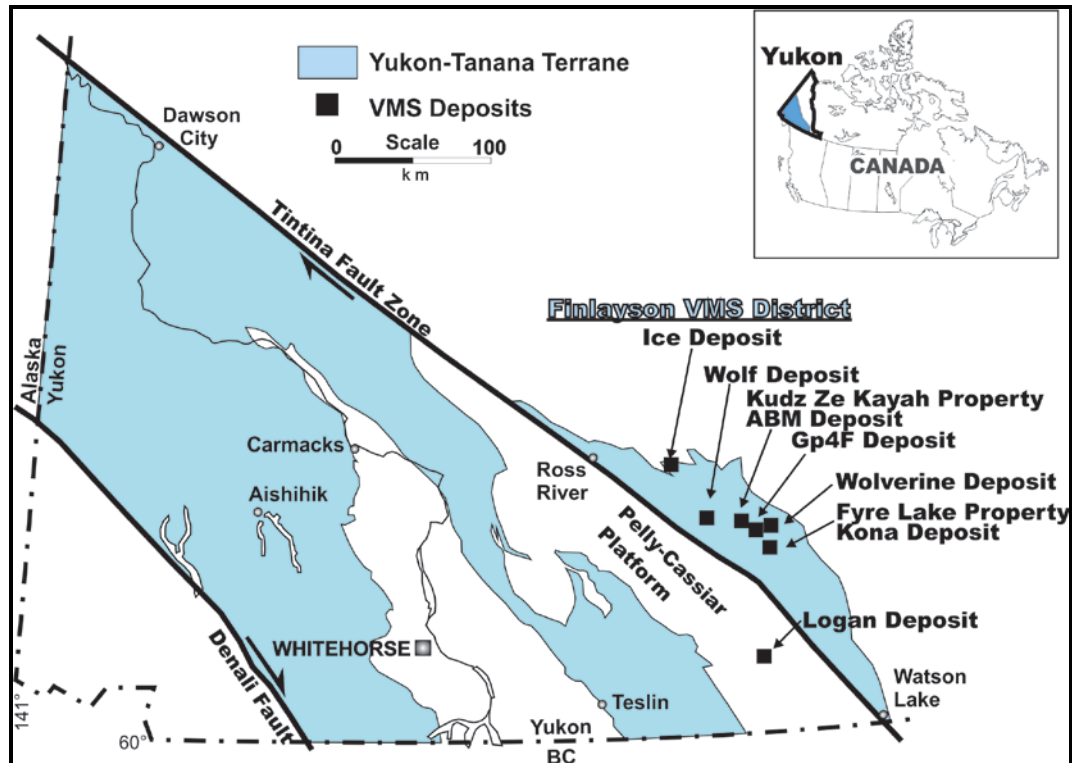
7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Property is a volcanic-hosted massive sulphide (VHMS) deposit in the FD, Yukon. The Wolverine deposit lies within the Yukon-Tanana pericratonic terrane (YTT), which is composed of poly-deformed metamorphosed sedimentary, volcanic and plutonic rocks derived from Late Devonian to Permian protolith arc assemblages. The FD comprises a dominant portion of the isolated outlier of YTT and Slide Mountain Terranes (SMT) northeast of the Tintina Fault. To date, 41 VMS occurrences and five deposits have been discovered at different stratigraphic levels within the FD and approximately 34 Mt of massive sulphides have been delineated since the district was recognized in the mid-1990s. Prior to the discovery of the Teck Resources ABM and GP4F deposits, Yukon Zinc's Wolverine and Ice deposits and Pacific Ridge Exploration's Kona deposit in the FD in the mid-1990s, VMS deposits were not widely recognized as significant deposit types in Yukon.

The FD forms a crescent-shaped 300 km x 50 km area that extends from Watson Lake (south) to Ross River (north). It is composed of several fault- and unconformity-bounded groups and formations of Devonian-Mississippian igneous-sedimentary rocks, juxtaposed against Proterozoic-Palaeozoic strata (Figure 7.1). These groups and formations include the strongly foliated and lineated layered metasedimentary and volcanic rocks of the Grass Lakes Group, the Grass Lakes plutonic suite. These are unconformably overlain by the carbonaceous metaclastic and volcanic rocks of the Lower Mississippian Wolverine Lake Group (Figure 7.2).

Figure 7.1 Yukon Tanana Terrane



The FD has been divided into three distinct successions separated by regional unconformities (Piercey 2001; Murphy and Piercey 1998, 1999; Murphy et al. 2001). The lower most unit, called the Grass Lakes Succession is comprised of mafic and felsic metavolcanic rocks, carbonaceous metaclastic rocks, marbles, and granitic orthogneiss. The Fyre Lake deposit is hosted in the lower portion of this succession. The ABM and GP4F deposits on the Kudze Kayah Property are located within the upper portion of this succession. The middle unit, called the Wolverine Succession, consists of carbonaceous argillite, felsic volcanics and high-level intrusions, and as well as exhalative carbonate and/or iron oxides. The Wolverine polymetallic VHMS deposit occurs near the base of Wolverine Succession. The uppermost unit, called the Campbell Range Succession, is comprised of mafic metavolcanic rocks and wackes. The Campbell Range Succession hosts the Ice VMS deposit (Figure 7.2, Figure 7.3 and Figure 7.4).

Figure 7.2 Regional Stratigraphy (after Piercey 2001)

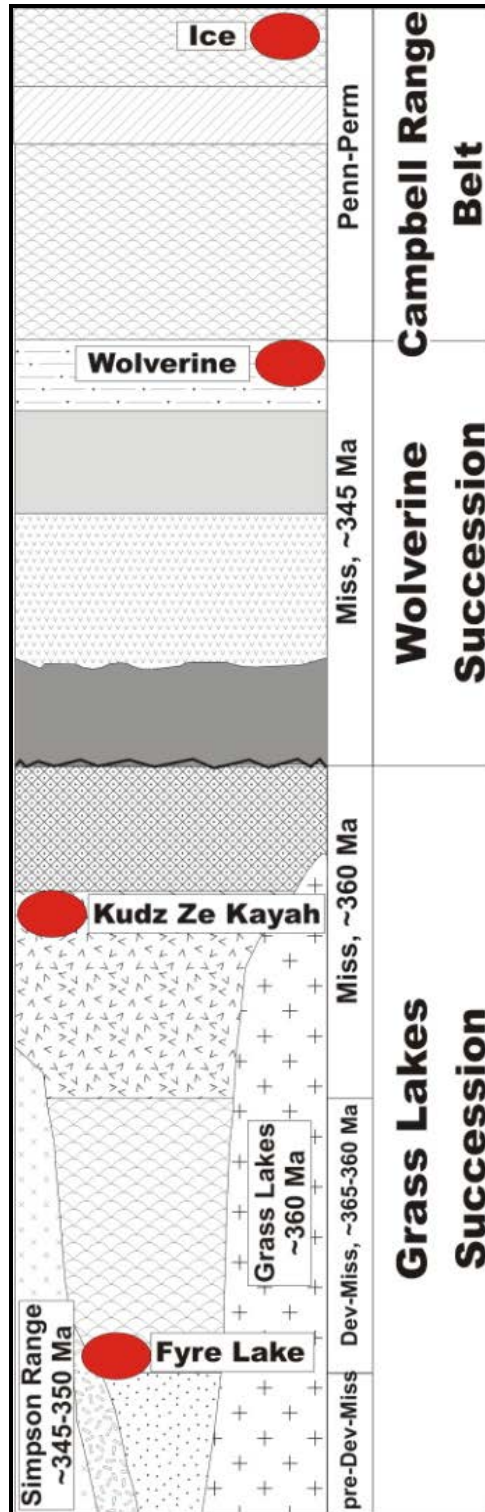


Figure 7.3 Geologic Environs of the FD (Murphy et al. 2006)

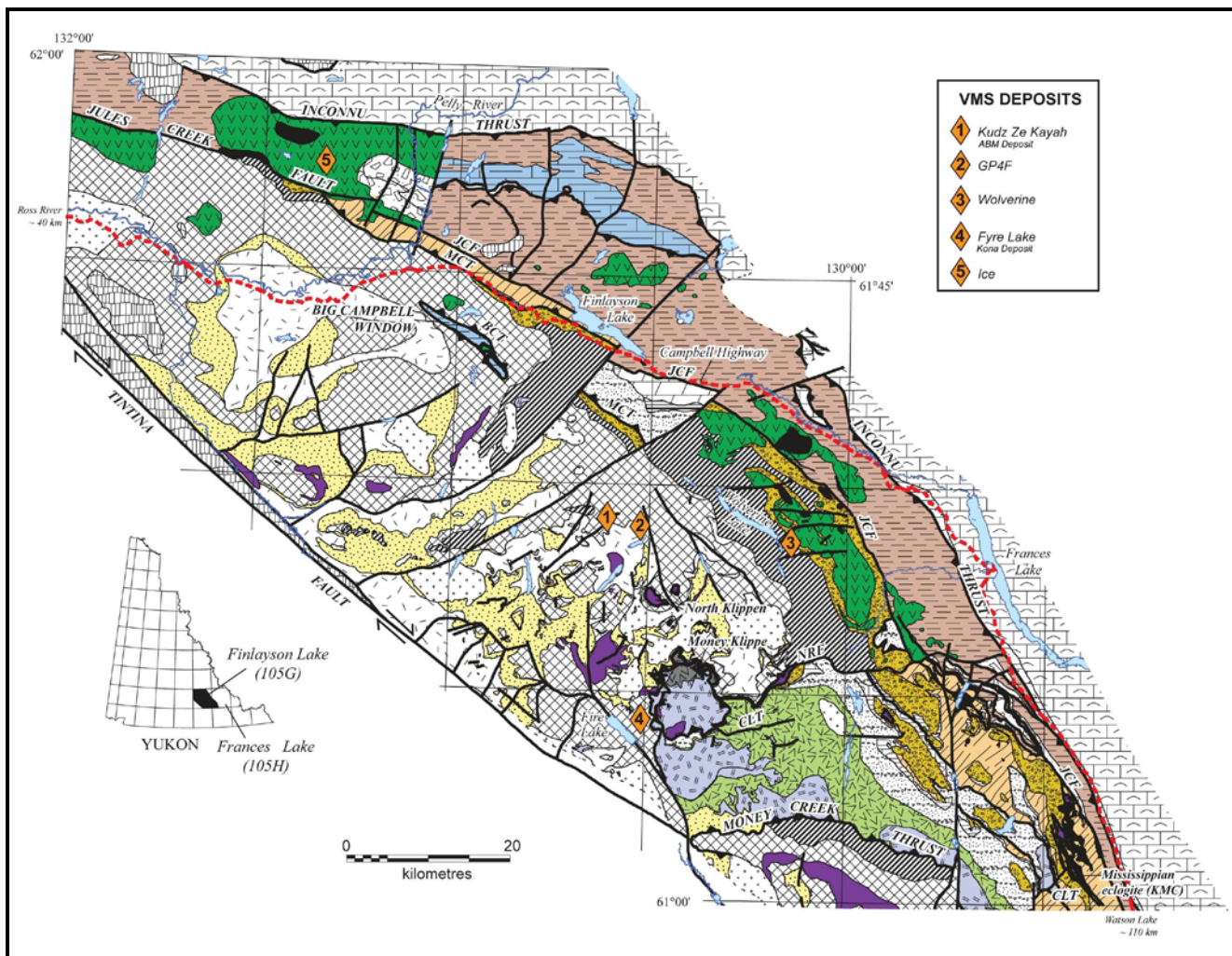
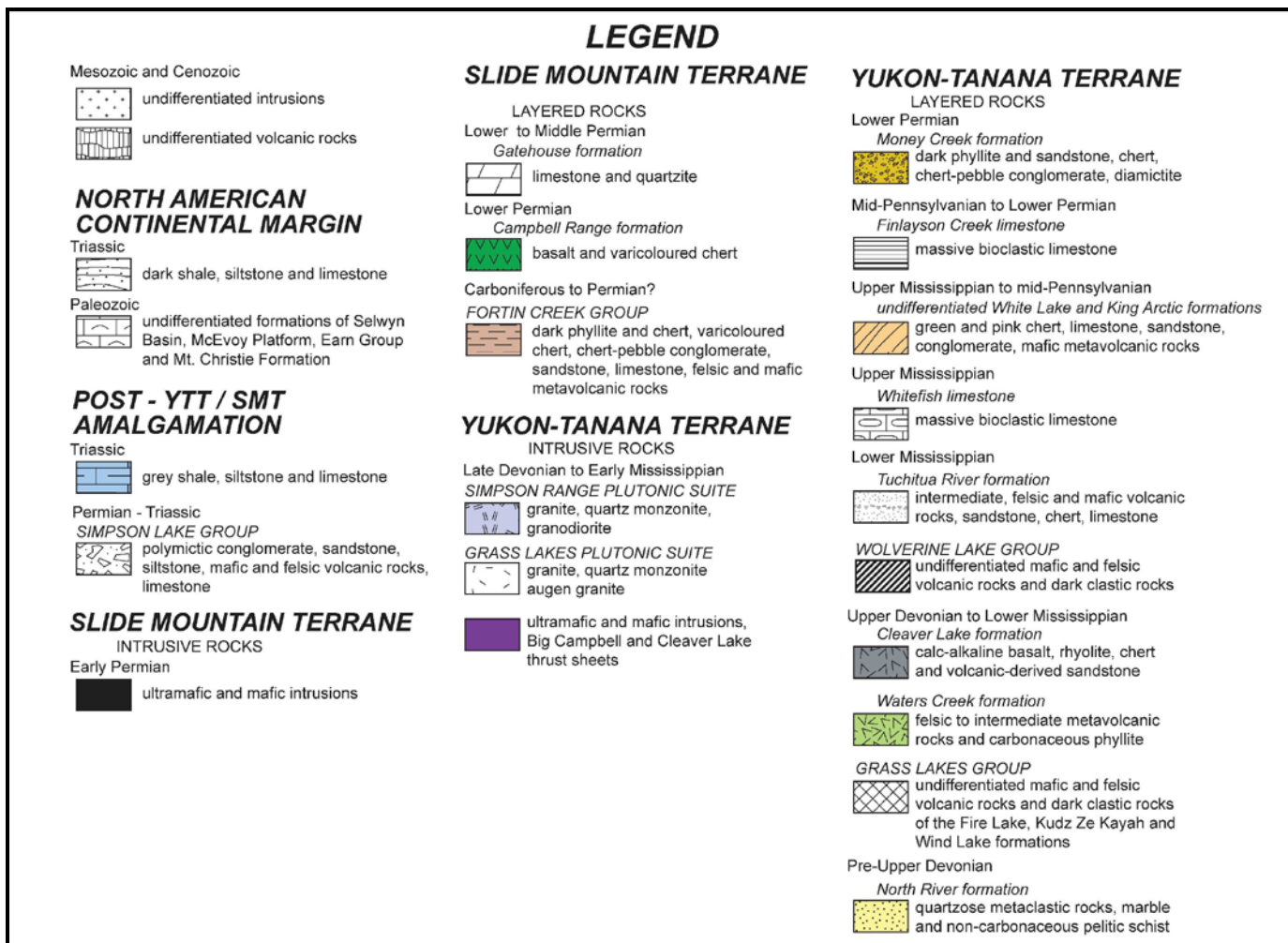


Figure 7.4 Legend of the Geologic Environs in the FD



7.2 DEPOSIT GEOLOGY

7.2.1 INTRODUCTION

The following discussion is from Bradshaw (2003). Minor edits have been made for spelling, grammar and consistency

The Wolverine polymetallic massive sulphide deposit is located in the FLD of southeastern Yukon Territory, Canada. The deposit occurs in a highly deformed, but coherent stratigraphic succession of Early Mississippian to Early Permian metavolcanic and metasedimentary rocks. Regional geological mapping and lithogeochemical studies are consistent with its formation on the margin of an evolving ensialic back-arc ocean basin between the Yukon-Tanana terrane and the ancestral North American craton. Local stratigraphy consists of four major rock units including (from oldest to youngest): (1) quartz- and feldspar-phyric volcanoclastic rhyolite, carbonaceous argillite and rhyolite porphyry; (2) interbedded argillite, aphyric rhyolite and magnetite-carbonate-pyrite exhalite; (3) fragmental rhyolite; and (4) interbedded carbonaceous argillite, greywacke, basalt and rhyolite.

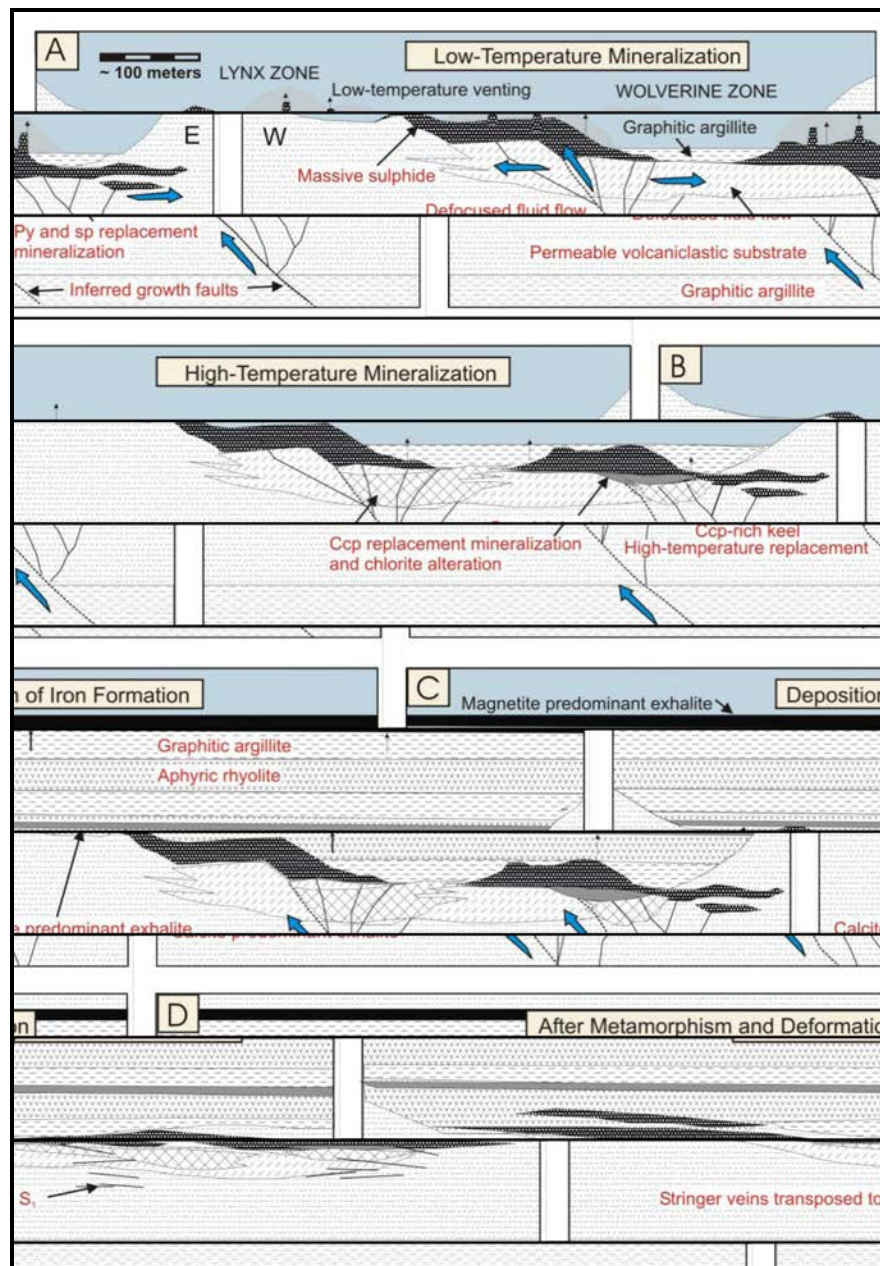
Sulphide mineralization occurs at the "Wolverine and Lynx horizons", which is located at a contact which marks a significant change in the character of volcanism between the footwall and the hanging wall of the deposit. The footwall consists of a sequence of carbonaceous shale, quartz- and feldspar-phyric volcanoclastic rhyolite, and minor feldspar-phyric rhyolite sills, whereas the hanging wall consists of intercalated aphyric rhyolite, carbonaceous shale, and carbonate-pyrite exhalite. Two separate zinc-lead-silver massive sulphide lenses, the Wolverine and Lynx horizons, situated in the same stratigraphic horizon, comprise most of the Wolverine deposit (Figure 7.5). They are laterally connected by stratabound, semi-massive replacement-style zinc-lead-silver mineralization, called the Saddle zone. Copper-rich mineralization commonly replaces zinc-lead-silver mineralization at the base of the lenses and in the footwall zones indicating that primary metal and mineral zonation are preserved (Figure 7.5). Multiple zones of sulphide stringer veins and regions of conformable chlorite-sericite-carbonate alteration are developed within permeable volcanoclastic rocks of the footwall. Barium-rich phengitic mica, biotite, magnesium (Mg)-rich chlorite and siderite are associated preferentially with massive sulphide mineralization.

The zinc-lead-silver massive sulphide lenses formed at 264 +/- 33 degrees celcius based on arsenopyrite geothermometry. The sulphide stringer zones formed at slightly higher temperatures (282 +/- 7 degrees celcius) based on estimates derived from hydrothermal chlorite geothermometry. Late-stage quartz-chalcopyrite-pyrite veins formed at temperatures between 265 degrees celcius and 353 degrees celcius. Based on fluid inclusion microthermometry, mineralizing fluids had relatively low salinities, and evidence of fluid boiling or phase separation is absent. Mineralization is estimated to have formed at water depths greater than 1 km

through a combination of seafloor hydrothermal venting from multiple sulphide mounds and sub-seafloor replacement processes. Sulphur stable isotopes indicate that sulphur was derived from a combination of biogenic and inorganic reduction of seawater sulphate within a partially closed, anoxic basin. The reduced nature of the ambient bottom water in the basin likely contributed to the preservation of the sulphide mounds.

The geological setting, styles of mineralization and physio-chemical conditions of the mineralizing fluids all indicate that the Wolverine deposit formed in a geological environment transitional between classic bimodal volcanic-hosted massive sulphide deposits and sedimentary exhalative massive sulphide deposits.

Figure 7.5 Wolverine Deposit Formation (Bradshaw et al. 2008)



7.2.2 WOLVERINE SUCCESSION STRATIGRAPHY

Unit 1 Footwall

One of the dominant units in the Wolverine stratigraphic column is the footwall rocks that are composed of volcaniclastic, carbonaceous sedimentary and porphyritic intrusive rocks. This heterogeneous unit contains the following lithologies:

- Green to grey quartz- and feldspar- crystal bearing rhyolite volcanoclastic rock with variable amounts of interbedded carbonaceous argillite that contains abundant flattened, fine- to coarse-grained fragments of rhyolite volcanic rock. This lithology can contain up to several volume percent of 1 to 5 mm diameter, subhedral to euhedral, K-feldspar crystals and 1 to 2% rounded greyish blue quartz “eyes”, although either, or both, may be absent. The relative abundances of feldspar crystals, quartz eyes, and volcanic rock fragments vary significantly in this lithology throughout the deposit.
- Black to grey, aphanitic to fine-grained, carbonaceous tuffaceous argillite. This rock commonly contains several volume percent grey tuff as layers or clasts and may contain up to several volume percent rounded blue quartz eyes up to 1 mm in diameter. This footwall argillite is similar to the carbonaceous argillites in the hanging wall but is distinguished from it locally by a slightly coarser grain size, lighter colour, and presence of quartz.
- Grey, weakly foliated, K-feldspar-phyric rhyolite porphyry. Contains 5 to 10% subhedral to euhedral, 5 mm to 1 cm diameter K-feldspar phenocrysts in a grey aphanitic siliceous groundmass. This unit only rarely contains quartz eyes (Tucker 1999; Bradshaw 2003).

The potassium feldspar-phyric porphyries occur an average distance of 20 m below the lower limit of the massive sulphide unit and are interpreted to be intrusions. The exact role of these intrusions with respect to the mineralizing event is unclear at the present time; however, the intrusions could be emplaced along the growth or scarp faults of the caldera (Tucker 1999; Bradshaw 2003). A more detailed description of the feldspar-phyric porphyries can be found in Piercey et al. (2001).

The massive sulphide lens typically lies immediately above and in some cases within the felsic volcanoclastic rocks; noting that there is a gradual transition from the upper felsic volcanoclastic rocks to the underlying carbonaceous argillite. Bradshaw (2003) noted that the felsic volcanoclastic rocks range in thickness from 1 to 60 m. The appearance of carbonaceous argillite unit in the footwall rocks marks the base of the sequence that is host to the Wolverine deposit. Bradshaw (2003) noted that the carbonaceous argillite unit in the footwall ranged in thickness from 30 to 50 m.

Unit 2 Mineralization

The Wolverine deposit has two areas of thicker mineralization known as the western Lynx and eastern Wolverine zones separated by a zone of relatively thin sulphide mineralization called the Saddle zone. Previous interpretation suggested that the Saddle zone is structurally controlled and bound by a sub vertical fault zone on either side. The Wolverine and Lynx zones are very similar in terms of their mineralization. They both contain four distinct styles of mineralization:

- Pyrite-rich Massive Sulphide: fine-grained, non-laminated pyrite-rich massive sulphide

- Stratiform Mineralization: fine-grained, laminated, sphalerite+pyrite+galena-rich massive sulphide
- Semi-massive Replacement Mineralization: fine-grained, sphalerite+pyrite-rich replacement style massive sulphide
- Stringer Zone Mineralization: chalcopyrite-rich stringer and disseminated semi-massive sulphide.

Volumetrically the laminated, sphalerite + pyrite-rich massive sulphide is the most abundant style sulphide mineralization and occupies the lower portion of the ore body, while the pyrite-rich massive sulphide generally occurs in the upper portion and flanks of the ore lenses. The replacement style mineralization and stringer zone occur below the ore zone and extend into the strongly chlorite altered footwall.

Pyrite-Rich Massive Sulfide

The pyrite-rich massive sulphide generally occurs in the upper portion and flanks of the ore lense. Pyrite in massive layered sulfide lenses typically occurs as fine-grained anhedral masses and as coarse-grained porphyroblasts. It also occurs as rare, very fine grained, anhedral, colloform masses and framboids near the hanging wall contact with graphitic argillite (Bradshaw et al. 2008).

Stratiform Mineralization

Stratiform sulphide lenses are generally uniformly fine grained and homogeneous, although a delicate bedding is commonly preserved. Pyrite is the principal sulphide mineral throughout the deposit and typically occurs as massive very fine anhedral grains, but also commonly occurs as coarser well-formed crystals (termed “buckshot” pyrite) set in a fine matrix of pyrite or sphalerite. Framboidal pyrite is present within the massive sulphide lenses, but is rare. Abundant fine grained sphalerite forms delicate, wispy sub millimetre to centimetre scale layers and is largely responsible for the bedded appearance of the massive sulphide mineralization. Sphalerite also commonly occurs as very fine grains interstitial to massive pyrite (Jambor 1996; Tucker 1999; Bradshaw 2003).

Semi-Massive Replacement Mineralization

The replacement mineralization tends to occur dominantly in the Saddle zone, as well as locally within the immediate footwall to both the Wolverine and Lynx zones where there is strong chlorite and/or carbonate alteration in both the hanging wall and footwall rocks.

Tucker et al. (1997) and Bradshaw (2003) also noted that the replacement zones are areas where sulphide minerals have replaced the porosity and permeability of the host rocks. This replacement results in fine grained, massive sulphide mineralization that ranges in thickness from several centimetres to approximately one metre. In most instances sphalerite is the most common replacement sulphide mineral. It occurs either as massive beds or as irregular blebs and disseminations with lesser

pyrite. Semi-massive pyrite and sphalerite replacement mineralization hosted by chlorite and carbonate altered rhyolite and/or argillite in the Saddle Zone can reach up to 13 m in thickness. There is also known to occur semi-massive pyrite and sphalerite mineralization in the footwall to the stratiform massive sulphide in the Wolverine zone that ranges from 6 to 8 m in thickness. Locally, chalcopyrite forms small massive replacement zones in the immediate footwall to the massive stratiform sulphide mineralization; noting that chalcopyrite replacement zones have well defined upper and lower boundaries and are on the order of 30 cm thick with 60 to 80% chalcopyrite and minor pyrrhotite, which is associated with intense and pervasive chlorite alteration.

Stringer Zone Mineralization

The stringer-style mineralization is dominantly associated with the replacement mineralization in the Saddle zone and in the immediate footwall to the thickest stratiform and semi-massive replacement sulphide mineralization in the Wolverine zone (Tucker 1999).

Well-developed stringer zones are present locally and are confined to relatively small specific areas which are interpreted to relate to upflow of hydrothermal fluids and associated with silicification of the host rock. Where well developed, stringer zones consist of 5 to 10%, 3 to 5 mm wide, randomly oriented veins containing quartz, pyrite, and sphalerite, with an alteration halo of silicified rock. Weakly developed stringer zones occur over a much wider area and have sparse quartz-sulphide veins which are typically 5 mm to 1 cm wide with no associated alteration halo. Well-developed stringer zones in the Wolverine zone range in thickness from 1 to 4 m, while sparse, smaller quartz chalcopyrite stringer veins can extend as many as 10 m down into the chlorite altered footwall.

SULPHIDE MINERALIZATION TEXTURES

The minerals galena, tetrahedrite-tennantite, and arsenopyrite are common, though difficult to distinguish from one another, and typically occur together as fine aggregates of anhedral grains within the sphalerite rich layers. Tetrahedrite-tennantite can also occur as very fine grains interstitial to and forming a matrix for pyrite, though this is only rarely observed in drill core due to the fine grained nature of the mineralization. Chalcopyrite is rare within the stratiform sulphide lenses, occurring almost exclusively as remobilized medium-grained masses on the edges of large blebs of quartz or calcite gangue (Jambor 1996; Tucker 1999; Bradshaw 2003)

Locally in the Lynx zone are breccia textures that occur on both upper margins of the massive sulphide lens and within the massive sulphide lens. Breccia facies include:

- a matrix supported breccia with 20 to 30%, rounded 3 to 5 mm diameter clasts of fine-grained pyrite within a matrix of similarly fine grained pyrite

- a clast-supported, jigsaw breccia with very little matrix and angular clasts of pyrite and sphalerite from one centimetre to several centimetres in diameter.

These breccias are interpreted as primary features associated with the precipitation of sulphides on the seafloor (Tucker 1999). Bradshaw (2003) noted that it is likely that the two clast populations listed by Tucker et al. (1997) are suggestive of local downslope remobilization; which resulted in the breccia textures.

Unit 3 Fragmental Rhyolites

Immediately above mineralization and sometimes extending into the footwall is a fragmental rhyolite with variable amounts of carbonate and silica. This is described as a grey rhyolite with a distinctive fragmental texture. There is also a grey to black, felsic volcanic rock with abundant (50 to 90 volume percent) siliceous lenses or clasts in a matrix of black carbonaceous argillite; this could also be termed as a lapilli tuff with an ash matrix.

The fragmental rhyolite contains the following facies variations:

- Grey rhyolite with a distinctive fragmental texture. This texture is defined by wispy sub-millimetre dark green to black, anastomosing micaceous bands which separate centimetre-sized felsic aphanitic rock domains. Fragments are sub-angular to sub-rounded irregular shapes with jagged boundaries. A variant of this unit has a distinctive greenish hue and 1 to 2% disseminated magnetite which may be a weak chloritic alteration.
- Grey to black, felsic volcanic rock with abundant (50 to 90 volume percent) siliceous lenses or clasts in a matrix of black carbonaceous argillite. The rock fragments are generally centimetre-sized, ovoid and flattened into the plane of foliation. This lithology is similar to the fragmental rhyolite described above, with the distinguishing criteria being a significantly greater sedimentary rock component. This felsic volcanoclastic unit occurs above the "mineralized sequence" and its base is generally marked by the presence of the uppermost magnetite-bearing iron formation. Its thickness throughout the deposit is fairly constant, averaging 80 m (Tucker 1999; Bradshaw 2003).

Unit 4 Hanging Wall

The hanging wall of the mineralized unit contains interbedded argillite, rhyolite, and magnetite-carbonate-pyrite exhalites which ranges from 85 to 160 m in width. The immediate hangingwall to the massive sulphide is most commonly black, strongly graphitic argillite. Rhyolites in the sequence are dominated by a grey, massive to flow-banded, aphanitic to very fine-grained, aphyric rhyolite. This is interbedded with a black, finely laminated, aphanitic, carbonaceous to strongly graphitic, argillite or ash tuff. Interbedded between the rhyolites and black argillites or ash tuff is a grey to white carbonate exhalite. In the upper portion of the sequence there is a grey to black, magnetite-rich exhalite (iron formation). The magnetite-rich exhalative rocks

are laterally extensive and are part of the stratigraphy on a regional scale for up to 12 km.

The sequence of rocks contains the following lithologies:

- Grey, massive to flow-banded, aphanitic to very fine-grained, aphyric rhyolite volcanic rocks. This rock is comprised of domains of aphanitic rhyolite with sub-millimetre to several centimetre-wide micaceous partings oriented parallel to the dominant S1 foliation. The rhyolite volcanic rock domains commonly contain minute feldspar microlites.
- Black, finely laminated, aphanitic, carbonaceous to strongly graphitic, argillite. This rock may contain several volume percent grey tuff layers or clasts. A strongly siliceous variant of this lithology contains abundant flattened and sheared veins of quartz and pyrite suggesting that the silicification may be an alteration phenomenon.
- Grey to black, magnetite-rich exhalite (iron formation). The iron-formation contains 5 to 60% disseminated to massive layered magnetite with lesser carbonate and chlorite and the layering is interpreted as primary. Magnetite is interbedded with quartz and lesser micaceous minerals on a millimetre scale.
- Grey to white carbonate exhalite. Contains up to 90% patchy to massive carbonate (calcite > ankerite > siderite) with lesser magnetite, chlorite and pyrite. Rarely exhibits fine laminations and is typically massive in texture (Tucker 1999; Bradshaw 2003).

UNIT 5 CAMPBELL RANGE

The uppermost unit is comprised of interbedded carbonaceous argillite and greywacke, with lesser basalt and rhyolite. These rocks are commonly green, massively fine-grained basalt with biotite and minor epidote on partings. There is also a more clastic unit of interbedded black aphanitic argillite and grey to black, slightly coarser-grained greywacke. This uppermost sequence probably represents the transition from the Wolverine stratigraphy to the overlying Campbell Range basalts.

At the top of the hanging wall stratigraphy is a unit composed of intercalated carbonaceous argillite and wacke with lesser mafic and felsic volcanic rocks. This unit contains the following lithologies:

- Green, massively fine-grained basalt with biotite and minor epidote on partings. These rocks are interpreted to be flows where massive and homogeneous, although a volcanoclastic origin is preferred where layering is prominent, and where they are interbedded with carbonaceous sedimentary rocks.

- Interbedded black aphanitic argillite and grey to black, slightly coarser-grained greywacke. Minor interbeds of felsic volcanoclastic rocks are present, and they are similar in appearance to the fragmental rhyolite volcanic rocks (Tucker 1999).

This uppermost sequence probably represents the transition from the Wolverine stratigraphy to the overlying Campbell Range basalts. The upper limit to this unit is unknown, but its thickness is at least 200 m. The bottom of this unit is recognized by the last occurrence of green basaltic volcanoclastic rock. It should be noted that there are also intercalated carbonaceous sedimentary rocks; which are extremely thin, on the order of centimetres. Individual basalt flow units can attain thicknesses of up to 40 m (Tucker 1999; Bradshaw 2003).

7.2.3 STRUCTURE

The following discussion is from Pearson and Giroux (2006):

Faulting is a common structural component of the Wolverine deposit area. Bradshaw (2003) noted that most apparent offsets in the stratigraphy appear to be less than one metre with no major offsets truncating any major units. Faults are commonly denoted by gouge zones that are most readily apparent in the argillite units; however, faults can also be denoted by strong fracturing and/or quartz-carbonate veining. Bradshaw (2003) also noted that the localization of the feldspar-phyric (FP) intrusions in the footwall rocks maybe controlled by syn-volcanic or scarp faults.

One style of faulting noted in 2005 exploration activities and theorized by Bradshaw (2003) is faulting that is sub-parallel to the primary foliation (S₁). Tucker et al. (1997) also noted that low-angle or sub-horizontal faults are poorly exposed on surface; noting the most accessible example is the contact between the Wolverine Succession and the overlying, younger Campbell Range rocks that denotes the separation between hanging wall felsic volcanic rocks and hanging wall mafic volcanic rocks. This sub-horizontal fault is denoted by a narrow quartz vein on surface. The sub-horizontal faults appear localized to ductile rocks such as argillite and massive sulphide mineralization. These 'horizontal' faults may provide an explanation for local, multiple argillite units intercalated with the massive sulphide mineralization; which could mean local structural thickening of the Wolverine deposit. Tucker et al. (1997) also noted that the sub-horizontal faults may also provide an explanation for thinning of the massive sulphide mineralization and that apparent hanging wall zones may be structural duplication. Rare kinematic indicators are suggestive of a south-directed thrust.

Dessureau (2006) noted that in the Saddle Zone portion of the Wolverine deposit, there are several generations of faults locally complicating stratigraphy; however, offsets are minimal. Generally there are three to four orientations showing minor displacement; which for the most part include orientations parallel to either S₁ or joint sets striking approximately 330°, 010°, and 040°, as well as striking 090°. The

subvertical fault zones show the most apparent displacement and provides the appearance of displacing the massive sulphide mineralization several meters when in fact, there may only be a small vertical offset less than a metre. A very common fault type is the planar extensional fault; which tends to strike 040° and dips subvertically. This orientation of faulting tends to show relatively small displacement that is generally between one centimetre and one metre.

Regionally, Murphy (1997) has observed both major and minor folds from the scale of rock exposures to drill core in the Grass Lakes area; however, it is important to note that the overall lack of fold hinges and overturned limbs facilitates stratigraphic correlations across the Wolverine deposit. The primary folding (F1) in the Wolverine deposit area verges to the southwest producing a 'S'-shape; which indicates that the sulphide mineralization occurs along the western limb of an open, upright fold structure. Bradshaw (2003) noted that the shape Wolverine deposit is strongly controlled by the S1 and to a lesser extent by the F1 with there being local thickening of units by the F1 folds; which were controlled by the primary regional deformation (D1). Only one area of minor folding was observed in the underground mapping through 2005 with Dessureau (2006) noting several pygmy folds that were generally gently inclined, south verging with northwest trending fold axis.

Tucker et al. (1997) noted that the S1 is approximately has a strike of 325° and a dip of 28° to the northeast. There is also an associated rodding lineation with a plunge orientation of 85° to the east and a dip of 15°. Compositional layering in the massive sulphide mineralization, represented by the S1 is consistent except in the area of fold hinges; however, Bradshaw (2003) noted that there are local syn-sedimentary structures that also complicate the S1 orientation. As well, sulphide stringers in the footwall rocks are transposed parallel to S1; noting it is theorized that the nature of the host rock may have prevented the formation of a brittle stockwork more commonly associated with stringer zones in VMS deposits.

Dessureau (2006) noted that from underground mapping there is one dominant foliation and two dominant, conjugate joint sets in the Wolverine Stratigraphy. The S1 is visible in all lithologic units within the Wolverine Stratigraphy; however, the S1 is more obvious in some units such as the argillite and less pronounced in others such as the exhalative units. Joint sets are also more pronounced in some units than others. The S1 is generally parallel to the primary bedding planes (S0) and strikes 330° and dips between 25° to 35° to the northeast. The two joint sets are much less pronounced, striking between 040° and 060° with a dip ranging from 70 and 80° for the first set and striking between 005° to 010° with a dip ranging from 80° to 90° for the second set. It should be noted that the foliation and joint sets are locally disrupted around the fault zones.

7.2.4 HYDROTHERMAL ALTERATION

The following discussion is from Pearson and Giroux (2006):

Tucker et al. (1997) and Bradshaw (2003) both noted that there are four main styles of hydrothermal alteration associated with the VMS sulphide system that formed the Wolverine deposit that occurs principally in the footwall rocks adjacent to the massive sulphide mineralization. The presence of the hydrothermal alteration beneath the massive sulphide mineralization signifies where the both structurally controlled and primary permeability and porosity of the footwall rock has been exploited by hydrothermal fluid passage. The alteration types in order of decreasing abundance are: (1) sericitization; (2) chloritization; (3) silicification; and (4) carbonatization.

Sericitization is typically evident as constrained to the footwall rocks that underlie the massive sulphide mineralization and is typically moderate to strongly pervasive in intensity. Sericite altered sedimentary and volcanoclastic rocks are always intensely foliated and extremely friable, as well as fissile. The hydrothermal alteration appears to be stratabound and typically found peripheral to chloritization; however, it is important to note that sericitization is the most widely distributed alteration facies reaching a thickness of approximately 50 m locally. Tucker et al. (1997) noted that sericite alteration is locally observed in hanging wall rocks, especially on the eastern fringe of the deposit.

Chloritization is commonly quartz or feldspar crystals set in a dark green chlorite matrix that is dominantly in felsic volcanoclastic rocks with and without intercalated argillite. Tucker et al. (1997) has noted that chloritization is best developed in the Wolverine Zone and is much less widespread than sericite alteration; however, spatially, chlorite alteration is very closely associated with sulphide mineralization. Within the chlorite alteration, massive replacement type mineralization is much more common than stringer-style mineralization. In most localities where chlorite alteration is present, there is a gradual transition in alteration types from chlorite-dominant to sericite-dominant further from the massive sulphide (and presumably heat and fluid flow).

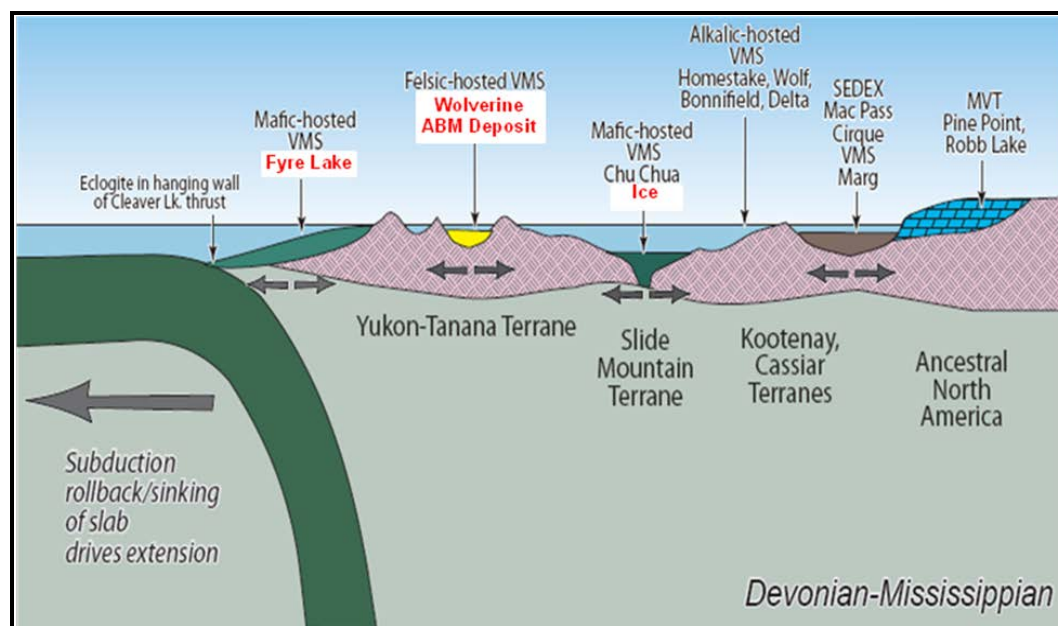
Silicification is characterized by the presence of quartz–sulphide (pyrite > sphalerite >> chalcopyrite) veins (more related to silica precipitation) and associated intense, pervasive silicification of the rocks immediately beneath the massive sulphide. Carbonate alteration is characterized by the formation of a carbonate assemblage that consists of varying degrees of white calcite, orange-brown ankerite, and brown siderite, in the fine to coarse grained felsic volcanoclastic lithology which in most places is the immediate footwall to the massive sulphide. The distribution of this alteration type is erratic and poorly understood (Tucker 1999).

8.0 DEPOSIT TYPES

8.1 VOLCANOGENIC MASSIVE SULPHIDE ORE DEPOSITS

VMS ore deposits are a type of metal sulphide ore deposit, comprising mainly copper-zinc-lead; they are associated with and created by volcanic-associated hydrothermal events in submarine environments. VMS deposits are both ancient and modern deposits; they are still forming on the seafloor around undersea volcanoes along many mid-ocean ridges, and within back-arc basins and forearc rifts. Mineral exploration companies are exploring for seafloor massive sulphide deposits; however, most exploration is concentrated in the search for land-based equivalents of these deposits (Figure 8.1).

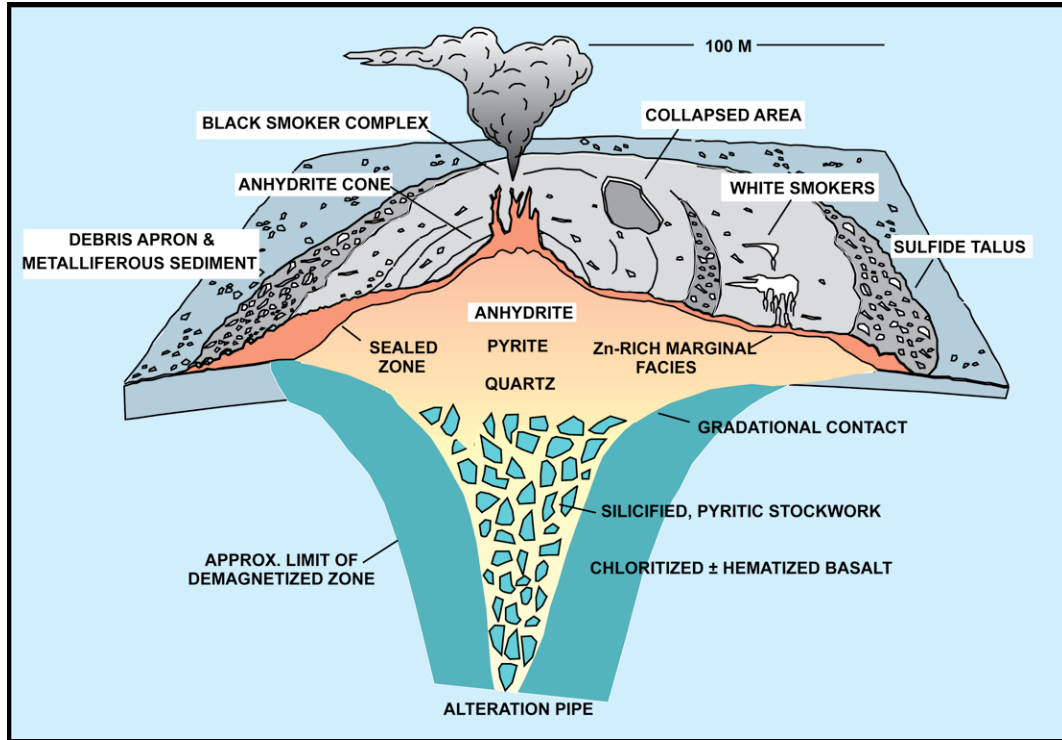
Figure 8.1 Generalized Cross Section Showing the Tectonic Setting for VMS Deposits



VMS deposits are predominantly layered accumulations of sulphide minerals that precipitate from hydrothermal fluids on or below the seafloor in a wide range of ancient and modern geological settings. In modern oceans they are synonymous with sulfurous plumes called black smokers (Figure 8.2). Other distinct deposit features include widespread alteration in rocks adjacent and often contemporaneous to chloritization, silicification and pyritization. Deposit specific features can be found such as sulfide-rich sediments adjacent to the deposit, accumulations of sulfate

minerals such as barite and anhydrite or collapsed areas containing siliceous sinter cones (Figure 8.2).

Figure 8.2 Generalized Section Showing the Volcanic Components of a VMS Deposit



VMS deposits occur within environments dominated by volcanic or volcanic-derived (e.g. volcano-sedimentary) rocks, and the deposits are contemporary and coincident with the formation of associated volcanic rocks. In mining district scale environment most, but not all significant VMS deposits occur in clusters that define major mining camps. In general, the deposit clusters are restricted to either linear rifts or calderas or eruptive centres. As a class, they represent a significant source of the world's copper, zinc, lead, gold, and silver ores, with cobalt, tin, barium, sulfur, selenium, manganese, cadmium, indium, bismuth, tellurium, gallium and germanium as mining by-products.

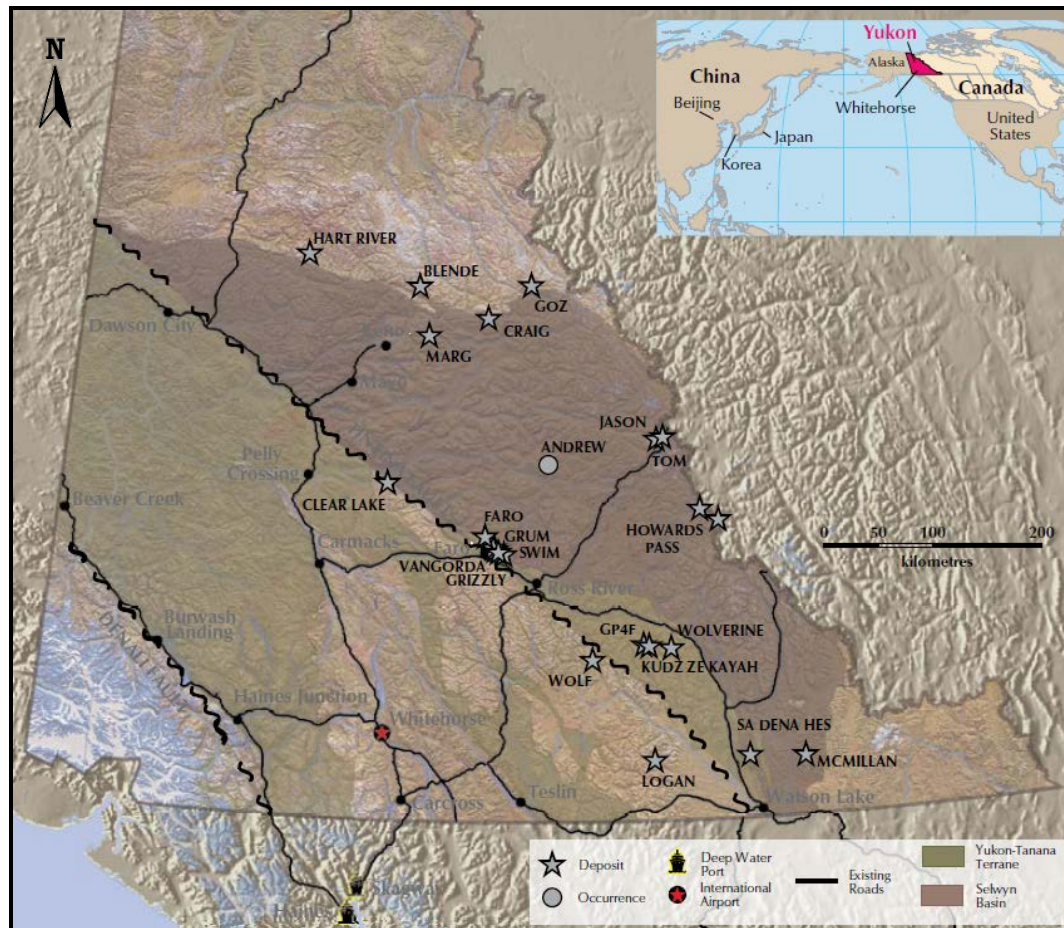
The close association with volcanic rocks and eruptive centers sets VMS deposits apart from similar ore deposit types which share similar source, transport and trap processes. Some VMS deposits are distinctive in that ore deposits are formed in close temporal association with submarine volcanism, and are formed by hydrothermal circulation and exhalation of sulphides, which are independent of sedimentary processes. This can set VMS deposits apart from sedimentary exhalative (SEDEX) deposits. Host rock assemblages, textural relationships in sulphide minerals and hydrothermal alteration mineralogy are the best tools for distinguishing SEDEX from VMS deposits (Robb 2005).

There is a subclass of VMS deposits, the volcanic- and sediment-hosted massive sulphide (VSHMS) deposits that do share characteristics that are hybrid between the VMS and SEDEX deposits. VMS deposits associated with siliciclastic sedimentary rock dominated settings with abundant felsic rocks and less than 10% mafic material. These settings are often shale-rich siliciclastic-felsic or bimodal siliciclastic. Notable examples of this class include the deposits of the Bathurst Camp, New Brunswick, Canada (e.g. Brunswick #12); the deposits of the Iberian Pyrite Belt, Portugal and Spain, and the Wolverine deposit, Yukon, Canada. Hybrid VMS-SEDEX deposits of the siliciclastic associations may be developed within interflow sediments or within units of sedimentary rocks which are present discontinuously throughout a larger and essentially contiguous volcanic package.

VMS deposits are typically located at the top of the felsic volcanic sequence, within a sequence of volcanoclastic tuffaceous epiclastics, cherts, sediments, or perhaps fine tuffs, which are usually related to the underlying volcanics. The hanging wall to the deposit is broadly related to a more mafic sequence of volcanic rocks, either andesite (e.g. Whim Creek and Mons Cupri, Western Australia or Millenbach, Canada), or basalt (Hellyer, Tasmania) or absent or sediments only (Kangaroo Caves, Western Australia). The hanging wall to the deposit can be volcanic units essentially contiguous and contemporary with the footwall rocks, indicating mineralization was developed in an inter-eruptive pause. VMS deposits are associated spatially and temporally with felsic volcanic rocks, usually present in the stratigraphy below the deposit, and often as the direct footwall to the deposit. Sediments are usually contiguous with VMS deposits in some form or another, and typically are present as (manganiferous) cherts, volcanogenic tuffaceous sediments and chemical sediments deposited within a submarine environment.

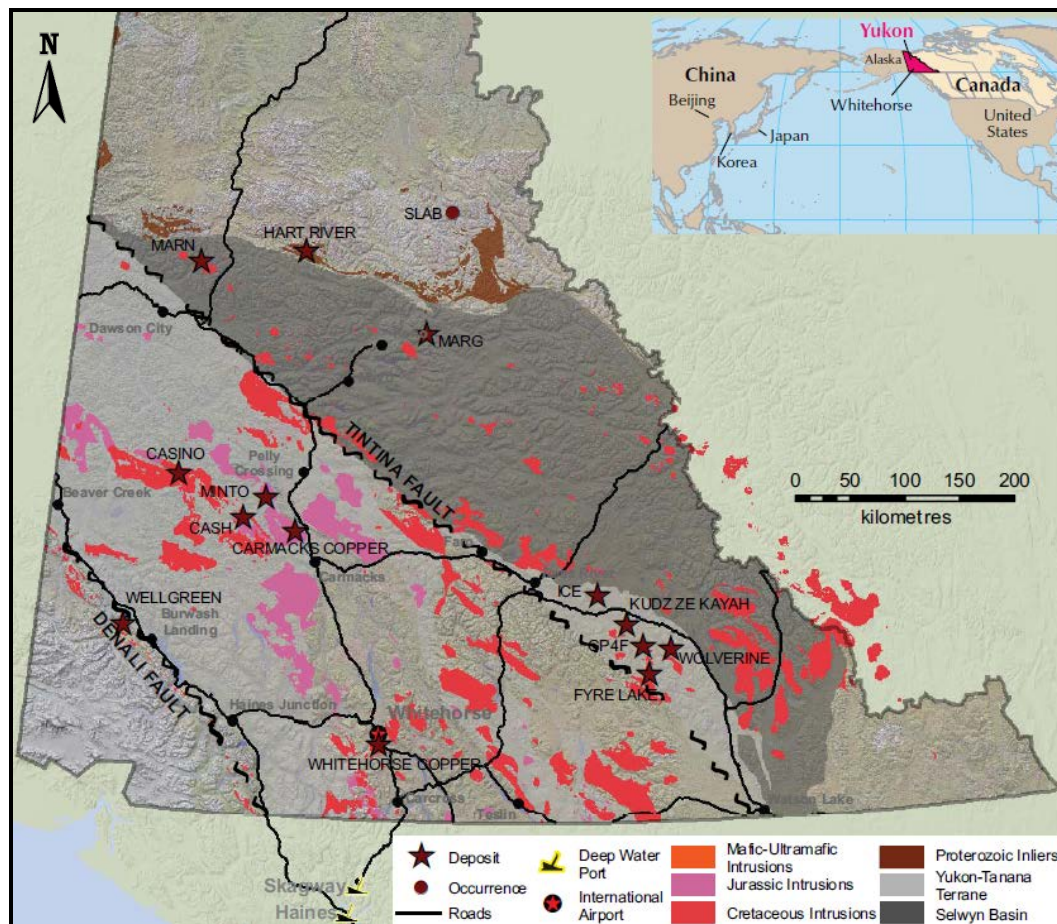
Altogether, these geological features have been interpreted to show an association of VMS deposits with hydrothermal systems developed above or around submarine volcanic centres.

Figure 8.3 Location Map for Zinc Deposits in Yukon Territory, Canada



Source: Government of Yukon (<http://www.gov.yk.ca/maps>)

Figure 8.4 Location Map for Copper Deposits in Yukon Territory, Canada



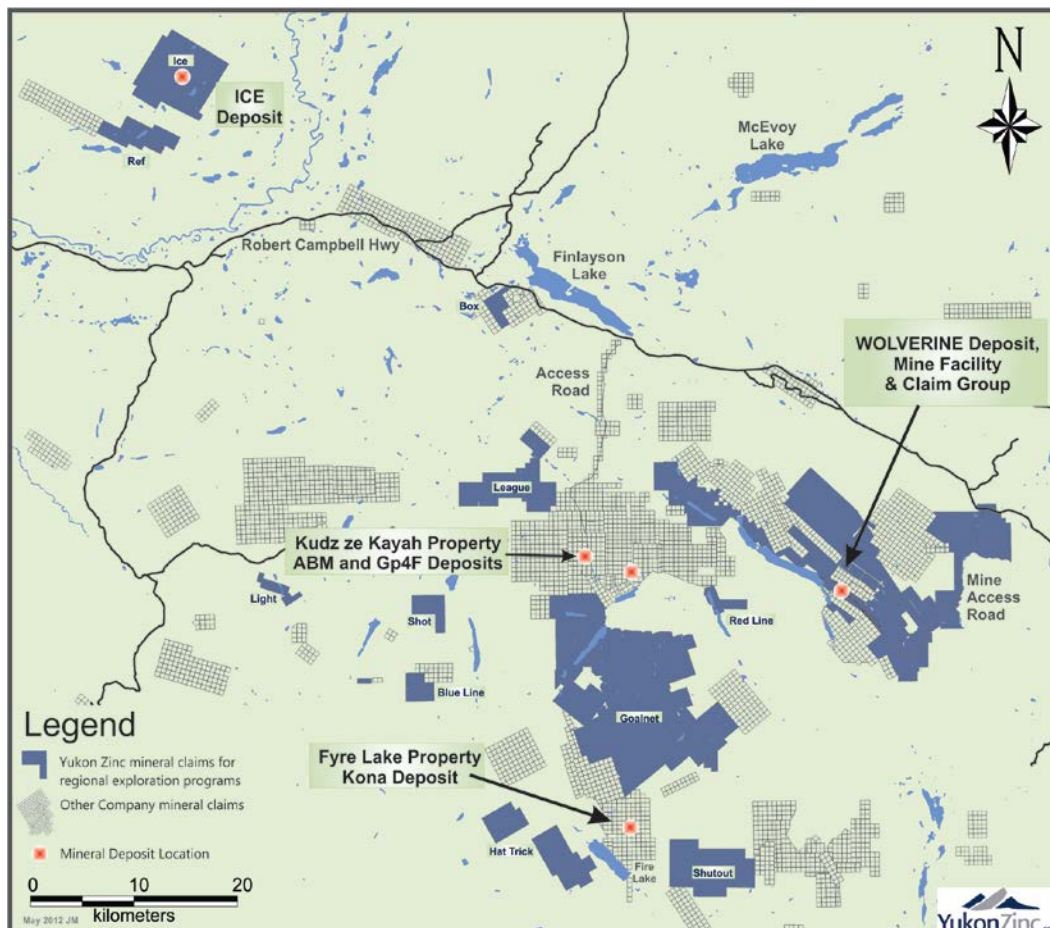
Source: Government of Yukon (<http://www.gov.yk.ca/maps>)

8.2 DEPOSITS TYPES OF THE FINLAYSON DISTRICT, YUKON

8.2.1 INTRODUCTION

The FD encompasses the region underlain by the stratigraphic successions that host the five VMS deposits discovered. The deposits include the new discoveries on the Kudz ze Kayah Property, the ABM and GP4F deposits, the Ice deposit, the Kona Deposit at Fyre Lake property and Wolverine. The district is notable for the age range and diversity of its deposits. Fyre Lake is Late Devonian in age and associated with chloritic phyllite and greenstone of boninitic composition. ABM, GP4F and Wolverine Lake are Late Devonian and Early Mississippian in age and associated with alkalic (A-type) felsic metavolcanic host rocks; and Ice occurs in Lower Permian basalt of mid-ocean ridge basalt (MORB) composition. The deposits occur in both YTT and SMT, with Kona, ABM, GP4F and Wolverine occurring in the former and Ice deposit occurring in the latter (Murphy et al. 2006; Piercey 2001) (Figure 8.5).

Figure 8.5 VMS Deposits of the Finlayson District



8.2.2 FYRE LAKE PROPERTY – KONA DEPOSIT

Pacific Ridge Exploration Ltd.'s 100%-owned Fyre Lake Property is host to a "Besshi-type" copper-cobalt-gold VMS deposit named the Kona deposit (Figure 8.5). Besshi-type massive sulphide deposits occur in strata consisting of sub-equal amounts of clastic sedimentary rocks and basalt and their setting is commonly close to a tectonic boundary. The Kona deposit is located at a similar stratigraphic position. The Kona deposit is also similar to many Besshi-type deposits in that the mineralization is predominately pyrite, chalcopyrite and pyrrhotite with lesser sphalerite (Foreman 1997). The Fyre Lake property is located in the FD, approximately 40 km southwest of the Wolverine Mine (Figure 8.5) (www.pacificridgeexploration.com/s/FyreLake.asp; Blanchflower 1997).

The Kona deposit occurs within the 365 Ma Fire Lake Formation of the Grass Lakes Group. The Fire Lake Formation consists of a metamorphosed volcanic sequence. The immediate host rock to mineralization is chlorite-quartz $\pm$ actinolite schists (metamorphosed mafic volcanics and volcanoclastics). The Kona deposit occurs at the transition from mafic volcanic rocks to overlying turbiditic sedimentary rocks in a

fore-arc setting. Mineralization consists of three parallel, tabular, northeast-trending sheet-like massive sulphide horizons composed of pyrite, pyrrhotite, chalcopyrite and sphalerite, with local lenses of massive magnetite.

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 to 40 m over a length of 1,500 m and a width of 250 m. Blanchflower, a consulting geologist with Minorex Consulting Ltd. (Minorex), operating as directed by Pacific Ridge Exploration Ltd., prepared a technical report in January 17, 2006 for the Kona VMS deposit. In this report, Minorex disclosed an Indicated Mineral Resource of 3.571 Mt grading 1.57% Cu, 0.10% Co and 0.61 g/t Au at a 1% Cu cut-off grade. In addition, they reported an Inferred Mineral Resource, at the same cut-off grade, comprised of 5.361 Mt grading 1.48% Cu, 0.08% Co and 0.53 g/t Au.

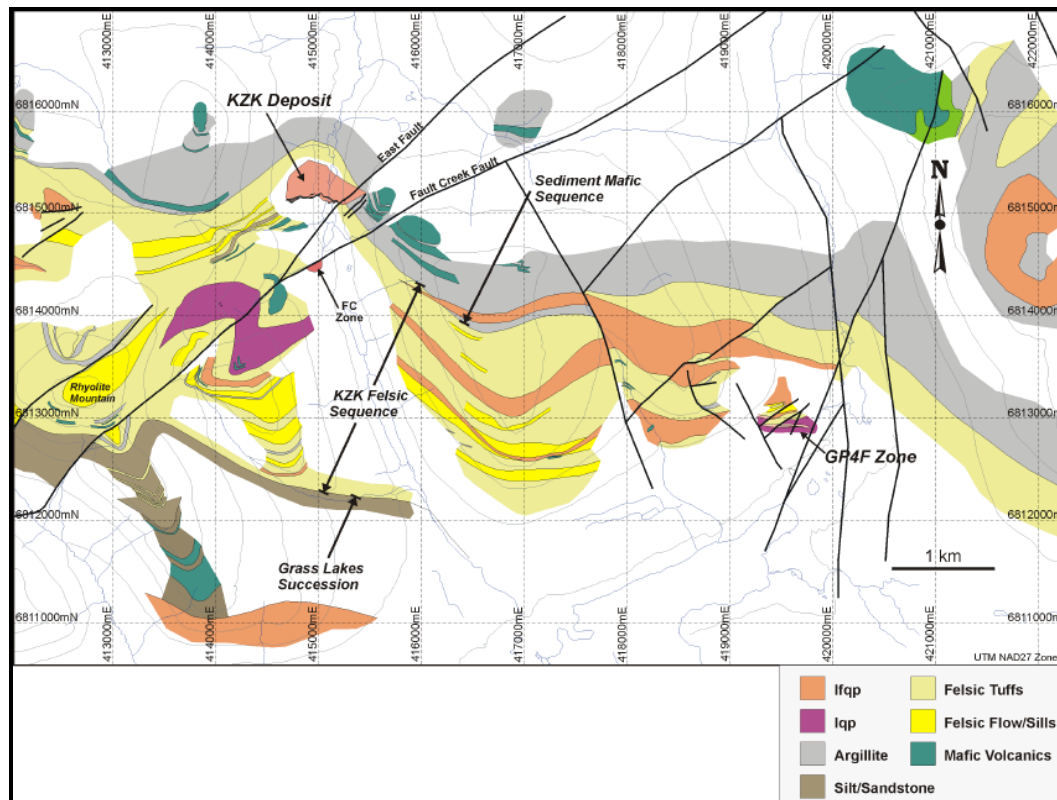
The QP has been unable to verify the information and that the information is not necessarily indicative of the mineralization on the property that is the subject of the technical report.

8.2.3 *KUDZ ZE KAYAH PROPERTY*

The Kudz ze Kayah Property is located 30 km west of the Wolverine Mine and is accessed by an all-season private road (Figure 8.3). It occurs within a sequence of predominantly felsic volcanic and volcanoclastic rocks with minor metasedimentary rocks of the Kudz ze Kayah Formation (Figure 8.6). The host-rock sequence is stratigraphically overlain by carbonaceous sedimentary rocks and mafic volcanic rocks of the Wind Lake Formation. The property hosts the ABM and GP4F deposits, and the Fault Creek zone which are all Mattabi/Kuroko hybrid VMS deposits that may have involved partial replacement of felsic tuff host rocks in addition to conventional seafloor sulphide deposition around a hydrothermal vent (Schultz 2001).

Cominco carried out a geochemical survey in 1977 in the FD area, but the survey was too wide-spaced to reveal evidence of the ABM deposit. In 1992, Cominco returned to the area to follow up on anomalous base metal stream silt samples which had been collected in 1988 by the Geological Survey of Canada. In late 1993, quartz-sericite altered rhyolite rocks and a 15 cm piece of banded massive sulphide-magnetite float were noted, but the source of mineralization was not found until geophysical surveys revealed a major anomaly under the valley. The initial discovery hole was drilled in April 1994.

Figure 8.6 Geology of the Kudz Ze Kayah Property



Source: Peter et al. (2007)

THE ABM DEPOSIT

The main part of the ABM deposit is a 500 m long, isoclinally-folded lens of massive to semi-massive sulphide that occurs within an overturned syncline. Mineralization extends down 400 m, averages 18 m thick, up to a maximum thickness of 34 m at the nose of the fold. Sulphide mineralogy is dominated by pyrite, then sphalerite, pyrrhotite, galena, chalcopyrite, and trace arsenopyrite. Gangue minerals include chlorite, quartz, magnetite, siderite, sericite, albite and barite.

The ABM deposit has a resource of 13.7 Mt at 6.0% Zn, 1.61% Pb, 0.9% Cu, 139.2 g/t Ag and 1.38 g/t Au (Hatch 2000, Shultz 2001).

The QP has been unable to verify the information and that the information is not necessarily indicative of the mineralization on the Property that is the subject of the technical report.

THE GP4F DEPOSIT

The GP4F deposit is located 4.5 km southeast of the ABM deposit (Figure 8.5 and Figure 8.6), within the Kudz ze Kayah Property. The GP4F deposit is within the

same package of Late Devonian-Early Mississippian rocks as the ABM deposit. It is hosted by a sequence of felsic volcanic and volcanoclastic rocks (Figure 8.5 and Figure 8.6).

The GP4F deposit is a composite zone comprised of thin, very high-grade massive sulphide lenses that vary from 0.1 to 3.2 m thick, and high-grade semi-massive sulphides in a mixed sulphide-silicate assemblage described as strong proximal alteration by Cominco. The GP4F deposit is internally diluted by unmineralized mafic sills/dykes and variably mineralized cherty felsic tuffs and quartz veins. As such, the GP4F Deposit ranges in true thickness from 1.9 to 8.9 m with an average true thickness of 5.3 m. Mineralization occurs within an overturned homoclinal sequence. It has a strike length of 200 m, dips moderately to the north and extends 350 m down dip. Immediately below the sulphide lens is a massive, foliated quartz-feldspar porphyry.

The mineralization, in comparison with the ABM deposit, is relatively copper-poor and lead-rich. Mineralization forms crudely banded coarse-grained pyrite in a matrix of sphalerite, pyrrhotite, galena and magnetite. The GP4F deposit has a historical resource of 1.5 Mt at 6.4% Zn, 3.1% Pb, 0.1% Cu, 90 g/t Ag and 2 g/t Au (Hatch 2000; Schultz 2001).

The QP has been unable to verify the information and that the information is not necessarily indicative of the mineralization on the Property that is the subject of the technical report.

8.2.4 *ICE PROPERTY AND DEPOSIT*

The Ice Property is 100% owned by Yukon Zinc. It is located 60 km east of Ross River in the northern part of the FD (Figure 8.5). The Ice Property is 80 km northwest of the Wolverine Mine (Figure 8.4). The Ice deposit is a Cyprus-type VMS deposit which is hosted by mafic volcanic rocks at the transition from underlying brecciated pillow basalt to overlying massive basalt and is interpreted to have formed in a back-arc basin/ocean basin setting.

The Ice claims were staked in February 1996 by Expatriate Resources Ltd. to cover a previously unstaked copper soil geochemical anomaly identified during a 1973 survey by Archer, Cathro & Associates Limited. High-grade secondary oxide copper mineralization on surface was discovered in May 1996 and additional claims were staked. Exploration work in 1996 and 1997 consisted of geological mapping, grid and reconnaissance soil sampling, airborne and ground magnetic and electromagnetic surveys. A total of 34 diamond drillholes (2704 m) in 1996 and 87 diamond drillholes (7,880 m) in 1997 were completed.

Geologically, the Ice Property is within the boundary of the Yukon-Tanana and Slide Mountain geologic terranes. The northeastern margin of the Ice Property is the Finlayson Lake Fault Zone, a complex zone of steep and shallow faults related to transpressive suturing. The southwestern boundary of the YTT is the Tintina Fault

zone, a major strike-slip fault with at least 450 km of dextral displacement during Late Cretaceous and/or Early Tertiary time (Tempelman-Kluit 1976).

The Ice deposit is underlain by Devonian to Triassic igneous and sedimentary rocks consisting of basalt, ultramafic and mafic plutonic rocks, ribbon chert and associated argillite, sandstone and marble. Most exploration to date has focused on a 600 m x 400 m area. The “Cyprus-type” deposit is hosted in a relatively undeformed ophiolite sequence belonging to Slide Mountain Terrane, consisting of a basalt breccia unit lying within a thick package of interlayered massive to pillowed basalts and deep-water sedimentary rocks. The best mineralization is contained within an area 350 m long and 50 m wide of thick massive sulphide accumulations. Primary mineralization consists of a lensoid massive sulphide horizon that is partially underlain by a discordant sulphide feeder zone. Average mineralization thickness is 4.94 m with a thicker central core of 15 to 16 m.

9.0 EXPLORATION

9.1 1993 TO 1995

The first discovery in the Finlayson District was the ABM deposit on the Kudze Kayah Property found by Teck Cominco in the summer of 1993. Atna acquired the 18 unit Foot claims in July 1993 to cover the favourable geology of the Fetish showing discovered by the Finlayson Joint Venture in 1973 and evaluated again in 1982 by Esso Minerals. These Foot claims host the Wolverine discovery. The Foot claims were one of three properties identified by Mark Baknes of Equity Engineering Ltd. in an evaluation of the Finlayson District for VMS deposits. Each property covered previously known showings or geochemical anomalies. Atna undertook prospecting and reconnaissance mapping and stream sediment geochemistry in late summer 1993. No additional fieldwork was done in 1994.

Westmin entered into an option to earn a 60% interest in the Foot and other two properties in January 1995 and commenced fieldwork in June. The Wolverine discovery was made in August 1995 in the first of four planned stratigraphic drill holes. Hole 1 intersected 5.3 m at 0.27 g/t Au, 52.4 g/t Ag, 0.27% Cu, 0.42% Pb and 9.26% Zn. Holes 2 and 3 intersected weak zinc mineralization at several stratigraphic levels. Follow-up Hole 4 was drilled 125 m down dip of Hole 1 and intersected 8.3 m of 7.63 g/t Au, 1,351 g/t Ag, 0.56% Cu, 3.45% Pb and 14.22% Zn (Figure 9.1, Table 9.1).

A second discovery was found in the Fisher zone of 2.4 m of semi-massive sulphides at 0.14 g/t Au, 66.3 g/t Ag, 0.12% Cu, 1.41% Pb and 2.84% Zn (Figure 9.1). Five more stratigraphic drillholes were completed and Westmin/Atna also completed an airborne geophysical survey.

Figure 9.1 Wolverine Property Drilling and Exploration Zones



9.2 1996 TO 1997

In 1996 Westmin/Atna continued to drill the Wolverine discovery, initially progressing downdip and to the southeast. In August 1996, geologists discovered a second lens, the Lynx zone. Stratigraphic drilling continued on the Wolverine and southward extensions to the Sable and Puck zones (Figure 9.1). Six drillholes were completed on Puck which defined the extension of the Wolverine horizon.

Work in 1997 consisted primarily of diamond drilling. Work primarily focused on defining the Wolverine and Lynx deposits and regional stratigraphic drilling. After the 1997 program, 71 drillholes had intersected the deposit (Table 9.1). The deposit was defined over a strike length of approximately 750 m and remained open downdip to

the northeast where it crossed onto the Wolverine claims now owned by Teck Cominco and called the Pelly Mountain property (Figure 9.1).

9.3 2000 TO 2004

A small exploration program was conducted during the summer of 2000 (Table 9.1). This program was part of the feasibility study Expatriate conducted as a joint venture with Teck Cominco (now Teck Resources) for the combined development of the Wolverine deposit and the ABM deposit (which is located on the Kudze Kayah Property). The agreement between Expatriate and Teck Cominco was terminated in October and the program was stopped. Exploration and development work lay dormant until the summer of 2004, when Expatriate began a large diameter drill program to acquire material to conduct metallurgical tests.

9.4 2005 TO 2009

In 2005, Expatriate changed its name to Yukon Zinc Corp. The 2005 drilling campaign was used to calculate a reserve and submitted as part of a NI 43-101 report(s) in 2006 and 2007 (Table 9.1). In 2008, Jinduicheng Molybdenum Group Ltd. and Northwest Non Ferrous International Investment Company Ltd. acquired all shares of Yukon Zinc, and made Yukon Zinc into a private mining company. Drilling work conducted in 2009 was part of a definition drilling program for underground mining development (Table 9.1).

Table 9.1 Drilling Completed

Year	Collars	Completed By	Number of Drillholes	Meterage and Location
1995	WV95-01 to WV95-24	Westmin/Atna	24	4454.7 m surface drilling
1996	WV96-25 to WV96-72	Westmin/Atna	47	12,012.1 m surface drilling
1997	WV97-73 to WV97-112	Westmin/Atna	39	5,558.9 m surface drilling
2000	WV00-113 to WV00-120 Wolverine drilling: WW00-001 to WW00-003	Expatriate Resources	11	957.8 m surface drilling
2004	WV04-121-WV04-129	Expatriate Resources	9	1,758.40 m surface drilling
2005	WV05-132 to WV05-190	Yukon Zinc	59	11,712.70 m surface drilling
2006	WV06-191 to WV06-201	Yukon Zinc	-	Monitoring well drilling south of deposit
2008	WV08-202 to WV08-208	Yukon Zinc	7	406.10 m of underground drilling
2009	WV09-209 to WV09-220	Yukon Zinc	11	395.91 m of underground drilling
2009	WV09-221 to WV09-249	Yukon Zinc	28	4,717.50 m surface drilling

10.0 DRILLING

Portions of this section have been taken from Pearson and Giroux (2006). Minor edits have been made for spelling, grammar and consistency.

From Pearson and Giroux (2006):

10.1 CORING SUMMARY

An initial drill program of 15 BQ and NQ2 drillholes were completed on the Wolverine deposit in 1995 totalling 3,684.9 m; which were part of a larger 24 drillhole, regional program that totalled 6,440.9 m. This was followed by 12,012.1 m of drilling in 1996 in 36 drillholes and 5,558.9 m of drilling in 1997 in 24 drillholes. 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 drillholes were completed in 1996 totalling 4,913.3 m and an additional 17 drillholes were completed in 1997 totalling 6,341.0 m.

In 2000, seven NQ drillholes 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 drillholes (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 eight drillholes were successfully completed totalling 1,758.2 m.

Definition drilling continued in 2005 with the successful completion of 59 NQ and NQ2 drillholes totalling 11,730.1 m. The following (Table 10.1) tabulates drilling completed for the Wolverine deposit.

Table 10.1 Drillhole Details for the Wolverine Deposit

BHID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Core Size	Hole Code	Heading
DW09-01	439,549.0	6,811,559.0	1,392.0	300		Water	Surface
DW09-03	439,768.0	6,811,107.0	1,387.0	200		Water	Surface
GT11-001	439,867.4	6,811,037.3	1,271.3	24	HQ3	Geotech	1270

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Core Size	Hole Code	Heading
GT11-002	439,867.4	6,811,037.4	1,271.9	39	HQ3	Geotech	1270
GT11-003	439,867.5	6,811,037.3	1,272.3	50.4	HQ3	Geotech	1270
GT11-004	439,867.6	6,811,037.3	1,272.7	39	HQ3	Geotech	1270
GT11-005	439,870.1	6,811,037.1	1,273.5	13	HQ3	Geotech	1270
GT11-006	439,874.0	6,811,037.9	1,273.8	13	HQ3	Geotech	1270
GT11-007	439,874.4	6,811,034.5	1,270.7	25.5	HQ3	Geotech	1270
GT11-007B	439,874.5	6,811,034.3	1,270.9	25	HQ3	Geotech	1270
GT11-008	439,874.5	6,811,034.3	1,271.5	36	HQ3	Geotech	1270
GT11-009	439,874.2	6,811,034.3	1,270.5	25.5	HQ3	Geotech	1270
GT11-010	439,873.6	6,811,033.8	1,271.4	23	HQ3	Geotech	1270
GT11-011	439,869.2	6,811,030.5	1,271.4	25.5	HQ3	Geotech	1270
GT11-012	439,867.2	6,811,033.6	1,271.5	29.5	HQ3	Geotech	1270
GT11-013	439,769.6	6,811,247.1	1,252.2	111	HQ3	Geotech	1250
GT11-014	439,769.6	6,811,247.1	1,252.5	103.5	HQ3	Geotech	1250
GT11-015	439,769.6	6,811,247.5	1,251.9	216	HQ3	Geotech	1250
GT11-016	439,769.5	6,811,247.5	1,252.2	99	HQ3	Geotech	1250
GT11-029	439,854.3	6,811,120.2	1,253.7	52.5	HQ3	Geotech	1250
GT11-030	439,854.1	6,811,120.0	1,253.5	33.3	HQ3	Geotech	1250
GT11-031	439,853.4	6,811,119.4	1,253.3	24	HQ3	Geotech	1250
GT11-032	439,852.1	6,811,118.3	1,253.5	18	HQ3	Geotech	1250
GT11-033	439,849.8	6,811,118.1	1,254.2	18	HQ3	Geotech	1250
GT11-034	439,850.1	6,811,118.1	1,255.4	22	HQ3	Geotech	1250
GT11-035	439,850.1	6,811,118.0	1,256.1	43.5	HQ3	Geotech	1250
GT11-036	439,850.1	6,811,117.9	1,256.9	25.5	HQ3	Geotech	1250
GT11-037	439,868.4	6,811,106.1	1,250.6	37.5	HQ3	Geotech	1250
GT11-038	439,868.4	6,811,106.0	1,250.2	24	HQ3	Geotech	1250
GT11-039	439,868.0	6,811,105.7	1,250.2	15	HQ3	Geotech	1250
GT11-040	439,866.6	6,811,099.0	1,250.9	13.5	HQ3	Geotech	1250
GT11-041	439,866.8	6,811,099.2	1,252.2	19.5	HQ3	Geotech	1250
GT11-042	439,866.9	6,811,099.2	1,252.8	42	HQ3	Geotech	1250
GT11-043	439,866.9	6,811,099.2	1,253.1	34	HQ3	Geotech	1250
GT11-044	439,867.2	6,811,099.2	1,254.3	16	HQ3	Geotech	1250
GT11-045	439,868.1	6,811,105.9	1,252.1	11	HQ3	Geotech	1250
GT11-046	439,876.7	6,811,086.4	1,250.9	51	HQ3	Geotech	1250
GT11-047	439,876.7	6,811,086.4	1,250.6	37.5	HQ3	Geotech	1250
GT11-048	439,876.6	6,811,086.3	1,249.9	24	HQ3	Geotech	1250
GT11-049	439,872.3	6,811,084.4	1,250.4	10.5	HQ3	Geotech	1250
GT11-050	439,872.8	6,811,084.4	1,252.4	31.5	HQ3	Geotech	1250
GT11-051	439,872.8	6,811,084.4	1,252.9	28.5	HQ3	Geotech	1250
GT11-052	439,873.8	6,811,084.8	1,253.6	14.5	HQ3	Geotech	1250
GT11-053	439,876.8	6,811,086.5	1,251.7	18	HQ3	Geotech	1250

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Core Size	Hole Code	Heading
GT11-054	439,886.7	6,811,068.9	1,251.4	30	HQ3	Geotech	1250
GT11-055	439,886.8	6,811,069.0	1,251.1	39	HQ3	Geotech	1250
GT11-056	439,886.7	6,811,068.9	1,250.1	14	HQ3	Geotech	1250
GT11-057	439,881.9	6,811,066.8	1,250.6	10.5	HQ3	Geotech	1250
GT11-058	439,881.9	6,811,066.9	1,252.1	20	HQ3	Geotech	1250
GT11-059	439,882.1	6,811,066.9	1,253.0	22.5	HQ3	Geotech	1250
GT11-060	439,882.3	6,811,067.0	1,253.8	19	HQ3	Geotech	1250
GT11-061	439,886.9	6,811,069.1	1,253.1	15	HQ3	Geotech	1250
GT11-062	439,886.8	6,811,069.0	1,251.9	20	HQ3	Geotech	1250
GT11-063	439,892.4	6,811,048.9	1,251.0	52.5	HQ3	Geotech	1250
GT11-064	439,892.4	6,811,049.0	1,250.4	30	HQ3	Geotech	1250
GT11-065	439,887.0	6,811,047.2	1,251.7	10.5	HQ3	Geotech	1250
GT11-066	439,887.1	6,811,047.3	1,252.6	40	HQ3	Geotech	1250
GT11-067	439,887.1	6,811,047.3	1,253.0	23	HQ3	Geotech	1250
GT11-068	439,893.4	6,811,035.6	1,253.9	13	HQ3	Geotech	1250
GT11-070	439,836.4	6,811,137.1	1,259.4	39	HQ3	Geotech	1250
GT11-071	439,836.4	6,811,137.1	1,259.1	37.5	HQ3	Geotech	1250
GT11-072	439,836.3	6,811,136.9	1,258.7	24	HQ3	Geotech	1250
GT11-073	439,836.1	6,811,136.7	1,256.9	16.5	HQ3	Geotech	1250
GT11-074	439,839.1	6,811,140.5	1,256.1	22.5	HQ3	Geotech	1250
GT11-075	439,839.2	6,811,140.6	1,256.6	34.5	HQ3	Geotech	1250
PW08-01	440,499.7	6,810,614.3	1,393.0	104	-	Service	Surface
PW08-02	440,369.4	6,810,688.5	1,394.0	90	-	Service	Surface
TH-01	439,697.2	6,811,198.5	1,302.1	23.5	AW	Test Hole	1300
TH-02	439,702.9	6,811,193.2	1,303.8	22.9	AW	Test Hole	1300
TH-03	439,703.5	6,811,183.9	1,306.1	10.05	AW	Test Hole	1300
TH-04	439,710.1	6,811,162.9	1,308.4	17.7	AW	Test Hole	1300
TH-05	439,836.7	6,811,183.6	1,241.8	20.1	AW	Test Hole	1230
TH-06	439,836.5	6,811,183.5	1,242.3	19.7	AW	Test Hole	1230
TH-07	439,837.6	6,811,182.2	1,241.8	21.6	AW	Test Hole	1230
TH-08	439,837.6	6,811,182.2	1,242.2	20.1	AW	Test Hole	1230
TH-09	439,751.3	6,811,169.5	1,291.9	23	AW	Test Hole	1280
TH-10	439,751.3	6,811,169.4	1,291.5	32.2	AW	Test Hole	1280
TH-11	439,753.9	6,811,167.9	1,291.1	20.1	AW	Test Hole	1280
TH-12	439,754.0	6,811,168.0	1,291.8	14.9	AW	Test Hole	1280
TH-13	439,752.8	6,811,229.7	1,267.3	27.4	AW	Test Hole	1260
TH-14	439,752.8	6,811,229.7	1,267.1	24.3	AW	Test Hole	1260
TH-15	439,753.3	6,811,230.0	1,267.3	31.7	AW	Test Hole	1260
TH-16	439,753.3	6,811,229.9	1,266.8	27.1	AW	Test Hole	1260
TH-17	439,762.5	6,811,262.0	1,248.6	17.1	AW	Test Hole	1250
TH-18	439,764.8	6,811,262.8	1,248.8	24.4	AW	Test Hole	1250

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Core Size	Hole Code	Heading
TH-19	439,764.8	6,811,262.8	1,248.6	25.9	AW	Test Hole	1250
TH-21	439,875.2	6,811,011.3	1,271.0	25.3	AW	Test Hole	1270
TH-22	439,875.2	6,811,011.3	1,271.0	29.9	AW	Test Hole	1270
TH-23	439,875.2	6,811,011.3	1,271.0	25	AW	Test Hole	1270
TH-24	439,636.3	6,811,298.6	1,299.0	29.3	AW	Test Hole	1300
TH-25	439,635.6	6,811,289.7	1,299.0	32.6	AW	Test Hole	1300
TH-26	439,640.3	6,811,260.9	1,298.7	25	AW	Test Hole	1300
TH-27	439,641.5	6,811,260.8	1,298.7	25.3	AW	Test Hole	1300
TH-28	439,642.3	6,811,261.2	1,298.7	30.9	AW	Test Hole	1300
TH-29	439,761.5	6,811,304.0	1,221.4	37	AW	Test Hole	1220
TH-30	439,750.9	6,811,314.2	1,220.2	37	AW	Test Hole	1220
TH-31	439,740.6	6,811,324.0	1,220.2	37	AW	Test Hole	1220
TH-32	439,729.9	6,811,333.9	1,220.2	37	AW	Test Hole	1220
TH-33	439,730.1	6,811,333.4	1,220.2	37	AW	Test Hole	1220
TH-34	439,719.7	6,811,336.2	1,220.3	37	AW	Test Hole	1220
TH-35	439,720.4	6,811,335.5	1,220.3	37	AW	Test Hole	1220
TH-36	439,730.8	6,811,325.8	1,220.1	37	AW	Test Hole	1220
TH-37	439,745.7	6,811,311.2	1,220.0	37	AW	Test Hole	1220
TH-38	439,864.4	6,810,962.5	1,269.5	26.2	AW	Test Hole	1270
TH-39	439,865.4	6,810,964.4	1,269.8	12.8	AW	Test Hole	1270
TH-41	439,803.3	6,811,246.9	1,213.2	33.83	AW	Test Hole	1200 Ramp
TH-42	439,811.7	6,811,229.2	1,216.7	38.1	AW	Test Hole	1200 Ramp
TH-43	439,828.4	6,811,216.1	1,219.6	32.61	AW	Test Hole	1200 Ramp
TH-44	439,829.0	6,811,290.7	1,210.5	35.97	AW	Test Hole	1200 Ramp
TH-45	439,832.6	6,811,287.4	1,210.4	28.04	AW	Test Hole	1200 Ramp
TH-46	439,833.4	6,811,292.8	1,212.0	19.8	AW	Test Hole	1210 Ramp
TH-47	439,833.4	6,811,292.8	1,212.0	24.7	AW	Test Hole	1210 Ramp
TH-48	439,605.6	6,811,324.1	1,212.0	19.8	AW	Test Hole	1210 Ramp
TH-49	439,612.1	6,811,304.6	1,299.5	16.2	AW	Test Hole	1300
TH-50	439,621.4	6,811,294.8	1,299.6	26.5	AW	Test Hole	1300
TH-51	439,599.0	6,811,337.5	1,300.0	35.1	AW	Test Hole	1300
TH-52	439,604.8	6,811,334.2	1,300.0	22.9	AW	Test Hole	1300
TH-53	439,605.6	6,811,324.1	1,299.9	14.9	AW	Test Hole	1300
TH-54	439,881.6	6,810,939.7	1,270.9	19.2	AW	Test Hole	1270
TH-55	439,880.8	6,810,939.1	1,270.8	24.4	AW	Test Hole	1270
TH-56	439,880.8	6,810,939.1	1,271.2	23.8	AW	Test Hole	1270
TH-57	439,880.0	6,810,938.8	1,270.9	18	AW	Test Hole	1270
TH-58	439,598.6	6,811,340.8	1,300.7	35.1	AW	Test Hole	1300FW
TH-59	439,598.3	6,811,340.4	1,300.7	36.9	AW	Test Hole	1300FW
TH-60	439,603.7	6,811,335.0	1,300.1	28.9	AW	Test Hole	1300FW
TH-61	439,889.1	6,811,044.4	1,251.2	35	AW	Test Hole	1250

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Core Size	Hole Code	Heading
TH-62	439,889.3	6,811,055.0	1,251.1	15.2	AW	Test Hole	1250
TH-63	439,885.7	6,811,070.6	1,250.9	18	AW	Test Hole	1250
TH-64	439,833.1	6,811,290.7	1,210.5	33.3	AW	Test Hole	1200RAMP
TH-65	439,830.7	6,811,286.7	1,210.5	27.4	AW	Test Hole	1200RAMP
TH-66	439,827.9	6,811,282.5	1,210.5	35.2	AW	Test Hole	1200RAMP
TH-67	439,827.9	6,811,282.5	1,210.5	26.8	AW	Test Hole	1200RAMP
TH-68	439,893.3	6,811,036.5	1,250.3	6.4	AW	Test Hole	1250
TH-69	439,892.0	6,811,032.1	1,251.3	6.4	AW	Test Hole	1250
TH-70	439,896.2	6,811,040.9	1,251.3	4.3	AW	Test Hole	1250
TH-71	439,896.1	6,811,036.1	1,251.3	6.7	AW	Test Hole	1250
TH-72	439,895.2	6,811,031.0	1,251.3	7.6	AW	Test Hole	1250
TH-73	439,885.8	6,810,926.7	1,270.0	27.7	AW	Test Hole	1270
TH-74	439,886.6	6,810,922.3	1,270.6	11.3	AW	Test Hole	1270
TH-75	439,884.9	6,810,917.4	1,271.0	34.1	AW	Test Hole	1270
TH-77	439,708.8	6,811,251.3	1,283.2	28	AW	Test Hole	1280
TH-78	439,712.8	6,811,250.5	1,283.2	28	AW	Test Hole	1280
TH-79	439,719.2	6,811,251.5	1,283.2	28.4	AW	Test Hole	1280
TH-80	439,656.2	6,811,337.1	1,263.0	40	AW	Test Hole	1260
TH-81	439,656.2	6,811,337.1	1,263.0	40	AW	Test Hole	1260
TH-82	439,656.2	6,811,337.1	1,263.0	40	AW	Test Hole	1260
TH-83	439,656.2	6,811,337.1	1,263.0	40	AW	Test Hole	1260
WV00-113	439,618.5	6,811,358.3	1,398.9	143	-	Wolverine	Surface
WV00-114	439,649.4	6,811,315.8	1,396.2	136.9	-	Wolverine	Surface
WV00-115	441,433.5	6,810,079.1	1,376.8	344.12	-	Sable	Surface
WV00-116	439,567.7	6,811,452.1	1,402.5	157	-	Wolverine	Surface
WV00-117	439,675.6	6,811,290.1	1,395.0	146	-	Wolverine	Surface
WV00-118	439,590.6	6,811,406.1	1,401.0	139.3	-	Wolverine	Surface
WV00-119	439,600.6	6,811,375.1	1,400.0	116.13	-	Wolverine	Surface
WV00-120	439,699.6	6,811,208.1	1,392.0	119.5	-	Wolverine	Surface
WV04-121	439,548.9	6,811,470.0	1,401.9	175.3	-	Wolverine	Surface
WV04-122	439,669.4	6,811,370.7	1,392.1	172.3	-	Wolverine	Surface
WV04-123	439,699.4	6,811,351.8	1,393.9	201.6	-	Wolverine	Surface
WV04-124	439,887.8	6,811,077.0	1,384.6	149.9	-	Wolverine	Surface
WV04-125	439,918.7	6,811,112.0	1,398.3	265.2	-	Wolverine	Surface
WV04-126	439,824.3	6,811,167.2	1,386.4	181.3	-	Wolverine	Surface
WV04-127	439,974.1	6,811,029.6	1,388.0	197.6	-	Wolverine	Surface
WV04-128	439,770.2	6,811,535.4	1,482.0	69.5	-	Wolverine	Surface
WV04-129	439,770.2	6,811,535.4	1,481.9	345.7	-	Wolverine	Surface
WV04-130	435,021.6	6,816,848.1	1,715.0	514.6	-	Fisher	Surface
WV04-131	435,096.6	6,816,685.1	1,695.0	487.8	-	Fisher	Surface
WV05-132	439,852.6	6,811,125.0	1,384.6	151.2	-	Wolverine	Surface

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Core Size	Hole Code	Heading
WV05-133	439,852.6	6,811,125.0	1,384.5	173.8	-	Wolverine	Surface
WV05-134	439,852.6	6,811,125.0	1,384.5	252.1	-	Wolverine	Surface
WV05-135	439,750.4	6,811,177.2	1,397.8	121	-	Wolverine	Surface
WV05-136	439,837.3	6,811,154.0	1,385.7	145.4	-	Wolverine	Surface
WV05-137	439,758.5	6,811,160.8	1,398.1	111.9	-	Wolverine	Surface
WV05-138	439,837.3	6,811,154.0	1,385.7	127.3	-	Wolverine	Surface
WV05-139	439,730.7	6,811,196.7	1,396.6	152.1	-	Wolverine	Surface
WV05-140	439,837.3	6,811,154.0	1,385.7	182	-	Wolverine	Surface
WV05-141	439,730.7	6,811,196.7	1,396.5	111.9	-	Wolverine	Surface
WV05-142	439,787.1	6,811,181.2	1,389.3	161.1	-	Wolverine	Surface
WV05-143	439,730.7	6,811,196.7	1,396.5	108.2	-	Wolverine	Surface
WV05-144	439,787.1	6,811,181.2	1,389.2	182	-	Wolverine	Surface
WV05-145	439,758.5	6,811,160.8	1,398.1	124.2	-	Wolverine	Surface
WV05-146	439,706.0	6,811,195.3	1,394.0	108.8	-	Wolverine	Surface
WV05-147	439,787.1	6,811,181.2	1,389.2	108.8	-	Wolverine	Surface
WV05-148	439,706.0	6,811,195.3	1,394.0	96.7	-	Wolverine	Surface
WV05-149	439,687.6	6,811,225.2	1,390.9	99.7	-	Wolverine	Surface
WV05-150	439,859.3	6,811,109.7	1,384.4	121	-	Wolverine	Surface
WV05-151	439,687.6	6,811,225.2	1,390.8	178.9	-	Wolverine	Surface
WV05-152	439,859.3	6,811,109.7	1,384.4	254.8	-	Wolverine	Surface
WV05-153	439,671.4	6,811,288.6	1,396.3	121	-	Wolverine	Surface
WV05-154	439,671.4	6,811,288.6	1,396.2	148.5	-	Wolverine	Surface
WV05-155	439,975.9	6,811,029.5	1,388.5	198.5	-	Wolverine	Surface
WV05-156	439,750.9	6,811,300.0	1,397.7	194.2	-	Wolverine	Surface
WV05-157	439,925.6	6,811,086.5	1,391.8	221	-	Wolverine	Surface
WV05-158	439,640.5	6,811,410.3	1,393.6	239.9	-	Wolverine	Surface
WV05-159	439,640.5	6,811,410.3	1,393.5	154.6	-	Wolverine	Surface
WV05-160	439,974.1	6,811,029.6	1,388.1	182	-	Wolverine	Surface
WV05-161	439,690.7	6,811,383.9	1,395.9	252.1	-	Wolverine	Surface
WV05-162	439,951.1	6,811,047.9	1,385.8	230.7	-	Wolverine	Surface
WV05-163	439,951.1	6,811,047.9	1,385.8	198.2	-	Wolverine	Surface
WV05-164	439,974.1	6,811,029.6	1,388.0	268.9	-	Wolverine	Surface
WV05-165	439,925.6	6,811,086.5	1,391.7	160.6	-	Wolverine	Surface
WV05-166	439,648.3	6,811,443.2	1,398.0	236.8	-	Wolverine	Surface
WV05-167	439,898.4	6,811,073.5	1,384.3	201.2	-	Wolverine	Surface
WV05-168	440,027.5	6,810,954.9	1,389.3	255.1	-	Wolverine	Surface
WV05-169	439,607.3	6,811,443.8	1,397.0	215.3	-	Wolverine	Surface
WV05-170	439,925.6	6,811,086.5	1,391.7	218.5	-	Wolverine	Surface
WV05-171	439,925.6	6,811,086.5	1,391.7	291.8	-	Wolverine	Surface
WV05-172	439,626.5	6,811,467.7	1,402.3	307	-	Wolverine	Surface
WV05-173	439,722.4	6,811,322.7	1,393.9	218.6	-	Wolverine	Surface

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Core Size	Hole Code	Heading
WV05-174	439,699.0	6,811,350.6	1,393.6	249.1	-	Wolverine	Surface
WV05-175	439,751.0	6,811,578.4	1,481.4	380.2	-	Wolverine	Surface
WV05-176	439,721.6	6,811,323.1	1,393.5	163.7	-	Wolverine	Surface
WV05-177	439,703.6	6,811,345.1	1,393.2	194.2	-	Wolverine	Surface
WV05-178	439,770.7	6,811,538.2	1,482.3	346.6	-	Wolverine	Surface
WV05-179	439,732.6	6,811,313.1	1,395.4	257.6	-	Wolverine	Surface
WV05-180	439,655.6	6,811,406.1	1,393.0	172.9	-	Wolverine	Surface
WV05-181	439,770.7	6,811,538.2	1,482.3	355.8	-	Wolverine	Surface
WV05-182	439,694.6	6,811,377.1	1,394.6	200.3	-	Wolverine	Surface
WV05-183	439,797.6	6,811,489.1	1,486.0	351.1	-	Wolverine	Surface
WV05-184	439,655.6	6,811,311.1	1,396.9	142.4	-	Wolverine	Surface
WV05-185	439,697.6	6,811,280.1	1,394.2	124.1	-	Wolverine	Surface
WV05-186	440,059.9	6,811,165.7	1,486.6	364.8	-	Wolverine	Surface
WV05-187	439,974.4	6,811,029.4	1,388.0	203.4	-	Wolverine	Surface
WV05-188	439,997.6	6,810,980.4	1,387.4	236.9	-	Wolverine	Surface
WV05-189	439,997.6	6,810,980.4	1,387.4	279.5	-	Wolverine	Surface
WV05-190	439,951.1	6,811,046.8	1,386.1	102.7	-	Wolverine	Surface
WV06-191	439,351.4	6,811,194.0	1,339.9	78.6	HQ	Water	Surface
WV06-192	439,351.4	6,811,194.0	1,339.9	171.1	HQ	Water	Surface
WV06-193	439,356.7	6,811,190.0	1,340.0	82.8	HQ	Water	Surface
WV06-194	439,362.4	6,811,186.5	1,340.1	22.9	HQ	Water	Surface
WV06-195	439,993.4	6,810,667.6	1,340.8	104.3	HQ	Water	Surface
WV06-196	439,988.4	6,810,670.9	1,340.6	19.8	HQ	Water	Surface
WV06-197	440,214.1	6,810,460.9	1,339.0	21.3	HQ	Water	Surface
WV06-198	439,747.0	6,810,819.9	1,332.5	21.3	HQ	Water	Surface
WV06-199	439,980.7	6,810,539.8	1,340.5	80.4	HQ	Water	Surface
WV06-200	439,980.1	6,810,537.3	1,340.8	21.3	HQ	Water	Surface
WV06-201	439,985.3	6,810,672.0	1,341.0	143.9	HQ	Water	Surface
WV08-202	439,715.5	6,811,140.0	1,308.7	40.8	BQTK	Geotech	Main Ramp
WV08-203	439,714.3	6,811,136.5	1,309.6	98.4	BQTK	Geotech	Main Ramp
WV08-204	439,772.0	6,811,126.4	1,315.4	50	BQTK	Geotech	Main Ramp
WV08-205	439,774.7	6,811,130.2	1,315.6	21	BQTK	Geotech	Main Ramp
WV08-206	439,787.7	6,811,108.2	1,322.5	91.1	BQTK	Geotech	Main Ramp
WV08-207	439,790.0	6,811,109.7	1,320.2	65.5	BQTK	Geotech	Main Ramp
WV08-208	439,724.2	6,811,127.4	1,309.4	39.3	BQTK	Geotech	Main Ramp
WV09-209	439,707.0	6,811,146.3	1,310.1	85.1	BQTK	Geotech	Main Ramp
WV09-210	439,796.8	6,811,098.5	1,323.6	33.2	BQTK	Geotech	Main Ramp
WV09-211	439,797.4	6,811,100.2	1,328.0	25	BQTK	Geotech	Main Ramp
WV09-212	439,707.0	6,811,146.3	1,308.6	24	BQTK	Geotech	Main Ramp
WV09-213	439,714.3	6,811,136.5	1,308.6	11.9	BQTK	Geotech	Main Ramp
WV09-214	439,707.0	6,811,146.3	1,310.9	24.08	BQTK	Geotech	Main Ramp

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Core Size	Hole Code	Heading
WV09-215	439,707.0	6,811,146.3	1,310.6	42.37	BQTK	Geotech	Main Ramp
WV09-216	439,707.0	6,811,146.3	1,310.4	68.58	BQTK	Geotech	Main Ramp
WV09-217	439,707.5	6,811,146.0	1,310.1	12.19	BQTK	Geotech	Main Ramp
WV09-218	439,716.4	6,811,141.7	1,312.1	10.36	BQTK	Geotech	Main Ramp
WV09-219	439,706.4	6,811,146.9	1,310.9	59.13	BQTK	Geotech	Main Ramp
WV09-220	439,589.3	6,811,405.1	1,399.7	187	NQTK	Wolverine	Surface
WV09-221	439,590.2	6,811,406.3	1,399.7	175.26	NQTK	Wolverine	Surface
WV09-222	439,603.6	6,811,381.6	1,398.6	188.48	NQTK	Wolverine	Surface
WV09-223B	439,651.5	6,811,317.1	1,395.7	131.09	NQTK	Wolverine	Surface
WV09-224	439,617.0	6,811,357.5	1,397.4	134.15	NQTK	Wolverine	Surface
WV09-225	439,617.0	6,811,357.5	1,397.4	124.97	NQTK	Wolverine	Surface
WV09-226	439,618.4	6,811,358.2	1,397.1	166.12	NQTK	Wolverine	Surface
WV09-227	439,641.4	6,811,345.6	1,396.5	152.4	NQTK	Wolverine	Surface
WV09-228	439,640.9	6,811,346.3	1,396.4	205.74	NQTK	Wolverine	Surface
WV09-229	439,651.5	6,811,317.7	1,395.5	179.22	NQTK	Wolverine	Surface
WV09-230	439,651.7	6,811,317.8	1,395.6	202.69	NQTK	Wolverine	Surface
WV09-231	439,666.7	6,811,294.4	1,394.3	120.09	NQTK	Wolverine	Surface
WV09-232	439,667.0	6,811,295.1	1,394.2	149.35	NQTK	Wolverine	Surface
WV09-233	439,679.4	6,811,268.5	1,392.1	109.73	NQTK	Wolverine	Surface
WV09-234	439,689.6	6,811,281.9	1,394.6	149.35	NQTK	Wolverine	Surface
WV09-235	439,694.3	6,811,244.4	1,391.5	120.4	NQTK	Wolverine	Surface
WV09-236	439,703.6	6,811,216.6	1,391.7	150.88	NQTK	Wolverine	Surface
WV09-237	439,700.0	6,811,209.5	1,391.6	108.2	NQTK	Wolverine	Surface
WV09-238	439,870.9	6,811,104.9	1,382.3	153.92	NQTK	Wolverine	Surface
WV09-239	439,891.2	6,811,044.3	1,382.0	128.02	NQTK	Wolverine	Surface
WV09-240	439,909.1	6,811,027.0	1,380.6	150.88	NQTK	Wolverine	Surface
WV09-241	439,921.9	6,810,999.2	1,377.7	146.34	NQTK	Wolverine	Surface
WV09-242	439,934.4	6,811,019.9	1,377.2	166.12	NQTK	Wolverine	Surface
WV09-243	439,934.5	6,811,020.1	1,377.2	191.42	NQTK	Wolverine	Surface
WV09-244	439,843.1	6,811,062.3	1,385.4	118.9	NQTK	Wolverine	Surface
WV09-245	439,692.4	6,811,286.6	1,394.7	217.93	NQTK	Wolverine	Surface
WV09-246	439,737.1	6,811,311.0	1,394.1	190.5	NQTK	Wolverine	Surface
WV09-247	439,750.6	6,811,293.6	1,395.4	179.83	NQTK	Wolverine	Surface
WV09-248	439,940.7	6,810,983.2	1,374.5	150.88	NQTK	Wolverine	Surface
WV09-249	439,957.4	6,810,963.1	1,372.6	167.64	NQTK	Wolverine	Surface
WV10-250	439,686.5	6,811,224.5	1,298.1	35	AW	Definition	1300
WV10-251	439,686.6	6,811,224.5	1,298.1	35	AW	Definition	1300
WV10-252	439,686.6	6,811,224.6	1,298.1	30	AW	Definition	1300
WV10-253	439,691.8	6,811,228.2	1,298.0	25	AW	Definition	1300
WV10-254	439,684.0	6,811,220.9	1,300.5	25.3	AW	Definition	1300
WV10-255	439,683.9	6,811,220.8	1,301.0	25	AW	Definition	1300

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Core Size	Hole Code	Heading
WV10-256	439,686.5	6,811,224.5	1,298.7	25	AW	Definition	1300
WV10-257	439,686.5	6,811,224.5	1,298.7	25	AW	Definition	1300
WV10-258	439,686.5	6,811,224.5	1,298.7	25	AW	Definition	1300
WV10-259	439,686.5	6,811,224.5	1,298.7	25	AW	Definition	1300
WV10-260	439,686.6	6,811,224.5	1,298.7	26	AW	Definition	1300
WV10-261	439,691.7	6,811,228.0	1,297.6	25	AW	Definition	1300
WV10-262	439,691.7	6,811,227.9	1,297.6	25	AW	Definition	1300
WV95-001	439,844.1	6,811,029.5	1,376.8	187.7	BQTK	Wolverine	Surface
WV95-002	440,151.3	6,810,765.1	1,370.0	160	BQTK	Sable	Surface
WV95-003	439,442.1	6,811,693.9	1,392.0	212.1	BQTK	Wolverine	Surface
WV95-004	439,918.6	6,811,111.8	1,400.3	226.2	BQTK	Wolverine	Surface
WV95-005	439,922.4	6,811,115.7	1,400.9	236.5	BQTK	Wolverine	Surface
WV95-006	433,887.9	6,817,491.8	1,679.3	233.5	BQTK	Fisher	Surface
WV95-007	434,561.0	6,816,857.4	1,713.3	465.7	BQTK	Fisher	Surface
WV95-008	439,852.7	6,811,125.0	1,384.6	172.5	BQTK	Wolverine	Surface
WV95-009	439,862.4	6,811,119.2	1,384.4	285.3	BQTK	Wolverine	Surface
WV95-010	439,944.3	6,811,065.3	1,385.3	229.8	BQTK	Wolverine	Surface
WV95-011	439,862.4	6,811,119.2	1,384.4	360.6	BQTK	Wolverine	Surface
WV95-012	439,944.3	6,811,065.3	1,385.3	257.9	BQTK	Wolverine	Surface
WV95-013	439,944.3	6,811,065.3	1,385.3	399.3	BQTK	Wolverine	Surface
WV95-014	436,590.0	6,814,223.2	1,297.2	404.8	BQTK	Fox	Surface
WV95-015	433,818.0	6,817,725.8	1,669.9	207.9	BQTK	Fisher	Surface
WV95-016	439,972.5	6,811,030.0	1,387.6	283.8	BQTK	Wolverine	Surface
WV95-017	438,207.6	6,813,654.3	1,380.0	109.1	BQTK	Fox	Surface
WV95-018	439,972.5	6,811,030.0	1,387.6	273.1	BQTK	Wolverine	Surface
WV95-019	440,224.8	6,810,860.4	1,402.5	412.4	BQTK	Sable	Surface
WV95-020	439,899.4	6,811,290.0	1,460.5	315.8	BQTK	Wolverine	Surface
WV95-021	439,899.4	6,811,290.0	1,460.5	322.8	BQTK	Wolverine	Surface
WV95-022	440,050.4	6,811,090.1	1,444.5	343.8	BQTK	Wolverine	Surface
WV95-023	439,931.9	6,811,048.3	1,380.7	177.1	BQTK	Wolverine	Surface
WV95-024	439,819.7	6,811,165.2	1,386.4	170.4	BQTK	Wolverine	Surface
WV96-025	440,059.6	6,811,164.2	1,487.4	437.1	BQTK	Wolverine	Surface
WV96-026	440,103.1	6,811,139.2	1,486.8	438	BQTK	Wolverine	Surface
WV96-027	440,059.6	6,811,164.2	1,487.4	416.8	BQTK	Wolverine	Surface
WV96-028	440,144.8	6,811,118.9	1,488.2	393.2	BQTK	Wolverine	Surface
WV96-029	440,213.0	6,811,033.4	1,465.4	633.1	BQTK	Wolverine	Surface
WV96-030	440,144.8	6,811,118.9	1,488.2	326.1	BQTK	Wolverine	Surface
WV96-031	440,182.7	6,811,078.4	1,483.4	516.3	BQTK	Wolverine	Surface
WV96-032	440,144.8	6,811,118.9	1,488.2	546.5	BQTK	Wolverine	Surface
WV96-033	440,868.9	6,810,156.4	1,365.9	302.1	BQTK	Sable	Surface
WV96-034	440,395.5	6,810,568.5	1,379.9	295.4	BQTK	Sable	Surface

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Core Size	Hole Code	Heading
WV96-035	440,000.6	6,811,258.1	1,502.9	513.4	BQTK	Wolverine	Surface
WV96-036	439,958.9	6,811,358.6	1,504.9	469.4	BQTK	Wolverine	Surface
WV96-037	433,999.3	6,817,599.3	1,675.0	380.7	BQTK	Fisher	Surface
WV96-038	433,887.2	6,817,792.8	1,670.0	508.7	BQTK	Fisher	Surface
WV96-039	439,998.1	6,811,256.2	1,502.9	463.3	BQTK	Wolverine	Surface
WV96-040	439,915.1	6,811,399.3	1,505.2	457.2	BQTK	Wolverine	Surface
WV96-041	434,363.1	6,817,060.6	1,710.0	539.5	BQTK	Fisher	Surface
WV96-042	434,166.4	6,817,489.3	1,705.0	392.9	BQTK	Fisher	Surface
WV96-043	439,913.5	6,811,396.8	1,505.2	404.5	BQTK	Wolverine	Surface
WV96-044	440,358.7	6,810,834.5	1,427.0	343.5	BQTK	Sable	Surface
WV96-045	434,339.0	6,817,387.7	1,680.0	405.1	BQTK	Fisher	Surface
WV96-046	440,531.2	6,810,498.3	1,370.7	323.7	BQTK	Sable	Surface
WV96-047	436,264.4	6,814,492.5	1,260.0	391.4	BQTK	Fox	Surface
WV96-048	439,886.6	6,811,446.9	1,506.0	400.8	BQTK	Wolverine	Surface
WV96-049	440,059.1	6,811,163.3	1,487.4	472.4	BQTK	Wolverine	Surface
WV96-050	439,886.6	6,811,446.9	1,506.0	397.5	BQTK	Wolverine	Surface
WV96-051	434,500.5	6,817,269.4	1,700.0	481.3	BQTK	Fisher	Surface
WV96-052	439,842.3	6,811,468.8	1,492.3	405.1	BQTK	Wolverine	Surface
WV96-053	439,842.3	6,811,468.8	1,492.3	341.1	BQTK	Wolverine	Surface
WV96-054	439,942.8	6,810,939.4	1,380.0	218.3	BQTK	Wolverine	Surface
WV96-055	435,283.1	6,816,275.6	1,680.0	546.8	BQTK	Fisher	Surface
WV96-056	439,801.9	6,811,500.2	1,489.0	370.9	BQTK	Wolverine	Surface
WV96-057	439,757.9	6,811,160.4	1,398.0	158.5	BQTK	Wolverine	Surface
WV96-058	439,704.6	6,811,261.4	1,394.0	168.2	BQTK	Wolverine	Surface
WV96-059	439,817.8	6,811,069.5	1,388.5	116.43	BQTK	Wolverine	Surface
WV96-060	439,648.3	6,811,443.2	1,398.0	288.34	BQTK	Wolverine	Surface
WV96-061	439,672.5	6,811,396.2	1,394.0	244.14	BQTK	Wolverine	Surface
WV96-062	439,581.5	6,811,438.1	1,402.0	211.83	BQTK	Wolverine	Surface
WV96-063	439,634.8	6,811,337.2	1,398.0	184.4	BQTK	Wolverine	Surface
WV96-064	439,688.7	6,811,601.0	1,472.0	405.1	BQTK	Wolverine	Surface
WV96-065	439,581.5	6,811,438.1	1,401.0	208.2	BQTK	Wolverine	Surface
WV96-066	439,605.7	6,811,383.9	1,399.0	175.9	BQTK	Wolverine	Surface
WV96-067	439,608.4	6,811,388.1	1,399.0	142.3	BQTK	Wolverine	Surface
WV96-068	439,703.0	6,811,349.8	1,393.5	177.7	BQTK	Wolverine	Surface
WV96-069	439,768.4	6,811,540.2	1,483.0	378.9	BQTK	Wolverine	Surface
WV96-070	439,667.2	6,811,295.5	1,395.0	139.3	BQTK	Wolverine	Surface
WV96-071	439,638.1	6,811,524.2	1,411.0	257.3	BQTK	Wolverine	Surface
WV96-072	439,632.1	6,811,333.0	1,397.0	134.7	BQTK	Wolverine	Surface
WV97-073	438,372.6	6,813,683.1	1,440.0	385.3	BQTK	Fox	Surface
WV97-074	439,704.7	6,811,260.6	1,394.8	151.5	BQTK	Wolverine	Surface
WV97-075	439,524.5	6,811,535.3	1,397.1	194.5	BQTK	Wolverine	Surface

table continues...

BHID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Core Size	Hole Code	Heading
WV97-076	439,720.7	6,811,192.3	1,395.7	151.5	BQTK	Wolverine	Surface
WV97-077	439,623.7	6,811,228.6	1,375.6	113.4	BQTK	Wolverine	Surface
WV97-078	437,888.6	6,813,657.1	1,330.0	193.2	BQTK	Fox	Surface
WV97-079	439,524.6	6,811,535.4	1,397.1	273.4	BQTK	Wolverine	Surface
WV97-080	439,580.0	6,811,252.4	1,375.4	85.34	BQTK	Wolverine	Surface
WV97-081	439,525.2	6,811,347.1	1,375.3	75.3	BQTK	Wolverine	Surface
WV97-082	439,720.7	6,811,192.3	1,395.7	160.6	BQTK	Wolverine	Surface
WV97-083	436,772.6	6,813,805.1	1,240.0	192.42	BQTK	Fox	Surface
WV97-084	439,548.7	6,811,469.2	1,401.7	183.2	BQTK	Wolverine	Surface
WV97-085	436,343.6	6,814,273.1	1,280.0	197.5	BQTK	Fox	Surface
WV97-086	439,552.5	6,811,472.2	1,401.7	261.2	BQTK	Wolverine	Surface
WV97-087	435,546.6	6,816,345.1	1,620.0	617.8	BQTK	Fisher	Surface
WV97-088	439,610.1	6,811,556.5	1,405.6	279.5	BQTK	Wolverine	Surface
WV97-089	439,562.4	6,811,587.0	1,396.1	145.4	BQTK	Wolverine	Surface
WV97-090	439,791.7	6,811,491.1	1,486.0	403.6	BQTK	Wolverine	Surface
WV97-091	435,222.6	6,816,593.1	1,693.0	610.8	BQTK	Fisher	Surface
WV97-092	441,124.6	6,810,051.1	1,348.0	276.4	BQTK	Sable	Surface
WV97-093	440,719.6	6,810,350.1	1,370.0	252.1	BQTK	Sable	Surface
WV97-094	439,720.7	6,811,192.3	1,395.7	197.2	BQTK	Wolverine	Surface
WV97-095	439,758.5	6,811,160.8	1,398.1	148.44	BQTK	Wolverine	Surface
WV97-096	434,868.6	6,816,752.1	1,715.0	572.4	BQTK	Fisher	Surface
WV97-097	439,791.7	6,811,118.6	1,395.0	163.7	BQTK	Wolverine	Surface
WV97-098	441,421.0	6,809,981.0	1,395.0	443.8	BQTK	Sable	Surface
WV97-099	439,704.7	6,811,260.6	1,394.8	148.4	BQTK	Wolverine	Surface
WV97-100A	440,005.6	6,810,894.1	1,370.7	242.93	BQTK	Wolverine	Surface
WV97-101	435,834.6	6,816,054.1	1,611.0	602.9	BQTK	Fisher	Surface
WV97-102	440,143.0	6,811,115.3	1,487.0	419.7	BQTK	Wolverine	Surface
WV97-103	435,531.6	6,816,652.1	1,575.0	549.9	BQTK	Fisher	Surface
WV97-103A	435,531.6	6,816,652.1	1,575.0	701.3	BQTK	Fisher	Surface
WV97-104	439,884.7	6,811,453.1	1,507.9	465.4	BQTK	Wolverine	Surface
WV97-105	440,184.1	6,811,077.3	1,484.2	423.4	BQTK	Wolverine	Surface
WV97-106	441,291.8	6,809,932.8	1,400.0	222.2	BQTK	Sable	Surface
WV97-107	441,210.4	6,809,991.5	1,352.0	213	BQTK	Sable	Surface
WV97-108	441,350.9	6,809,914.6	1,355.0	173.4	BQTK	Sable	Surface
WV97-109	441,342.8	6,810,019.2	1,370.0	207	BQTK	Sable	Surface
WV97-110	441,382.3	6,809,995.7	1,360.0	258.8	BQTK	Sable	Surface
WV97-111	440,124.7	6,810,989.2	1,428.4	341.1	BQTK	Wolverine	Surface
WV97-112	440,184.1	6,811,077.3	1,484.2	483.7	BQTK	Wolverine	Surface
WW00-001	439,742.6	6,811,676.1	1,512.7	463.3	BQTK	Wolverine	Surface
WW00-002	439,742.6	6,811,676.1	1,512.7	684.6	BQTK	Wolverine	Surface
WW00-003	439,770.6	6,811,634.1	1,513.0	516.94	BQTK	Wolverine	Surface

10.2 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.

10.3 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 drillhole collar locations had been surveyed (Universal Transverse Mercator (UTM) northing, UTM easting, and elevation) in North American Datum (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.

10.4 DOWN HOLE SURVEYS

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

10.5 POST-2006 DRILLING

Post 2006 drilling is referenced from J. Moore, 2009 Diamond Drilling Report For the Wolverine Project Claims in the Watson Lake Mining District, Yukon Territory, Canada, May-October, 2009.

In 2005, Expatriate changed its name to Yukon Zinc Corp. A drilling program was completed finishing 59 drillholes totaling 11,712.5 m. These and previous drillholes were used to calculate a reserve and submitted as part of a NI 43-101 report(s) in 2006 and 2007. Underground development commenced, and totaled 450 m of underground advancement. This included 110 m of development within the massive sulphide mineralization.

Yukon Zinc contracted Kluane Drilling Company to diamond drill WV09-221 to WV09-249 drillholes (Table 10.2). The drillholes were focused in the Wolverine Deposit area, in the Lynx and Wolverine Zones. Drill results for the 2009 program confirmed mineralization from previous drilling programs and infilled needed information between previously drilled holes. All apparently mineralized intersections were assayed as well as 3 to 5 m into the hanging wall and footwall. The 2009 drilling program had satisfactory results and confirmed previous work.

Table 10.2 2009 Diamond Drilling Program

Hole ID	North	East	Elevation	End of Hole (m)	Core Size	Zone
WV09-220	6811405.12	439589.25	1399.71	187.00	NTQ	Lynx
WV09-221	6811406.26	439590.20	1399.65	175.26	NTQ	Lynx
WV09-222	6811381.61	439603.64	1398.58	188.48	NTQ	Lynx
WV09-223b	6811317.08	439651.46	1395.70	131.09	NTQ	Lynx
WV09-224	6811357.52	439617.03	1397.40	134.14	NTQ	Lynx
WV09-225	6811357.50	439617.04	1397.36	124.97	NTQ	Lynx
WV09-226	6811358.20	439618.40	1397.10	166.12	NTQ	Lynx
WV09-227	6811345.62	439641.40	1396.45	152.40	NTQ	Lynx
WV09-228	6811346.25	439640.85	1396.37	205.74	NTQ	Lynx
WV09-229	6811317.65	439651.49	1395.54	179.22	NTQ	Lynx
WV09-230	6811317.78	439651.67	1395.62	202.69	NTQ	Lynx
WV09-231	6811294.42	439666.68	1394.33	120.09	NTQ	Lynx
WV09-232	6811295.07	439667.04	1394.21	149.35	NTQ	Lynx
WV09-233	6811268.45	439679.35	1392.06	109.73	NTQ	Lynx
WV09-234	6811281.86	439689.56	1394.62	149.35	NTQ	Lynx
WV09-235	6811244.36	439694.31	1391.54	120.40	NTQ	Lynx
WV09-236	6811216.64	439703.56	1391.68	150.88	NTQ	Lynx
WV09-237	6811209.51	439699.97	1391.63	108.20	NTQ	Lynx
WV09-238	6811104.90	439870.92	1382.30	153.92	NTQ	Wolverine
WV09-239	6811044.27	439891.17	1382.03	128.02	NTQ	Wolverine

table continues...

Hole ID	North	East	Elevation	End of Hole (m)	Core Size	Zone
WV09-240	6811026.96	439909.06	1380.61	150.88	NTQ	Wolverine
WV09-241	6810999.24	439921.86	1377.71	146.34	NTQ	Wolverine
WV09-242	6811019.87	439934.42	1377.17	166.12	NTQ	Wolverine
WV09-243	6811020.08	439934.48	1377.24	191.42	NTQ	Wolverine
WV09-244	6811062.31	439843.09	1385.36	118.87	NTQ	Wolverine
WV09-245	6811286.63	439692.39	1394.69	217.93	NTQ	Lynx
WV09-246	6811311.00	439737.07	1394.09	190.50	NTQ	Lynx
WV09-247	6811293.61	439750.61	1395.38	179.83	NTQ	Lynx
WV09-248	6810983.20	439940.71	1374.47	150.88	NTQ	Wolverine
WV09-249	6810963.12	439957.39	1372.64	167.64	NTQ	Wolverine

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

From Cook (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 Lager 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 Yukon Zinc'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.

11.1 ANALYTICAL PROCEDURES – 2004 TO 2008

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 inductively coupled plasma (ICP) package with a perchloric, nitric, and hydrofluoric acid decomposition that has an inductively coupled plasma-atomic emission spectroscopy (ICP-AES) method. Detection limits have a wide range because of the 25 elements.

Over limits for copper (Cu-OG46), lead (Pb-OG46), zinc (Zn-OG46), arsenic (As) (As-OG46) and antimony (Sb) (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 atomic absorption (AA) analysis with an upper detection limit of 0.01%.

Mercury is not part of the ME-ICP61a package, so mercury is analyzed by mercury (Hg)-CV41 that is an aqua regia digestion and cold-vapour 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.

11.2 ANALYTICAL PROCEDURES – 2009

From Moore (2009):

Acme Analytical Laboratories Ltd. on 1020 Cordova St. East, Vancouver, BC was used for the Wolverine Project summer 2009 surface drilling program. Acme is currently registered with ISO 9001:2000 accreditation. This is a global standardization of quality assurance for products and services. Mr. Clarence Leong, a BC Certified Assayer and Acme General Manager supervised the analytical process with the project geologist and Phil Mulholland, CPG.

11.2.1 PROCESS

The project geologist supervised the sample shipment procedure. Samples were shipped in doubled rice bags on pallets, which were then plastic sealed, from the Wolverine Project and received in the loading bay at Acme. R & L Expediting, Twilight Expediting and Manitoulin Trucking Limited were used as shipping agents and shipments were tracked by the project geologist and office staff. A request for analysis is submitted with each sample shipment, which outlines the analytical method that has been requested and the samples that were shipped. Analytical packages that have been requested are Group 7AX and Group 6 Fire Assay for Au-Ag and for overlimits Group 7AR.

11.2.2 STANDARDS

For all three analytical methods standard reference materials are used, analysis are repeated and duplicate analysis of sample pulps are analyzed. The analytical resultant values are used to estimate analytical accuracy and precision.

Acme Analytical Laboratories provides standard reference materials which are used in the analytical process to estimate analytical accuracy. In Group 7AX analysis standard SF-3A are used. In Group G Fire Assay OXP39 and OXP61 are used. In Group 7AR analysis standards CZN-3, GBM997-6, CCU-1C and PTC-1A are used.

Internal standards were purchased at CDN Laboratories and these were inserted every 10 samples in the sample stream.

11.2.3 REPEATS

For repeat samples, a sample is crushed, pulverized and split into two portions. Two samples will be taken from one portion of the split. The second sample is called the repeat. Analysis of a sample are thus 'repeated' and resultant values compared, this is a control measure for precision and accuracy in analytical procedures. Overall, the data is satisfactory.

11.2.4 DUPLICATES

When the sample is submitted to the lab it is crushed, pulverized and split. One sample is taken from either side of the split. A duplicate analysis is performed on a subsample of the sample. The analytical results from duplicate samples provide a measure of the variability inherent in the sampling and analysis process. The analyses are compared as part of the quality checking process and the element zinc is reported on. The performance of the duplicate samples and analysis in Wolverine Project drilling is satisfactory.

11.2.5 SECOND LABORATORY

When the sample is submitted to the lab it is crushed, split and pulverized. One sample pulp is analysed. A selection of low, medium and high samples were sent to ALS Chemex for repeat analysis of the pulps. The analytical results from repeat samples provides a measure of the variability inherent in the sampling and analysis process. The analyses are compared as part of the quality checking process and the element zinc is reported on. The performance of the repeat samples and analysis in Wolverine Project drilling is satisfactory.

11.3 QUALITY CONTROL AND QUALITY ASSURANCE

From Pearson and Giroux (2006):

11.3.1 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 11.1. 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 11.1 Statistics for Standards

STD 1	Copper	Lead	Zinc	Silver	Gold	STD 2	Copper	Lead	Zinc	Silver	Gold
Average	24163	12377	216500	98	2.6	Average	25630	32083	240100	183	4.4
Median	24250	12300	216500	99	2.2	Median	25600	31900	241000	184	4.8
S.D.	589	336	50000	5	1.7	S.D.	346	855	50000	9	1.3
Minimum	22300	11850	208000	88	2.0	Minimum	25000	30200	227000	163	2.2
Maximum	25100	13300	232000	109	11.0	Maximum	26400	33400	248000	195	6.0
STD 3	Copper	Lead	Zinc	Silver	Gold						
Average	37747	28121	216300	218	6.4						
Median	39800	29500	215000	210	5.6						
S.D.	6094	5767	73000	75	3.9						
Minimum	24000	12200	207000	101	2.3						
Maximum	47200	35300	241000	482	21.0						

11.3.2 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.

11.3.3 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 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.

11.3.4 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.

11.3.5 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 closely 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 11.2. 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 11.2 R²-Values for ALS Chemex vs. ACME Comparison for Metal Values

Element	Drillhole				
	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	1.0000	0.9981	0.9958	0.9988	0.9994
Ag	0.9985	0.9861	0.9984	0.9737	0.9974
Au	0.8844	0.7764	0.9910	0.9859	0.9760

11.3.6 WOLVERINE MINE SITE LABORATORY

A new assay laboratory is nearing completion at the Wolverine site. It is currently assaying a number of different materials from the mine and mill for base metals and silver. A circuit to undertake fire-assay with an AA finish will soon be implemented as well. Previously, all assays were completed off-site by ALS Chemex. Photographs of the new site assay laboratory are presented below (Figure 11.1 to Figure 11.4).

The new site-based laboratory includes trained and experienced staff who are well versed in handling high-grade ore samples. Contamination is always a priority, and the new laboratory will use standards, blanks and duplicates to monitor assay quality.

Main assayed elements will include iron, zinc, lead, copper, silver and gold.

Samples for low grade assay include, but are not limited to:

- face samples
- diamond drill core
- final tails
- pre-float concentrate
- zinc rougher tails
- zinc scavenger tails
- zinc cleaner scavenger tails
- copper cleaner scavenger tails
- zinc feed
- copper feed

- lead feed
- cyclone overflow.

Samples for high grade assay include, but are not limited to:

- no. 1 copper column
- copper concentrate
- copper bags – percent moisture
- lead concentrate
- lead bags – percent moisture
- zinc concentrate
- zinc filter - percent moisture
- zinc trucks - percent moisture.

In addition, Yukon Zinc has a purpose-built core facility in the former assay laboratory for logging, sampling and cutting diamond drill core on site. This facility was not visited, but indicates that Yukon Zinc is organized for ongoing underground drill core logging, sampling, processing and assaying.

Figure 11.1 Sample Receiving Area in Wolverine Mine Assay Lab



Figure 11.2 Sample Grinding Stations in Wolverine Mine Assay Lab



Figure 11.3 Atomic Absorption Unit in Wolverine Mine Assay Lab



Figure 11.4 Wet Chemical Analysis (Titration) for Base Metal Assays in Wolverine Mine Assay Lab



12.0 DATA VERIFICATION

From Wardrop (2007):

The 2005 definition drill program on the Wolverine deposit consists of 59 diamond drillholes for a total of 11,730 m. Drilling confirmed the extent of sulphide mineralization across the orebody and completely filled in the 25 m drill spacing required for the resource calculation.

12.1.1 CORE DRILLING

Between 1995 and 2005, the Wolverine Project drilled approximately 46,956 m of core in 159 surface drillholes. Throughout the exploration history, several drill contractors have been on the property. All contractors used diesel/wire line diamond-drill rigs with 10 ft core barrels to extract the core. The orebody delineation from 1995 to 1997 used BQTK-, BQ-, and NQ-sized core equipment. The definition drill program, which also supplied core for numerous DMS, metallurgical, and environmental tests, was taken from NQ-, NQ2-, and HQ-size core drilled in 2000, 2004, and 2005.

Core was extracted from the core barrel and then placed in sequentially numbered 5 ft core boxes and carefully brought to the core shack by the drill contractors or the geologist for the geologist to examine. Core recoveries were generally adequate. The hanging wall argillites have lower recovery while the massive sulphides, iron formation and calcite-pyrite exhalites and the andesite flows exhibit very good recoveries.

Underhill Geomatics Ltd. (Underhill), a registered survey firm in Whitehorse, Yukon, surveyed drill collars from 1995 to 2000. Collar locations were provided with a UTM northing, UTM easting, and elevation, in NAD 27 projection, Zone 9. In 2005, Procon Mining and Tunnelling Ltd. (Procon) surveyed the recent and existing collars to verify their locations. Underhill audited the Procon survey work and found no discrepancies. This survey validated the collar locations on the property.

Downhole surveys were performed with a single shot Sperry-Sun (photographic and magnetic) and acid dip tests between 1995 and 2000. Yukon Zinc purchased a Reflex EZ Shot survey tool (digital-magnetic) in 2004. In 2004 and 2005, holes were generally surveyed with 50 m spacing. However, drillholes greater than 300 m depth were surveyed every 30 m to accurately record the downhole deviation.

12.1.2 GEOLOGICAL LOGGING AND SAMPLING

Drill core was logged in the core shack by a team of geologists using project nomenclature on an Yukon Zinc drill log form. Specific information was recorded on lithology, alteration, sulphide mineralization, and structure including recovery within the lithology. Logs were manually entered into a logging software program designed to incorporate drillhole location, geology, geochemistry and downhole surveys. This program can export the database to Access if the data are required in another program. Data validation occurred weekly and at the end of the drill program.

A geological technician collected geotechnical logs for 2004 and 2005 drillholes to assist with project mine planning. The information included core recovery (per 3 m run), rock quality designation (RQD), fracture intensity, joint roughness and alteration, weathering, core angles, and rock strength. Detailed rock mass rating (RMR) information carried out under the instruction of a rock mechanic engineer was captured for a selection of drillholes from 2005.

Sampled intervals continued a minimum of 6 m into the hanging wall, the sulphide mineralized zone, and the footwall. The geologists sampled core for geochemical analyses after it had been logged. Sampling parameters were outlined by Yukon Zinc and followed to provide representative and consistent sample intervals. Parameters are as follows:

- lithology changes were broken out*
- maximum/minimum sample size were observed*
- geological or mineralogical variability.*

Sample intervals were clearly marked on the core box for the core cutter. Prior to cutting, the core was color photographed for archival purposes. Core was halved using a rock saw. One-half was sent to ALS Chemex for analyses and the other half was carefully replaced in the core box maintaining orientation.

12.1.3 CORE OR SAMPLE CHAIN AT CUSTODY

At least once a day, the drill contractor or geological personnel delivered the core to the core shack. Core preparation, logging, sampling, cutting, and sample bagging was carried out by geologists or geological technicians. Core boxes were marked with coded ALS Chemex assay tags when the core was halved. The sampled half was placed in a 6 mm poly bag with the corresponding ALS Chemex assay lab tag and sealed with a Zap-strap tie. Samples from individual drillholes were put in an addressed rice sack and sealed with a shipment summary statement detailing the number of sample bags and shipment dates. Rice sacks were flagged with different colors to designate different drillholes.

The rice sacks were stored in the core shack until sent to the Wolverine airstrip to be flown out on the chartered Alkan Air flight to Whitehorse. Samples were unloaded and stored in the locked cargo containment area until freight pick-up by Air North. Air North sent the samples to Vancouver and couriered them directly to the ALS Chemex laboratory. ALS Chemex received and promptly processed the samples.

The archived half of the core was returned to the core box and stored outdoors on the Wolverine property. Core boxes were stored in the upper core storage on racks and cross-piled on pallets. Boxes were labelled with drillhole number, box number, and meterages on embossed aluminium tape secured to the edge of the core box.

12.1.4 ASSAYING

Samples were entered into the ALS Chemex tracking system with sample weights upon arrival at the laboratory. The core samples were dried and crushed to pass through a 75 µm sieve, yielding a 250 g sample. Remaining sample and coarse rejects were weighed and stored at the laboratory.

The samples were processed with high sulphide sample techniques:

<i>Gold and Silver.....</i>	<i>30 g fire assay with gravimetric finish</i>
<i>Base Metals.....</i>	<i>25 element ICP package with perchloric, nitric, and hydrofluoric acid decomposition (that has an ICP-AES method)</i>
<i>Copper, Lead, Zinc & Arsenic....</i>	<i>Ore-grade aqua regia digestion with instrument finish</i>
<i>Antimony Over-Limit.....</i>	<i>Hydrochloric acid and potassium chlorate digestion and AA analysis</i>
<i>Mercury.....</i>	<i>Aqua regia digestion and cold-vapour AA analysis</i>
<i>Selenium.....</i>	<i>XRF, pressed pellet analysis</i>
<i>Specific Gravity.....</i>	<i>Pycnometer</i>

12.1.5 QA/QC

ALS Chemex employed various methods to ensure quality control of the data obtained from the Wolverine drill core. ALS Chemex quality control data are available for 2004 to 2005. Prior to this time, the property was controlled by Atna Resources Ltd. and Westmin Resources Ltd., thus the quality control procedures for that period are not available.

QA/QC procedures employed are described below.

- Standards, blanks, and lab pulp duplicates checked the analytical accuracy of ALS Chemex.*
- Duplicate assaying of pulps checked the precision of ALS Chemex, as determined by the reproducibility of lab results.*

STANDARDS

A total of 76 standards were submitted for analysis together with the drill core. There are three possible standards from the Myra Falls underground. Standards were randomly included within every 20 core samples. Thirty STD 1, twenty-six STD 2 and twenty STD 3 standards were submitted in 2005; the summary statistics of copper, lead, zinc, silver, and gold are presented in Table 12.1. Future standards will be derived from representative underground samples at the Wolverine deposit.

Table 12.1 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	2.6	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	21,500	210	5.6
SD	589	336	50,000	5	1.7	346	855	50,000	9	1.3	6,094	5,767	7,300	75	3.9
Minimum	22,300	11,850	208,000	88	2.0	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	11.0	26,400	33,400	248,000	195	6.0	47,200	35,300	241,000	482	21.0

Note: All measurements in parts per million (ppm).

BLANKS

Blanks were included in sampling to verify lab accuracy. One blank, composed of rhyolite from near the Wolverine property, was included in approximately 20 samples. A total of 71 blanks were submitted in 2005.

DUPLICATES

Duplicates of drill core were included in sampling to verify lab accuracy. Duplicates were indicated on the core box sample tag. The halved core for assay was split again using the rock saw and each quarter placed into a sample bag. One duplicate sample was included per 20 samples for a randomly chosen interval. A total of 59 duplicates were submitted in 2005.

INTERNAL QUALITY CONTROL ALS CHEMEX

ALS Chemex provided Yukon Zinc with an internal QA/QC report. The report included the originals of all geochemical results submitted to the laboratory, as well as the results from an analysis of certified and in-house standards, blanks, and duplicates that was conducted to maintain quality control for the laboratory. The chief technician verified all results and information before the report was received by Yukon Zinc.

ANALYTICAL CHECK LABORATORY

The analytical accuracy of the ALS Chemex laboratory was checked against another laboratory's results. From five drillholes, 105 samples were sent to ACME laboratories and treated with the same suite of preparations and analyses used at ALS Chemex. Pulp splits were shipped from the North Vancouver, BC lab to ACME in Vancouver, BC. The analytical techniques at ACME were chosen from a QA/QC procedural report (Cook, 2005) written for Yukon Zinc.

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. Any outliers present were identified by the magnitude of the sample pair variances with R^2 values presented in Table 12.2. Correlation between sample pairs for copper, lead, zinc, and silver is high, with outliers representing a 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, there were an acceptable number of duplicate drillholes submitted and the duplicates demonstrate an acceptable level of analytical accuracy between ALS Chemex and ACME.

Table 12.2 R^2 Values for ALS Chemex vs. ACME Comparison for Metal Values

Element	Drillhole				
	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	1.0000	0.9981	0.9958	0.9988	0.9994
Ag	0.9985	0.9861	0.9984	0.9737	0.9974
Au	0.8844	0.7764	0.9910	0.9859	0.9760

12.1.6 COMPUTER DATABASE

Drillhole data were stored in a logging database program that collects all drillhole location information, geology, downhole surveys, and geochemistry. This program can export information to a spreadsheet or database program, if necessary, for modelling or other applications. Data validation occurred weekly in the field and a thorough data validation occurred at the end of the 2005 field season.

Geochemistry results from the laboratory were emailed to the Vice-President of Exploration and passed to the project geologist for direct assimilation into the database program.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

The metallurgical test work used in the design of the process facilities of the Wolverine Mine was reviewed in the Feasibility Study and Independent Technical Report on the Wolverine Project – Finlayson District, Yukon issued by Wardrop in 2007. From that time, the process design was frozen and the plant was subsequently erected in 2009. During this period, no further test work was conducted by Yukon Zinc except for one investigation and report conducted by the School of Chemistry and Chemical Engineering, Central South University, Yuelu Mountain, Changsha, Hunan, P.R. China. This report was titled “Flotation Testing of Lynx Composite Samples, Yukon, Canada” (The Central South University Report) dated May 2010, and was made available to Tetra Tech by Yukon Zinc on November 10, 2011. In essence, this is an excellent report providing information, which was previously not available for consideration during the design phase of the process plant.

A brief review of the contents of this report follows in Section 13.2. The review incorporates a description of the sample used for the test work, the head grade assay values recorded, the results obtained from the mineralogical investigation, and the results obtained from sequential flotation tests and from bulk copper/lead flotation test work. Discussions regarding the results obtained, and particularly a comparison of the two flotation methods, and subsequent recommendations, are also presented.

13.2 REPORT REVIEW

13.2.1 *SAMPLE MATERIAL*

The material tested by The Central South University originated from the Lynx deposit at the Wolverine Mine. The University received four samples in April 2010. These samples were prepared for the test program by combining them into a single composite sample.

13.2.2 *COMPOSITE SAMPLE HEAD GRADE*

The individual samples showed significant variations in head grade values:

- Copper grades ranged from 0.42 to 3.17% Cu.
- Lead grades varied between 0.11 and 1.81% Pb.

- Zinc assays varied from 0.87 to 15.68% Zn.
- The head value of the composite sample was 1.98% Cu, 1.41% Pb, and 12.00% Zn. These head grade values were confirmed by x-ray fluorescence analysis.
- The gold and silver assay values of the composite sample were recorded as 1.45 g/t Au and 290 g/t Ag.

The following impurity elements were also recorded:

- antimony, 0.16% Sb
- arsenic, 0.11% As
- cadmium, 0.13% Cd
- carbon, 0.15% C
- cobalt, 0.007% Co
- iron, 25.08% Fe
- nickel, 0.005% Ni
- selenium, 0.126% Se
- sulphur, 31.45% S.

Among these impurity elements, antimony, arsenic, cadmium and selenium are noteworthy and could possibly complicate the recovery. The sample was not analysed for mercury content.

13.2.3 MINERALOGICAL EVALUATION

The following sulphide minerals were identified in the sample and are listed in order of abundance in Table 13.1. The general range of grain sizes observed during the investigation of the minerals is also shown.

Table 13.1 Sulphide Minerals Identified and Typical Particle Size

Sulphide Mineral	Particle Size Range (µm)
Pyrite	40 to 2,000
Sphalerite	30 to 600
Chalcopyrite	20 to 400
Galena	15 to 100
Boulangerite	10 to 60
Tetrahedrite	20 to 100
Pyrrhotite	-

table continues...

Sulphide Mineral	Particle Size Range (µm)
Clausthalite	5 to 100
Arsenopyrite	-
Gold/electrum	about 15

During the quantification of the sulphide minerals present in the sample, surprisingly a lead oxide content of about 20% of the total lead was recorded to be present with the balance generally being lead sulphide. However, neither the identity of this lead oxide phase, nor the mineral itself, or its host, was discussed in the report. Similarly, the sample was also reported to contain about 7% zinc oxide of the total zinc present. Again, no further details were recorded, except that the mineral franklinite, a zinc-spinel, was identified as a constituent in the gangue. This mineral would explain the presence of the zinc oxide.

An analytical review of the method of analysis for lead and zinc would be required to ascertain whether the lead present in the “lead oxide” phase reports as part of the total lead analysis. If this is the case, then the attainable recovery of lead cannot be as high as was originally estimated since the lead oxide-containing phase may have different flotation characteristics and may not necessarily report to the lead sulphide concentrate. Similar considerations apply for the zinc oxide phase.

The sphalerite has been found to contain up to 10% Fe and 1% Cd within its crystal structure, and has a maximum zinc content of about 55% Zn.

The clausthalite has been reported to be closely associated with the lead oxide phase, and is itself reported as consisting of two phases, one being a silver/antimony-rich phase. However, given the economic importance of the silver component of this mineral, further test work will be required to characterise the behaviour of this mineral in the flotation process.

The main silver-bearing minerals were determined to be clausthalite and tetrahedrite, with minor occurrences of the silver-bearing gold mineral electrum.

The gangue minerals were determined to consist mainly of chlorite, quartz, mica, hornblende, feldspar and calcite.

The mineralogical study concluded that in order to liberate 90% of the sphalerite, a grain size of 37 µm was required, while for chalcopyrite, the equivalent size was 26 µm.

13.2.4 SEQUENTIAL FLOTATION TEST WORK

The original process design for the Wolverine Mine was based on the Locked Cycle Test F13 (PRA Test F13) conducted by PRA. This procedure was a sequential flotation process initially producing a copper concentrate, followed by a lead

concentrate, and finally a zinc concentrate. This sequential flotation procedure was also followed by the Central South University in the test program, together with subsequent tests using the bulk flotation procedure. Generally, the preparation and reagents used in the test work program also followed the original process design parameters. Typical results are shown in Table 13.2 for a locked-cycle test conducted as a sequential flotation test, and these results are compared in the table with the results obtained from the equivalent bulk flotation locked-cycle tests. The design concentrate grades and recoveries for each component of the flotation circuit are also given in Table 13.2.

The sequential flotation test sample had initially been ground to a primary P₈₀ grind size of 62 µm, and the regrind P₈₀ size for each metal was 19 µm.

13.2.5 BULK FLOTATION TEST WORK

Test work was also conducted to determine whether a bulk copper/lead concentrate could be produced initially, followed by the separation and upgrading of the copper from the lead minerals, with the subsequent flotation of the zinc mineral into a saleable concentrate. Typical results of tests using comparable reagents and grind conditions are also presented in Table 13.2 for a locked-cycle test conducted as a bulk flotation test for comparison with the results obtained from the sequential flotation procedure.

In this case, the primary P₈₀ grind size was 38 µm, with a regrind particle size P₈₀ value of 19 µm for the copper and lead circuits, and 21 µm for the zinc circuit.

A number of grind and regrind sizes were tested, while different flotation reagents and dosage rates were also tested during the flotation test work program. However, the optimal results were obtained from the grind sizes as recorded.

Table 13.2 Flotation Test Results per Flotation Circuit

Copper Circuit	Copper Grade (%)	Copper Recovery (%)	Gold Recovery (%)	Silver Recovery (%)
Design	21.3	79.9	33.0/33.4	52.6/60.0
Sequential Flotation	24.5	80.8	53.6	64.4
Bulk Flotation	24.6	90.3	56.5	70.5

Lead Circuit	Lead Grade (%)	Lead Recovery (%)	Gold Recovery (%)	Silver Recovery (%)
Design	21.3/22.3	46.0	23.2	12.9/14.9
Sequential Flotation	25.6	55.7	13.5	15.2
Bulk Flotation	48.1	42.8	7.5	11.5
Bulk Flotation Middlings	16.1	24.2	7.0	1.0

Zinc Circuit	Zinc Grade (%)	Zinc Recovery (%)	Gold Recovery (%)	Silver Recovery (%)
Design	54.2	89.4/89.9	10.9	10.2
Sequential Flotation	50.8	89.5	9.5	11.4
Bulk Flotation	50.7	91.4	9.2	9.5

As can be seen from the results presented in Table 13.2, it is apparent that both flotation procedures have resulted in copper concentrate grades and recoveries exceeding the original design parameters for grade and recovery values as originally established by the PRA Test F13 test. The gold and silver recovered into the copper concentrate is also better than that of the original results obtained by the PRA test work. The results for the lead flotation circuit are generally better than the original PRA results, especially if the lead middlings fraction is incorporated into the main lead concentrate product. The results for both flotation methods for the zinc circuit are comparable, although both sets of zinc circuit test results are slightly inferior when compared with that which was obtained from the original PRA test results.

A comparison of the sequential flotation method with the bulk flotation method indicates that the copper circuit results for the bulk flotation procedure are generally better, and also for gold and silver recovery. The lead flotation results are approximately comparable if the middlings fraction is combined with the main lead concentrate. The zinc circuit results are also very similar for the zinc concentrate grade and recovery, as well as the gold and silver recoveries.

A further comparison of the two flotation procedures regarding the impurity content of the respective concentrates indicates similar results being attained. The bulk flotation procedure yielded lower lead and zinc contents in the copper concentrate than the sequential method.

Regarding the lead concentrate, the bulk flotation procedure yielded lower copper content, but a higher zinc content. For the zinc concentrate, the bulk flotation procedure yielded a lower copper and a lower lead content.

13.2.6 CONCENTRATE ANALYSIS

The Central South University Report also presented a detailed elemental analysis of the three concentrate products obtained from the bulk flotation procedure. These

results were discussed in the report in relation to the quality of the concentrates obtained by the test work and compared these results with the concentrate quality standards used in China to categorize these products. In China, the quality grade standard of a concentrate is determined by the grade of the main metal, together with the impurity content associated with that particular concentrate product. Based on the results obtained, the respective concentrates have been categorized within the Chinese concentrate grading system as follows:

- copper concentrate – 7th grade
- lead concentrate – 6th grade
- zinc concentrate – 6th grade.

13.3 DISCUSSION OF REPORT

The Central South University report was very comprehensive and detailed regarding the test work activities that were investigated. The conclusions derived from the test work are both very interesting and useful in confirming the previous test results and design parameters as based on PRA Test F13. The results are also useful for further process development considerations and for potential plant modifications. Some specific comments regarding the test work conducted are summarized below:

The geological and geographic location of the samples were not provided in the report; and it is therefore unclear how the Lynx sample which was tested fits into the LOM plan. A detailed test program similar to this investigation, using Wolverine and Saddle deposit sample material, should also have been undertaken in order to obtain a complete metallurgical understanding of the differences which may exist between the various ore types at the Wolverine Mine.

A part of each of the four samples used to prepare the composite sample, should have been retained to conduct variability flotation test work, once the basic process flowsheet had been confirmed. This is particularly important since it would result in a better understanding of the control of the process given the significant variations noted in the head grades for copper, lead and zinc.

No grinding tests or abrasion tests were conducted.

The test work program included the use of the reagent potassium chromate. Although potassium chromate is a very effective depressant reagent, it is very unlikely that the environmental permitting process will allow the use of this reagent in any commercial application.

The test program conducted by the Central South University did not incorporate a pre-flotation stage. The reasons for the omission of this process stage in the test program has not been recorded, despite the inclusion of the pre-flotation circuit in the plant design. It is of interest that the report did not allude to the presence of

carbonaceous material in the sample, and it is possible that the Lynx deposit contains lower, or less significant, quantities of carbonaceous material.

Although very good metallurgical results were obtained in using the bulk flotation concept, these results should be regarded as confirmation that both flotation procedures could be utilized and will likely provide generally similar results. However, unless additional evidence is obtained that supports the supposition that the bulk flotation procedure gives significantly improved results over the present sequential flotation design method, Tetra Tech recommends that the plant configuration remain unchanged from the present design, which uses the sequential flotation process.

13.4 RECOMMENDATIONS FOR FUTURE METALLURGICAL TEST WORK

It is recommended that in any future program of metallurgical testing, variability flotation test work be undertaken to investigate the differing responses of Lynx, Wolverine and Saddle zone ore.

Future test programs must also incorporate a pre-flotation stage so that a full characterization of the material can be established, even if the effect does not ultimately or materially impact the results obtained.

Any future flotation test program should incorporate carbon assays and product distributions.

Any future test program should include the establishment of the distribution for silver and gold, together with the distributions for copper, lead, and zinc for a full characterization of the metals in all the various products.

14.0 MINERAL RESOURCE ESTIMATES

14.1 DRILLHOLE DATABASE

Tetra Tech received drillhole data from Yukon Zinc, which included geological logs, assay, and density results from 203 holes.

A total of 5,954 assay and 2,133 density results were imported into the drillhole database.

A summary of all assay composite statistics is presented in Table 14.1, Table 14.2 and Table 14.3.

Table 14.1 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
Standard Deviation	1.88	388.1	7.22	1.43	2.00
Minimum	0.003	0.5	0.01	0.01	0.001
Maximum	16.63	5067.0	50.90	9.26	23.10
Coefficient of Variation	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
Standard Deviation	0.327	62.68	1.40	0.19	0.50
Minimum	0.001	0.10	0.01	0.01	0.001
Maximum	6.86	1999.0	34.50	6.53	17.30
Coefficient of Variation	4.36	5.32	4.22	4.56	5.46

Table 14.2 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	340.0	11.93	1.53	1.11
Standard Deviation	1.65	310.6	5.56	1.18	1.24
Minimum	0.024	2.43	0.04	0.01	0.001
Maximum	10.28	2177.0	32.95	7.01	8.83
Coefficient of Variation	1.00	0.91	0.47	0.77	1.12
Waste					
Number	3568	3568	3568	3568	3568
Mean	0.052	7.67	0.20	0.03	0.048
Standard Deviation	0.164	33.35	0.57	0.07	0.177
Minimum	0.001	0.10	0.01	0.01	0.001
Maximum	3.10	621.9	6.45	1.21	2.49
Coefficient of Variation	3.16	4.34	2.88	2.45	3.70

Table 14.3 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
Standard Deviation	1.34	267.0	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
Coefficient of Variation	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
Standard Deviation	1.48	318.7	5.36	1.18	1.45
Minimum	0.03	5.0	0.04	0.02	0.004
Maximum	8.53	2172.0	25.10	7.01	7.73
Coefficient of Variation	1.11	1.17	0.55	0.95	1.10
Wolverine Zone					
Number	140	140	140	140	140
Mean	2.25	411.8	12.71	1.66	0.88
Standard Deviation	2.02	343.6	5.00	1.23	0.88

table continues...

	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	Cu (%)
Minimum	0.07	16.7	2.22	0.08	0.005
Maximum	10.28	1973.8	25.94	6.81	5.37
Coefficient of Variation	0.90	0.83	0.39	0.74	0.99

14.2 GEOLOGICAL INTERPRETATION

The Wolverine deposit was modelled by Tetra Tech using surfaces to represent the hanging wall and footwall contacts of the sampled massive sulphide zones. The model honoured the drillhole data in 3D.

14.2.1 ASSAYS

The modelled wireframe contains a total of 1,141 samples collected from 178 drillholes. Descriptive statistics are presented in Table 14.4.

Table 14.4 Descriptive Statistics on Raw Assay Drillhole Data within Mineralized Zones

	Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	Length (m)
Valid Cases	1,141	1,141	1,141	1,141	1,141	1,141
Mean	353.42	1.71	1.34	1.60	12.18	0.93
Standard Error of Mean	10.72	0.06	0.06	0.04	0.21	0.02
Variance	131,233.36	3.63	3.80	2.03	48.40	0.27
Standard Deviation	362.26	1.90	1.95	1.43	6.96	0.52
Variation Coefficient	1.03	1.12	1.46	0.89	0.57	0.56
Rel. V. Coefficient (%)	3.03	3.31	4.31	2.64	1.69	1.66
Skew	3.92	2.68	4.33	1.40	0.45	1.53
Kurtosis	31.93	10.06	26.93	3.16	0.75	5.07
Minimum	0	0	0	0	0.01	0.06
1st Percentile	2.5	0.01	0.0142	0.01	0.0884	0.1
5th Percentile	21	0.08	0.08	0.05	0.875	0.2
10th Percentile	44.08	0.17	0.16	0.12	2.466	0.3
25th Percentile	143.8	0.56	0.36	0.435	7.51	0.6
Median	256.2	1.13	0.77	1.3	12.1	1
75th Percentile	453	1.94	1.5	2.395	16.625	1.1
90th Percentile	735	4.11	2.776	3.448	20.9	1.5
95th Percentile	970	5.386	4.636	4.248	24.1	1.9
99th Percentile	1,705.52	9.1716	10.3926	6.2916	29.89	2.9406
Maximum	5067	16.63	23.1	10.84	50.9	4.4

14.2.2 COMPOSITES

In order to normalize the assay data, raw assay values are often composited to a standard length. Tetra Tech composited the raw assay values to a 2 m length starting at the drillhole collars, honouring the modelled geological boundaries. Any composite less than 1 m in length was added to the previous composite.

Intersections less than 1 m but greater than 0.5 m were included, with the exception of two standalone composites less than 0.5 m that were deleted, thereby creating a dataset of composites ranging from 0.5 m to 3 m in length. Descriptive statistics of the composites are presented in Table 14.5.

Table 14.5 Descriptive Statistics on 2 m Composited Drillhole Data within Mineralized Zones

	Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	Length (m)
Valid Cases	547	547	547	547	547	547
Mean	353.63	1.72	1.26	1.67	12.47	1.94
Standard Error of Mean	12.95	0.07	0.06	0.05	0.24	0.02
Variance	91796.07	2.88	2.08	1.44	30.22	0.13
Standard Deviation	302.98	1.70	1.44	1.20	5.50	0.36
Variation Coefficient	0.86	0.98	1.14	0.72	0.44	0.18
Rel. V. Coefficient(%)	3.66	4.21	4.87	3.08	1.88	0.79
Skew	2.46	2.23	3.36	1.10	0.21	-0.84
Kurtosis	9.33	6.03	16.51	1.90	0.57	3.32
Minimum	1.60	0.00	0.00	0.00	0.02	0.49
1st Percentile	10.38	0.05	0.02	0.01	0.12	0.72
5th Percentile	31.82	0.10	0.13	0.13	3.06	1.17
10th Percentile	80.24	0.34	0.24	0.27	5.13	1.44
25th Percentile	173.03	0.73	0.43	0.77	8.99	2.00
Median	273.08	1.20	0.84	1.47	12.71	2.00
75th Percentile	440.47	2.00	1.44	2.36	15.93	2.00
90th Percentile	710.74	4.08	2.79	3.22	18.72	2.20
95th Percentile	922.12	5.57	3.96	3.86	21.25	2.51
99th Percentile	1595.81	8.64	7.71	5.58	27.21	2.91
Maximum	2217.68	11.51	13.60	7.76	34.19	3.00

14.2.3 CAPPING

All assay and composite data were reviewed prior to estimating resources for possible erratic high values or outliers. The data were plotted on histograms and cumulative distribution curves as well as being subjected to a decile analysis to identify possible outliers. Tetra Tech concluded that capping was not warranted for the Wolverine dataset.

14.3 RESOURCE BLOCK MODEL

The geological interpretation and resource model were carried out using 3D geological modelling software provided by Gemcom Software International Inc. (Gemcom) of Vancouver. Modelling was carried out in GEMS™, Version 6.2.3. The model is oriented so that model north is parallel to the strike of the Wolverine deposit.

The block model is comprised of blocks measuring 6 m length x 6 m width x 4 m thickness with an origin of:

- $x = 439817.06$, $y = 6810680.213$, $z = 1381$
- with 120 columns, 160 rows, and 115 levels.

The block model has been rotated 35° counter-clockwise around the origin to align the columns parallel to the strike of the Wolverine deposit. The model coordinates are in the UTM NAD83 coordinate system.

14.3.1 SPATIAL ANALYSIS OF GRADES

Geostatisticians use a variety of tools to describe the pattern of spatial continuity, or strength of the spatial similarity of a variable with separation distance and direction. The semi-variogram measures the correlation between data values as a function of their separation distance and direction. The distance at which the semi-variogram reaches the maximum variance is called the "range of correlation" or simply the range. The range of the semi-variogram corresponds roughly to the more qualitative notion of the "range of influence" of a sample; it is the distance over which sample values show some persistence or correlation. The shape of the semi-variogram describes the pattern of spatial continuity. A very rapid decrease near the origin is indicative of short scale variability. A more gradual decrease moving away from the origin suggests longer scale continuity.

For the Wolverine deposit, experimental semi-variograms were generated at 16 orientations within the plane of mineralization. A 2D map of the semi-variogram values for the various distance and directions were plotted outward from a common centre. The direction of maximum continuity was identified within the plane of mineralization and toward the north. Experimental semi-variograms were then computed at the two mutually perpendicular directions (semi-major and minor axis).

A semi-variogram model was fitted to the three experimental semi-variograms directions. Individual semi-variogram models are summarized in Table 14.6 for zinc, silver, copper, gold, and lead; all were modelled with a nugget and one spherical structure except for zinc, which was modelled with two nested spherical structures. Rotation angles are set to correspond to Gemcom's azimuth-dip-azimuth rotational convention.

Table 14.6 Semi-variogram Models

	Domain	Model	Primary Azimuth	Primary Dip	Secondary Azimuth	X Range	Y Range	Z Range
Zn	Main Zone	$C_0=6.82635$						
		$C_1=9.54346$	353.74	-18.61	94.26	21.6	9.5	4.7
		$C_2=13.14849$	353.74	-18.61	94.26	82.5	36.4	17.8
Ag	Main Zone	$C_0=5460.45$						
		$C_1=45634.82$	353.62	-18.54	94.12	79.9	28.5	4.6
Cu	Main Zone	$C_0=0.24383$						
		$C_1=1.81426$	353.78	-18.63	94.3	129.7	20.9	10
Au	Main Zone	$C_0=0.24176$						
		$C_1=1.46135$	354.43	-18.99	95.06	78.9	27.9	4.5
Pb	Main Zone	$C_0=0.36792$						
		$C_1=0.82662$	353.9	-18.7	94.45	54.8	19.6	10.1

14.3.2 INTERPOLATION PLAN

Grades were interpolated within the block model using the ordinary kriging (OK) interpolation method with weighting parameters based on the semi-variogram data presented in Table 14.6.

Grades were interpolated using the 2 m composite values. The grades were interpolated in four separate passes. Sample selection was based on search ellipse scaled to a proportion of the ranges from the semi-variogram analysis for each grade element: Pass 1 at two-thirds of the variogram range, Pass 2 at the variogram range, Pass 3 at twice the range, and Pass 4 at three times the range.

Grades were only interpolated in the first three passes if at least three samples were found within the search ellipse, and a maximum of sixteen samples were used to interpolate any block. The fourth pass required a minimum of one sample and a maximum of sixteen samples. Sample selection for grade interpolation was also restricted by domain; thus, any main zone block was interpolated using only samples found within the main zone, and any footwall zone block was interpolated using only samples found within the footwall zone.

14.3.3 MINERAL RESOURCE CLASSIFICATION

Mineral resources were classified in accordance with definitions provided by the CIM as stipulated in NI 43-101. The Wolverine mineral resources are classified by Tetra Tech as Indicated and Inferred.

Blocks were classified as Indicated Mineral Resource if at least three composites from two holes were found within a search ellipse scaled to the modelled semi-variogram ranges for zinc.

All other blocks within the wireframe were classified as Inferred Mineral Resource if at least one composite was found within a search ellipse scaled to twice the modelled variogram range for zinc.

The classification model was visually examined for reasonableness. The classification was modified to eliminate small pockets of Indicated resources found within Inferred zones and pockets of Inferred Resources found within Indicated zones.

14.3.4 MINERAL RESOURCE TABULATION

Tetra Tech estimated that the Wolverine deposit contains 4.140 Mt of Indicated Resources averaging 12.91% Zn, 374 g/t Ag, 1.3% Cu, 1.83 g/t Au, and 1.7% Pb, at a US\$150 RMV cut-off (Equation 14.1). The recoverable metal value cut-off refers to the minimum value of recoverable metal in a tonne of ore at which the ore is expected to be economically feasible to mine. In addition, Tetra Tech also estimated that the Wolverine deposit contains 1.635 Mt of Inferred Resources averaging 11.8% Zn, 368 g/t Ag, 1.2% Cu, 1.56 g/t Au, and 1.6% Pb. The mineral resources are reported in Table 14.7.

Table 14.7 Mineral Resources at US\$150 RMV Cut-off

Classification	Cut-off (US\$)	Tonnes ('000 t)	Volume (m ³)	Zn (%)	Ag (g/t)	Cu (%)	Au (g/t)	Pb (%)
Indicated	150	4.140	1,045	12.9	374	1.3	1.83	1.7
Inferred	150	1.635	413	11.8	368	1.2	1.57	1.6
Contained Metals				Zn ('000 t)	Ag ('000 g)	Cu ('000 t)	Au ('000 g)	Pb ('000 t)
Indicated	-	-	-	534	1,548,360	54	7,576	70
Inferred	-	-	-	193	601,680	20	2,567	26

The RMV formula, presented in Equation 14.1, was based on metal prices and recoveries as outlined in Table 14.8.

Equation 14.1 Recoverable Metal Value

$$RMV = Zn\% * 15.5007 + Ag \text{ g/t} * 0.2155 + Cu\% * 35.2740 + Au \text{ g/t} * 12.6620 + Pb\% * 6.0848$$

Table 14.8 RMV Calculation Factors

Metal	Metal Price (US\$)	Recovery (%)
Zn	0.79/lb	89
Ag	11.17/oz	60
Cu	2.00/lb	80
Au	703.27/oz	56
Pb	0.60/lb	46

Metals prices used in block model and ore resource generation are generally set at a level that will ensure good selectivity of the block model. Metals recoveries were based on the overall metal recoveries as indicated in the prior metallurgical test work. These prices and recoveries are considered conservative and are not always the same prices and recoveries used in the eventual economic analysis of the Project. Current metal prices are significantly higher than the prices used in the 2010 block model, which were selected on the basis of long-term metal prices considered appropriate at the time of the modelling.

14.3.5 BLOCK MODEL VALIDATION

The block model was validated by examining the block model results in order to compare with the drillhole composite and raw assay data on sections. The validation of the block model grades by section agreed well with the drillhole data.

Tetra Tech interpolated grade using the inverse distance and nearest neighbour methods, which produced similar results at a global scale.

Tetra Tech also compared the 2010 estimation with the estimate prepared in 2006 and both estimates were similar; however, the 2010 estimate was smaller due to the consideration of the north-eastern property boundary.

15.0 MINERAL RESERVE ESTIMATES

15.1 STATEMENT OF MINEABLE ORE RESERVES

In 2011, Tetra Tech prepared the ore reserve estimate using the Mine2-4D suite of software for design and block model interrogation. Table 15.1 shows the Probable Ore Reserve Estimate. Table 15.2 shows the contained metal.

Table 15.1 Ore Reserve Estimate

Volume (m ³)	Tonnes	Category	Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)
1,003,062	3,751,452	Probable	327.29	1.60	1.17	1.47	11.55

Table 15.2 Contained Metal

Ag (kg)	Au (kg)	Cu (t)	Pb (t)	Zn (t)
1,227,812	6,002	43,892	55,146	433,293

15.2 CUT-OFF CRITERIA

Using the 2011 long-term forecast metal prices listed below, and the operating costs described in Section 21.0, Tetra Tech derived a cut-off grade of 12.24% zinc equivalent (ZnEq). Cut-off grade refers to the minimum grade of contained metal in a tonne of ore at which the ore is expected to be economically feasible to mine. Zinc equivalent is used to express the total value of the metal content of polymetallic ore in terms of zinc. This zinc equivalent grade was used to determine the minable ore reserves.

- Ag: \$19.00/oz
- Au: \$1139/oz
- Cu: \$2.87/lb
- Pb: \$0.98/lb
- Zn: \$1.04/lb

Zinc equivalent was calculated using the following formula:

$$ZnEq = \frac{[(ValueofSilverContained) + (ValueofGoldContained) + (ValueofCopperContained) + (ValueofLeadContained) + (ValueofZincContained)]}{(Zincpricepergram)}$$

Numerically, this is achieved through the following formula:

$$ZnEq = \frac{[(Ag(g/t) * 19 * 0.032) + (Au(g/t) * 1139 * 0.032) + (Cu\% * 22.04 * 2.87) + (Pb\% * 22.04 * 0.98) + (Zn\% * 1.04 * 22.04)]}{(1.04 * 22.04)}$$

15.3 STOPE DESIGN

Stope design was based on a minimum mining width of 3 m. The stope outlines were digitized every four vertical metres to represent the planned height of individual cuts. The design includes five cuts per level; therefore a single level accesses 20 m of vertical mineralization.

15.3.1 GEOMETRICAL RECOVERY

Geometrical recovery is defined as the percentage of mineralization that can be mined due to the constraints of the geometry of the deposit, and the mining method or equipment used. Figure 15.1 shows the recoverable area in a cross section typical of the Wolverine and Lynx deposits. Figure 15.2 shows the recoverable area in a cross-section of the Saddle deposit, the illustration highlights the portion that can be mined.

The geometrical recoveries calculated are 93% for the Wolverine and Lynx deposits, and 79% for the Saddle deposit.

Figure 15.1 Geometrical Recovery for the Wolverine and Lynx Deposits

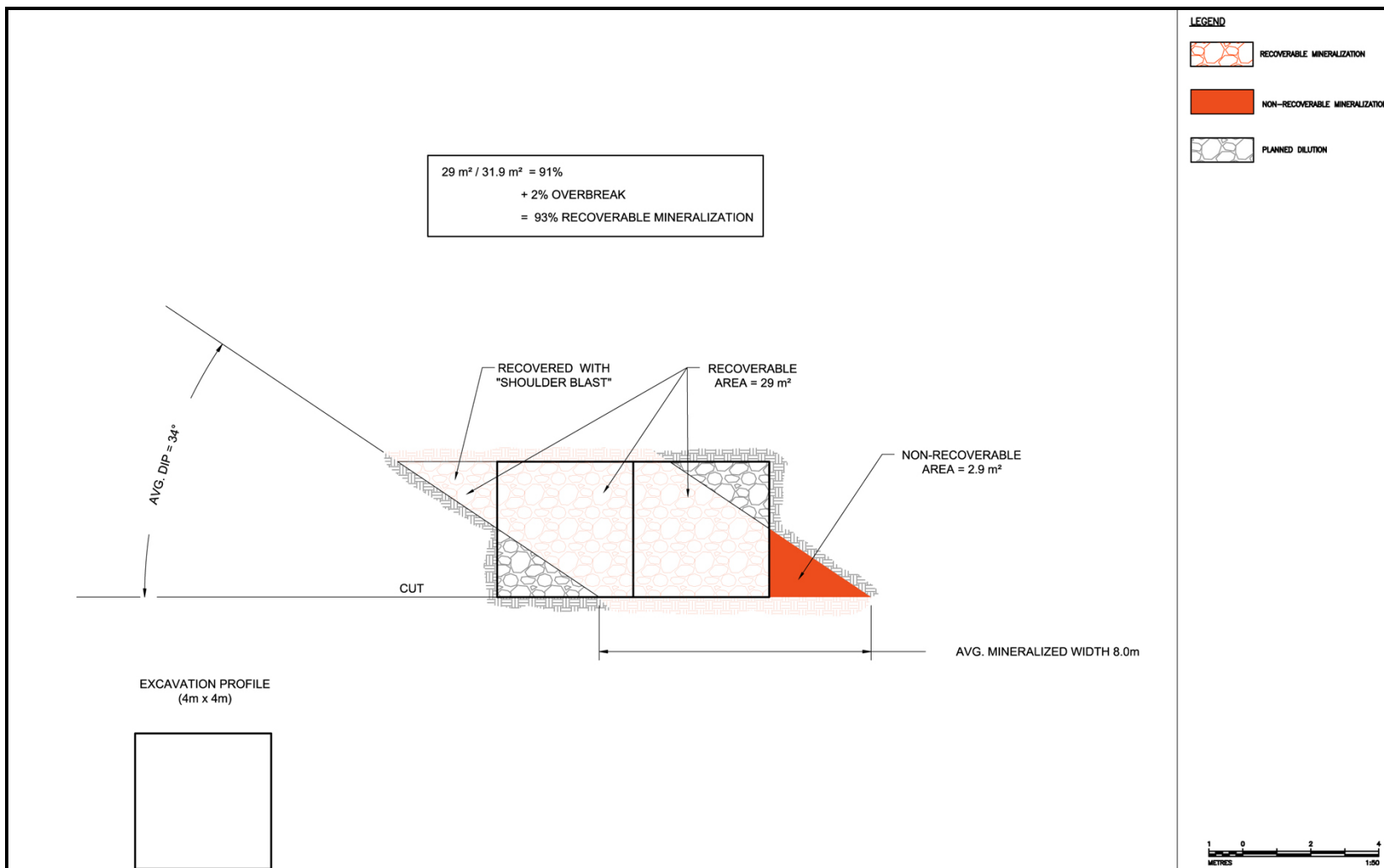
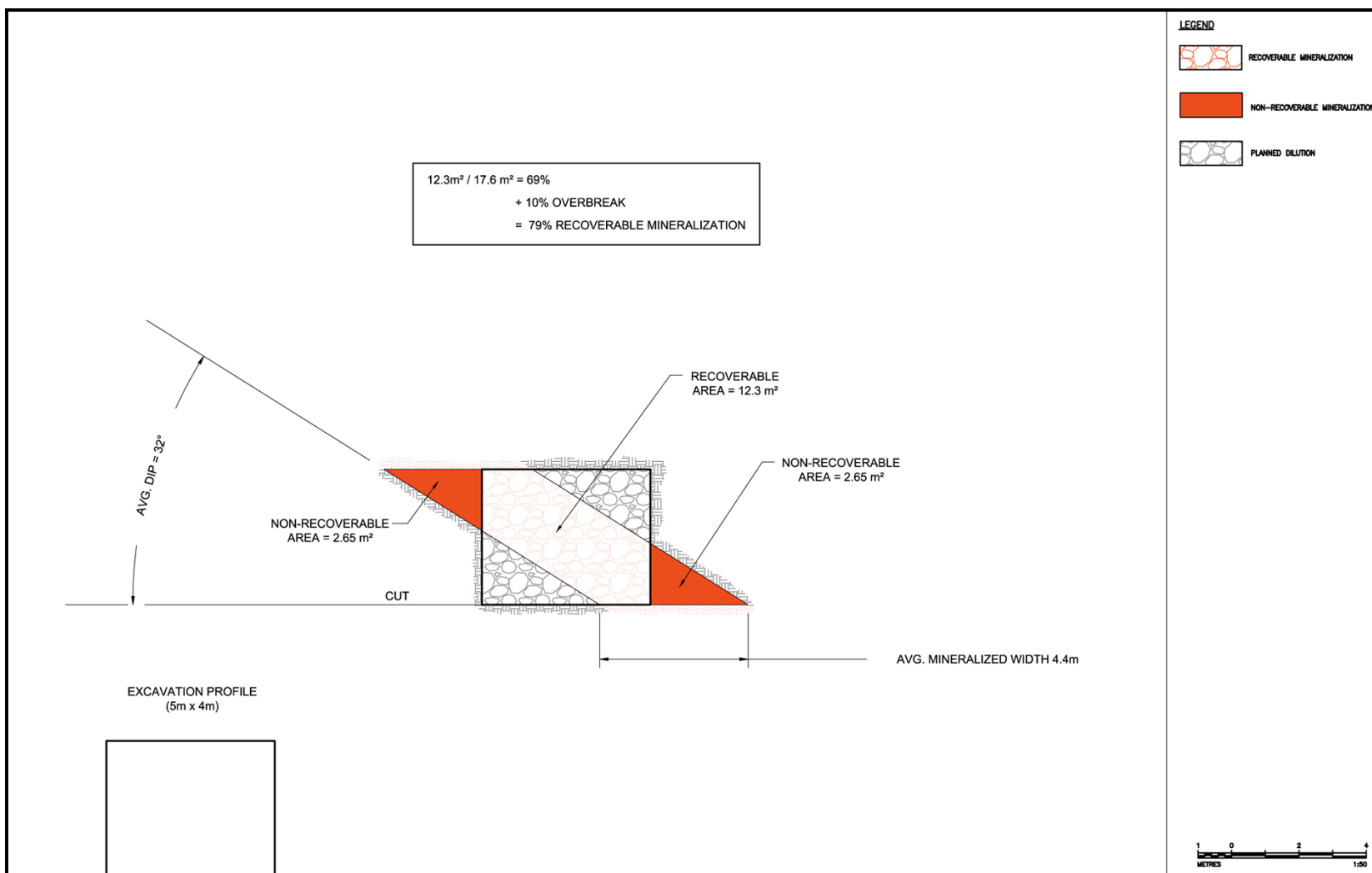


Figure 15.2 Geometrical Recovery for the Saddle Deposit



15.4 DILUTION

15.4.1 WOLVERINE AND LYNX DEPOSITS

Planned dilution was estimated by laying out typical 4 x 4 m drift outlines in cross-section, with an average orebody width of 8 m and a dip of 34°. The percentage of planned dilution taken during mining was calculated by measuring the cross-sectional area that is waste compared to the cross sectional area mined. The planned dilution used for the Wolverine and Lynx deposits was estimated at 17%. An additional 2% was added as unplanned dilution to account for overbreak during blasting operations.

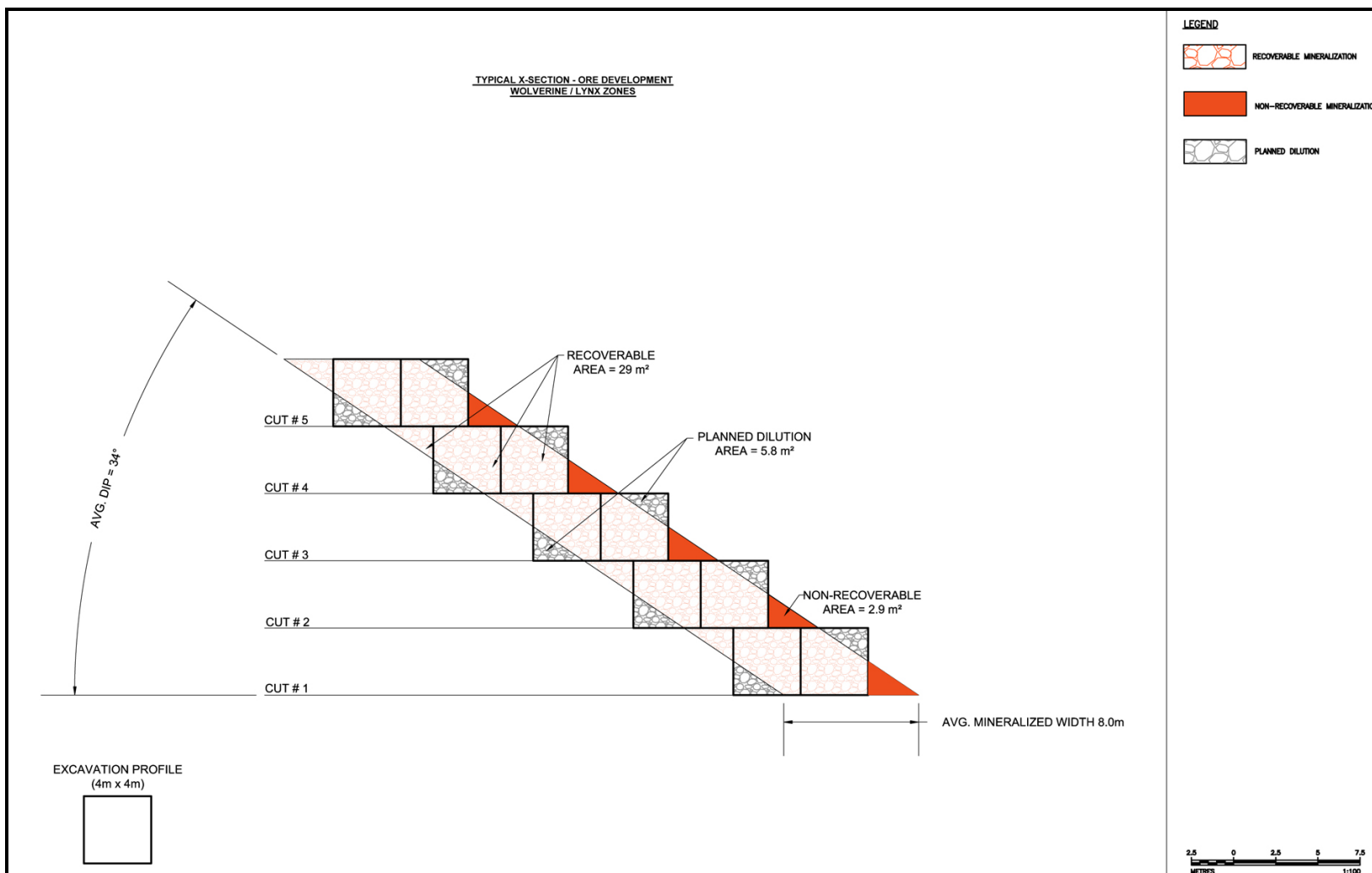
Figure 15.3 shows the typical cross section of the Wolverine and Lynx deposits used to determine the average planned dilution and geometrical recovery for the mining method.

Paste fill is another source of dilution as some paste from lower cuts will inadvertently be scraped up when the LHDs are cleaning the cut floor. Table 15.3 shows the factors used in calculating the amount of fill dilution from mucking paste off of a lower backfilled cut.

Table 15.3 Paste Fill Dilution Factors

Cut Number	Dilution (%)
Cut 5	6
Cut 4	4
Cut 3	4
Cut 2	4
Cut 1	0

Figure 15.3 Typical Cross Section of the Wolverine and Lynx Deposits



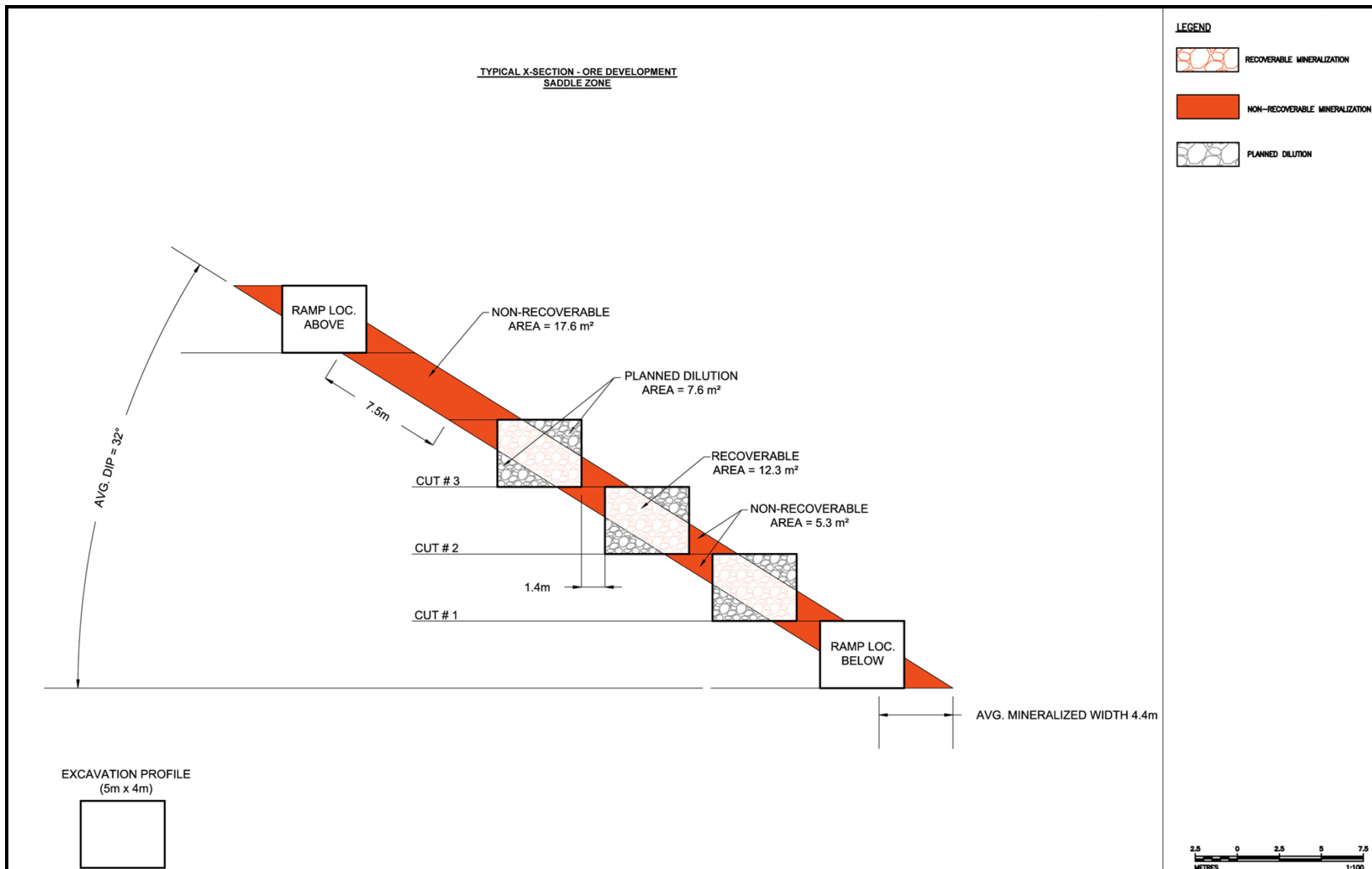
15.4.2 SADDLE DEPOSIT

Planned dilution was estimated by laying out a typical 5 x 4 m drift outline in cross section with an average orebody width of 4.4 m and dip angle of 32°. The percentage of planned dilution taken during mining was calculated by measuring the cross-sectional area that is waste, compared to the cross-sectional area mined. The planned dilution used for the Saddle deposit was estimated at 38%. An additional 2% was added as unplanned dilution to account for overbreak during blasting operations.

Figure 15.4 shows a typical cross section of the Saddle deposit used to determine the average planned dilution and geometrical recovery for the mining method.

No paste fill dilution was accounted for in the Saddle deposit because successive cuts are offset sufficiently that there is no mining conducted on top of fill.

Figure 15.4 Typical Cross Section of the Saddle Deposit



15.5 MINING RECOVERY

Mining recovery is a function of mineralized material left behind due to operational constraints typical to underground cut and fill processes, such as ragged floors and loss of fines in undercuts. An average mining recovery was determined for all three deposits and is described in the following sections.

15.5.1 *WOLVERINE AND LYNX DEPOSITS*

A typical cut-and-fill recovery of 90% was assigned to cuts two to five, to account for the following items:

- blasted mineralized material left behind on the floor on rock
- blasted mineralized material left behind on paste fill
- blasted mineralized material that is not able to be mucked.

For the first cut, a recovery of 92% was assigned because no material will be lost into backfill or void in a lower cut.

15.5.2 *SADDLE DEPOSIT*

A typical cut-and-fill recovery of 95% was assigned to all cuts due to:

- Minimal blasted mineralized material is left behind because the cut does not daylight to a lower cut but is instead a single unique heading.
- Minimal blasted mineralized material is being mucked off rock instead of paste fill.

Due to the main ramp going through the Saddle deposit, only three of the four potential cuts will be mined thereby allowing for a pillar around the ramp for access when the Saddle deposit is being mined. This results in only 75% of the remaining Saddle deposit being excavated throughout the current LOM mine plan.

15.6 RECONCILIATION OF IN SITU MATERIAL TO MINABLE RESERVE

Table 15.4 shows the in situ material that is contained within stope wireframes.

Table 15.4 In Situ Material Contained within Stope Wireframe Shapes

Deposit	Tonnes	Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)
Wolverine	1,784,867	389.09	2.12	1.15	1.43	11.66
Lynx	2,226,925	314.44	1.38	1.27	1.67	12.72
Saddle	164,245	427.84	1.90	1.86	1.68	11.14
Total	4,176,037	350.80	1.72	1.24	1.57	12.20

Wolverine mineralization already mined has been depleted from future reserves as shown in Table 15.5

Table 15.5 Material Mined from Stopes

Deposit	Total Tonnes Removed from Reserve
Wolverine	63,228
Lynx	114,043
Saddle	7,071
Total	184,342

Once the recovery and dilution factors have been applied to the in situ material and the Table 15.5 material has been depleted, the remaining material is considered to be reserve. Table 15.6 shows the minable reserve for the Wolverine Mine.

Table 15.6 Ore Reserves Estimate

Volume (m ³)	Tonnes	Category	Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)
1,003,062	3,751,452	Probable	327.29	1.60	1.17	1.47	11.55

16.0 MINING METHODS

16.1 GEOTECHNICAL EVALUATION

For geotechnical evaluations, Tetra Tech is relying on reports and work by other experts. This section provides a summary of the reports completed by the consultants used by Yukon Zinc as well as evaluations and extractions from the GCMP for underground mining, put in place by Yukon Zinc.

16.1.1 CONSULTANT REPORTS

An investigation and evaluation of the rock mass quality and ground support design was completed by a mining consultancy firm called Rockland Ltd. in 2010. Their report was used as a basis for the current comprehensive underground GCMP for the Wolverine Mine, describing the risk assessment process, roles and responsibilities, ground support design, communication plan, training, incident investigations, monitoring, mapping, auditing and document control policies. Malcolm Swallow, P.Eng. independent mine consultant, authorized the GCMP.

The objectives of the GCMP, which are currently implemented by Yukon Zinc, are to reduce the risk of uncontrolled ground failure; contribute to the development and maintenance of a safe working environment and contribute to efficient extraction of ore reserves. These objectives are being achieved through:

- identification of hazardous areas and assessment of associated risks
- design and implementation of appropriate ground control systems
- communicating known hazards to the workforce in advance of both development and production
- design and implementation of systems to detect and control change
- design and implementation of procedures associated with ground control including standard operating procedures for the installation of the ground support
- providing clear and unambiguous definitions of roles and responsibilities for individuals working under the plan
- external and internal auditing to assess the effectiveness and degree of compliance with the GCMP and assist in identifying improvement requirements (Shin 2011).

The GCMP is a living document that is implemented by Yukon Zinc and is subject to periodic update dependent on varying conditions underground. Yukon Zinc employs a full time ground control expert to implement and update the GCMP, as a fundamental part of the mining process.

16.1.2 *GROUND SUPPORT*

The support regimes identified at the Wolverine Mine address all aspects of the mine including the main ramp, stope access, ore drive, raises and intersections.

Support types are to be determined from recommendations in the GCMP Report, using geological information, contact orientation and Ground Strength Index (GSI).

The GCMP Report employs the empirical Q-support guideline (Grimstad and Barton 1993) method to establish minimum support requirements and then defines detailed ground support guidelines after examination of the model by use of Phase² software from Rocscience Inc. (Rocscience 2010).

The GCMP Report summarizes by stating that the rock mass quality for the ore is “fair”, while the footwall and hanging wall rock mass quality is “very poor”, with numerous shallow dipping clay filled faults.

Using the Q system the GCMP Report recommends the use of 3.7 m Super Swellex on 1 m x 1 m spacing in the ramp for both the back and side wall of the ramp. The report also recommends 2.4 m regular Swellex placed in a 1 m x 1 m grid for drifts in the ore. The stand-up time for drifts partially in ore with hanging wall contact is estimated at 12 h and drifts partially in ore with footwall contact are estimated at 24 h. This means that support must be put in place within 12 or 24 h of completion of blasting and before any other development work can take place.

Additionally, the unsupported stand-up time of the main access ramp at the contact to the footwall and hanging wall is assessed to be zero, because in these specific areas, the openings are located in extremely poor ground. A zero stand-up time means that the installation of pre-support is required before excavation can take place. For the main access ramp in the area of the footwall or hanging wall contacts, where such very weak ground conditions exist, pre-support is utilised, using rebar or hollow core bar in “forepoling”, a type of spiling method. This pre-support technique is very common for drifting/tunnelling through weak ground that has the potential to unravel.

16.1.3 *GEOMECHANICAL ASPECTS OF MINING*

Three different variants of the drift and fill mining methods are proposed for use at the Wolverine Mine. They include drift and fill mining with side slash; drift and fill with retreat panels; and drift and fill with primary and secondary panels. All three are geomechanically viable and are detailed in Section 16.3.

All mine workings will be tight backfilled to improve stability within the stope and reduce stress build-up in mine pillars. The backfill will be a combination of pastefill and development rock (Wardrop 2007).

16.1.4 *GEOTECHNICAL MONITORING*

Geotechnical assumptions are based on the results of geotechnical borehole data, numerical modelling, field observation during the mining process and engineering judgment.

Current monitoring results confirm that existing mining support is appropriate and within the expectations of the existing ground support standard. The location of instrumentation at the Wolverine Mine used to monitor ground response is discussed in detail in the GCMP Report. Any moves away from the current ground support standard will require assessment through monitoring to assess ground response.

Current monitoring includes:

- seventeen extensometers at the main ramp crossover
- nine tilt meters at the steel sets in the portal area to monitor ground movement related to slope sliding
- daily convergence measurements of stopes and ore drives to test brow stability
- visual shotcrete crack monitoring
- regular rock mechanic inspections and numerical modelling using Rocscience Phase², Unwedge and Dips software.

The GCMP report recommends a formalized monitoring recording schedule be documented and maintained that contains the location and type of monitoring to be carried out, whom is to perform the reading, time last read and the reading frequency (Shin 2011).

16.2 MINE DESIGN AND ACCESS

All of the lateral development work in the mine is by drill jumbos, mechanized rock bolters and LHD vehicles. This section will describe the types of excavations used to mine the Wolverine, Lynx, and Saddle deposits.

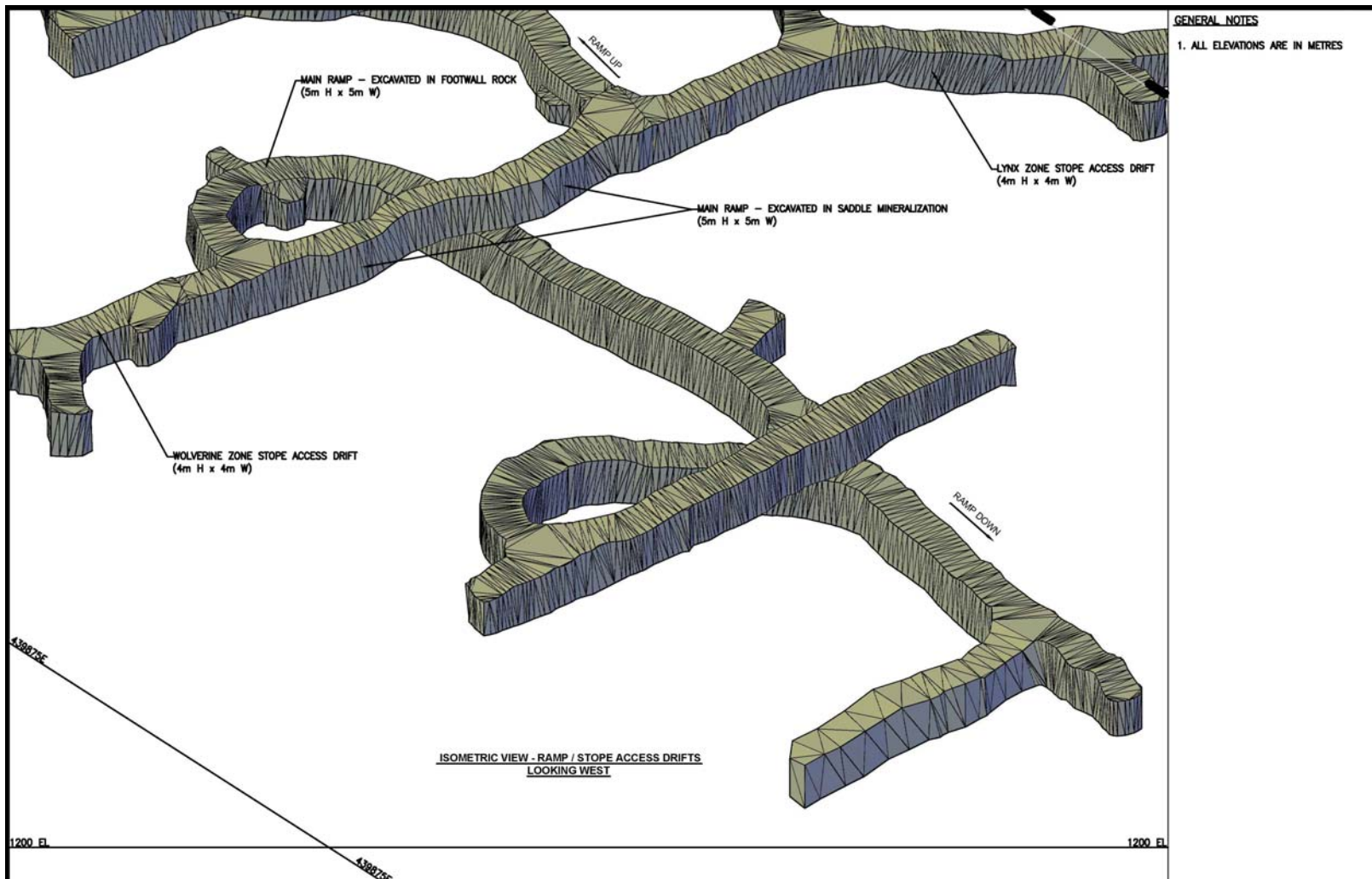
16.2.1 *MAIN ACCESS RAMP*

The mine is accessed by a single ramp from surface (elevation 1345) at an average gradient of 14%. Access dimensions average 5 m wide and 5 m high. The ramp is located in the Saddle deposit area which is central to both the Lynx and Wolverine

deposits. All personnel, material and equipment use this ramp for mine activity including access, material and supply handling.

The ramp is developed moving from the footwall, through the Saddle zone and into the hanging wall material, in a series of spiral repeats. The ramp enters the Saddle deposit every 20 vertical metres where a separate drift is excavated toward the Lynx deposit. The ramp is then excavated toward the Wolverine deposit to achieve an ore access 10 m below the previous Lynx deposit access. Once the access elevation is reached, the ramp is turned into the footwall to reach the next level below. Figure 16.1 shows the development in an isometric view of the ramp.

Figure 16.1 Isometric View of Ramp



16.2.2 *STOPE ACCESS DRIFTS*

Stope access drifts are started every ten vertical metres, alternating between the Lynx deposit and Wolverine deposit. Stope access drifts within individual deposits are spaced 20 vertical metres apart. Figure 16.1 shows the layout of the stope access via drifts, going into the Wolverine and Lynx deposits from the ramp.

The stope access drifts are driven 4 m wide, 4 m high and follow the footwall contact. The bottom stope access drift is driven at minus 15% grade to reach the bottom elevation for the first cut.

16.2.3 *DIAMOND DRILL DRIFTS*

Currently, there is no plan to develop drifts dedicated entirely to diamond drilling. Any definition diamond drilling is presumed to be done from the initial footwall drift in the stope, or from one of the “P” cross cuts as the stope develops.

16.2.4 *VENTILATION ACCESS DRIFTS*

Ventilation access drifts, or stubs, are driven from the main ramp every 20 vertical metres for fresh air raise connection to the stoping levels. The stubs are approximately 4 m wide x 4.4 m high. More detail on ventilation is provided in Section 16.5.

16.2.5 *RAISES*

Raises with a cross sectional area of approximately 10.5 m are constructed for fresh air and secondary egress from the mine. The raises are driven conventionally using hand held drills and mucked using slushers. The raises are sequenced in a leapfrog pattern to enable the fresh air to be carried in the direction of the ramp progression. More detail on ventilation raises is provided in Section 16.5.

16.2.6 *MISCELLANEOUS DEVELOPMENT*

- Remucks are excavated on every spiral of the ramp to help speed up the development mucking cycle. A maximum of 150 m separates the remucks which are typically driven 5 m wide, 4 m high and 15 m long.
- Safety bays are excavated every 30 m along the main access ramp. They are driven 1.5 m wide, 2 m high and 1.8 m long.
- Water collection sumps are located on every level.
- A main sump is planned for the bottom of the current ramp.
- There are storage areas for both detonators and explosives underground.

16.3 MINING METHODS

Overhand drift and fill mining is the method currently being used at Wolverine due to:

- the flat dipping nature of the deposits (average of 34°)
- the deposits being relatively narrow
- potentially poor ground conditions at the hanging wall contact
- varying width of the deposits
- selectivity required to achieve acceptable amounts of waste dilution.

In this method, access into the ore occurs from a stope access drift driven at 15% down, from the main ramp, in the Saddle deposit area. Horizontal cuts are then mined in ore 4 m high. Once a cut is mined out, it is filled with cemented pastefill, piped into the cut from the backfill plant on surface. After the cut is completely filled it is given time to cure sufficiently, so that the next cut above can be excavated. The roof of the stope access ramp is then slashed down to allow access to the lift above and the next 4 m slice, or cut, is then mined. The last cut of a level will be mined partially under pastefill from the mined out level above. Hence, it is very important that the backfill in the first cut of each level has the proper strength to be undercut by mining from below.

The initial drift follows the footwall contact. Behind the advancing face, a portable diamond drill is used to drill horizontal test holes towards the hanging wall so the width of the mineralization can be confirmed. If the deposit is wider than one drift width, that ore can be slashed and recovered remotely as mining retreats, or panels can be excavated until the hanging wall is reached. Due to the varying deposit thickness on any single cut, it is expected that a number of variations to the basic mining method will be used.

Guidelines for the mining method versus deposit thickness are provided in Table 16.1.

Table 16.1 Thickness Versus Mining Method

Horizontal Ore Thickness	Mining Method
<5 m	Drift and fill with side slash (DFSS)
5 to 8 m	Drift and fill with retreat panels (DFRP)
>8 m	Drift and fill with primary and secondary panels (DFPS)

The various drift and fill mining methods are further described in the following sections.

16.3.1 *DRIFT AND FILL WITH PRIMARY AND SECONDARY PANELS*

This mining method is used in areas that are more than 8 m wide on the level. Once the footwall drift is complete along the footwall contact, 5 m wide primary stope panels are excavated leaving temporary 5 m pillars in between. The panels are developed from the footwall drift at approximately 60° from the footwall azimuth and driven until the hanging wall contact is reached. Figure 16.2 shows the plan view of the drift and fill mining method.

The panels are extracted using a primary/secondary sequence where all of the primary panels are mined and tight filled before the secondary panels are excavated between the filled panels. The secondary panels are then filled as tightly as possible once mining is complete.

To avoid excessive dilution at the end of the panels, when the hanging wall contact is reached, shotcrete may need to be applied to prevent the hanging wall from sloughing. This is done as required, dependent on local ground conditions.

The panels are backfilled individually which can cause some logistical challenges due to the number of fill bulkheads required and the relatively small amounts of fill required for each pour.

16.3.2 *DRIFT AND FILL WITH RETREAT PANELS*

Where the deposit is less than 8 m wide (two drift widths), shorter panels are excavated from the footwall drift to the hanging wall contact after the footwall drift has been completely excavated. The panels are sequenced in a retreat fashion instead of primary/sequence fashion.

The panel furthest from the ramp access are mined first. Once this panel is mined out and filled, the adjacent panel is excavated – not leaving a pillar in between.

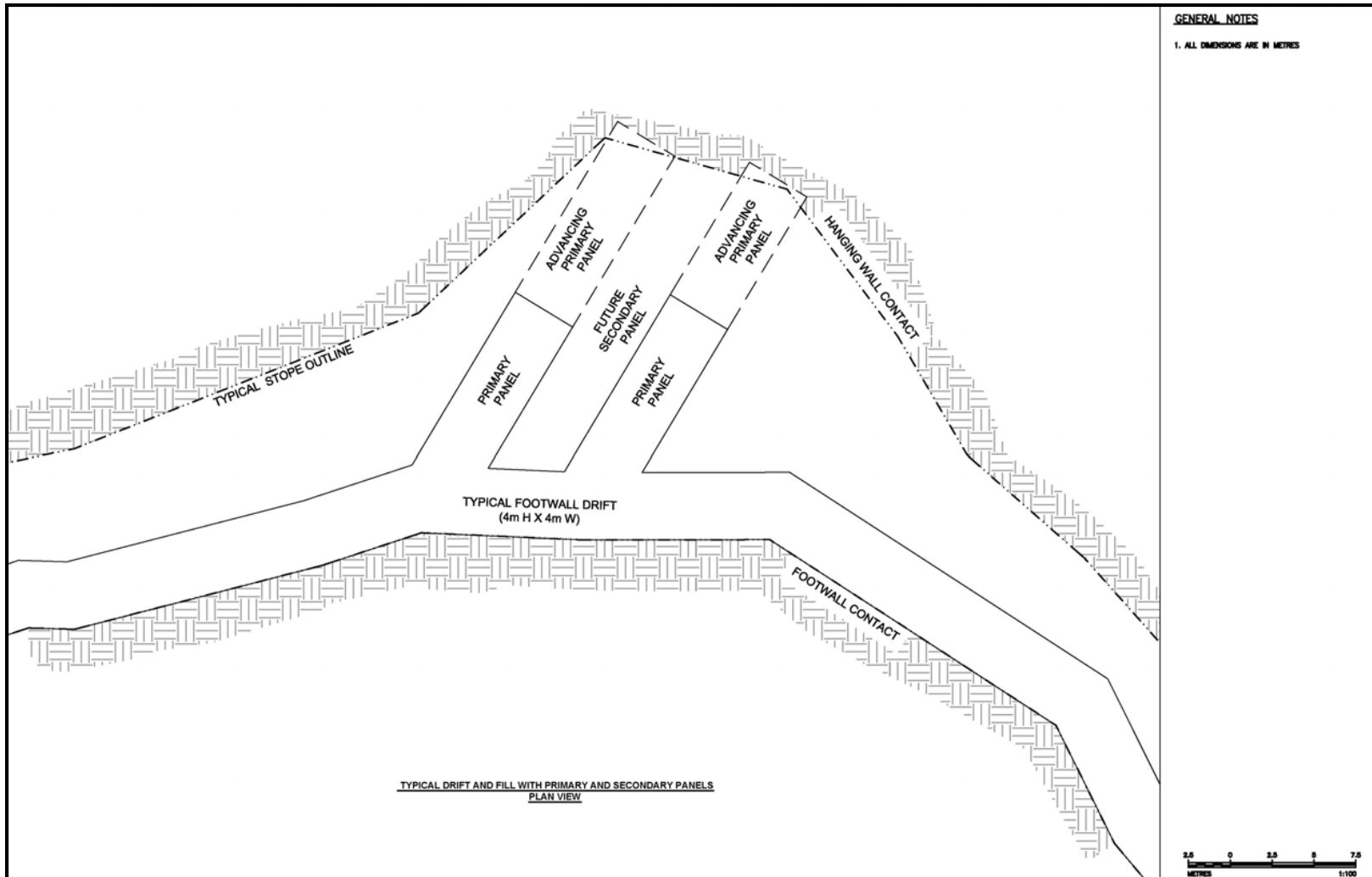
Using this retreat method allows for the effective span to be managed easier.

16.3.3 *DRIFT AND FILL WITH SIDE SLASH*

In areas where the deposit is less than 5 m wide, the deposit is mined in two stages. The initial drift follows the footwall contact to the extent of the deposit and any mineralization left behind in the walls is slashed using the drill jumbo in a retreating manner towards the ramp access.

The hanging wall contact is not bolted in this method but may be shotcreted to prevent unraveling or other failures. Geotechnical considerations should be the major driving force when choosing the amount or length of slash to blast at one time.

Figure 16.2 Drift and Fill Plan View



16.3.4 NAMING CONVENTION

Stoping blocks are named by the elevation of the floor of the bottom cut of the stope. The stoping elevations are offset 10 m for both the Lynx deposit (e.g. 1300 Level, 1280 Level, etc.) and the Wolverine deposit (1290 Level, 1270 Level, etc.).

Cuts within a stoping block are named from the first bottom cut known as the sill cut to the fifth and final cut known as the top cut. Panels within a cut are numbered increasingly starting with the panel closest to the ramp access (P1, P2, P3, etc.).

16.3.5 BLOCK MINING SEQUENCE

For both the Lynx and Wolverine deposits, the initial footwall drift commences from the stope access and follow the footwall contact of the mineralization. While the footwall drift is being excavated, test holes drilled with either a mobile diamond drill or drill jumbo are used in defining the width of mineralization along the cut.

After the footwall drift has reached the end of the economic mineralization, a retreat plan is then executed. This plan uses the results of the test holes that were drilled from the footwall drift to determine the appropriate locations for the different mining methods.

Once the retreat plan has been executed for the entire cut, the portion of the cut that is void is filled with pastefill. Once that portion of the pastefill has had sufficient time to cure, the next cut access can be excavated by slashing the roof of the stope access and the new footwall drift can commence.

16.4 MINE EQUIPMENT

The current fleet at Wolverine Mine is rented from and operated by Procon, the mining contractor on site. Table 16.2 shows a list of all mobile equipment on site as supplied by the contractor; only a small proportion of the fleet is operated underground, as required by the current mine plan, at any given time.

Table 16.2 Underground Mining Equipment

Quantity	Equipment Type	Manufacturer	Specification
Production and Development			
2	Truck	JCI	26 t
2	Truck	Toro	35 t
1	LHD	Atlas Copco	4 yd ³
2	LHD	EJC	3.5 yd ³
2	LHD	Wagner	6 yd ³
1	LHD	Elphinstone	3 yd ³

table continues...

Quantity	Equipment Type	Manufacturer	Specification
1	LHD	EJC	-
1	LHD	Elphinstone	7 yd ³
1	Jumbo	-	-
1	Jumbo	Tamrock	2 boom
1	Jumbo	Tamrock	2 boom
3	Bolter	MacLean	-
1	Bolter	Sandvik	-
1	Bolter	Pluton	-
1	Transmixer	-	-
1	Shotcreter	Normet	-
1	Shotcreter	Mine Master	-
-	Explosives Loaders	Service Vehicles	-
Service and Support			
1	Boom Truck	HIAB	-
1	Scissor Lift	JUT	-
1	B Truck	-	-
1	Grader	-	-
1	Scissor Lift	-	-
2	Man Carrier	Toyota	-
1	Front End Loader	Cat	3 yd ³
2	Mancarrier	Jeffery	-
1	Service Vehicle	Kubota	-
1	Service Vehicle	Kubota	-

The mobile equipment requirements will change in the future once the mine goes to full production. Table 16.3, provided by Malcolm Swallow, P.Eng., a consultant retained by Yukon Zinc, contains a preliminary list of diesel powered equipment that will be required for use underground, at full production. Surface mobile equipment is not included in this list.

Table 16.3 Proposed Underground Mining Equipment

Equipment Type	Quantity
6 yd LHD	4
30 t Truck	3
Drill Jumbo	3
Bolting Jumbo	4
Shotcreter	2
Transmixer	3
Scissor Lift	2
Lube Truck	1
Hiab Truck	1

table continues...

Equipment Type	Quantity
Explosives Loader	2
Utility Tractor	2
Fluid Dispensing System	1
Fuel Truck	1
Utility Vehicle	2
Four Wheel Drive Forklift	1
Flat Bed Service Vehicle	1
Mechanics Truck	1
Toyota Mancarrier	2

16.4.1 MINE BACKFILL

Mine pastefill is fabricated on surface from a combination of processed tails and cement binder. This dense material is pumped from the plant through 125 mm pipe to the appropriate mine cuts. Screen and fabreen barricades have been designed to hold back the pastefill, while allowing excess flush water and pore water to seep through the barricade without creating significant hydraulic head. Cut heights of 4 m will produce a maximum hydraulic head of 42 kPa (6 psi). Once a cut has been deemed as full, the pour can be shut off, or diverted towards the next section to be filled through appropriate use of knife gate valves.

16.5 MINE VENTILATION AND SECONDARY EGRESS

Yukon Zinc has provided Tetra Tech with the following resources to examine the ventilation plan of the Wolverine Mine:

- Yukon Zinc Corporation Wolverine Project Mine Ventilation Study, W.V. Smyth, P.Eng., December 2008
- Amended Technical Report on the Wolverine Property – Finlayson District, Yukon, Wardrop, October 22, 2007
- Malcolm J.A. Swallow, P.Eng., Consulting Engineer, 2011. Memo: The Case for Owner Operated Mining. November 30, 2011
- Underground Ventilation Plan – September 28, 2011 – Compressed Sketch.

The ventilation study (Smyth 2008) and the proposed equipment fleet summary will be used as the primary sources of information in the analysis of the current ventilation plan. The ventilation section of the amended technical report (Wardrop 2007) is considered superseded by the report from W.V. Smyth.

In the development of the current ventilation plan the following design criteria were examined:

- air volume required during the operation of the mine
- the most efficient ventilation scheme for the primary and auxiliary system
- the effect of raise diameter on the primary ventilation circuit
- the optimal primary fan size, location and specifications
- the optimal mine air heater size, location and specification
- energy consumption of the ventilation system and mine air heaters.

To reduce the energy consumption of the ventilation system, a number of auxiliary ventilation systems were examined. These included various auxiliary ducting sizes, couplings and truck loading locations. The air distribution and quantity determine the mines ventilation requirements.

The ventilation model prepared by W.V. Smyth was not available for analysis. A simplified ventilation model was created to determine the adequacy of the primary and secondary ventilation systems.

16.5.1 VENTILATION DESIGN

The primary ventilation is a push system. Air enters the mine through a single 9250-AMF-5000 650 hp fan near the portal. Fresh air for each active face is pushed down one of two fresh air raises and blown to the working faces with auxiliary ducting. Air from working faces flows through the stopes and is exhausted up the ramp to surface. Air distribution is based on a single-pass system.

The auxiliary ventilation system consists of one fan per production level. Auxiliary fans are mounted in ventilation bulkheads, dividing the active mine from the fresh air raise. Flexible ducting is installed the entire length of the stope. This setup maximizes the utilization of air entering each level. Variable frequency drives (VFD) installed on the fans further reduces the overall energy consumption of mine ventilation system.

The mine air heaters are capable of providing 36 MBTU/h, which heats the current air volume from -45°C to 5°C. Based on the current ventilation quantities and historical temperature data from the Faro weather station, the mine air heater is suitable for the typical temperature ranges.

VENTILATION QUANTITIES

The required ventilation quantities were estimated using the mobile equipment fleet and the number of active levels in a “worst case” scenario. In the worst case scenario, 12 of 16 available production faces are in operation at any given time to a depth equivalent of 1070 Level. The ventilation report (Smyth 2008) calculated ventilation requirements based on diesel engine permits and an equipment utilization of 100%. This study proposed that 274,000 cfm would be required when the mine is

at full depth. Calculating underground air volumes according to engine permitting is not in compliance with the Yukon Occupational Health and Safety Act as currently written. The act explicitly requires underground mines to ventilate at 100 cfm/hp for equipment in use (see Yukon Mining Regulations Section below).

Ventilation based on equipment utilization percentages and the Yukon Occupational Health and Safety Act requires 323,280 cfm when the mine is operating at a depth equivalent to 1070 Level. See Table 16.4 for the proposed equipment list and ventilation requirements when the mine is at 1070 Level.

Table 16.4 Mine Equipment Ventilation Requirements

Equipment	Fleet Units	Power (hp)	Utilization (%)	Utilized Power (hp)	Utilized Air Volume (cfm)
30 t Truck	3	393	100	1,179	117,900
6 yd ³ LHD	4	295	100	1,180	118,000
Transmixer	3	185	25	555	13,875
Drill Jumbo	3	148	25	444	11,100
Bolter	4	125	25	500	12,500
Shotcrete Machine	2	-	60	-	24,000
Scissor Lift	2	148	30	296	8,880
Crewcab 4 X 4	2	130	25	260	6,500
Service Tractor	1	115	25	115	2,875
Toyota Mancarrier	2	130	25	260	6,500
Underground Grader	1	115	10	115	1,150
Total	27	1,784	-	4,904	323,280

The mine currently uses a VFD 9250-AMF-5000 650 hp fan which is capable of providing 326,000 cfm when the mine has reached the 1070 Level. As of September 2011, the mine fan is providing 175,000 cfm underground at a reduced power setting. The maximum airflow capacity of the fan assumes that the auxiliary fans will only overcome the resistance of the ducting and not provide additional pressure to the primary ventilation system. The auxiliary fans should be able to provide an additional unknown volume of air to the mine. If additional airflow is required above the current capacity of 326,000 cfm, the current fan motor could be replaced by a larger model.

YUKON MINING REGULATIONS

The following is an excerpt from Smyth 2008 and is still applicable to the Wolverine Mine ventilation system.

The Yukon Regulatory requirement of 0.06 m³/s kW (100 cfm/bhp) was not used as the design basis for primary or secondary flow requirements. With current diesel engine technology, Permit Airflows for the specified Wolverine fleet are 41 to 62% of the .06 m³/s kW specified in 15.61 (2) (b) of the Yukon Regulations.

Article 15.58 (3) (c) states: For each piece of diesel-powered equipment operating underground, a record containing....the prescribed ventilation rate in cubic metres per second required.

Article 15.61 (2) states: Prior to the operation of diesel-powered equipment in an underground mine or project, the ventilation system shall be mechanically controlled, and capable of supplying to the operating area an air current:

- a. of sufficient volume to dilute all the exhaust gases of the equipment to a concentration that does not exceed the permissible levels as in the Occupational Health Regulations,*
- b. of at least 0.06 cubic metres per second for each kilowatt (100 cfm/bhp) of power of the diesel-powered equipment operating in the workplace, and*
- c. that will reduce the time-weighted average exposure of a worker to airborne respirable combustible dust to not more than 1.5 milligrams per cubic metre of air.*

Wolverine operations feedback was that the Yukon Mine Inspectors are satisfied with the airflows as specified on the equipment permits. It is recommended that Yukon Zinc obtain formal confirmation of this as acceptable. Permit airflows are such that contaminate levels are below those specified in the Yukon Occupational Health Regulations. One would speculate that the Yukon Regulations have not yet been revised to consider the advances in diesel engine technology that have led to engine permit values in the 0.03 cubic metres per second per kilowatt range. Permit airflows will ensure compliance with 15.61 (2) (a) and (c). Additionally, the system specified for Wolverine provides uncontaminated fresh air to every workplace where 15.61 (2) (b) could become a compliance issue.

No formal confirmations of engine permitting airflows have been provided. Additionally, regulations similar to the Yukon's Article 15.61 (2) (c) have been amended in other provinces to a lower threshold limit value (TLV). It is expected that the Yukon will also reduce the acceptable TLV of respirable combustible dust.

16.6 UNDERGROUND MINE SERVICES

16.6.1 MINE POWER

The primary underground feed voltage at Wolverine Mine is 4,160 V. Electrical power is produced at the diesel generator farm on surface and distributed underground from a main feeder cable installed in the main ramp. Once underground, the primary voltage is stepped down to 600 V at 5 kVA substations located on 1280, 1260 and 1220 Levels for distribution to the underground mining equipment.

16.6.2 COMPRESSED AIR

Compressed air is supplied to the mine from a bank of four 600 cfm Atlas Copco electric compressors located in the mill building. The compressors are connected to two air receiver banks in the mill, one being used for dry air and the other being used for un-dried air. A total of 2,400 cfm of un-dried air is piped to the underground mine workings at a nominal pressure of 110 psi via a 6 in. diameter heated line. Final distribution from the 6 in. mine air main distribution line, located in the main decline, is by 4 in. and 2 in. diameter pipe running on to each sublevel.

16.6.3 COMMUNICATIONS

A leaky feeder system provides primary communications throughout the underground mine workings with Femco wired phones being used as a backup if required.

16.6.4 MINE WATER SUPPLY

Process mine water is collected in No. 2 Sump, which is located adjacent to the lower crusher pad, for re-use as mine service water. Water from this sump is pumped back underground via a 100 mm (4 in.) diameter line using three 9.7 kW (13 hp) Flygt pumps. The storage capacity of No. 2 Sump is approximately 1,000 m³. Headers on each sublevel distribute process water to the working areas through 50 mm (2 in.) diameter piping. There are no pressure reducing valves required underground due to the shallow depth of the mine.

16.6.5 MINE DEWATERING

The mine dewatering system presently consists of a staged pumping system. Dirty water is collected on the existing sublevels in small single compartment sump boxes and pumped with 13 hp pumps to a dual compartment sump located at approximately 1205 Level on the main ramp. A 30 hp pump moves clean water from the overflow side of this sump to surface while the underflow side is used to settle out and collect solids. Additional pump boxes equipped with 30 hp centrifugal pumps are located on the main ramp in several sump cut-outs; mine water collected in these installations is also pumped to the main dual compartment sump.

The permanent mine dewatering system will be comprised of a full dual compartment sump located at the bottom of the main ramp. It will be arranged similarly to the existing 1205 sump, with the clean water side collecting overflow from the settling compartment and the underflow side being used to settle out solids from the mine water.

16.6.6 SUPPLY STORAGE

There are no dedicated storage areas located underground. However, unused remucks are employed for consumables storage as needed.

16.6.7 *POWDER AND DETONATOR STORAGE*

There are no permanent powder or detonator storage facilities located underground. However, there are two 35,000 kg powder magazines and one 50,000 detonator magazine located on surface. Small cut-outs on the main ramp are presently being used to temporarily store caps and powder. Powder is moved underground on an as needed basis and consumed immediately.

16.6.8 *FUEL STORAGE AND DISTRIBUTION*

All underground mobile equipment is refuelled on surface from a single 49,000 L double walled safety tank, located near the mine portal entrance. A refuelling station will be constructed underground when the main ramp has been completed.

16.6.9 *CENTRAL BLASTING*

Central blasting used at the Wolverine Mine allows the operation to initiate several blasts remotely, from a safe control point on the surface.

16.6.10 *EQUIPMENT MAINTENANCE*

There are no dedicated maintenance facilities underground. All equipment maintenance is done on surface in the main mine truck shop facilities. More detail on the surface infrastructure is given in Section 18.0.

16.6.11 *MINE SAFETY*

FIRE SUPPRESSION

There are no maintenance or fuel storage facilities underground that require fire suppression. All mobile equipment is equipped with standard fire suppression equipment.

MINE RESCUE

Wolverine Mine has its' own mine rescue staff and equipment on site. All rescue personnel have been trained and certified. An on-going program of development of rescue personnel is underway.

REFUGE STATIONS

A permanent 30 man refuge station sufficient to meet maximum active manpower underground at any one time, is located on the 1240 Level. Breathable air is supplied either via the mine's compressed air system, or from oxygen bottles in the

refuge station. In addition, the refuge station contains food and medical supplies, communications equipment and carbon dioxide absorbent sheets.

EMERGENCY EGRESS

The secondary mode of egress from the mine is through an Escapeway located in the main fresh air raise.

PASTEFILL DISTRIBUTION

Pastefill is distributed underground via a 125 mm (5 in.) schedule 80 trunk line located in the main ramp and escapeway while secondary distribution to the stopes is via 100 mm (4 in.) high-density polyethylene (HDPE) pipe.

16.7 WORK FORCE

The Wolverine Mine is being managed and operated by the Owner. Labour and equipment for use underground are provided by mining contractors for underground production and development. An earthworks contractor supplements the mines own fleet of surface equipment and a housekeeping and catering contractor is used to run the camp. A fully serviced camp exists at site sufficient to meet the needs of mine Owner and contractor personnel. All non-management personnel are flown in and out of the mine site on a 21 day-in, 21 day-out roster. Management personnel work a 14 day-in and 14 day-out schedule.

Yukon Zinc mining staff schedule:

- two weeks-on, two weeks-off
- 12 h per shift
- two shifts per day.

Contractor schedule:

- three weeks-on, three weeks-off
- 11 h per shift
- two shifts per day.

16.7.1 YUKON ZINC PERSONNEL

All management, technical, health, safety and environmental (HS&E), mill, and human resources (HR) staff is directly employed by Yukon Zinc. Table 16.5 shows the positions directly employed by Yukon Zinc.

Table 16.5 Yukon Zinc Personnel

Yukon Zinc Personnel	Total
Management	
Mine General Manager	1
Surface Operations Manager	1
Mill Manager	1
Underground Mine Manager	1
Senior Mine Accountant	1
Health and Safety Superintendent	1
Environmental Superintendent	1
Human Resources Superintendent	1
Subtotal	8
Technical Services	
Technical Services Superintendent	1
Senior Engineer	2
Mine Engineer	2
Co-op Student	2
Senior Geologist	2
Mine Geologist	2
Geology Technician	2
Senior Mine Technologist	1
Mine Technologist	2
Mine Surveyor	4
Subtotal	20
Mill	
Chief Metallurgist	1
Senior Metallurgist	2
Mill Metallurgist	2
Mill Technician	2
Mill Superintendent	1
Mill General Foreman	1
Mill Shift Supervisor	4
Control Room Operator	5
Flotation Operator	8
Filter Operator	4
Reagent/Paste Backfill Operator	4
Crusher Operator	2
Operator Trainee	7
Mill Helper	7
Mill Maintenance Foreman	2
Maintenance Planner	2
Millwright	7

table continues...

Yukon Zinc Personnel	Total
Electrician	4
Instrumentation	2
Subtotal	67
HS&E	
Medic and Health Coordinator	1
Safety and Training Coordinator	2
Medic and Safety Technician	2
Security Officers	4
Environmental Coordinator	2
Environmental Technician	1
Subtotal	12
Total	107

16.7.2 CONTRACTOR PERSONNEL

For production Yukon Zinc has hired Procon as the mining contractor. Procon has been engaged as the underground mine contractor at the Wolverine Mine site in Yukon since 2005. Work by Procon began by mobilizing to site on the winter road, establishing the decline portal, and excavating 1,200 m of ramp, ore drift, and exploration drift for bulk sampling. Procon has a total of four crews, two of which work on site at any one time. Current work by the contractor, under the overall direction of Yukon Zinc, includes all aspects of underground production, including main ramp development, lateral ore development, stoping, paste filling and service functions.

The Procon personnel list is provided in Table 16.6.

Table 16.6 Procon Contractor Personnel

	Day Shift	Night Shift	Daily	Total
Management				
Superintendent	1	0	1	2
Production Shift Boss	1	1	2	4
Construction Shift Boss	1	0	1	2
Safety and Training	1	0	1	2
Trainer	1	0	1	2
Surveyor	1	0	1	2
Clerk	1	0	1	2
Maintenance				
Master Mechanic	1	0	1	2
Lead Hand Mechanic	1	1	2	4
Mechanic	2	1	3	6

table continues...

	Day Shift	Night Shift	Daily	Total
Beat Mechanic	1	1	2	4
Electrician	1	0	1	2
Development (Ore and Waste)				
Jumbo Operator	2	2	4	8
Scoop Operator	2	2	4	8
Bolter Operator	4	4	8	16
Truck Operator	3	3	6	12
Shotcrete Operator	6	6	12	24
Raise Miner	2	0	2	4
Support				
Construction	4	4	8	16
Surface Labour	1	0	1	2
Drilling				
Drillers	1	1	2	4
Total	38	26	64	128

Arctic Construction Inc. (Arctic) was originally hired to construct the surface infrastructure for the mine development. They constructed a 29 km site access road and the site earthworks (tailings pond, ditches and retaining walls). Currently they are working as a general contractor around the site supplementing Yukon Zinc personnel as necessary.

Yukon Zinc have a staff of surface operators who provide all surface service functions. Yukon Zinc surface personnel are listed in Table 16.7.

Table 16.7 Yukon Zinc Surface Services Personnel

Surface Services Personnel	Total
Management	
Maintenance Foreman	1
Warehouse	
Warehouse Supervisor/Site Buyer	2
Worker	4
Waste and Water Treatment	
Waste and Water Treatment Coordinator	1
Operator	2
Power House	
Electrical Foreman	1
Operator	1
Mechanic	4
Welder/Fitter	2

table continues...

Surface Services Personnel	Total
Site Services	
Foreman	1
Equipment Operator	20
Camp Maintenance	5
Yard Labourer	9
Total	53

16.8 DEVELOPMENT AND PRODUCTION SCHEDULES

16.8.1 *STOPE PRODUCTIVITY ASSUMPTIONS*

A heading size of 4 m wide x 4 m high with 10% overbreak was assumed. It is also assumed that the effective broken round length is 3 m.

Given these assumptions and using an average density of 3.7 (average of ore and waste), the amount of material in one drift round blast is 195 t. For the basis of scheduling and using round numbers, 200 t per blast was used.

The annual production rate per stope is based on an average of four drift rounds (800 t) per week in each stope and assumes that sufficient mining faces are available to maintain this rate. In this manner, the rate also accounts for delays such as bulkhead construction and backfilling.

16.8.2 *LIFE-OF-MINE OPERATIONS AND PRODUCTION SCHEDULES*

This section shows the production schedule and LOM plan that is used as a basis for the financial analysis covered in Section 20.0. Table 16.8 shows the LOM schedule in tonnes and grade by level for the Wolverine Mine. To demonstrate the grade by year for the mine plan Table 16.9 was added. Table 16.10, Table 16.11, and Table 16.12 show how the ramp, fresh air and return raises and associated development are constructed.

Table 16.8 LOM Plan

Orebody	Level	LOM Plan Tonnes	Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	ZnEq2 (%)	2012	2013	2014	2015	2016	2017	2018	2019	2020
Lynx	1340	10,631	270.08	1.07	0.74	1.38	8.39	19.82	-	-	-	-	10,631	-	-	-	-
Lynx	1320	108,803	266.59	1.04	1.77	1.32	10.69	24.76	-	-	-	-	41,600	41,600	25,603	-	-
Lynx	1300	217,688	382.42	1.90	1.65	2.07	13.34	31.84	41,600	41,600	41,600	41,600	41,600	9,688	-	-	-
Wolverine	1290	33,918	324.50	1.34	1.15	0.90	8.16	22.00	-	33,918	-	-	-	-	-	-	-
Lynx	1280	229,771	358.14	1.95	1.22	1.81	13.28	29.83	41,600	41,600	41,600	41,600	41,600	21,771	-	-	-
Wolverine	1270	43,543	577.32	3.26	0.85	1.51	10.89	33.33	41,600	1,943	-	-	-	-	-	-	-
Lynx	1260	168,421	372.49	1.75	1.12	1.77	11.40	27.70	41,600	41,600	41,600	41,600	2,021	-	-	-	-
Wolverine	1250	88,053	730.30	4.29	0.71	2.17	12.47	40.37	41,600	41,600	4,853	-	-	-	-	-	-
Lynx	1240	129,160	360.46	1.81	1.07	1.94	10.57	26.68	41,600	41,600	41,600	4,360	-	-	-	-	-
Wolverine	1230	129,828	569.65	3.53	0.62	1.93	13.32	35.73	41,400	41,600	41,600	5,028	-	-	-	-	-
Lynx	1220	138,244	362.01	1.62	0.70	2.14	10.91	25.94	41,600	41,600	41,600	13,044	-	-	-	-	-
Wolverine	1210	141,303	399.33	2.52	0.82	1.67	13.44	30.59	35,200	41,600	41,600	22,903	-	-	-	-	-
Lynx	1200	177,708	233.00	0.92	1.09	1.42	12.59	23.90	30,000	41,600	41,600	41,600	22,908	-	-	-	-
Wolverine	1190	148,089	400.18	2.56	1.35	1.71	14.01	32.74	25,000	41,600	41,600	39,889	-	-	-	-	-
Lynx	1180	206,785	197.54	0.71	1.17	1.03	12.03	22.04	20,000	41,600	41,600	41,600	41,600	20,385	-	-	-
Wolverine	1170	135,594	294.30	1.55	2.01	0.91	10.89	26.68	15,000	41,600	41,600	37,394	-	-	-	-	-
Lynx	1160	190,283	225.16	0.78	1.27	1.26	12.24	23.50	12,000	33,000	41,600	41,600	41,600	20,483	-	-	-
Wolverine	1150	183,211	385.86	1.78	1.29	1.11	8.84	25.37	-	27,000	41,600	41,600	41,600	31,411	-	-	-
Lynx	1140	140,619	233.28	0.84	1.46	1.33	12.97	25.10	-	15,939	41,600	41,600	41,480	-	-	-	-
Wolverine	1130	240,719	347.11	1.68	1.01	1.40	10.02	24.96	-	-	34,000	41,600	41,600	41,600	41,600	40,319	-
Lynx	1120	151,423	279.08	0.97	1.22	1.41	12.78	25.62	-	-	-	41,600	41,600	41,600	26,623	-	-
Wolverine	1110	342,995	223.71	1.02	0.97	1.06	9.83	20.41	-	-	-	82,000	83,200	83,200	83,200	11,395	-
Lynx	1100	104,362	293.57	1.04	0.87	1.45	12.15	24.52	-	-	-	-	41,600	41,600	21,162	-	-
Wolverine	1090	117,305	184.76	1.01	1.12	0.81	10.41	20.21	-	-	-	-	38,000	41,600	37,705	-	-
Lynx	1080	15,231	425.31	1.50	1.05	1.85	12.69	29.78	-	-	-	-	15,231	-	-	-	-
Wolverine	1070	48,467	187.30	0.86	1.13	0.88	8.58	18.32	-	-	-	-	33,000	15,467	-	-	-
Saddle	-	110,204	384.80	1.71	1.40	1.44	8.99	26.00	-	-	-	-	-	-	-	45,000	65,204
Average Grades	-	-	327.30	1.60	1.17	1.47	11.55	26.4	-	-	-	-	-	-	-	-	-
Total Tonnes	-	3,752,359	-	-	-	-	-	-	470,000	611,000	621,253	621,018	620,871	410,405	235,893	96,714	65,204
Tonnes per Day	-	-	-	-	-	-	-	-	1,288	1,671	1,702	1,701	1,701	1,127	646	265	179

Table 16.9 LOM Production Schedule

	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Tonnes	470,000	611,000	621,253	621,018	620,871	410,405	235,893	96,714	65,204	3,752,358
Zn (%)	12.24	11.97	12.02	11.77	11.29	10.91	10.59	9.52	8.99	11.55
Cu (%)	1.05	1.15	1.19	1.23	1.21	1.17	1.11	1.19	1.40	1.17
Pb (%)	1.77	1.61	1.57	1.42	1.32	1.24	1.18	1.38	1.44	1.47
Ag (g/t)	418.235	372.674	344.363	304.574	274.976	269.605	256.414	350.106	384.802	327.305
Au (g/t)	2.263	1.939	1.747	1.452	1.190	1.178	1.134	1.617	1.712	1.597

Table 16.10 Ramp and Associated Development Schedule (m)

Level to Level	Ramp Development Including Miscellaneous	2011	2012	2013	2014	2015
1200 to 1180	209	159	50	-	-	-
1180 to 1160	202	-	202	-	-	-
1160 to 1140	192	-	192	-	-	-
1140 to 1120	415	-	-	300	115	-
1120 to 1100	194	-	-	-	194	-
1100 to 1080	191	-	-	-	-	191
Total Ramp (m)	-	159	444	300	309	191

Table 16.11 Fresh Air Development Schedule (m)

Level to Level	Fresh Air Raise Development (including Lateral Connection)	2011	2012	2013	2014	2015
1250 to 1230	40	40	-	-	-	-
1230 to 1210	42	42	-	-	-	-
1220 to 1200	36	36	-	-	-	-
1210 to 1190	26	26	-	-	-	-
1200 to 1180	31	-	31	-	-	-
1190 to 1170	47	-	47	-	-	-
1180 to 1160	42	-	42	-	-	-
1170 to 1150	33	-	33	-	-	-
1160 to 1140	31	-	31	-	-	-
1150 to 1130	34	-	-	34	-	-
1140 to 1120	37	-	-	-	37	-
1130 to 1110	34	-	-	-	34	-
1120 to 1100	45	-	-	-	45	-
1100 to 1080	35	-	-	-	-	35
Total Ventilation (m)	-	144	184	34	116	35

Table 16.12 Return Raise Development Schedule (m)

Level to Level	Return Air Raise Development	2011	2012	2013	2014
1280 to 1260	34	-	-	34	-
1260 to 1240	26	-	-	26	-
1240 to 1220	27	-	-	37	-
1220 to 1200	25	-	-	-	25

17.0 RECOVERY METHODS

17.1 ORIGINAL DESIGN – INTRODUCTION

The original process plant design was based on an underground mine plan that provided for a maximum annual design plant feed of 1,702 t/d (621,308 t/a). The initial design for a tonnage of 1,702 t/d included 302 t/d slated to be waste/barren dilution rock and this material was to be discarded using the DMS process. The DMS portion of the circuit was subsequently excluded from the design, because a revised (then new) mining plan and geological report determined that all mined material would be ore and would be processed in the plant. As a precautionary measure, a pre-flotation stage was incorporated into the design to which would serve to remove the bulk of any carbonaceous material entering the plant with the ore feed.

The unit processes selected for the plant design were based on metallurgical test work conducted at the PRA laboratories located in Vancouver, BC (PRA Report, June 2000), specifically PRA Test F13.

The metallurgical processing procedures and the plant design parameters were chosen to produce saleable grade copper, lead, and zinc concentrates. The copper and lead concentrates were expected to contain an associated valuable metal content of gold and silver. In addition, the unit processes selected for the recovery of the metals used conventional size reduction and mineral beneficiation methods. The process was designed to incorporate approximately 60% of the flotation tailings to be used underground as paste backfill, with the balance impounded in the tailings pond facility. Water utilization was designed to maximize the re-use of water as process water, and to recover water from the tailings pond. Fresh water from the underground dewatering wells was to be used for gland service and reagent preparation. Excess process water was to be pumped to the tailings pond facility.

The unit operations for the process plant design included:

- run-of-mine (ROM) ore feed hopper and ore reclaim
- primary crushing and screening
- secondary crushing and screening
- two-stage rod and ball mill grinding with cyclone classification
- pre-flotation, pre-flotation cleaning, and discharge of the pre-flotation concentrate to tailings
- copper rougher flotation, regrinding and cleaner flotation stages

- copper concentrate thickening, filtration and dispatch
- lead rougher flotation, regrinding and cleaner flotation stages
- lead concentrate thickening, filtration and dispatch
- zinc rougher flotation, regrinding and cleaner flotation stages
- zinc concentrate thickening, filtration, and dispatch
- tailings filtration
- tailings disposal alternately by paste backfill and deposition in the tailings pond
- water reclamation from tailings pond for the process water circuit
- treatment of tailings pond reclaim water for discharge to the environment
- fresh water circuit.

Figure 17.1 shows a simplified process flowsheet. Table 17.1 shows the major process design criteria. As mentioned previously, the design criteria are all based on the results of the test procedure of Locked Cycle PRA Test F13. Vendor information was incorporated into the design criteria where applicable.

Figure 17.1 Simplified Process Circuit Flowsheet

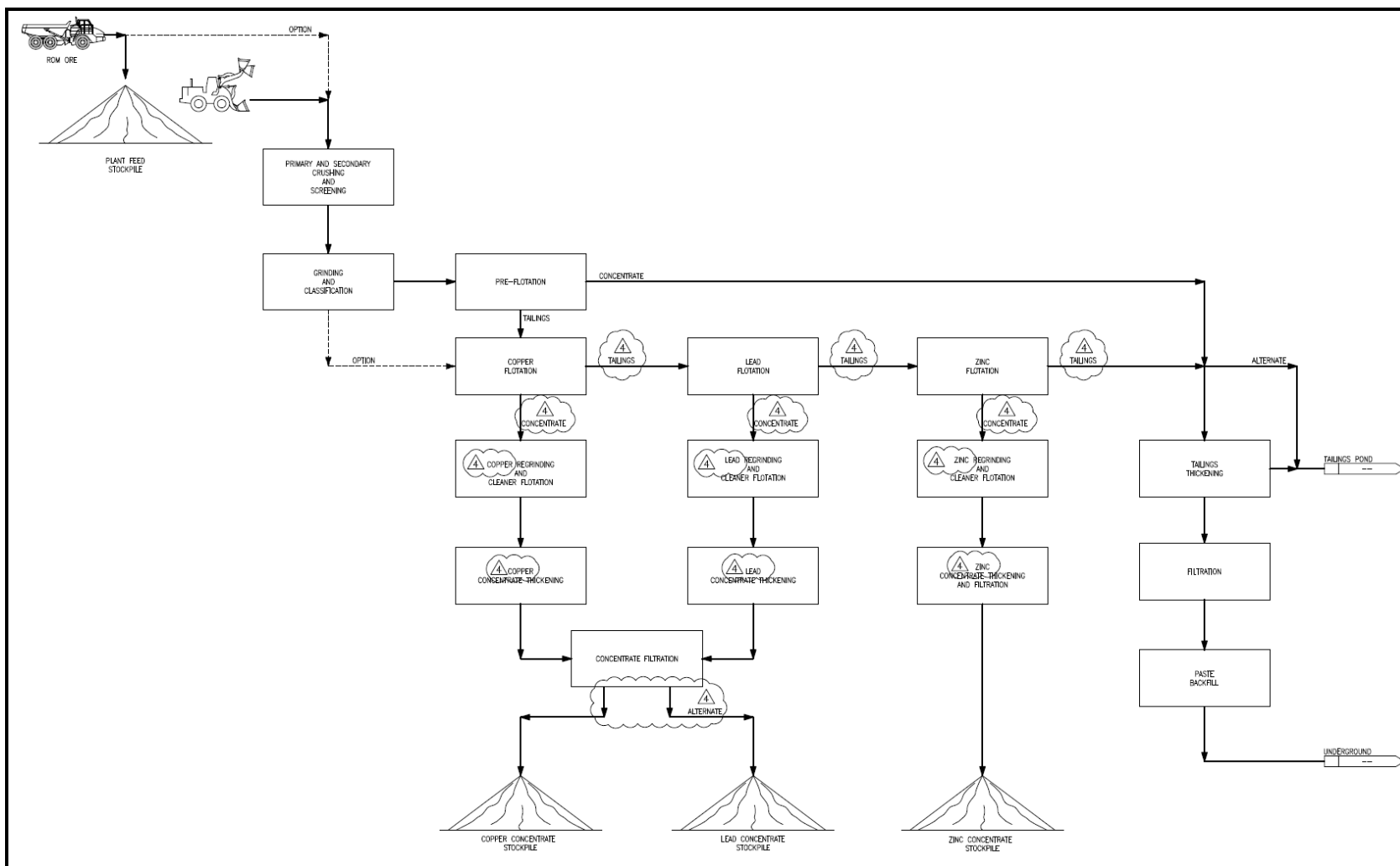


Table 17.1 Process Design Criteria

General	Unit	Value
Type of Deposit: Cu/Pb/Zn Polymetallic Sulphide Ore		
Shifts/Day	-	2
Hours/Shift	h	12
Hours/Day	h	24
Days/Year	d/a	365
Availability/Utilization		
Annual Overall Processing Rate	t/a	621,308
Daily Processing Rate	t/d	1,702
Crusher Plant Availability	%	40.0
Grinding and Flotation Plant Availability	%	92.0
Production		
Ore Specific Gravity	-	3.71
Ore Bulk Density	t/m ³	2.30
Ore Moisture Content	%	5.0
Head Grade	%Cu	0.91
	%Pb	1.26
	%Zn	9.66
	g/t Au	1.36
	g/t Ag	282
Recovery	%Cu	79.9
	%Pb	46.0
	%Zn	89.4
Total recovery to Cu, Pb and Zn concentrates combined	%Au	67.5
Total recovery to Cu, Pb and Zn concentrates combined	%Ag	83.1
Cu Concentrate Grade	%Cu	21.3
	%Pb	2.3
	%Zn	3.7
	g/t Au	11.3
	g/t Ag	4,409
Pb Concentrate Grade	%Pb	22.3
	%Cu	2.0
	%Zn	12.4
	g/t Au	13.5
	g/t Ag	1,625
Zn Concentrate Grade	%Zn	54.2
	%Cu	0.4
	%Pb	1.0
	g/t Au	0.7
	g/t Ag	151

table continues...

General	Unit	Value
Cu Concentrate Mass Recovery	%	3.90
Cu Concentrate Production	t/a	19,929
Pb Concentrate Mass Recovery	%	2.30
Pb Concentrate Production	t/a	11,753
Zn Concentrate Mass Recovery	%	19.30
Zn Concentrate Production	t/a	98,623
Impact Bond Work Index	kWh/t	5.7
Abrasion Index, Ai	-	0.046
Rod Mill Bond Work Index	kWh/t	11.4
Feed Size, P ₈₀	µm	19,000
Ball Mill Bond Work Index	kWh/t	11.4
Product Size, P ₈₀	µm	53
Copper Regrind Product Size, P ₈₀	µm	20
Lead Regrind Product Size, P ₈₀	µm	20
Zinc Regrind Product Size, P ₈₀	µm	20

17.2 PLANT DESIGN

17.2.1 OPERATING SCHEDULE AND AVAILABILITY

The original plant was designed to operate based on two 12-hour shifts per day, for 365 days per year.

The crushing circuit was designed for 40% overall availability and utilization, while the grinding and flotation circuit running time was designed for 92% availability. These availabilities were selected in order to allow for sufficient downtime for scheduled plant equipment maintenance.

The design plant feed rate was to be 1,702 t/d of ore with no allowance made for dilution rock, or waste rock.

17.2.2 ORE CHARACTERISTICS

The plant was designed for ore delivery at a specific gravity of 3.71 g/cm³ and a bulk density of 2.3 t/m³. The design allowed for a nominal 5% moisture content and a maximum ore size of 450 mm.

The crushing/impact Bond work index was determined to be 5.7 kWh/t. The rod mill and ball mill Bond work index both varied from 9.6 to 11.4 kWh/t. The work index value of 11.4 kWh/t was assumed for the design criteria for both the rod mill and the ball mill. An abrasion index of 0.046 was used.

The plant design used an average ore head grade of 0.91% Cu, 1.26% Pb, 9.66% Zn, 1.36 g/t Au, and 281.79 g/t Ag. These values were obtained from the mining plan production schedule.

Table 17.2 shows the distribution of metal recoveries anticipated into the three respective concentrates, together with the respective assay grade values. These values were also based on the results from PRA Test F13.

Table 17.2 Concentrate Recovery Distribution and Concentrate Grades

Metal	% Recovery to the Concentrates			Concentrate Composition, % or g/t		
	Copper Concentrate	Lead Concentrate	Zinc Concentrate	Copper Concentrate	Lead Concentrate	Zinc Concentrate
Copper	79.9	4.3	6.7	21.30	1.97	0.36
Lead	8.0	46.0	17.2	2.25	22.30	0.98
Zinc	1.2	2.4	89.4	3.74	12.40	54.20
Gold	33.4	23.2	10.9	11.30	13.50	0.74
Silver	60.0	12.9	10.2	4,409	1,625	151
Mass	3.9	2.3	19.3	-	-	-
Feed	-	-	-	0.91	1.26	9.66

The plant was designed to produce and handle the following daily amounts of concentrates:

- copper concentrate between 50 and 59 t/d with an average production rate of 55 t/d
- lead concentrate between 32 and 40 t/d, with an average production rate of 34 t/d
- zinc concentrate between 255 and 295 t/d with an average production rate of 270 t/d.

Concentrate mass changes were attributed to the probable variations in the feed grade of the ore being delivered from the different stopes. This affected the amount of concentrate production that would require treatment in the cleaner and filtration circuit.

17.2.3 PRIMARY CRUSHING

Underground ore was to be delivered to the primary crusher for size preparation suitable for feeding to the secondary crusher. The primary crushing plant was designed with following equipment:

- a stationary grizzly
- a rock breaker

- a vibrating grizzly feeder
- a jaw crusher
- conveyor belts
- a conveyor belt magnet
- cranes.

Underground haulage trucks with 50 t maximum load capacity, subsequently revised to 30 t capacity, were to bring ROM ore from the underground adit to the feed hopper or surge bin. The ore was to be dumped directly from the trucks into the feed hopper for feeding the crusher. In the event of unscheduled crusher plant stoppages or breakdowns, ore would be dumped onto a temporary stockpile located adjacent to the crusher. The feed hopper was designed to have a stationary grizzly with 450 x 450 mm square apertures, to prevent oversize rocks from entering the feed hopper. A rock breaker was to be installed to break up any rocks larger than 450 mm in size. The feed hopper/surge bin was designed to have a nominal live capacity of 60 t equivalent to a volume of 26 m³ based on the bulk density of 2.3 t/m³. The temporary stockpile would also serve as an area to blend the ore types in the event that the ore grades from different stopes had significant variations in values. The surge capacity in the surge bin was designed to provide a steady feed of ore to the crusher.

The surge bin was to be equipped with a vibrating grizzly feeder. The undersize material would report to the conveyor feeding the cone crusher sizing screen, and the oversize material was to be the feed to the primary jaw crusher. The jaw crusher was designed to crush the sized ROM ore to a product rock size with a P₈₀ of 75 mm, using a closed size setting of 70 mm. The crushing circuit was designed on the basis of the ore having a moisture content of 5.0%.

Jaw crusher discharge was to join the undersize material from the vibrating grizzly on the conveyor to the cone crusher sizing screen.

17.2.4 SECONDARY CRUSHING

The secondary cone crusher was designed to crush ore to a suitable size for milling. The main equipment items for this circuit included:

- conveyor belts
- a cone crusher
- a vibrating sizing screen
- a metal detector
- a belt magnet
- a vibrating feeder

- a cone crusher dust collection system.

The pre-screened and jaw crushed ore was to discharge onto the conveyor feeding the sizing screen ahead of the cone crusher. The cone crusher discharge was also to join the stream to be screened. The screening operation was to be conducted on the material on an as-received basis (i.e. not wet-screened).

The screen undersize material was to discharge onto the conveyor feeding the fine ore bin. The screen oversize ore fraction would feed the cone crusher via a feed chute and a vibrating feeder.

The fine ore bin was designed to have a live capacity of 1,700 t, equivalent to the daily plant throughput.

A dust extraction system was to be installed to control the dust generated by the crushers. The dust collected by the system was to be deposited onto the conveyor feeding the fine ore bin for subsequent processing.

17.2.5 GRINDING CIRCUIT

The grinding circuit was designed to reduce the size of the crushed ore to the particle size required for the flotation process. The grinding process chosen was a 2-stage operation with the rod mill in open circuit and the ball mill in closed circuit with classification cyclones fed from the common mill discharge sump. The grinding circuit was designed to include the following equipment:

- conveyor belt feeders
- conveyor belts
- a conveyor belt weigh scale
- a rod mill
- a ball mill
- a mill discharge pumpbox
- cyclone feed slurry pumps
- cyclone classifiers
- a mass flow meter
- a sampler system.

The ore in the fine ore bin was to be drawn out of the bin under controlled feed rate conditions using belt feeders. These feeders were to discharge the material onto a conveyor belt feeding the rod mill. A belt scale was to be installed to control the feed to the rod mill. Water was to be added to the rod mill feed material to assist the grinding process at a controlled rate.

The rod mill discharge was designed to flow into the mill discharge pumpbox. The discharge from the ball mill was also to flow into this pumpbox. The combined discharge slurry was to be pumped to a cyclone cluster for classification. The cut size for the cyclones was at a P_{80} size of 53 μm , and the design circulating load to the ball mill was 300%. The cyclone underflow was to return to the ball mill as feed material at a design pulp density of about 70%. The cyclone overflow was to discharge into the pre-flotation cells prior to the flotation process at a design pulp density of about 30% solids. The mills were designed to treat feed material at a rate of 64 t/h of post-DMS feed. This was also the design rate to the flotation circuit.

Reclaim water from the tailings pond was to be added as dilution to the grinding circuit as required.

Provisions were made for reagent addition to the grinding process if required.

17.2.6 FLOTATION CIRCUIT

The circuit design was to incorporate four stages of sequential flotation. The pre-flotation stage was included to remove the naturally-floating carbon, which may influence the subsequent flotation stages. Following pre-flotation, three stages of sequential flotation were included to recover copper, lead, and zinc minerals into their respective metal concentrates.

Provision was made throughout the process to allow reagent addition within the flotation circuits, as required.

The pre-flotation circuit was to consist of the following equipment:

- flotation reagent addition facilities
- a sampling system
- rougher flotation cells
- cleaner flotation cells
- standpipes
- a pumpbox
- slurry and solution pumps.

The flotation circuit was designed so that the cyclone overflow from the classification cyclones in the grinding circuit feeds the pre-flotation circuit. The pre-flotation rougher concentrate was to be upgraded in a pre-flotation cleaner stage. The pre-flotation cleaner concentrate was to be discarded to the tailings circuit. The pre-flotation tailings were then to proceed to the copper flotation circuit as new feed to the circuit. The pre-flotation circuit was designed to remove approximately 1% mass as cleaned “product”. This pre-flotation concentrate product was expected to contain the bulk of the organic carbon present in the plant feed material.

Each of the three subsequent mineral concentrating flotation circuits was designed to include the following equipment:

- conditioning tank(s)
- flotation reagent addition facilities
- a sampling system
- rougher flotation cells
- a regrind mill and classifier
- cleaner flotation cells or columns
- pumpboxes
- slurry and solution pumps.

Following the pre-flotation stage, the pre-flotation tailings would enter the copper flotation circuit as new feed. The slurry would be conditioned in the copper conditioning tank. The conditioned slurry was designed to overflow into the copper rougher/scavenger flotation bank of cells. The rougher/scavenger copper concentrate was to feed the first cleaner flotation column. The first cleaner tailings and, optionally, the scavenger concentrate product was to be reground and sent to the feed of the first cleaner scavenger flotation circuit. The first cleaner scavenger tailings was to report to the feed of the rougher circuit, and the first cleaner scavenger concentrate was to report to the feed of the first cleaner flotation circuit.

The feed to the second cleaners was to be the first cleaner concentrate with the second cleaner concentrate reporting as final concentrate. The second cleaner tailings would report to the first cleaner feed. The final copper concentrate was to be pumped to the copper concentrate thickener.

The copper tailings slurry was to be pumped to the lead conditioning circuit as flotation feed to the lead flotation circuit.

Provision was made for the use of water from the copper concentrate thickener overflow to be re-used in the copper flotation circuit providing this does not have a deleterious effect on the flotation of copper minerals.

The lead flotation circuit layout was to be similar to the copper flotation circuit and was to include rougher flotation, a regrind mill and two stages of concentrate cleaning. The flotation cell volumes were to be similar, although the unit numbers of the cells in the various stages were to be different. The circuit was to differ in that all of the rougher and scavenger concentrate was to be reground prior to cleaning together with material from the first cleaner scavenger concentrate. The lead second flotation cleaner concentrate was to be the final concentrate product and was to be pumped to the lead concentrate thickener. The other major difference in this circuit was that the flotation cells chosen for the cleaner flotation circuit were mechanical cells.

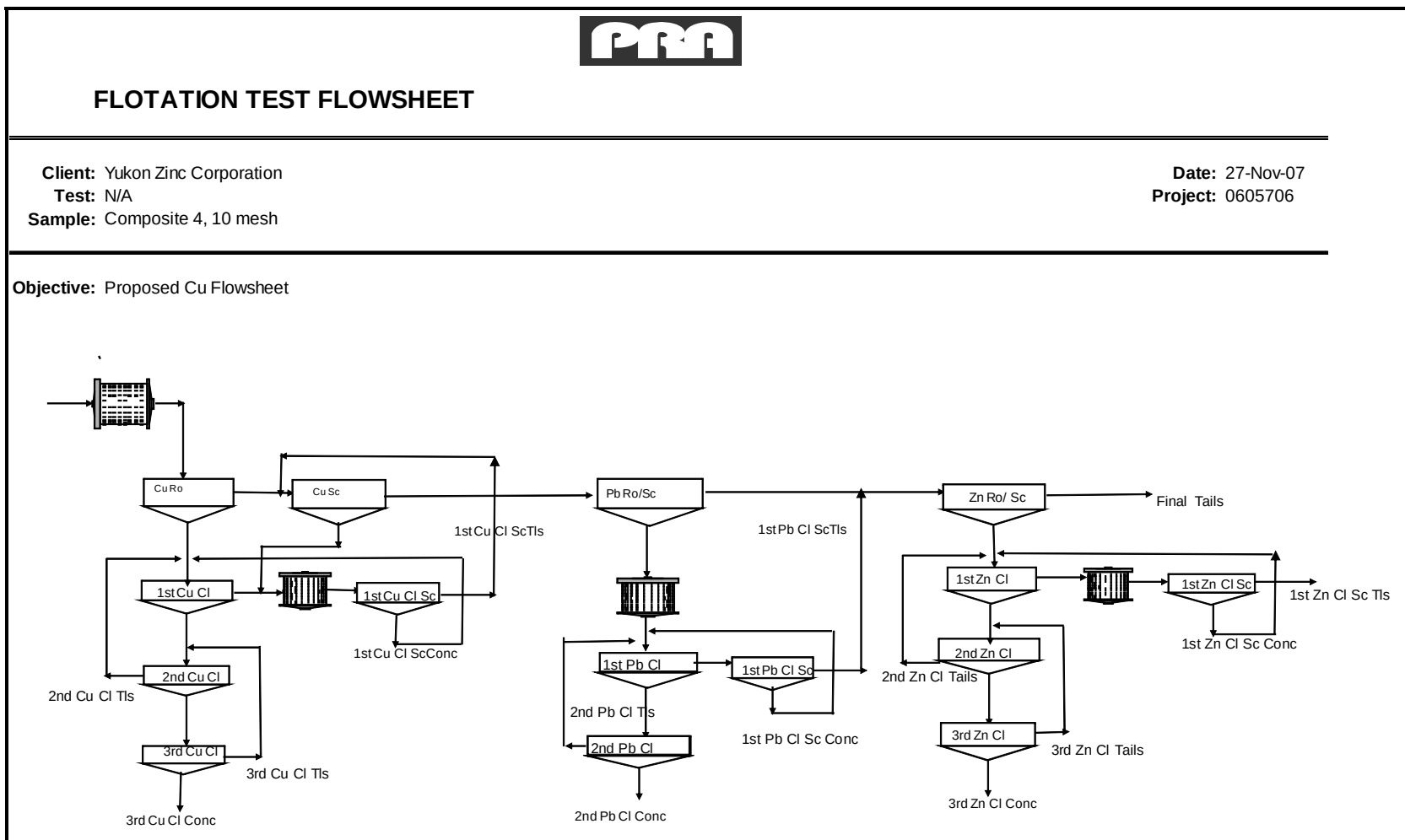
The lead flotation tailings stream was to constitute the feed material to the zinc flotation circuit. Water recycling from the lead-concentrate thickener overflow was to be re-used where applicable.

The zinc flotation circuit was also to be similar to the copper and lead flotation circuits, but was designed with different numbers and volumes of flotation cells in the various flotation stages. The zinc flotation circuit was designed with three conditioning tanks. For the zinc circuit, the third cleaner concentrate was to be the final concentrate product. This was to be pumped to the zinc concentrate thickener. The other major difference in the zinc circuit was that the flotation cells chosen for the cleaner flotation circuit were a combination of mechanical cells for the first cleaners followed by column cells for the second and third cleaners. In the zinc flotation circuit, the first cleaner tailings were to be reground with an option in place to regrind the rougher-scavenger concentrate.

The zinc flotation tailings stream was to constitute the final tailings leaving the plant. This tailings material was to be pumped to the tailings pond facility, or to the paste backfill plant. Water recycling from the zinc-concentrate thickener overflow was to be re-used in the zinc flotation circuit where applicable.

The basic flotation circuit design was based on the flowsheet initially based on the PRA test work results and subsequently developed by Metso following additional test work undertaken by PRA and agreed upon by Yukon Zinc. The design was incorporated following the flotation testing and is shown in Figure 17.2. Note that the PRA flowsheet in Figure 17.2 is correct for the copper and zinc regrind circuits, but incorrect for the lead regrind circuit which should show the first cleaner scavenger concentrate also reporting to the regrind circuit mill. Although this would appear to be inconsistent in that this material has already been reground, this configuration was agreed upon by Metso, PRA and Yukon Zinc for incorporation into the plant process design (see also Section 17.5.3).

Figure 17.2 Flotation Flowsheet November 27, 2007



17.2.7 CONCENTRATE HANDLING

The flotation cleaner concentrates for each metal were to be thickened and then filtered and stored prior to shipping to the smelters. Each of the three concentrate handling circuits were to have the following equipment:

- concentrate thickeners
- concentrate slurry pumps
- process water pumps
- process water tanks
- concentrate stock tanks
- a thickener overflow standpipe
- concentrate filter presses
- concentrate storage and dispatch facilities
- a dust collection system.

The flotation concentrate from each of the three metals were to be treated in a similar manner. The concentrate produced would be pumped from the cleaner flotation stage to its respective concentrate thickener. Flocculant would be added to the thickener feed to aid the settling process. The thickened concentrate would be pumped to the concentrate stock tank using thickener underflow slurry pumps at a design underflow density of 60% solids. Pressure filters were chosen for dewatering the concentrate products. A swing filter press unit was chosen to treat the copper and lead concentrates as batch products and a separate zinc filter press was chosen for the zinc concentrate product. The filter press was to dewater the concentrate to produce a concentrate with a moisture content of about 8% with the filter press solids to be discharged to its respective concentrate stockpile. Each of the dewatered concentrates was to be stored in a portion of a designated storage facility. Each concentrate was to be periodically loaded into trucks for dispatch off the property.

The thickener overflow solution from each concentrate thickener was to be collected in the respective process water tank for recycling in the flotation circuit, while excess solution was to be discharged to a common concentrate thickeners overflow standpipe and pumped to the final tailings pumpbox for pumping to the tailings pond.

17.2.8 TAILINGS DISPOSAL

The flotation tailings were to be treated for intermittent use as paste backfill in the underground workings, and alternatively deposited in the tailings pond facility. The tailings handling circuit was to have the following equipment:

- a tailings thickener

- slurry pumps
- a filter feed storage tank
- vacuum disk filters
- vacuum pumps
- a conveyor belt
- a filtrate receiver
- a paste backfill plant
- a cement silo.

The zinc flotation circuit tailings would be the final plant tailings. The tailings were to be used as paste backfill when needed, and during the periods that the paste backfill was not required, the tailings were to be pumped directly to the tailings pond from the final tailings pumpbox. Approximately 60% of the tailings would be used for the paste backfill operations.

Solution from the concentrate thickener overflows, plus filtrate from the filter presses and the tailings disc filters would be mixed with the zinc tailings and pumped to the tailings pond. This would allow the various reagents to oxidize and/or degrade in the tailings dam prior to returning the water to the plant as reclaim process water.

The paste backfill plant would be used to mix the dewatered tailings with cement to obtain the consistency and strength required for underground backfill operations. The paste was to be pumped into the mined-out areas to provide underground support.

17.2.9 REAGENT PREPARATION

Chemical reagents would be added to the process stream to facilitate the separation and flotation processes. The preparation of the various reagents would require the following equipment:

- bulk handling systems
- mix and holding tanks
- metering pumps
- transfer pumps
- a flocculant preparation facility
- a lime slaking and distribution facility
- a cement storage facility
- safety equipment.

Various chemical reagents were to be added to the grinding and flotation circuit to modify the mineral particle surfaces to enhance the floatability of the mineral particles into selective concentrate products. Fresh water was to be used in the making up or the dilution of the various reagents that would be supplied in powder/solids form, or which require dilution prior to the addition to the slurry. These prepared solutions would be added to the points of addition of the various flotation circuits and streams using metering pumps.

Each reagent was to be prepared within its own segregated sump area in order to contain spillage. A dedicated spillage pump was to be installed to return spillage from the sump to the mixing tank of each specific reagent for re-use in the plant.

17.2.10 WATER SYSTEM

Fresh water was required for the reagent preparation and for gland service to the slurry pumps. Fresh water was to be supplied from the underground dewatering wells. The water will be pumped to the fresh and firewater tank for storage and distribution.

Some process water generated in the individual flotation circuits as concentrate-thickener overflow solution was to be re-used in the respective flotation circuit. Reclaimed water was to be pumped from the tailings pond facility to the water reclaim tank for distribution to the points of usage.

The supply of potable water was designated to come from a well situated in the vicinity of the operating facility.

17.3 SEPTEMBER 2011 AUDIT REPORT SITE VISIT AND SUBSEQUENT PROCESS PLANT MODIFICATIONS

17.3.1 INTRODUCTION

The process description, as given in the previous section, has been followed in the construction of the Wolverine Mine plant facility treating the multi-mineral sulphide ore and producing a saleable copper concentrate, lead concentrate, and a zinc concentrate, each also containing revenue contribution silver and gold which adds to the value of the respective concentrate.

However, as a result of a shortage of ore as plant feed while the mine is still in the production ramp-up stage, the plant is presently not running at design capacity. Although the individual unit processes could be considered as having been commissioned operationally, design throughput treatment rates for extended periods of time have not been attained to date. However, the throughput is ramping up with each succeeding month.

The following section describes some of the changes and modifications made to the plant during the commissioning period as completed by the staff at the Wolverine Mine. The list is not exhaustive but serves to highlight changes and modifications which have had to be made to the plant and which were brought about because of inadequate or non-suitable performance of the various aspects of the plant during the initial months of plant operations.

Although detailed and descriptive, this section on plant modifications and changes serves to illustrate and reinforce the commitment by the Yukon Zinc and Wolverine Mine mill staff to ensure that this processing facility will function operationally as per the basic design.

17.3.2 *PRIMARY CRUSHING CIRCUIT*

The primary crushing section has experienced start-up problems during the commissioning phase. A brief description of the problems experienced during the start-up phase, and some of the modifications made, are given in the following section.

FEED STOCKPILE

At the time of the audit, the plant feed stockpile was not yet utilized for blending the ROM ore, to the extent that was initially planned, as a result of ore supply shortages. The present arrangement of dumping ore and picking it up again to feed the plant is inefficient and costly, but is a result of the design of the grizzly/feed hopper which does not easily facilitate the dumping of ore directly from the ore haulage trucks, the need to blend ore, and the present erratic ore supply from underground during this production ramping-up stage of the mine. The ore material on the ROM plant feed stockpile was seen to be fine-grained with very few sizeable competent rocks or boulders present. The material was very dark to black indicating relatively high carbonaceous matter content. At the time of the visit, it also appeared to have a relatively high moisture content estimated to be 6 to 8% moisture, a situation which would complicate the handling in the crushing and grinding section as a result the fineness of the material. This aspect has since been remedied and currently taken moisture determinations (as at May 2012) indicate an ore moisture content of the feed material of around 4%.

Some minor segregation in colour of the material was visually observed indicating that material sourced from different stopes was arriving at the plant. However, no blending for grade control was being carried out at that time, although the longer-term plan includes the physical blending of ores on the stockpile using front-end loaders. This was also in keeping with the original design of the plant facility which included a pad for the blending of the ore from different stopes.

The feed material from underground was also finer (and wetter, as mentioned) than the original design at the time of the audit resulting in severe blockages and ore

transfer movement restrictions. This problem has been resolved by making modifications to the feed bin in order to facilitate ore feed to the primary crusher and ore transfer through the system and by reducing the moisture content of the ore received from underground. The original bin design was criticized as being too square resulting in too much dead space in relation to the slot area of the feeders to the grizzly and the jaw crusher. The additional effect of the significant amount of dead space was the loss of effective bin surge volume.

ROM FEED BIN

The jaw crusher vibrating grizzly feeder installation is very constrained making access for maintenance and clean-up very difficult. The vibrating grizzly feeder to the jaw crusher should have had VFD controls. At present there is a lack of control of material being fed to the jaw crusher resulting in an erratic feed supply and spillage. A VFD control has subsequently been installed to improve the original design, and this has improved the feed control to the jaw crusher and reduced spillage.

WASTE ROCK STOCKPILE

At present the waste rock from underground development is brought to surface and is then deposited on the waste rock stockpile. The mine plan calls for this material to be returned to underground sites for the backfilling and support of the mined-out stopes. However, consideration is presently being given to treating this material in the mill as incremental feed material since it does contain metal values albeit at lower grades than the ore, and this would be a less expensive option than re-loading this material and transporting it to specific underground sites. It would also boost the mill throughput, which at present has spare capacity, and the tailings could be returned underground for deposition as backfill paste. However, the metallurgical response of this material needs to be studied before this step is taken as this may be problematical with the anticipated higher carbonaceous matter content together with the lower metal grades. Since the waste rock stockpile has only been permitted on a temporary basis, a decision will be required relatively soon for its disposal either via treatment in the mill, or by increasing the ore stockpile capacity, or by transportation back underground as fill.

CONE CRUSHER FEED VIBRATING SCREEN

The original vibrating screen has not withstood the duty required in the application and has already failed. This was repaired as a temporary measure, but it is slated for replacement by a new screen from an alternate supplier. Also, its layout location above the cone crusher has been criticised by the operating staff as being difficult to access for cleaning, inspection and maintenance. The screen feed bin located above the screen has also been questioned regarding its size (too large) and difficulty of access. From a maintenance point of view, the cone crusher layout is very constrained and a great deal of time is required in preparation for access for

maintenance work to be performed. The feeder to the cone crusher should have been designed as a feed chute from the screen. The belt feeder complicates the operations, is more difficult for layout and requires more maintenance than a feed chute.

CRUSHER BUILDING

The overall layout of the crusher building has been deemed to be acceptable with good lighting and access.

Initially, there was no door provided at the truck tipping point into the feed hopper/crusher ore feed bin to shut off the building when no crushing is taking place. A door has been subsequently installed and has made a substantial temperature difference to the operating conditions in the area at the feed hopper during the periods of cold and inclement weather.

CRUSHER FEEDER STOP BLOCK

A new stop block has been installed at the point where the loader tips the plant feed material into the ROM ore feed hopper. The unit initially installed was deemed to be inadequately designed for the purpose.

DUST COLLECTION

Although initially a spray mist system was designed for dust control in the crusher building, this was subsequently changed during the latter part of the design phase to a standard air filter dust control system. This dust control system is working satisfactorily.

CRUSHER OPERATOR STATION

The crusher operator station situated to overlook the ROM ore feed bin has been positioned very well with regards to the flow of ore.

CONVEYOR BELTS

A totally enclosed conveyor gallery would have been a preferred conveyor system. The installed partially covered system results in difficult access to the furthest areas of the belt and the idlers and support system, particularly during maintenance and cleaning. However, it is appreciated that this was a major cost-saving item and that the conveyor system has been constructed in a sufficiently robust manner to minimise downtime, and maintenance requirements.

17.3.3 GRINDING CIRCUIT

ROD MILL FEED BIN

The rod mill feed bin, or fine ore bin, did not operate as originally designed and choked during operations, or at best operated below design expectations because of the build-up and compaction of ore in the dead space and with subsequent rat-holing in the bin. This led to major modifications having to be incorporated. Amongst others, the sides of the bin were steepened thereby removing the dead space where material compaction and plugging occurred. High angle steelwork siding walls were constructed within the bin eliminating much of the dead space and ensuring that the wall angle was steep enough to force the material to gravitate to the rod mill belt feeders. An air cannon was also installed. The modifications were costly and the outcome of these bin wall modifications was that the live capacity was reduced from the design value of 1,700 t, to about 1,200 t live capacity. Apart from the loss of some of the live capacity capability, the modified bin system now operates very well.

This failure of the bin design in not permitting uninterrupted feed to the rod mill was the subject of detailed discussions with the engineering company, Wardrop Engineering Inc., involved in the original design of the system. Yukon Zinc believed that the problem was caused by poor bin design, while the engineering company contended that the bin design parameters were based on the design parameters which required drier and coarser-sized ore material than what was actually entering the bin during the plant start-up operations when the problem became apparent.

During the audit, the coarsest material observed on the rod mill feed belt was estimated to be 12 to 14 mm, and the moisture content was visually estimated to be between 7 to 9%. Since that time, the moisture content of the material fed to the mill averages about 4%. The original design moisture content of the ore was 5.0%, while the rod mill feed was designed for a particle size of minus 19 mm and a P_{80} of 16 mm.

ROD MILL CHARGER

The rod charger was installed on a pivot in front of the rod mill to facilitate rod charging. There were many discussions during the design phase about how to “store” this rod charger off the operating floor or in an out-of-the-way area, but no obvious solution was apparent. However, since the rod charger is currently used only once every two or three days, it does obstruct that area of the mill floor.

ROD MILL FEED PIPE

The internal diameter of the rod mill feed pipe as supplied by Metso was less than 200 mm (8 inches) in diameter and was prone to blocking by the mill feed material. Removal of the wear liners inside the feed pipe in order to increase the diameter of

the pipe to about 270 mm (10.75 inches), and the fitting of water sprays, has facilitated the feeding of the rod mill.

PRIMARY GRIND PARTICLE SIZE ANALYZER

The primary on-stream particle size analyzer had not been commissioned at the time of the site visit for the audit. The primary grind P_{80} value of 53 μm has been routinely attained operationally, and these size grading analyses are being conducted manually at this stage.

OPERATIONAL CIRCUIT UTILIZATION

Although the grinding circuit is currently operating on a full-day campaign basis when ore is available, the design tonnage of 1,700 t/d has not yet been attained mainly as a result of mining constraints. At the time of the plant visit, the throughput was being throttled back to about 1,000 t/d, but subsequently the recent daily throughput has varied between 1,200 and 1,450 t/d, and has since reached 1,500 t/d on occasion during May 2012 – see Table 17.4 for monthly data.

CIRCUIT LAYOUT

The overall grinding area layout was spacious and not cramped, with sufficient space available for access to the equipment for maintenance and inspection purposes. There is also good access between the mill area and the flotation area. Floor space has also been provided for maintenance work and facilities.

CONTROL ROOM

The location of the main control room, and the viewing angle requirement of the mill and the flotation floor, was the subject of many discussions during the design phase of the plant. Eventually a reasonable compromise regarding available viewing angle and the location of the pipe racks was reached. However, the appearance of the control room creates the impression that a better arrangement could have been constructed, and a full view window should have been considered. Also, the offices adjoining the plant should have had windows provided for viewing the operations and not a wall even though there are pipe racks along that wall, i.e. a better layout design could have been implemented.

17.3.4 *PRE-FLOTATION CIRCUIT*

INTRODUCTION

Initially, the process plant design included a DMS circuit in the design in order to remove the barren and lighter fraction of the mined material which contained little or no metal values. However, this DMS circuit became a relatively high capital cost

item in the mill design during preparation by Tetra Tech. Also, the revised mine plan at the time indicated that the amount of barren waste material, or dilution, which would be mined together with the valuable ore component, would constitute only a very small portion of the total feed to the plant. This dilution was indicated to be significantly less than originally estimated at the start of the design of the project. This newly revised mine plan therefore essentially allowed for 1,700 t/d of ore to be supplied to the plant, rather than the 302 t/d of waste and 1,400 t/d of ore for a total of 1,702 t/d.

Therefore, as a consequence of this reduction in the amount of waste material reporting as feed to the plant, and as a cost-saving measure, the DMS circuit was deleted from the design of the processing plant. The necessary adjustments were subsequently made to the crushing circuit to accommodate the changes in the feed to the grinding circuit, and the flotation and filtration circuit was re-designed to accommodate 1,700 t/d of ore and not the originally planned 1,400 t/d of ore. As a preventative measure, and in the place of the DMS circuit, the pre-flotation stage was incorporated into the plant design, and this was deemed adequate at dealing with the anticipated relatively small amounts of carbonaceous matter which would be present in the feed material to the plant which could occur either within the mined ore envelope, or in the barren waste material mined with the ore. This pre-flotation circuit was installed as a safeguard since it was known that if this carbonaceous material was not removed at the start of the flotation process, this material would cause higher than normal flotation reagent consumptions and potentially reduce the overall flotation efficiency. Also of interest as a chronology of the plant design was that immediately prior to the deletion of the DMS circuit, both the DMS circuit and the pre-flotation circuit were still present in the plant design.

The pre-flotation circuit was designed to be a rougher stage together with a single cleaner flotation stage. According to test work, the cleaner pre-flotation concentrate produced was to contain relatively low-grade material equivalent to plant feed grade or lower, and this concentrate would then be discarded with the final plant tailings.

PRE-FLOTATION

During the first year of mine production, the pre-flotation circuit has been found to be removing a significantly higher mass as the pre-flotation concentrate than the original design parameters. This pre-flotation concentrate material has also been found to contain significant values of silver and gold as well as copper, lead and zinc which are therefore lost to the final tailings. Mass recoveries of up to 10% have been obtained in the plant operations compared with about 1% which was obtained during the test work programs. The metal content of this concentrate has been found to be significantly higher than the original test work had indicated.

As a result of the financial value of the contained metal, and particularly silver and gold, in the pre-flotation concentrate, test work has been initiated by the Wolverine Mine to determine how the contained metal values losses can be minimized within the existing pre-flotation circuit configuration. Although a water drip-tray system was

installed at the time of the site visit as a test to allow a fine stream of water drops to fall on the pre-flotation concentrate prior to its collection and discharge to tailings in order to assist in the washing away of entrained gangue particles, this was subsequently discontinued as unsuccessful.

Other test work initiated by the Wolverine Mine in order to limit the loss of valuable metal values in the pre-flotation concentrate includes:

- Screening of the rod mill discharge material to avoid overgrinding.
- Using spirals to remove low SG and carbon-bearing material by gravity concentration.
- Centrifugal gravity concentration of the pre-flotation concentrate to recover valuable metal content prior to discharge to tailings. This included the testing of the pre-flotation concentrate with a Falcon centrifugal concentrator to recover valuable minerals included in the pre-flotation concentrate. Test results from Met-Solve Laboratories, Vancouver, BC, on recently conducted test work, have indicated that the pre-flotation concentrate stream is amenable to the upgrading and recovery of the valuable mineral components using centrifugal gravity concentration. The results obtained are presented in Table 17.3. Based on these results obtained, plant trials are currently underway to recover valuable mineral components utilizing a non-fluidized Falcon Centrifugal Concentrator.

Table 17.3 Pre-flotation Concentrate Centrifugal Gravity Concentration Test Results

Pre-flotation Concentrate Calc. Heads		Gravity Concentrate at 20% Mass (Grade)	Percent Distribution Gravity Concentrate (%)	Gravity Tails at 80% Mass (Grade)	Percent Distribution Gravity Tails – to Final Tails Box (%)
Metals\Elements	Calc. Grade				
Au	91.1 g/t	31.8 g/t	70	3.41 g/t	30
Ag	1,010 g/t	1,468 g/t	29	896 g/t	71
Cu	7.2%	9.2%	25	6.7%	75
Zn	3.2%	3.6%	22	3.1%	78
Pb	1.3%	1.4%	22	1.3%	78
Total C	7.0%	2.9%	8	8.1%	92
Graphite	1.3%	0.8%	13	1.4%	88
C-Inorganic	0.6%	0.4%	12	0.7%	88
C-Organic	5.2%	1.7%	7	6.0%	93

- Wolverine Mine is also planning the retrofitting of a DMS circuit. This is discussed in Section 17.5, and originally discussed at the time of the plant visit, test results obtained recently have been sufficiently encouraging to provide the go-ahead for this process modification. Test work completed

early in 2012 by Maquinaria Technica Atlantica, Spain, using run-of-mine samples from the Wolverine Mine, has demonstrated the significant metallurgical benefits and financial viability of installing a DMS circuit in the processing circuit at the mine. The test work sample used was a minus 18 mm feed material screened at 1 mm, and this constituted the feed to a 250 mm (10 inch) diameter DMS cyclone using a Warman pump. The minus 1 mm material was further separated into fines and slimes, with the fines collected and treated separately with a spiral concentrator in order to remove the heavier mineral components in this fines fraction. This DMS and spiral concentrator combination rejected a total of 24.7% of the feed material with metal losses in the 2 to 4% contained metal range. In addition, a significant amount of the carbon present in the sample was also removed in the rejects stream fraction. An important observation was made during the test work, namely that the DMS cyclone Warman feed pump was generating a significant amount of the slimes in the minus 1 mm fraction by the action of the pump itself in feeding the screen and the DMS cyclone. It was concluded that although the test work was not optimized, the DMS process circuit will contribute markedly to the overall efficiency of the processing plant by removing a significant proportion of the barren component of the plant feed material thereby reducing the carbon content of the material reporting to the flotation circuit and which is deemed to be the major reason for the plant not performing optimally.

A further discussion on the pre-flotation circuit is also provided in Section 17.5.4.

17.3.5 FLOTATION CIRCUIT

CONDITIONING TANKS

The slurry inflow pipes for the conditioning tanks are too close to the overflow slurry outflow ports exiting the tank resulting in short-circuiting and a reduced conditioning time. The inflow pipe should be situated on the opposite side of the outflow, or should be introduced into the tank via a downcomer arrangement.

BLOWER AIR CONTROLS

The flotation blower air pipe and valves for each row of flotation cells are situated on the side furthest away from the walkway making access for adjustments and maintenance difficult. Having a walkway on only one side of each row of the flotation cells was a design decision at the time and this decision was made to limit cost and building space requirements. In the meantime, the mine staff have manufactured a special tool to assist the operating staff with valve setting changes.

The control of blower air for flotation has been problematical. It was decided by the Yukon Zinc during the design phase that the control of air to each flotation cell would be made manually. However, the adjustment to any particular cell leads to air flow

“re-adjustment” to the other cells making control changes difficult. Since changes in slurry density contributes to air volume changes, Tetra Tech believes that an automatic control system on the supply of air to the flotation cells would be advantageous.

PH MEASUREMENT/CONTROL

For the lead circuit, a pH meter has been installed in the conditioning tank, but it was providing inaccurate information regarding the pH of the slurry feeding the lead rougher flotation circuit because of lag control effects. The pH meter has since been relocated to the lead flotation feed stream and is now providing the required information accurately.

FLOTATION CELL PADDLES

Although claimed to contribute to an improved operational efficiency of flotation cells, it was observed that only some of the flotation cells paddles were functioning during the plant visit. The flotation cell manufacturer recommends the use of paddles for the type and sizes of cells installed. However, it is not clear whether the operation of these paddles is not regarded as a maintenance priority, or whether their operation is considered unnecessary for the successful operation of each of the flotation circuits.

REGRIND CIRCUIT

The stirred media detritor (SMD) mills used for regrinding are operating satisfactorily at this stage, although the design particle size of 20 µm has not yet been attained in either of the copper, lead, or zinc flotation regrind circuits. In all three cases, the present regrind size being attained is about 37 µm. Plant staff are currently working towards the objective which is to attain the design regrind size of 20 µm for each of the flotation circuits.

Further discussions on the regrind circuit are provided in Section 17.5.2.

ON-STREAM ANALYSER

The plant was designed to have fully automatic on-stream analyses conducted on 22 product streams for copper, lead, zinc, iron and silver. To date, as at the time of the plant audit, six of the most significant product streams have been commissioned and are considered to be fully-functioning. At this stage, the plant staff are unsure whether all 22 product streams will be commissioned, as it appears that this aspect of on-stream analysis has been over-designed and the ultimate value of the complete on-stream sampling system is not apparent. However, the present commissioning plan is to incorporate the use of possibly up to 12 streams.

One of the drawbacks of the large number of samples being taken is that in certain cases, whole product streams are used as the samples for analysis. The result is

that a significant number of pipes have been installed, as well as specific product pumps to deliver and return products from special sumps, and the resulting maintenance is outweighing the potential benefits of the on-stream analyses.

In addition, some of the on-stream samplers have been incorrectly selected for the proposed duty and gate valves have had to be introduced into certain pipes to provide pressurized lines for the sampling to take place.

Despite the large number of on-stream sample points in the design, the plant staff have indicated that one major sampling point was omitted during the design, namely a sampler on the final tailings i.e. the tailings thickener underflow material. Although the final tailings grades can be calculated from the zinc rougher tailings stream, and the zinc first cleaner scavenger tailings streams and the respective mass flows, the pre-flotation concentrate is not sampled although it can be calculated from the cyclone overflow/flotation feed sample and the copper flotation feed sample. However, this is considered to rely on too many other samples and operational factors and is therefore not accurate enough. Also, since this is a final tailings sample, a metallurgical quality sample is required, and this cannot be obtained by depending on other conditions. Since the pre-flotation concentrate can significantly influence the value of the final tailings, the Wolverine Mine is considering the installation of a sampler for the final tailings material. At present hand samples of the final tailings are taken on a frequent basis.

REGRIND CIRCUIT AND COLUMN CELL AREA LAYOUT

This area of the plant is extremely cramped and access to equipment is tight.

COLUMN CELLS

The column cells for the upgrading of the copper and zinc flotation concentrates are performing satisfactorily. Two modifications have been introduced, namely a drip-water trough to “wash” away entrained gangue, and an overflow pipe has been installed for improved column cell level control.

REAGENT PREPARATION

The reagent preparation area, including the three-tier system adopted for certain reagents, is generally functioning well. However, unexpectedly, an additional reagent, sulphuric acid, has had to be accommodated in the general reagent area, since the process water pH used in the flotation circuit was found to be too high. The reason for the high pH value in the process water is the significant amounts of shotcrete material (containing lime and cement) being included with the ore and which increases the pH of the process water, including the reclaim water returning from the tailings disposal facility. The recent change over to the use of wet shotcrete in the mine should further reduce the amount of shotcrete reaching the mill as wet shotcrete is a more efficient and economical process compared with the present

system in use. Also, as the mine moves to second and third cuts, the amount of shotcrete used as a percentage of the ore fed to the mill will decrease, thereby reducing the influence on the pH value of the water used in the plant.

Although a lime silo and slaking unit is available, at the time of the plant visit, the Wolverine Mine was still using its supply of quicklime for pH control when this was needed. This quicklime was ordered as part of the Wolverine Mine start-up cycle. Since the depletion of the quicklime, the mill now uses hydrated lime which is trucked to the Wolverine Mine in bags and then pneumatically transferred to the lime silo.

The cyanide for the zinc flotation circuit is prepared in its own isolated and locked enclosure although a jib crane had to be installed to assist with the reagent off-loading and delivery to its make-up/storage area.

MAINTENANCE ACCESS

The dart valves used for level control on the flotation cells were installed with no access for maintenance in certain applications. The Wolverine Mine mill staff subsequently relocated and installed certain walkways to facilitate unimpeded access to these valves.

Pump access, particularly in the SMD and cleaner column flotation area, is very constrained and difficult and insufficient area/space has been allocated for maintenance activities. Typically, v-belt changes to certain pumps has been very difficult, and the lack of space affects maintenance and inspections. Also, access to a number of pumps was not possible since a walkway was installed over the top of the pumps, which left insufficient clearance for maintenance activities.

GENERAL ACCESS

Apart from these maintenance access problems which were not given sufficient consideration at the time of the original design, overall access to the equipment is generally good. However, in one instance, the mill staff had to install an elevated walkway from the rougher flotation cells area to the regrind/column cell area. This was inadvertently overlooked during the design, but was deemed necessary for ease of movement of the operating and maintenance staff. In addition, although there was minimal available open space during the layout design phase, some enclosed building space is wasted because of a lack of access and/or location.

17.3.6 CONCENTRATE HANDLING

CONCENTRATE THICKENERS

Although not under production design load, the three concentrate thickeners are operating acceptably. However, the zinc concentrate thickener was not producing a clear solution overflow at the time of the visit. Since all three concentrates produced

are currently at a coarser grind than the design regrind size, this may lead to increased amounts of flocculant required in the future for possibly all the concentrate thickeners in order to ensure that clear overflow water is obtained.

CONCENTRATE FILTERS

The concentrate filters are operating well under the present conditions. The swing copper-lead filter and the zinc filter are handling the present production rate of concentrate produced. Cycle times are presently about 13 to 15 minutes. While this is longer than originally specified, this is not a problem at this stage of the ramping up operation. Typical moisture contents recorded have been between 8.6 and 9.6% moisture. However, when the plant operates at the design regrind size of 20 μm , it is anticipated that the cycle times may increase and that the moisture contents may also increase.

CONCENTRATE STORAGE

The concentrate storage facility has been modified so that both the copper and lead concentrates are now currently bagged using a mobile bagging system. Previously, the copper concentrate produced was conveyed from the filter and stockpiled in its respective concentrate storage area, prior to loading into the concentrate trucks. The concentrate bagging system is deemed more efficient with respect to handling and transportation to the smelter for the current concentrate quality. As the quality of the copper concentrate improves, the operation will revert to bulk handling of the concentrate. The zinc concentrate is stockpiled in the designated area before being bulk-loaded into concentrate trucks using a front end loader as per the original design.

17.3.7 TAILINGS HANDLING

TAILINGS THICKENER

In contrast to the three smaller concentrate thickeners, the tailings thickener was located outside of the mill building and was the subject of many discussions regarding insulation and the minimizing of heat losses during the design phase. The ultimate decision was for the tailings thickener to be completely enclosed with an enclosed access corridor from the mill building to the thickener.

The tailings pumpbox was found to be under-sized during the start-up of operations, particularly when no paste backfill was being produced, because of the inflow of multiple slurry and solution streams. The sides of the pumpbox have been raised to increase the capacity and thereby improve control of the tailings pumps.

17.3.8 *HOUSEKEEPING*

The general appearance of the plant was clean, and there was no accumulated spillage anywhere. The crusher circuit still contained spillage of crushed material from the previous shift in various places in the crushing circuit. Apparently, the routine clean-up had not yet taken place at the time of the audit visit. There were small, steel fibre pieces which were being removed by the magnet, but tended to spill onto the floor, and which has the potential to subsequently block screens and other orifices. These pieces are being introduced into the plant from the underground wire-reinforced shotcrete and comes in with the plant feed material. As mentioned, the use of wet shotcrete and the reduction of steel fibre reinforced shotcrete will alleviate this problem.

The mill area was clean and the level of housekeeping was of a high standard.

The flotation area was also clean and the housekeeping was also of a high standard. No spillage was observed on the flotation floor area.

The filtration and concentrate thickener area was also of a high housekeeping standard.

The concentrate storage building was of a good standard of housekeeping, although at the time of the visit, bags containing concentrate were stored outside alongside the walls of the concentrate shed and between the concentrate shed and the plant building.

No scrap or used equipment or equipment parts were seen to be lying around.

The sloping floors to separate sumps under the flotation cells has been successful in diverting spillage to the respective sump pump, and no walls or berms have been required to contain any operationally arising spillage.

The assay and metallurgical testing laboratory is generally tidy. However, the assay laboratory floor was not as clean as would be expected in a facility where contamination is one of the prime considerations in the production of reliable assay data. Tetra Tech strongly advises that Yukon Zinc address laboratory cleanliness and restrict all foot-traffic in the assay laboratory to minimize or the possibility of contamination.

17.3.9 *ASSAY AND METALLURGICAL LABORATORY*

The assay laboratory is very spacious and has been well-equipped for its role in supplying the Wolverine Mine and mill with assay information from underground samples and the plant products.

At present, the assay facility consists of a sample receiving area, a high-grade and low-grade sample preparation area, and drying ovens, followed by crushers, pulverizers and splitters.

The assay laboratory also consists of the following sections and equipment:

- An Atomic Absorption Spectrophotometer (AAS) set up for analysing copper, lead, zinc, iron, and silver, with a second machine on order. The incorporation of this second AAS unit will allow for the separate analyses of high grade and low-grade samples as well as serving as a general standby unit.
- An ICP Spectrometer was being installed at the time of the audit visit, and was due to be commissioned imminently. This ICP unit has since been commissioned and is in everyday routine use, and is being used for samples requiring trace element analyses, as well as providing indication information for environmentally-related samples.
- A fully equipped classical wet-chemistry laboratory.
- A micro-balance room with the required equipment.
- Two fire assay furnaces for gold analyses plus attendant equipment.
- Safety ventilation hoods for the hotplates.
- Water treatment facility to neutralize all laboratory effluents.
- A still for the production of demineralized water as required in the assay facility.

During the audit visit, the assay staff highlighted the need for an analytical instrument to enable analyses for sulphur to be conducted, and to have this unit suitable to analyse for carbon as well. Given the importance of recognizing and planning for the presence of carbon in the Wolverine Mine samples and plant products, this type of analytical instrument was considered by Tetra Tech to be an essential item for the assay laboratory, and that a suitable analytical unit should be purchased without delay. In response, an Eltra Carbon and Sulphur Analyzer has been purchased and commissioned, and was in routine use by mid-April 2012.

The two fire assay furnaces had recently been purchased and installed, and were being commissioned at the time of the visit. The presence and recovery of gold had not received prior importance regarding its distribution in the ore, nor its financial contribution to the cash flow of the Wolverine Mine during the design phase. It had been assumed by the client during the design phase that special samples requiring gold assays would be sent to a specialized contracting assay laboratory. Given the financial importance of gold and its by-product value, particularly to the copper and lead concentrates, routine assaying for gold at the assay laboratory at Wolverine Mine commenced during January 2012.

The metallurgical laboratory is also well-equipped with a crusher, laboratory-sized mill, flotation machines, screens and a particle size analyzer, bottle roller, and other miscellaneous equipment. This laboratory will be very valuable in supporting metallurgical investigations.

17.4 METALLURGICAL PERFORMANCE

17.4.1 INTRODUCTION

The plant performance during 2011, and the first four months of 2012, that produced concentrates and gave measurable metallurgical data, has been presented in Table 17.4. The results are discussed in Section 17.4.2.

Table 17.4 Metal Recoveries and Grade Values

2011	Cu Circuit				Pb Circuit				Zn Circuit				Total Ag Rec. (%)	Total Au Rec. (%)	Mill Feed (t)
	Rec. (%)	Cu Grade (%)	Ag Rec. (%)	Au Rec. (%)	Rec. (%)	Pb Grade (%)	Ag Rec. (%)	Au Rec. (%)	Rec. (%)	Zn Grade (%)	Ag Rec. (%)	Au Rec. (%)			
January	n/d	n/d	n/d	n/d	n/d	n/d	n/d	n/d	n/d	n/d	n/d	n/d	n/d	n/d	0
February	77.5	10.9	55.1	n/d	2.9	6.2	1.4	n/d	67.2	33.7	13	n/d	69.5	n/d	3,276
March	51.8	15.2	34.8	n/d	34.3	13.0	14.4	n/d	49.7	38.2	13.5	n/d	62.7	n/d	9,019
April	47.0	16.2	27.6	n/d	28.4	9.6	15.8	n/d	63.0	37.6	17.1	n/d	60.5	n/d	12,485
May	47.2	14.4	39.2	n/d	29.2	7.9	16.1	n/d	64.3	46.5	9.9	n/d	65.2	n/d	11,045
June	51.0	17.7	35.8	n/d	29.7	16.5	12.2	n/d	71.6	47.6	8.8	n/d	56.8	n/d	18,408
July	51.7	15.9	39.5	n/d	44.3	23.0	16.6	n/d	67.3	50.0	11.0	n/d	67.1	n/d	1,517
August	50.7	15.7	40	15.7	39.8	22.1	14.7	2.3	70.1	41.4	12.3	n/d	67.0	18	23,661
September	41.5	11.2	34.2	- n/d	38.7	17.0	23.8	- n/d	76.6	40.9	12.4	- n/d	70.4	- n/d	18,556
October	25.6	12.0	14.0	- n/d	44.5	14.3	37.2	- n/d	67	41.6	11.5	- n/d	62.7	- n/d	15,998
November	37.4	13.3	29.0	- n/d	46.7	16.0	23.8	- n/d	76.3	42.4	11.2	- n/d	64	- n/d	18,076
December	26.5	13.5	18.2	6.9	44.9	14.9	27.3	5.2	76.8	43.6	16.2	10.3	61.7	22.4	21,311
2011	45.2	14.9	30.3	-	38.5	15.5	21.5	-	70.0	42.3	12.4	-	64.2	-	153,352
Design/Budget I	79.9	21.3	60.0	33.4	46.0	22.3	12.9	23.2	89.4	54.2	10.2	10.9	83.1	67.5	-
Design/Budget II	79.9	21.3	52.6	33.0	46.0	21.3	14.9	10.0	89.9	54.2	0.0	-	67.5	43.0	-

table continues...

2012	Cu Circuit				Pb Circuit				Zn Circuit				Total Ag	Total Au	Mill Feed
	Rec. (%)	Cu Grade (%)	Ag Rec. (%)	Au Rec. (%)	Rec. (%)	Pb Grade (%)	Ag Rec. (%)	Au Rec. (%)	Rec. (%)	Zn Grade (%)	Ag Rec. (%)	Au Rec. (%)	Rec. (%)	Rec. (%)	(t)
January	33.8	13.5	26.7	5.0	53.8	16.2	24.9	7.9	78.2	43.5	11.3	6.8	63.0	12.9	24,841
February	32.3	14.3	24.1	10.8	47.1	15.4	23.7	19.8	72.2	44.9	12.8	6.8	60.6	30.7	29,846
March	29.2	14.7	17.6	13.6	35.1	12.2	18.6	8.9	76.5	45.7	16.4	6.3	52.6	22.5	29,413
April	28.8	14.0	21.1	18.3	43.6	16.7	21.2	8.6	81.5	43.3	16.9	7.9	59.2	27.0	40,298
2012 YTD (Ave/Total)	30.6	14.2	22.4	12.5	45.5	15.4	22.1	11.2	77.6	44.2	14.6	7.0	59.0	23.7	124,398

Note: n/d = no data, Rec. = recovery

In addition, a comparison of the various ore feed grades is given in Table 17.5, together with the set of actual plant feed data recorded by the Wolverine Mine.

Table 17.5 Plant Feed Grade Values

Feed Grade	Actual I*	Actual II**	Mine Plan Design I	Test PRA F13 Feed Assay Design II	Test PRA F13 Design III
Copper (%)	0.98	0.90	0.85	0.91	1.03
Lead (%)	0.97	1.00	0.76	1.26	1.10
Zinc (%)	8.32	8.29	8.78	9.66	11.70
Silver (g/t)	230.00	214.05	135.40	282.00	285.00
Gold (g/t)	1.50	-	0.67	1.36	1.31

Note: *Actual I = off-site assay of cyclone overflow/flotation feed samples averaged over five sets of readings from the period March 2 to August 13, 2011

**Actual II = actual grades from the mine assays averaged over the same period

From Table 17.5, it is apparent that the differences between the actual plant assay feed grade assay values are not significant, namely, less than 10%.

The assay results for the PRA Test F13 both as the assayed head grade values designated as Design II in Table 17.5, which has been assumed to be the basis of the plant design, as well as the re-calculated head grade values for PRA Test F13 designated as PRA Test F13 Design III in Table 17.5 are shown. This clearly demonstrates the different values attained during metallurgical test work.

It is apparent that the plant performance to date has generally not attained the original design performance targets (as of the time of the plant visit) with regards to metal recoveries, nor attained the design product concentrate grades. The lead circuit has on occasion met, or nearly attained, the design parameters, especially with respect to the recovery of silver in the lead concentrate. Although the results are steadily improving over the operating months, the copper and zinc circuits still need to improve further, although the recovery of silver, which is the highest value component metal in the ore, has achieved the budgeted recovery values.

17.4.2 ANALYSIS OF GRADE AND RECOVERY PERFORMANCE DATA

The following comments regarding the plant performance data are relevant.

PLANT PERFORMANCE DATA

The actual metallurgical performance plant data for the three circuits is given in Table 17.4. The mill throughput data have been included which gives the actual tonnage treated for the months January to December 2011, and January to April 2012.

AVERAGE VALUES

The recovery and grade data given in the rows titled '2011' and '2012 YTD' in Table 17.4 reflect the actual arithmetic averaged values for the relevant periods. These are only for purposes of ease of comparison of information, and these values have no other statistical significance.

DESIGN PARAMETERS

The values given in the rows titled 'Design/Budget I' and 'Design/Budget II' in Table 17.4 reflect the target/budget concentrate grades and metal recoveries obtained by the PRA Test F13 which formed the basis of the plant design. The design recovery for silver and for gold in the respective concentrates has also been shown in Table 17.4. Note that some differences exist between Design/Budget I and Design/Budget II numbers depending on the source of information. Both have been shown in Table 17.4.

ACTUAL ORE FEED GRADE VALUES

The actual plant feed grade data given in Table 17.5 reflects the information obtained from the Wolverine Mine during the site visit. The difference between the sets of data has been described in a previous section.

DESIGN AND FEED GRADE VALUES

Various feed grade values have been given in Table 17.5 as being the design feed grade of the ore supplied to the Wolverine Mine. The PRA Test F13 is provided as two slightly different sets of data. The 'official' analysis of the material tested is as indicated (termed 'Design II'), while the second set gives the re-calculated feed grade values for the tests are shown out of interest (termed 'Design III'). These values demonstrate the degree of variation which could occur during actual mill operations. The mine plan feed grades (termed mine plan 'Design I') are relatively similar for copper, lead and zinc, but has had the silver and gold feed grades have been reduced to approximately half of that given by PRA Test F13 for reasons which are not known.

COMPARISON OF FEED GRADE VALUES

A comparison of the design feed grade values with the actual feed grades indicates that the copper feed grade value is approximately the same, although the original mine plan value for copper is lower. The copper feed values are respectively, 0.91% Cu (Test No. F13 Design II, which is the basis for design), 0.98% Cu (actual from Wolverine Mine for the period March to August 2011) and 0.85% Cu (original mine plan Design I). However, the lead feed grade values show significant variations month-on-month, namely 1.26% Pb (Design II), 0.97% Pb (actual March to August 2011), and 0.76% Pb (mine plane - Design I), respectively.

The feed grade values for zinc show relatively minor variations for the month-on-month actual data, and comparatively give respectively 9.66% Zn (Design II), 8.32% Zn (actual March to August 2011) and 8.78% Zn (mine plan - Design I).

Also of note is that the feed grade for silver varies significantly. In this case, the design feed value is 282 g/t Ag (Design II), the actual March to August 2011 feed value is 230 g/t Ag while the original mine plan indicated 135.4 g/t Ag (Design I). Similarly, the feed grades for gold also vary significantly, namely 1.36 g/t Au (Design II), 1.50 g/t Au (actual March to August 2011) and 0.67 g/t Au (Design I), respectively. The reason for the original mine plan silver and gold values being so low compared with the PRA Test F13 design data is not known.

It is apparent that some differences in the actual feed ore grades have been experienced compared with the design values, but that only in the case of zinc could it be considered to be a significantly lower value. In this case, it would be expected that lower zinc recoveries would have been obtained.

Also of significance is the fact that no specific allowance has been made for characterising the pre-flotation concentrate as was done for the copper, lead, and zinc concentrate products.

17.5 DISCUSSION

17.5.1 *MILL THROUGHPUT*

At the time of the visit, the mill was only processing a nominal $\pm 1,000$ t/d of material and not the attained the designed 1,700 t/d. Apart from the ramping up constraint of the mined ore supply, this lower than design reduced throughput is apparently also related to limitations arising from the timing of paste fill required in the underground workings. Since the time of the visit, production has steadily increased as more ore has become available. In April 2012, the mill averaged 1,343 t/d for the full month, and on a number of days during the month of April, the mill operated in excess of 1,400 t/d, thereby indicating that the mill can successfully handle the ever-increasing amount of feed material.

At the time of the audit visit, the Wolverine Mine was only expected to achieve the full production rate of 1,700 t/d towards the end of 2012, which in the opinion of Tetra Tech, seemed to be a very long ramping-up period. The concern was that the plant cannot really be considered fully commissioned and operational until the target throughput of 1,700 t/d is achieved. As an alternative, it is recommended that ore be stockpiled for production run campaigns and that when a mill treatment campaign is conducted, to push the plant throughput to achieve 1,700 t/d during that campaign period, even if it shortens that particular campaign period. This concern has been alleviated to a great extent with the demonstrated success in handling the increased ore production and mill throughput rate.

Also at the time of the visit, it was stated that the feed to the rod mill was being curtailed to the $\pm 1,000$ t/d throughput rate. This implies that the grinding circuit cannot be considered to be fully commissioned, and will also result in inferior mill performance efficiency at these lower tonnages, with the possibility of over-grinding certain ore components, together with the potential to increase the loss of metal in the pre-flotation circuit. The lack of a VFD controller on the rod mill motor means that the mill speed cannot be reduced to limit the potential for over-grinding. Attaining the design throughput of 1,700 t/d is very important to ensure that the basic plant design is sound. During the design phase when the grinding mills were purchased in order to reduce the long delivery time for mills, Metso indicated that the grinding circuit has apparently been designed to handle a maximum of about 2,100 t/d. However at this present stage, even a throughput of 1,700 t/d has not been accomplished or proven yet. In addition, all the other unit processes in the plant must also demonstrate that 1,700 t/d can be handled consistently and for extended periods of time, and that includes the crusher plant, all the flotation circuits including the regrind units, the concentrate thickeners, and the concentrate filters in order to confirm that no bottlenecks or other unforeseen problems exist in the plant. Furthermore, with the current reduced throughput, it would be expected that concentrate grades and recovery values would be close to, or meet, the design values during this ramping up to full tonnage stage of the operations, but this had not taken place at the time of the visit, and, until the treatment rate of 1,700 t/d has been achieved and the plant has operated successfully at that rate, the plant cannot be regarded as fully operational. However, as mentioned previously, the mill throughput rate is ramping up with each passing month.

17.5.2 REGRIND CIRCUIT- I

The regrind circuit is producing concentrates after a regrind of selected product streams to a size of 37 μm . This is coarser and different from the design regrind value which was set at 20 μm for all three copper, lead and zinc circuits and which was the design criteria required to achieve the targeted concentrate grade and recovery values.

The initial PRA test work was originally based on the rougher concentrate of each of the three flotation circuits being reground to 20 μm . However, PRA in conjunction with Metso and Yukon Zinc changed the regrind configuration for copper and zinc in particular during November 2007, as discussed in the process description in Section 17.2.6 and shown in Figure 17.2.

The present plant regrind configuration for all the circuits requires an individual review as to whether:

- the design regrind size of 20 μm is still valid for all three circuits particularly given that the primary grind is a P_{80} value of 53 μm .

- the regrind circuit is presently regrinding the most effective product streams (i.e. should the rougher concentrates only be reground, or only certain specific product streams – see Section 17.5.3 for further discussion).
- some mineralogical evaluation of the liberation of the minerals will be required to confirm the required regrind size, probably also from different areas of the mine.
- the effect of the release of additional carbonaceous material during the regrinding process requires quantification and qualification.
- the Wolverine Mine should also be operating the regrind circuit to the design P_{80} values of 20 μm in order to assess the potential benefits for grade and recovery, and also to confirm that the equipment is able to operate at this grind continuously. The downstream effects on settling and filtration should also be carefully monitored. This activity should be undertaken during this ramping up to full tonnage stage, while there is very little operational or production throughput pressure on the plant. Corrective action can then be taken in a more opportune manner.

In a partial answer to the above, test work conducted historically indicated a regrind size of 20 μm for each circuit. Subsequent test work conducted in China and reported in May 2010, also confirmed that 20 μm was the optimal regrind size for the copper and lead flotation circuits, while a slightly coarser size of about 22 μm was required for the zinc flotation circuit. Since the most recent test work was conducted on a sample from the Lynx deposit only, this requires confirmation for material from the Wolverine and Saddle deposits as well, but serves to indicate that the finer regrind to about 20 μm is necessary. Once these investigations have been completed, the regrind circuit can be modified to best suit the operational conditions.

17.5.3 REGRIND CIRCUIT- II

The initial regrind circuit design for each flotation circuit was for the respective rougher concentrates to be reground to a P_{80} size of 20 μm prior to the cleaner stages. This design was subsequently changed by Yukon Zinc after test work was conducted by PRA under guidance from Metso. The changes that were made are shown in Figure 17.2 and this was described in Section 17.5.6.

Part of the rationale for changing the regrind circuit configuration was the fact that an apparently acceptable rougher concentrate grade and recovery was attainable in each of the respective rougher concentrates. This rougher concentrate material would then be easily recovered and upgraded in their respective cleaner circuits without the need for regrinding. It was considered that the middlings streams contained a sufficiently high metal grade but required a regrind to liberate the contained valuable minerals. Only these product streams requiring the regrind step were therefore selected for regrinding. Unfortunately, a full review of all the PRA/Metso directed test work is beyond the scope of this audit on plant performance. However, it does raise two concerns namely:

- that possibly the copper and zinc regrind circuits in particular have been installed and are regrinding product streams of lesser overall metallurgical importance
- do the regrind mills have the required design capacity to treat the rougher concentrate streams, and not a variety of possibly smaller tonnage streams.

The whole regrind circuit requires a review as soon as the circuit has been declared fully operational with respect to the suitability of the product streams being reground, the design treatment rate of the regrind circuits, and the required regrind P_{80} particle size as mentioned before. Interestingly enough, the Central South University Report showed that the P_{80} regrind size of 20 μm was optimal for copper and lead, and about 22 μm for zinc which serves to confirm the regrind size required, although it was a sample from the Lynx deposit only that was tested..

17.5.4 PRE-FLOTATION CIRCUIT

It is apparent that the decision to delete the DMS circuit from the process flowsheet design has had a major impact on plant operations. Whether this can be attributed solely to this decision, or whether the plant feed material composition has changed, or whether other metallurgical process effects are also contributing to the lower recovery values being attained, is unclear at this time in the absence of test data. However, based on the plant performance data to date, the pre-flotation circuit on its own cannot be considered to be a successful substitute for the DMS circuit because of the high losses of copper, lead and zinc and particularly silver and gold, presently being discarded to the tailings in the high volumes of pre-flotation concentrate.

Presently, there is a high pre-flotation concentrate mass recovery and high metal content contained in the pre-flotation concentrate. This high pre-flotation concentrate mass is a result of the much higher than expected levels of carbonaceous material in the feed to flotation. Whether this is a function of mineralogy, mining method, mining the wrong (incorrect) rock type, the recycling of reagents in the recycled process water, or the activation of the carbonaceous material by grinding, has yet to be established.

The current recovery level of pre-flotation concentrate mass and grade values were not envisaged in the original design criteria which predicted that only about 1% mass recovery would be obtained in this circuit. The pre-flotation circuit was even designed with a by-pass facility as it was considered that it would not be necessary to operate this circuit at all times. In the design, it was considered that this 1% pre-flotation concentrate mass would be carrying with it relatively low metal values at the ore feed grade or lower. In other words, the metal losses associated with the pre-flotation concentrate were deemed to be negligible.

17.5.5 DMS CIRCUIT-I

As has already been discussed, the original process design included a DMS circuit which was deleted as a capital cost-saving measure. At that time, both a DMS circuit and a pre-flotation circuit were incorporated into the overall process design. After the deletion of the DMS circuit, the pre-flotation circuit was retained in order to treat the then “stated-to-be” minor amounts of barren/waste low specific gravity rock type material in the feed to the plant. It was anticipated that the pre-flotation circuit would produce a concentrate with a mass recovery of about 1% of the total feed, and contain metal values with grade values equal to, or lower than, the ore feed grade.

The inclusion of the pre-flotation circuit was debated during the design phase since some of the test work conducted indicated that the pre-flotation circuit was superfluous to operational and metallurgical needs. This test work did not indicate that there would be any savings in reagent costs by the removal of the carbonaceous matter in the pre-flotation concentrate. Nor did it indicate the reduced power requirements brought about by reducing barren rock inclusion in the feed to fine grinding. The final process design retained the pre-flotation circuit, but made provision for the inclusion of a by-pass piping option in order that the pre-flotation circuit could be circumvented.

Earlier test work did not indicate the extent of the problem that would be faced by at the Wolverine Mine, namely that there would be a major deleterious effect on plant operations caused by this carbonaceous material which would no longer be rejected by the DMS plant. In fact, routine carbon assaying was not done during the test work phase since it was never recognized to be a problem, or a potential problem. That the plant is presently recording poor results (high metal losses) in the pre-flotation circuit appears to suggest that the samples tested during the laboratory test programs were different to the ore that is presently mined and treated. Alternatively, a plant condition may exist which could be leading to the pre-flotation circuit reporting the high metal concentrate values. In this regard, it is considered that the water currently used for process make-up water may contain sufficiently high amounts of various reagents that may not have been neutralized, oxidized, sufficiently diluted, or dissipated in the tailings pond. These residual reagents could therefore be acting as a collector reagent in the pre-flotation circuit. This aspect should be tested at site as a matter of priority to eliminate the possibility that reagent re-circulation and build-up is contributing to the pre-flotation circuit concentrate loss problem.

17.5.6 DMS CIRCUIT-II

At the time of the audit visit, apparently the Yukon Zinc Company was considering the retrofitting of a DMS circuit at the process plant at the Wolverine Mine. This matter was only made known to Tetra Tech on November 15, 2011 and this was not formally discussed as a planned process option until that time, although it had been mentioned during the plant visit that various metallurgical test programs had been initiated.

However, this installation is worth pursuing once it has been established that all future ore treated at the plant will in fact have a significant carbonaceous content. In this regard, it has to be mentioned that not many carbon analyses were conducted on samples over the years of testing, but that a database of the source of ore and the carbon content could be a valuable source of information.

Since the time of the plant visit, and as mentioned in Section 17.3.4, DMS testwork was conducted in Spain, and the results were sufficiently encouraging for YZC to confirm the viability of the DMS retrofit concept.

The DMS process has been shown to be an effective means of removing low-SG metal-barren material even though no carbon assays were performed on any of the test products. Two sets of tests were conducted by different facilities and these both confirmed the effectiveness of the DMS process, and in addition, there has been the latest successful DMS test work conducted in Spain.

The installation of the proposed DMS circuit should not be seen as fully replacing the pre-flotation circuit and totally resolving all the carbonaceous-related problems. The DMS circuit contains a 1 mm aperture screen which removes all the minus 1 mm fines from the feed material to the DMS process since these fines interfere with the DMS process. This minus 1 mm stream will be discharged to spiral concentrators and a desliming circuit. Although the DMS-spiral concentrator circuit has not been completely defined, the spiral concentrate and the DMS sinks material will be fed to the grinding circuit, and will subsequently report to the flotation circuit. The DMS floats (barren material) and the spiral discard products will be discarded as tailings material. Some slimes material containing carbonaceous matter will enter the main processing circuit. In addition, it is anticipated that each of the regrind circuits will liberate some carbonaceous matter. The extent of the influence of this additionally liberated carbon cannot be estimated at this stage. Similarly, even with the installation of a DMS circuit ahead of the grinding circuit, it is not possible to state with any degree of certainty that the design concentrate product grades and recoveries will be achieved since there are too many other unquantified factors present that may affect the process performance. The removal from the circuit of the barren DMS floats and spiral tailings and slimes generated during this processing stage, together with the layout of this circuit, requires additional design consideration particularly since it has been noted that rotary pumps are enhancing the production of excessive amounts of slime material. It is recommended that the overall DMS circuit design should be undertaken in conjunction with the vendor selected for this process in order to rely on the vendors' experience and testing facilities prior to the installation of this circuit.

However, despite the fact that the DMS circuit design has not been optimized to date, it is apparent that the DMS circuit will contribute significantly to controlling metal values presently lost in the pre-flotation circuit, will improve the efficiency of the other flotation circuits, and will also reduce the feed tonnage to grinding and flotation, resulting in significant operational savings.

The inclusion of a DMS circuit in the overall process will also have added benefit of making available space/tonnage in the main process for treating additional material since the processing sections after grinding have been designed to treat 1,700 t/d of ore. In this regard, it is of interest to consider that the initial design treatment rate of the plant was 1,702 t/d, of which 302 t/d, or 17.67% of the total, was slated to be barren material. This was subsequently reduced to “negligible amounts” which contributed to the decision to delete the DMS circuit from the process design. The plant was re-designed to accommodate a full treatment rate of 1,700 t/d of ore. At the time of the design, it had been stated that the planned amount of the barren material would be 9% which is equivalent to approximately 150 t/d. This is a significant amount which could either be replaced with an increase in feed to the plant, namely up to about 1,870 t/d, and which after removal of the 9% waste will result in the design treatment rate of about 1,700 t/d. Alternatively, less material could be treated in the grinding and flotation circuit but possibly more efficiently. This could result in significant savings in power costs, provided that the rod and ball mills can be fitted with the necessary VFD control to reduce the speed of the mills. On the basis of the recent test results from Spain, an amount of 24.7% of the total feed was rejected, which is equivalent to 420 t of potential reject material. This could be replaced with an increase in plant feed of up to about 2,100 t/d resulting in the design treatment rate of about 1,700 t/d for the rest of the process plant.

The DMS circuit will also have the added benefit of upgrading the ore feed grade which should be beneficial to the subsequent flotation process. The separate treatment of the anticipated high-carbon content minus 1 mm screen undersize material from the DMS feed screen should also be considered. Possible alternative treatment options prior to flotation would include spiral concentrators, or centrifugal concentration to retain the higher density metal bearing fraction while disposing of the lower grade, higher carbon content material.

In summary, control of the barren material, and the associated high-carbon content slimes material, with a DMS/spiral concentrator circuit will constitute a major process design change to the existing mill circuit. It is currently expected that this anticipated plant modification/retrofit should be installed, commissioned and operating resulting in further improvements to the quality and quantity of all three flotation concentrates produced with respect to metal recoveries and grades.

17.5.7 GOLD-SILVER RECOVERY

When the process facilities were designed in 2007, the Wolverine Mine was seen as a copper, lead and zinc mine with by-product silver and some gold contained in some of the concentrates that would contribute to the value of the product. Gold was not a primary product and much of the test work omitted to characterize the metallurgical distribution of the gold in the products, and even the silver results were sporadic. Since that time, metal prices have changed significantly to the point where the major cash contributor to the Wolverine Mine is now the revenue generated by the silver contained in the concentrates. Similarly, the 1.5 to 1.8 g/t Au content of the ore in

the feed material to the plant has a significant metal value and its highest practical recovery should be receiving urgent attention.

The Wolverine Mine is now rightly focussing on gold and silver as being very important revenue generators. The metallurgical characterization of gold and silver should therefore be reviewed and a database of information should be compiled. The geological aspects relating to plant feed material should also include gold and silver. As mentioned in Section 17.3.4, test work is currently underway using a centrifugal concentrator to recover as much as possible of the silver and gold currently lost in the pre-flotation concentrate.

17.5.8 BUDGET/DESIGN RECOVERY AND GRADE VALUES

Further, and continuous ongoing test work needs to be performed particularly during the period that the plant ramps up to full production, and when new ore zones are encountered, so that it is possible to establish whether the original budget recoveries and grades are reasonable. Possibly these budgeted values need to be revised. The reasons are as follows:

- To confirm whether the feed grade of the ore to the plant is the same as the feed grade of the material tested.
- To confirm whether the feed material currently fed to the plant is representative of the mineralogy of the samples which were tested. Each of the three ore zones in the Wolverine Mine namely, Saddle, Wolverine and Lynx deposits, have been reported in the various test work programs as having mineralogical differences and differing metallurgical responses.
- Review the laboratory test results to take into account commercially operational factors. The laboratory test results were accepted as the design values for recovery and concentrate grades, without considering scale-up and operational factors. Since laboratory results are generally achieved under the most ideal processing conditions, the industry convention is to reduce the recoveries attained, depending on the complexity of the operation, but especially during the initial stages of production. By way of example, for the production of a single concentrate, the anticipated initial recovery could be as much as 6% lower than the laboratory/design result. As expertise and familiarity with operations improves, this recovery is reduced to a difference of about 2% after approximately three years of operation.
- Also, the design/budget grade and recovery values have been based on the grades and recoveries obtained by the PRA Test F13. This test did not include a pre-flotation stage, and apparently did not indicate any necessity for its inclusion in the plant design, nor that of a DMS circuit.
- No systematic variability tests across the overall mine deposit were conducted.

- A revision of the design/target/budget values is recommended also possibly including the effect of the origin of the mined material and variations in feed grade. Additionally, future budgets should be varied to take account of the effect of the start-up of operations.

It is expected that the installed DMS circuit will improve plant performance figures to the level of the original budget grade and recovery values.

17.5.9 CARBON ANALYSES

As stated earlier, no systematic carbon analyses were conducted during the original metallurgical test programs. It is therefore not possible to compare the historical test work with the actual results being obtained, or to use carbon assays as a guide for operating the plant, particularly the pre-flotation circuit.

17.5.10 METALLURGICAL LABORATORY

The focus of the metallurgical and laboratory staff appears to be getting the plant fully operational. As mentioned previously, although the laboratory is well-equipped, no metallurgical test work was being conducted at the time of the visit. At this stage it is imperative that the focus be on declaring the plant fully operational and functioning at the design throughput rate of 1,700 t/d, even if this is only done on a campaign basis while the underground mine ore supply is being developed to be able to supply this rate in the future. However, it is foreseen that the planned future series of test work programs will produce significant dividends to the plant operations by providing the results which will serve as the basis for improving mill performance.

17.5.11 RETURN WATER

The pH of the return water at the time of the visit to the process plant was very high. As mentioned, the Wolverine Underground Mine will need to control the amount of shotcrete going with the ore to the plant in order to reduce its contribution to the high pH values being recorded. This may also be contributing to enhancing the conditions where the residual reagents are more active, and/or not being neutralized/oxidized to the anticipated extent. However, the control of the shotcrete reporting with the ore may be a very difficult task from an operational point of view. Some control may be possible by limiting the amount of wasted shotcrete which is dumped into the ore passes, and the changeover to wet shotcrete should also have a positive effect on shotcrete loss and rebound.

The Wolverine Mine may need to install a water treatment plant on the reclaim water recovered from the tailings pond since reagents may not be neutralized or dissipating or oxidizing in the time available between deposition in the tailings dam and the water being returned for re-use in the plant. This may be contributing to the high mass and metal recoveries noted in the pre-flotation circuit, and may also be

detrimentally affecting the copper and/or lead and/or zinc flotation circuits with respect to the concentrate grade and/or recovery.

17.5.12 *MILL FEED*

As previously mentioned, the ore feed material on the stockpile appeared to be fine-grained with no competent rock component. It is unclear whether this condition is a result of the present mining practice, or whether this is an inherent geological feature of the rock type.

It has been commented that at this stage of the mining operations, the bulk of the ore production is coming from primary drifts and fill operations which is inherently finer sized material than that which will be supplied to the mill at a later stage of mining when secondary and tertiary stoping operations are undertaken because there are more blast holes in drift and fill mining than in open stope mining methods. The lack of competency in the material on the ore tip is a mixture of the finer blasted material and the inclusion of waste material from the hanging and footwalls, which also contains the increased amount of carbonaceous material reporting to the mill feed. Changes to underground mining practices, including the use of controlled blasting and using lower velocity explosives, will inevitably help in this regard, as will improved control of the blasting envelope to minimize or avoid the inclusion of excessive waste material.

17.5.13 *REAGENTS*

After the plant has been shown to be able to mill and produce flotation concentrates of a consistent grade and recovery, a review of the reagents used and the consumption rates will be helpful with respect to operating costs and reagent recycling in the process water.

17.5.14 *HOUSEKEEPING*

Housekeeping must be maintained throughout the plant, particularly in the assay laboratory.

17.5.15 *OBSERVATION*

The Wolverine Mine is making a concerted effort to meet and maintain staffing levels during the ramping up period to full production, while all the systems and unit processes are being commissioned and made operational. Consultants have also been used where deemed necessary to facilitate the start-up of mine operations.

18.0 INFRASTRUCTURE

18.1 INTRODUCTION

The facilities and infrastructure for the Wolverine Mine include the following:

- site access road
- airstrip
- fuel storage and dispensing
- power generation and distribution
- process buildings
- assay laboratory
- wet shotcrete plant
- waste rock storage pads
- truck shop
- mining office complex
- administration buildings and dry
- first aid building and mine rescue station
- personnel accommodation and mess (camp).

The facilities have been placed on the site to take into account the functional requirements, traffic patterns, and personnel movements required for an operating mine. Figure 18.1 and Figure 18.2 indicate the location of the facilities at the Wolverine Mine.

Figure 18.1 Site Location General Arrangement

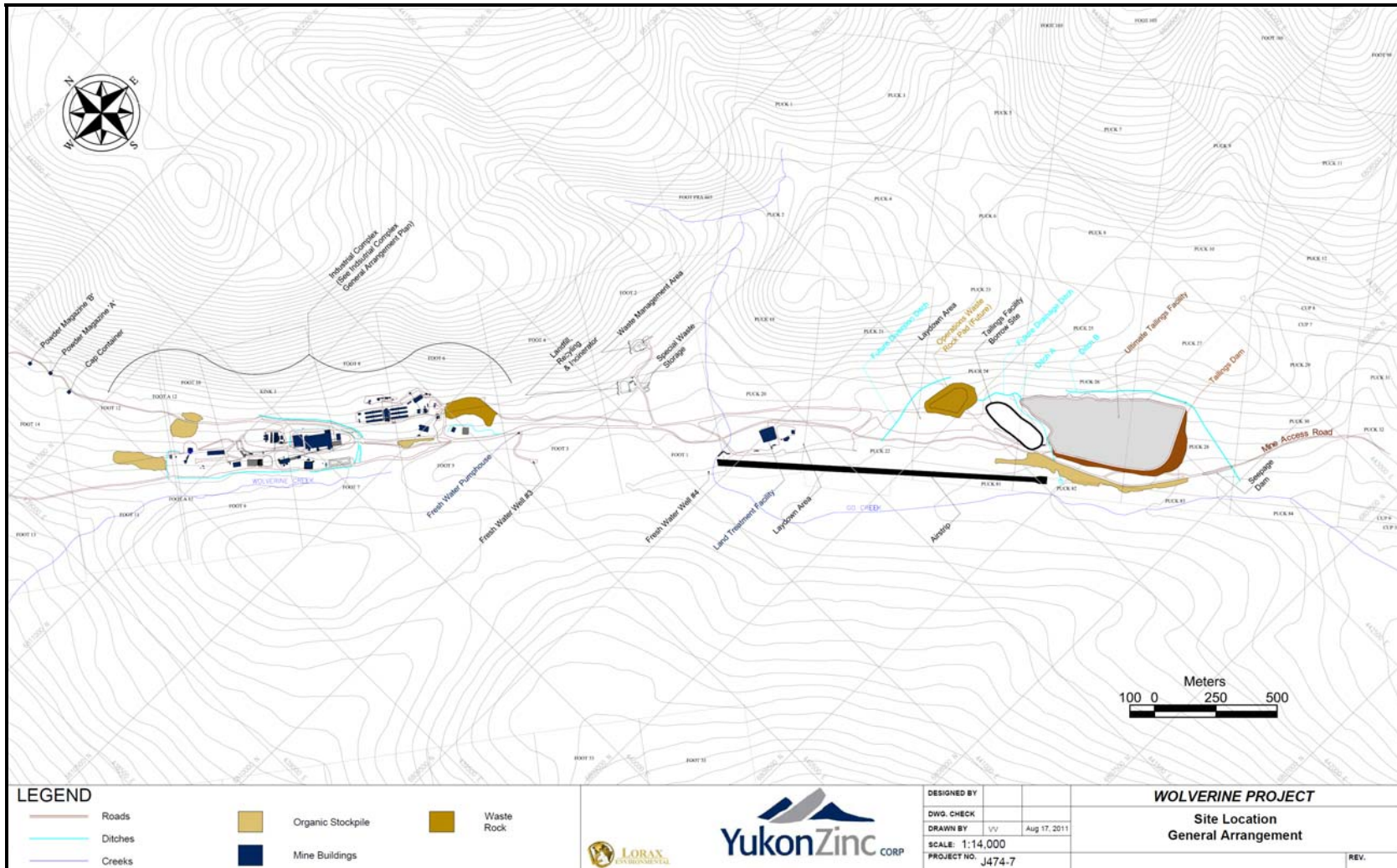
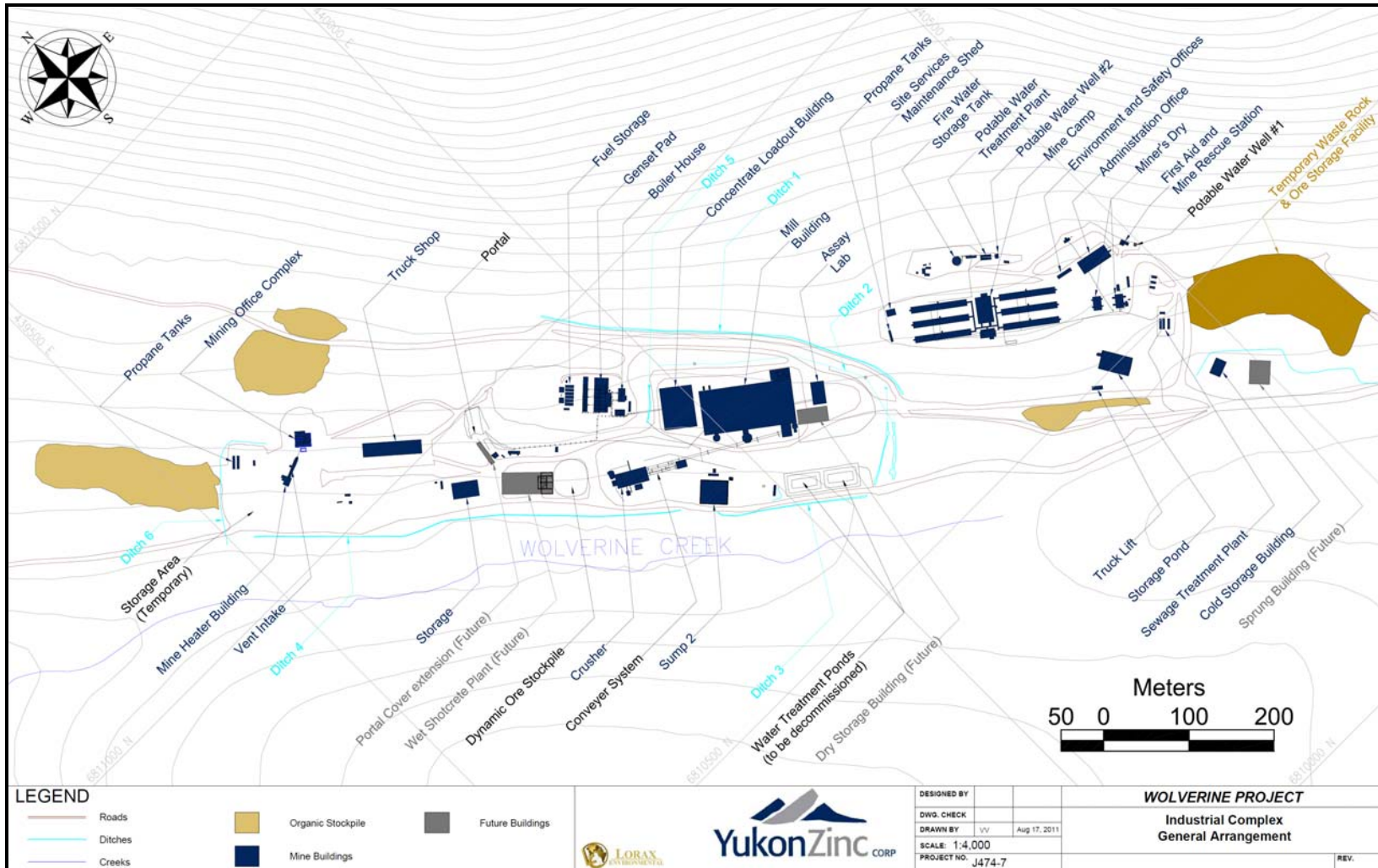


Figure 18.2 Industrial Complex General Arrangement



18.2 SITE ACCESS ROAD

In 2005 and 2006 Yukon Zinc retained Yukon Engineering Services Inc. (YES) to complete the route selection, preliminary design, geotechnical assessment, detailed design, tendering, construction management and quality assurance. YES retained the services of EBA Engineering Consultants Ltd. (EBA) to provide professional geotechnical evaluations of the route and fill sources, undertake a roadway geotechnical testing program, complete laboratory testing of samples taken along the route and from granular sources, and to provide construction recommendations resulting from their investigations and laboratory results. AMEC Earth & Environmental developed the environmental (acid rock drainage/metal leaching (ARD/ML)) testing protocol.

In 2007, Yukon Zinc completed the first phase of access road construction from approximately km 190 of the Robert Campbell Highway (Highway 4) to the mine site. In 2010, Yukon Zinc completed road improvements to ensure the safe use by larger vehicles including concentrate haul trucks and other service and supply vehicles. The main improvements included grade reduction, re-sloping of cut and fill slopes, road widening, the addition of pull out bays in certain locations, addition of a granular surface in specific areas, and progressive reclamation activities. A remotely-controlled gate was installed at the start of the access road to prevent unauthorized use.

The all-weather access road was constructed for use by concentrate haul trucks and service vehicles during the operations phase. The road is a private, single lane road with passing bays, has restricted access and is operated under radio control. The road is used year round with minimal load restrictions. The design of the access road meets Transportation Association of Canada (TAC) RLU 60 Single Lane Resource Road specifications.

18.3 AIR STRIP

The gravel airstrip is approximately 1,340 m long at an elevation of approximately 1,303 masl, with a width of 18 to 30 m. The 280 m of runway at the south end of the airstrip is an extension required to provide for an emergency stop way to allow an aircraft to land on the airstrip with a full passenger load.

The airstrip is intended for restricted use only by day during good visual meteorological flight (VFR) conditions. There are currently no instrument flight rules (IFR) procedures for the area. The site is located within areas of uncontrolled airspace. The airstrip is located at approximately N61° 24' 44", W130° 05' 49". The surrounding area is moderately mountainous with numerous hills with lower wetland areas. The airstrip is currently able to accommodate a plane that seats up to 18 passengers as well as luggage and cargo.

18.4 DIESEL FUEL STORAGE AND DISTRIBUTION

Diesel fuel requirements for the underground mining equipment, surface mobile equipment, and power generators are supplied from six diesel fuel storage tanks (75,000 L capacity each) and an 8,500 L gasoline tank. These tanks are located northeast of the process plant (Figure 18.2) and gravity flow is used to supply the end use points. Fuel is delivered by 45,000 L capacity B-Train tanker trucks. Storage is based on two weeks of reserve in the event of road problems and restrictions. A fuel truck transports diesel fuel and lubricant to the various mobile equipment as required. Fuel for the power generators flows by gravity and is stored in day tanks located at the powerhouse.

18.5 POWER GENERATION AND ELECTRICAL DISTRIBUTION

18.5.1 GENERAL

The electrical system has been sized to take into account the mining loads, process loads, and ancillary loads, such as the camp and service facilities. Spare capacity is available within the electrical distribution system to allow for stand by and limited future increase in load. An additional generator is planned to be installed in 2012 to allow for planned maintenance during times of maximum load.

18.5.2 POWER GENERATION

The Wolverine Mine is situated in a remote location and is served with on-site power generation. The electrical system is designed in accordance with the following:

- Canadian Electrical Code
- Use of Electricity in Mines Regulations(CSA M421-00)
- Institute of Electrical and Electronics Engineers Standards.

PLANT LOAD

The total connected plant load is estimated to be 10,870 kW. Table 18.1 shows the operating loads.

Table 18.1 Operating Loads

Time Period	Maximum Demand (kW)	Average Demand (kW)	Total Period (kWh)
Summer	7,741	6,558	28,724,641
Winter	7,897	6,826	29,896,245
Total	-	-	58,620,886

The load is served by eight diesel generator sets, each rated 1,200 rpm, 1.26 MW continuous, 1.45 MW prime power generating at 4,160 V for a total installed generating capacity of 10.08 MW continuous, 11.60 MW prime power.

The generators are configured for a spare set (n+1 criteria) to allow for one set to be down for standby or preventive maintenance and service. This provides redundant capacity which is critical for a remote site.

SINGLE LINE DIAGRAM

The single line diagrams (A0-18-001, A0-18-002 and A0-18-003) are shown in Figure 18.3, Figure 18.4, and Figure 18.5 and show the overall site power distribution.

Figure 18.3 Overall Site Single Line Diagram (1 of 3)

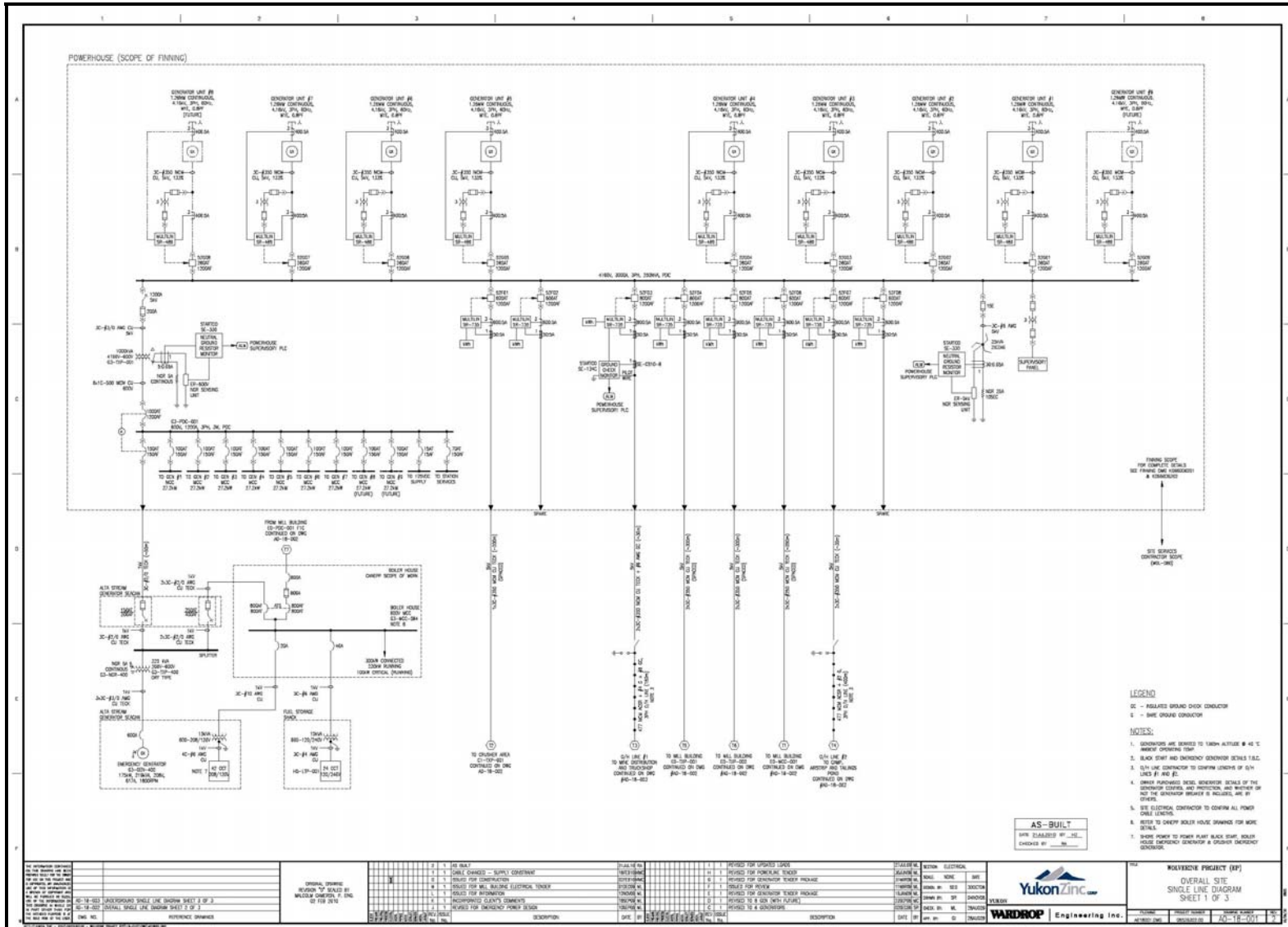
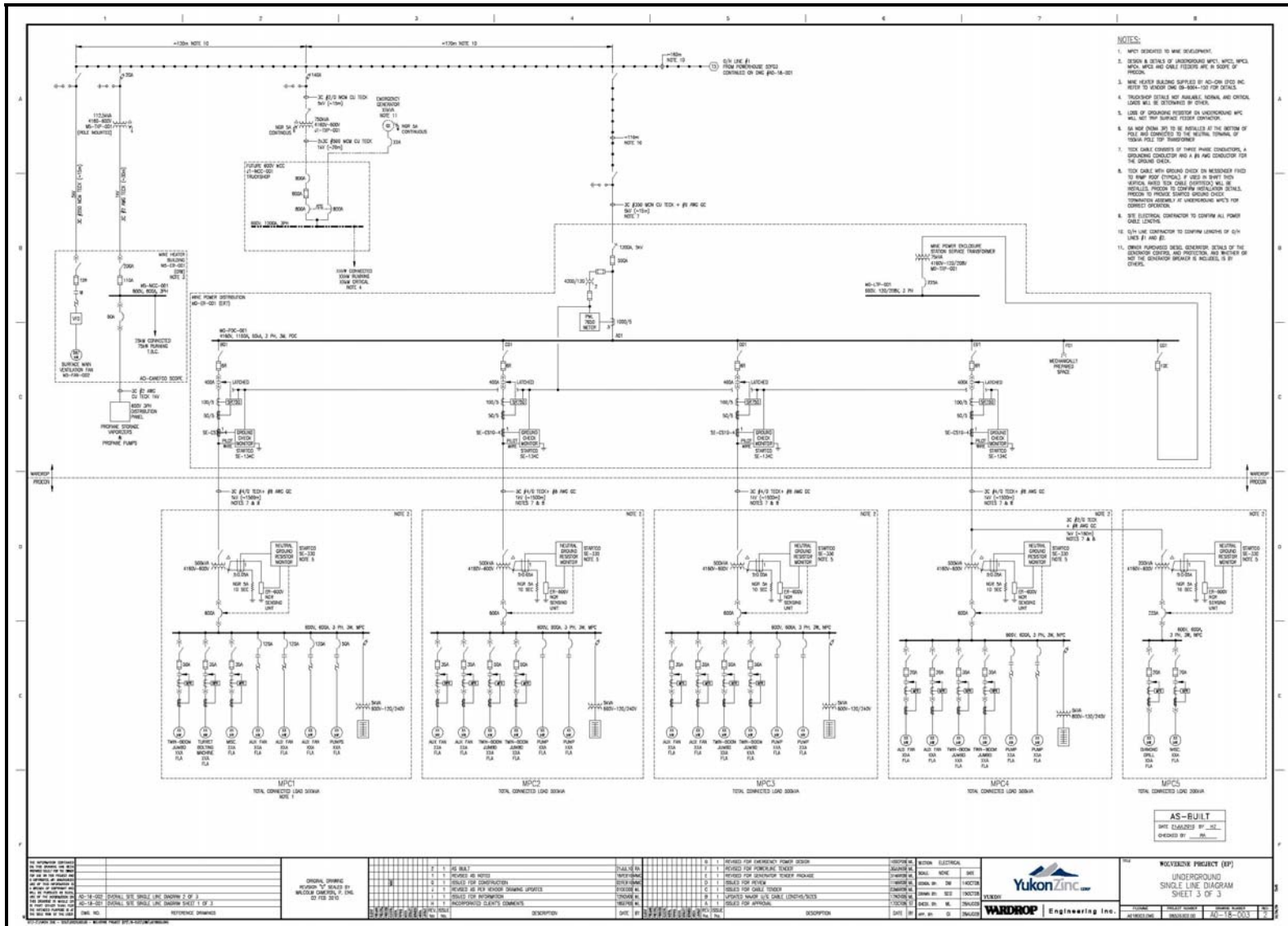


Figure 18.5 Overall Site Single Line Diagram (3 of 3)



VOLTAGE LEVELS

Power is generated and distributed around the site at 4,160 V. Large motors (generally above 225 kW) are connected directly to the 4,160 V system, whereas smaller motors (generally between 200 to 0.5 kW) are connected directly to the 600 V system.

High bay lighting is provided at 347 V. All other loads, including heat tracing, are provided at 120 or 208 V.

POWER DISTRIBUTION

Power is distributed using 5 kV cables in trays for nearby locations and overhead line for remote loads. All 5 kV cables are shielded.

The main 5 kV switchgear is located in the generator powerhouse.

LARGEST MOTORS

The large motors connected to the 4,160 V system are as follows:

Primary Ball Mill.....	1100 kW
Mine Ventilation Fan.....	600 kW
Zinc Regrind Mill.....	355 kW
Primary Rod Mill	400 kW
Vacuum Pump.....	300 kW
Plant Air Compressors x 4.....	260 kW

The Glycol pumps are fed from the powerhouse 600 V system.

EMERGENCY POWER

The camp is equipped with a standby diesel power generation system in case of an outage of the overhead line, which is fed from the powerhouse. Other emergency generators provide power to critical systems such as the reclaim and booster pumping stations.

OPERATING COSTS

The largest operating cost in the power system is for diesel fuel. The expected energy conversion is about 3.3 kWh/L of diesel. Hence, the annual consumption of diesel for power generation is expected to be: 58,620,886 kWh x 1 L / 3.3 kWh = 17,763,905 L.

HEAT RECOVERY

The generators are equipped with heat recovery systems. For generators of this size, the maximum recoverable heat is about 80% of the electrical output. Hence the annual recoverable heat is estimated to be approximately $0.8 \times 58,620,886 \text{ kWh} = 46,896,708 \text{ kWh}$ at $3,412 \text{ BTU/kWh} = 160,011 \text{ MBTU}$.

Current uses of the waste heat within the glycol system include heating of the following buildings:

- process buildings (see Section 18.6), which is complemented by the auxiliary boiler house
- power house
- crushing building
- assay laboratory.

In addition to generator heat being utilized, waste oil is consumed in the waste oil burner to heat the truckshop.

Emergency propane heating is also available.

UNDERGROUND ELECTRICAL AND COMMUNICATIONS

Various underground routes are established throughout the site for the purpose of power distribution and communications.

18.6 PROCESS FACILITIES

The process facilities consist of a crusher building, mill building and concentrate load-out (CLO) building. Feed conveyors connect the crusher building to the mill (Figure 18.2) and transport crushed ore to the rod and ball mills. Ancillary facilities include a wet shotcrete plant, assay laboratory, mining office complex, truck shop and camp, which also includes administration offices and a first aid and mine rescue station.

The CLO building has a 15 t electric overhead bridge crane to service all areas. The crane has sufficient capacity to assist in the maintenance of the grinding mills. A 7.5 t overhead crane services the reagent and concentrate storage areas. The mill building has several full-size overhead doors to allow access by service vehicles. Man doors are provided in accordance with building code requirements.

The mill process produces tailings that are pumped from the mill to the tailings management facility through an 8 in diameter HDPE pipeline. The tailings pipeline is installed in a shallow ditch alongside the access road so that any leakage or spillage

can be contained. The tailings discharge pipeline is connected to a distribution point near the centre of the dam.

Almost 99% of the mill water requirements are met through recycles supernatant from the tailings impoundment. A barge-mounted pump station in the tailings pond pumps water to the reclaim water tank located inside the mill building via insulated and heat-traced 8 in diameter pipe following the same route as the tailings discharge line.

18.6.1 *WET SHOTCRETE PLANT*

The wet shotcrete plant is installed under a fabric structure west of the mine portal that manufactures shotcrete for use in the underground mine using aggregate sourced from local borrow pits. The aggregate is combined with cement, micro-silica, fibre, steel fibres and sand to form the shotcrete. The on-site shotcrete plant eliminates the need to import dry shotcrete from a supplier in bags, reduces the resulting cost of transportation. Also, manufacturing shotcrete on-site prevents having to dispose of the used shotcrete bags, as these are a one-time use item.

18.6.2 *ASSAY LABORATORY*

The assay laboratory is located adjacent to the mill building (Figure 18.2). It is an 125 ft x 50 ft pre-engineered structure, similar to the process buildings. The total building area is 580.6 m² containing four offices, a lunchroom, grinding/screening room, metallurgical laboratory, sample preparation area, wet laboratory section, instrument room, fire assay room, and washrooms for men and women.

18.6.3 *TRUCK SHOP*

The truck shop is located northwest of the mine portal (Figure 18.2), and is a pre-engineered structure which houses an overhead crane, wash bay, with oil separator system and under slab waste oil radiant heating system. The truck shop is a 226 ft x 52.5 ft pre-engineered structure, with a cement floor to collect drippings which are directed to a sump for collection.

18.6.4 *CAMP AND ADMINISTRATION OFFICES*

Camp infrastructure at the Wolverine Mine consists of six 41-man dormitories, a kitchen, recreation hall, administration office, environment and safety offices, first aid office, mine rescue station and dry facilities. Additional support infrastructure includes a maintenance shed, firewater tank, potable water treatment plant, communication station, and sewage treatment plant. The sewage, potable water and fire water lines are buried pipelines installed prior to the placement of the dorm buildings. Power is supplied to the camp via overhead power lines connected to the power generation station.

There is a buried firewater loop installed throughout the industrial complex and camp with fire hydrants for full coverage of all facilities. The loop extends to the fuel storage facilities and to the mining office complex. A jockey pump installed which will start supply water flow when required. A diesel operated pump will also be installed to start in case of power failure when there is a demand for firewater. The fire water tank is located at the camp complex on the same pad as the potable water treatment plant (Figure 18.2) and holds 400 m³ of water.

18.7 WATER MANAGEMENT

The ore processing facilities use water reclaimed from the tailings impoundment for much of the requirements. Fresh water from surface wells is used for process requirements where reclaim water is not suitable. Other water sources include underground mine discharge and surface groundwater wells for potable water (potable water well #2 and #2 in Figure 18.2). Collection ditches collect surface runoff from the industrial complex and divert it to collection Sump #2 (Figure 18.2). Diversion ditches divert clean surface runoff from upslope of the industrial complex around the site and into Wolverine Creek (Figure 18.2). Similarly, diversion ditches are installed at the tailings facility to divert the water around the facility and into Go Creek (Figure 18.1).

A positive water balance exists at the site and excess water is stored in the tailings management facility (TMF). Tailings supernatant is predicted to have high concentrations of copper, lead, selenium, silver, and zinc and will be treated at a water treatment plant prior to release to Go Creek. Water will be stored in a retention pond to confirm that water quality meets the requirements of the Type A Water Use License QZ04-065. The water treatment plant process is described further in Section 18.8.

18.8 WATER TREATMENT

The TMF has sufficient capacity to store approximately three years of process water and tailings within the impoundment. Accordingly, water treatment operations, and effluent discharge, will not be required until 2013 or 2014 depending on the volume of tailings discharged to the pond.

A biological reduction treatment system will be used to produce effluent of acceptable water quality for discharge. The bioreactor will be designed to treat up to 15 L/s (approximately 55 m³/h). The bioreactor tanks will contain a granular carbon substrate. Water movement through the bioreactor system will occur in a modified plug up-flow configuration. The water from the tailings pond (influent) will be heated during certain periods of the year to ensure appropriate microbial growth. The water will also receive a nutrient amendment designed to facilitate the growth of the microbial community inoculated on the carbon substrate. The water discharged from the bioreactor will be fed through a sand filter to remove suspended solids.

Treated water from the water treatment plant will initially be directed to a retention pond for testing. If water in the retention pond does not meet discharge limits, the water will be recycled back to the tailings impoundment or the water treatment plant until treatment is sufficient. Water meeting permitted discharge limits will be released to Go Creek at the final discharge location. Testing will be conducted routinely to ensure discharge limits are achieved. Initial bench scale work on the tailings water treatment plant has been completed, with a pilot scale operation being completed in summer 2012. Initial results indicate successful removal of copper, lead, selenium, silver, and zinc.

18.9 HEALTH, SAFETY, ENVIRONMENT AND SECURITY

Yukon Zinc has overall responsibility for the development of HS&E and security policies and implementation. A project HS&E plan details the requirements for HS&E, which is in full compliance with the laws and regulations of Yukon. All contractors, sub-contractors, suppliers, vendors and consultants are required to comply with these requirements. Yukon Zinc employs both a health and safety superintendent and an environmental superintendent to manage HS&E and security at the Wolverine Mine. The following sections give an overview of the staffing and policies in place to ensure a safe working environment.

18.9.1 INDUSTRIAL HYGIENE

Industrial hygiene policies at the Wolverine Mine require personal protective equipment (PPE) to comply with Canadian Standards Association (CSA) Standards including Z94.2-07 Hearing Protection Devices, Performance, Selection, Care and Use and Z94.4-02, Selection, Use and Care of Respirators. Respirator selection is based on potential hazards of the operation identified in a qualitative risk assessment. A site wide comprehensive exposure monitoring plan is in place to ensure workplace protections meet or exceed the requirements of the Yukon Occupational Health and Safety Act and Health Regulations. In addition to PPE, engineering controls are implemented to reduce exposures to potentially dangerous substances. Worker health for exposures to lead and silica is monitored through medical examinations in accordance with Yukon standards.

18.9.2 SECURITY

Access to the Wolverine Mine is limited to the access road and the airstrip. As such, security is focussed on these two entry points. Entrance via the access road is radio-controlled with a locked gate. On arrival at site, either via road or air, all site visitors are subject to search of personal belongings for drugs or alcohol and violators are removed from the site immediately. Instruction and orientation on Yukon Zinc policies and procedures is provided for visitors (short one hour orientation) and for workers (one day orientation). Underground personnel have a further orientation on the policies and procedures specific to underground operations.

Safety and security services are available 24 h/d. Additionally, camp facilities and administrative offices are provided with automatic and manual fire suppression and alarm systems. Fire extinguisher training and an ample supply of maintained fire extinguishers provide initial response capacity to incipient fires. The surface site has a heated fire suppression 6 in ring main with a dedicated firewater tank on the hill above. On the fire main are fire hydrants with indicating control valves and strategically placed monitors to suppress and contain large fires throughout the surface facilities. Hand fire hose connections are provided by building entrances and hoses are maintained in heated locations. Emergency response personnel are trained in the use of these suppression systems and on call while on site. Appropriate liquid fuel fire suppression capacity is provided in the fuel storage area and in the commercial grade kitchen.

18.9.3 *FIRST AID*

The Wolverine Mine employs one certified primary care paramedic, one advanced care paramedic and a further three certified Industrial First Aid Level 3 with transportation endorsement. The medical staff supports the general health of the camp including water quality monitoring, seasonal flu vaccinations and immunizations for blood borne pathogen exposures. In addition, supervisors and emergency responders have a minimum of first aid training. A fully equipped ambulance for advanced life support is maintained on site and is supplemented by an advanced first aid transport. In addition to the full-time medical staff, the Wolverine Mine retains the services of a medical director based in the nearby community of Watson Lake. Patient transport to advanced care facilities is provided by Med-Evac air ambulance services (helicopter and fixed wing) based out of Whitehorse and Watson Lake. Specific protocols are in place to determine escalation of level of care, evacuation and destination.

The underground and mill operations have daily safety meetings before commencement of operations, plus weekly formal safety meetings. Safety meetings are both department and job specific. The Wolverine Mine has a site wide Occupational Safety and Health Association (OSHA) committee that meets monthly and provides peer review of incidents during the prior month, plus a formal walk through and review of the safety aspects of all the areas of the mine, involving worker representatives from all departments and senior safety personnel.

18.9.4 *TRAINING*

The Wolverine Mine has established formal training programs in workplace safety, environmental protection, equipment operation, emergency response, and job procedures.

Job specific training is provided by subject matter, using in house, or consultant experts, in specific operating areas as required and is fully documented. Training is an ongoing process with training programs in any new procedure, preceded by a Job Hazard Analysis (JHA). The training specific to underground mining operations is

provided by a full time mine trainer for the purposes of improving operating standards in the underground mine.

All employees are trained in the use of risk based JHA techniques to evaluate work procedures and identify and mitigate threats. JHA's are developed for all standard procedures and reviewed periodically with employees. Unique conditions or non-routine jobs require the use of a JHA to evaluate the work conditions before the task commences. The site has established special hazard training programs for work involving hot work, confined space, fall protection, working alone, energy isolation and hazardous materials management.

The mine maintains a rescue centre with full mine rescue re-breathing sets, plus air breathing sets and also hazmat suits and the appropriate support equipment, including hydraulic cutting and lifting equipment and other standard emergency response gear. Mine emergency response training is provided by trainers certified in mine rescue training. Training in maintenance of breathing apparatus is provided by manufacturer certified trainers.

18.9.5 *ENVIRONMENT*

Environmental protection is a corporate priority for Yukon Zinc and it is the company's commitment to strive to preserve and protect the environment for the future, while providing a safe and responsible operation. Yukon Zinc has developed and operates the Wolverine Mine to minimize the geographic extent, duration and magnitude of effects of the mine on valued ecosystem and cultural components. The mine has been designed for eventual permanent passive closure and will be managed to minimize risk and potentially harmful incidents.

19.0 MARKET STUDIES AND CONTRACTS

19.1 LONG TERMS SALES

Yukon Zinc's long-term sales plan for zinc, lead and copper concentrate is to market to suitable buyers and smelters in the world. There are opportunities to optimize and enhance Yukon Zinc's existing sales contracts from their current position during ramp up to the planned production and higher concentrate qualities. This will enable Yukon Zinc to open more avenues with smelters and buyers. Yukon Zinc has 10 long-term proposals in place from smelters and international trading companies for mine life production, based on production of the 2010 design specification of concentrates.

19.2 SHORT TERMS SALES (1-2 YEARS)

With the current quality of concentrate and limited current production quantities during ramp up of production, Yukon Zinc has sold zinc and lead concentrates to only one international buyer, Trafigura AG with a quantity of 4,000 wmt of zinc and 5,800 wmt of lead. Yukon Zinc also sold 1,500 wmt of copper to Trafigura and 600 wmt of copper concentrate to local smelter Xstrata Canada. Additionally, Yukon Zinc has finalized an extension agreement with Trafigura for an additional 40,000 wmt of zinc, 7,000 wmt of lead and 2,000 wmt of copper concentrate for shipment in 2012.

19.3 TRANSPORTATION OPTIONS AND COSTS

Transportation of bulk zinc and bagged lead concentrates is by truck from Wolverine Mine to Stewart Port, then by bulk vessel from Stewart Port to China Port. Bagged copper concentrate goes by truck to Stewart and then by rail from Stewart to local smelters.

The distance from the Wolverine site to the port at Stewart, BC is 864 km. The transportation cost to Stewart presently reflects bulk hauling of zinc at \$100/t and bag hauling of copper and lead at \$100/t. Port handling and loading services with Stewart Bulk Terminals is at a rate of \$10/t. Shipping bagged concentrate does provide Yukon Zinc with cost advantages depending on markets. The operation has the ability to ship bagged concentrate from Stewart via barge directly to Prince Rupert, BC and/or Vancouver, BC ports.

19.4 PAYABLE TERMS

ZINC CONCENTRATE

Payables:

Zinc: 85% subject to a minimum deduction of eight units at the LME SHG settlement London Metal Exchange.

Silver: Deduct 3 oz/dmt and pay for 70% of the balance at the LBB Spot

LEAD CONCENTRATE

Payables:

Lead: 95% subject to a minimum deduction of three units

Silver: 95% subject to a minimum deduction of 50 g/dmt

Gold: 95% subject to a minimum deduction of 1 g/dmt

COPPER CONCENTRATE

Payables

Copper: 96.5% subject to a minimum deduction of one unit

Silver: 93%

Gold: 92% subject to a minimum deduction of 1 g/dmt

The payable is on the basis of the current quality from the early stage production ore and as the quality of the concentrates increase with Yukon Zinc's production ramp-up there are significant opportunities to realize higher payables.

Marketing costs in the financial model are estimated to be 0.5% of gross revenue less smelting, refining and transportation costs.

19.5 ROYALTIES

There are three royalty agreements currently in effect:

- Atna Royalty:
 - The project is 100% owned by Yukon Zinc subject to a sliding scale royalty payable to Atna (Royal Gold) on that portion of the NSR attributed to payable silver and gold, that is gold value net of the estimated precious metals share of smelter treatment and refining

charges, plus concentrate transport costs. The royalty rate is zero at silver prices less than US\$5.00/oz, 4% if silver prices are between US\$5.00 and US\$7.50/oz and 10% at silver prices over US\$7.50/oz (refer to Table 19.1).

Table 19.1 Atna Royalty Rate

Silver Price	Nominal Royalty Rate (%)
<US\$5.00/oz	Nil
US\$5.00/oz to US\$7.50/oz	4.0
>US\$7.50/oz	10.0

- Equity Exploration:
 - A NSR royalty of 0.5% on all metal production from the Foot claims is payable to Equity Exploration. Yukon Zinc can buy out this royalty entirely with a payment of \$500,000. It is assumed that Yukon Zinc will buy this royalty out in 2012.
- Kink Royalty:
 - The Kink royalty is currently held by Strategic Metals (formerly Nordaq Resources) and consists of a NSR royalty of 1% on all metal production from the Kink 3 claim. The royalty reduces to 0.5% after cumulative payments of \$500,000 have been made. The Kink 3 claim covers approximately 40% of the Wolverine deposit.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL PERMITTING

The Wolverine Mine is fully permitted for production. An environmental assessment for the Wolverine Mine was completed in 2006. QML-0006 was issued December 5, 2006 by the Yukon Department of Energy, Mines and Resources. Type A WUL QZ04-065 was issued October 4, 2007 by Yukon Environment. These two licenses outline the major regulatory and operating requirements for the Wolverine Mine. Various minor permits, including permits issued by Yukon Environment, Yukon Workers' Compensation Health and Safety Board, and the Canadian Nuclear Safety Commission outline operating conditions for activities at the Wolverine Mine not regulated by the QML and WUL.

The licenses and the QML in particular, stipulate the preparation of numerous plans for review and approval. Yukon Zinc has satisfied these requirements with the approved versions listed in Table 20.1.

Table 20.1 Currently Approved Operating Plans as of December 6, 2011

Plan Name	Version/Revision Number	Approval Date
General Site Plan	2011-05	September 9, 2011
Reclamation and Closure Plan	2009-03	May 17, 2010
Heritage Resource Protection Plan	2007-01	December 11, 2007
Monitoring and Surveillance Plan	2011-03	November 30, 2011
Spill Contingency Plan	V2010-03	March 25, 2011
Waste Management Plan	V2010-03	March 25, 2011
Wildlife Protection Plan	2009-01	May 13, 2009
Mill Operation Plan	2010-02	June 4, 2010
Mine Development and Operation Plan	2010-02	August 9, 2010
Tailings and Infrastructure Design and Construction Plan	2009-02	March 20, 2009

20.2 ENVIRONMENTAL MONITORING AND REPORTING

An extensive environmental monitoring and surveillance plan for the mine has been developed and implemented. The most recent version of this plan is dated October 2011. This program includes the following elements:

- surface water including water quality sampling, hydrology and acute lethality testing
- groundwater including water quality sampling, water levels and water table (piezometers)
- weather including temperature, precipitation, snowpack, and evaporation;
- sediment, benthos and periphyton
- tailings storage facility monitoring
- engineering inspections for physical stability
- environmental effects monitoring (as per *Metal Mining Effluent Regulations* (MMER) (MMER, 2006))
- geochemical test work
- underground mine monitoring
- reclamation program effectiveness monitoring and Bio-pass system design.

The Wolverine Mine became subject to the MMER under the *Fisheries Act* on June 3, 2009, based on being a “mine under development”. The first Environmental Effects Monitoring (EEM) biological study design was submitted in June 2010, and revised in October 2010 following a review by Environment Canada. Although some effluent discharges of mine contact water occurred in 2009, this was discontinued in October 2009 with the completion of the tailings storage facility, as all contaminated or contact water has since been directed to the facility for storage. The first EEM interpretive study was completed in September 2011, with the study report submitted to Environment Canada in December 2011. The second EEM study design is due for submission in Q1 2013.

20.3 ENVIRONMENTAL MANAGEMENT

The Wolverine Mine has specific obligations to protect wildlife and heritage resources, to manage waste generated by the project, to manage hazardous materials, and to manage the quality and quantity of any liquid effluent discharges. These obligations and the associated management measures are detailed in the applicable plans (Table 20.1). Specific matters related to water, waste rock, and tailings management are discussed below.

20.3.1 *WATER MANAGEMENT*

Process water, underground discharge, collected surface runoff and treated sewage effluent reports to the TMF. Water is reclaimed from the TMF for ore processing. The TMF is currently operated as a zero discharge facility. The TMF has approximately three years of storage, at which time water will be treated using a bioreactor based water treatment plant, as outlined in Section 18.

20.3.2 *WASTE ROCK MANAGEMENT*

Waste rock brought to surface is considered as potentially acid generating (PAG) and is temporarily stockpiled on two lined pads. Waste rock brought to surface is returned to the underground as backfill during operations, or before mine closure. No waste rock will be closed out on surface. The schedule for returning waste rock to the underground remains to be finalized.

The original project plan included provision for one waste rock stockpile (as permitted by Type B WUL QZ01-051, and mining land use permit LQ00140), based on the expectation that rock could start being returned underground as backfill in 2010-2011. However, with the pad near full in summer 2011 and underground development not yet to a point that would allow waste rock to start being used as backfill, the QML was amended to allow for the establishment of a second waste rock stockpile with a total capacity of 325,000 t. The construction of the second waste pad was completed in December 2011.

Runoff from the waste rock stockpiles is collected in sumps and transferred to the TMF for storage and reclaim to the mill process.

20.3.3 *TAILINGS MANAGEMENT*

The ore body is a sulphide bearing rock and the tailings are PAG if exposed to oxidizing conditions. The TMF for the project has been designed to contain approximately half the tailings volume (1,267,000 m³) generated from the mill. The remainder of the tailings volume will be returned to the mine underground as paste backfill.

The TMF has been constructed in a natural depression on the northeast valley slope of Go Creek. The TMF is being developed in two stages, with the 19.5 m high starter dam constructed in 2009/2010 and the raise to the ultimate height of 26.5 m scheduled for the 2012 construction season. The TMF is lined with a 40 mil linear low density polyethylene (LLDPE) geomembrane under the impoundment and along the interior face of the dam. The TMF incorporates an emergency spillway for routing of the 1:10,000 return period flood event; seasonal storage for average conditions of 60,000 m³; 200-year flood storage consisting of 200 year wet month precipitation with snowmelt of 37,500 m³; and 2 m of freeboard, of which 0.5 m is used for passage of the 1:10,00-year event. Tailings deposition is by end-of-pipe

discharge, creating a shallow-sloped (1% slope) beach. Tailings are kept wet through a combination of new tailings deposition and rising water level.

Tailings are transferred to the TMF and reclaim water is returned to the mill in polyurethane insulated double walled HDPE pipe, with the inner pipe being 8 inches in diameter and the outer pipe being 12 inches. The pipes are not heat-traced, but have the ability to be if necessary. The pipes generally grade toward the tailings facility, so that should a breakage or a shutdown of the mill occur, the tailings and effluent within the pipes will drain via gravity back to the TMF. A water recirculation system is to be installed to avoid pipe freezing during extended winter shutdowns.

20.4 CLOSURE AND RECLAMATION

The closure and reclamation plan for the project has been approved by the Yukon Government Department of Energy, Mines and Resources (Table 20.1). The plan details the measures to be taken during both a temporary suspension of operations and permanent closure of the Wolverine Mine.

20.4.1 TEMPORARY SUSPENSION

A temporary suspension of mining and processing activities could result when factors such as changing market conditions or mine related factors occur. A state of inactivity may evolve into a state of permanent closure if prevailing conditions for the resumption of operations are not favourable. QML-0006 also stipulates that where temporary closure exceeds three continuous years, the site will be considered permanently closed and the final closure plans must be implemented.

Closure activities for a temporary shutdown have been planned to ensure that all safety and environmental standards are achieved. During a temporary closure, Yukon Zinc intends to be a responsible steward of the site and demonstrate its commitment to re-opening the site by continuing to:

- ensure physical and chemical stability of the site
- monitor and maintain buildings and facilities
- maintain the site and main access roads
- maintain security and access protocols
- dewater the mine to prevent flooding of the underground workings
- collect site runoff from the industrial complex and waste rock pad
- operate and maintain water management structures and treatment facilities to ensure no uncontrolled discharges occur.

Therefore, surface facilities will only be accessible to Yukon Zinc personnel, or designated representatives, and equipment and facilities will remain essentially intact on site.

20.4.2 *PROGRESSIVE RECLAMATION*

Yukon Zinc has committed to progressive reclamation (i.e., reclamation in the course of operation) of several project elements as soon as possible in the project cycle, including:

- construction phase laydown areas
- footprints where temporary structures and redundant components were located
- borrow sites
- the downstream face of the tailings dam
- selected areas along access corridors.

A key element of the progressive reclamation plan is the planned decommissioning of the waste rock stockpiles. Yukon Zinc has committed to returning waste rock to the underground and reclaiming the waste rock stockpiles. The schedule for this phase of progressive reclamation remains to be established.

Implementation of progressive reclamation will require completion of the reclamation research program to identify borrow areas for reclamation materials and finalize re-vegetation measures with respect to successful and appropriate seed mixes and any necessary fertilization or soil amendments that may be required to support establishment of a self-sustaining vegetative cover.

20.4.3 *CLOSURE*

The timing of facility closures is dependent on a number of factors including the purpose of the facility and its future use and environmental considerations. Site decommissioning activities are anticipated to commence during the final stages of operations after Year 9. Table 20.2 provides a list of anticipated activities that will be required during the decommissioning and closure period. Decommissioning of the waste rock pad is not included as this is currently scheduled for Year 4.

A number of personnel will be required onsite to implement the various decommissioning, closure and reclamation tasks. The water treatment plant will remain in operation to treat excess water accumulating in the TMF during the early closure phase until TMF water meets discharge criteria. The following infrastructure and equipment will remain operational or onsite to support water treatment plant operations:

- onsite roads to the camp, industrial complex, tailings facility and airstrip

- main access road and associated light duty vehicles
- discharge pipelines and pumps
- fuel storage facility
- power generating facility with adequate capacity to power the water treatment plant, pumping systems, and camp
- small maintenance workshop
- reagent storage facility
- communication system.

During decommissioning and the early closure period, the soils will be tested for contaminants in all areas where ore, concentrate, waste rock, solid wastes, special wastes, fuel and chemicals were stored or handled at the site. If contamination is found, the contaminated soil will be removed from the area and either temporarily disposed of in the land treatment farm near the airstrip or hauled directly off site to an approved facility. The selected disposal method will be in accordance with the Yukon Environment Act and Contaminated Sites Regulation, and Special Waste Regulation.

Once infrastructure is removed, slopes will be stabilized by contouring and leveling to provide land forms which conform to the surrounding terrain and provide suitable seedbeds. Erosion features will be minimized on re-sloped surfaces, runoff will be diverted away from steep slopes, and settling ponds and diversion ditches will be used as necessary.

Table 20.2 Activities Associated with the Decommissioning and Closure Phases

Component	First Year of Closure and Decommissioning	Year 2 to Year 3 of Closure and Decommissioning Phase
Mine workings	Install hydraulic plugs as main ramp is backfilled to stratify underground water Cement/grout ventilation opening Install portal barrier	
Industrial complex area	Dismantle and remove mill building and supporting infrastructure, including buried tanks, pipes and underground services Cover concrete foundations with overburden and re-vegetate Market mining and mill equipment Transport explosive and cap magazines offsite Deactivate most gensets leaving adequate power for the camp and water treatment facility Conduct remediation programs Recontour, replace organic layer and seed disturbed areas Remove freshwater pond	Construct portal discharge ditch to Biopass Remove site runoff collection ditches once area is reclaimed and vegetated Remove remaining gensets and transmission lines once all activities are complete Remove demolition waste or dispose of in authorized landfill. Remove all hazardous waste from site Market and dispose of all assets once support for water treatment and reclamation activities are no longer needed
Tailings facility	Dismantle tailings discharge pipeline and dispose of in tailings facility Decommission diversion ditches A and B Cover tailings with 0.5 m coarse inert material during winter	Cover tailings with 0.5 m coarse inert material (winter placement) Dismantle tailings reclaim line only after water quality in pond meets criteria Remove seepage dam
Water treatment plant	In use for treatment of excess water from tailings facility	Decommissioned only after tailings pond water meets discharge criteria
Camp	Remove modular components not required to support ongoing activities.	Progressively remove all buildings Market sewage treatment plant Decommission and seal water wells
Airstrip	Will not be decommissioned	Will not be decommissioned
Onsite and access road	In use for removal of material and import of supplies; road access and traffic control maintained	Access and traffic controls in place until the end of the closure period; road route deactivated and reclaimed once water treatment is no longer required
Land treatment farm	Use soil in reclamation activities if deemed remediated; haul offsite if contaminated	
Landfill, incinerator, waste storage areas	Remove incinerator from site Close landfill and storage areas by contouring, capping and seeding	

Closure planning includes a contingency plan for mitigating the potential for poor quality groundwater originating in the underground workings and discharging into Wolverine Creek. A portion of the groundwater that contributes to the flow in Wolverine Creek is naturally elevated in zinc, cadmium and selenium. For the treatment of metals and selenium in groundwater that contributes to Wolverine Creek, a passive biological treatment system (termed Biopass) will be developed and tested. The Biopass system represents passive biological treatment where dissolved metals are precipitated as metal sulphides and dissolved selenium (selenate, selenite) is reduced to solid phase elemental selenium through microbial reduction.

The Biopass system will be constructed in the Wolverine Creek channel along the stretch of the creek (e.g. 400 m long) that could potentially receive groundwater with high selenium and other metal concentrations derived from mine water. Clean water in Wolverine Creek, upstream of the Biopass channel, will be diverted along the western margin of Wolverine Creek in a lined channel and re-introduced into Wolverine Creek in the lower reach that is not adversely affected by poor quality groundwater.

The Biopass will be approximately 2.5 m deep and 3 m wide and will collect groundwater that naturally discharges to Wolverine Creek. Deep groundwater that does not enter into Wolverine Creek will flow towards Little Wolverine Lake where no impacts to water quality are predicted to occur. The excavated channel that intercepts groundwater will be filled with an organic substrate to support the sulphate reducing bacteria and microorganisms that reduce dissolved selenium to solid phase elemental selenium. Organic materials that are being tested as substrate amendments include mixtures of mushroom compost, manure, alfalfa, and sewage sludge along with natural peats in the area. Overlying the organic substrate is a gravel layer (0.15 m) that facilitates lateral drainage of the upward flowing treated effluent. The drainage layer overlying the organic substrate is sealed at the top by an impermeable geomembrane liner that limits exchange of water and gas with the overlying cover (0.25 m) of topsoil to protect the liner. Rainfall and surface runoff that infiltrates into the cover layer is collected in lateral sand drains and removed to prevent water saturation of the topsoil cover.

The groundwater that discharges into the more permeable organic substrate is biologically treated while moving in a down-slope direction. Collected groundwater will flow upwards and through the organic substrate where treatment will occur. Before joining the non-diverted section of Wolverine Creek, the Biopass system will merge into a french drain that discharges into Wolverine Creek. While the water exiting from the Biopass system is expected to be depleted of oxygen, this water will be combined with the diverted well-oxygenated water from upper Wolverine Creek. Moreover, this combined flow will then traverse steep terrain for approximately a kilometer before reaching the mouth of Wolverine Creek and is expected to be fully oxygenated in this fish-bearing reach.

The groundwater discharge rates that can be treated will depend on the porosity of the organic substrate layer and the total length of the Biopass. For example, by

using a Biopass with a total length of 400 m, it will be possible to treat groundwater at discharge rates of approximately 2 L/s; groundwater discharge rates in the treatment area are expected to be on the order of 0.5 L/s to 1 L/s, following installation of the hydraulic plugs in the portal.

Closure of the TMF is based on static testing (acid base accounting) of composite tailings samples that indicated that the tailings contain significant quantities of sulphide-sulphur and lesser quantities of neutralization potential. As such, Wolverine tailings are characterized as potentially acid generating. Kinetic testing of two tailings samples in laboratory humidity cells have been ongoing for approximately 340 weeks each.

To prevent oxidation of the tailings solids and subsequent acid generation, the tailings impoundment has been designed to be a water retaining structure underlain with an impermeable liner. This design permits the tailings to remain completely saturated, both during operations and at closure, and will eliminate the potential for acid drainage from the facility. Moreover, the liner construction also greatly reduces the potential for seepage of tailings water and the concomitant potential for groundwater contamination occurring both during operations, closure and at post-closure. The tailings closure strategy involves the placement of an inert cover material over the tailings solids and the maintenance of a water cover over the entire facility to provide a stable cover for the tailings and reduce the potential for remobilization and resuspension of tailings solids from wind induced wave action. Yukon Zinc currently proposes to quarry coarse inert material for use in covering the tailings. The borrow location for this coarse inert material will be evaluated and full acid rock drainage testing (e.g. acid-base accounting and shake flask testing) will be performed to confirm there is no potential for acid generation and/or metal leaching from the material.

Upon cessation of operations, the coarse inert material cover will be placed over the surface of the tailings in two lifts. During the first winter of the closure period, a 0.5 m thick layer of coarse inert material will be placed on the ice within the tailings pond. As the ice melts, the material will gradually settle over top of the submerged tailings. The objective is to place the cover with minimal disturbance to the tailings interface. Following complete settling of the material, an underwater survey will be conducted to ensure adequate cover over the tailings. During the second winter, a second 0.5 m thick layer of material will be laid over the ice to complete the 1.0 m material cover.

Ultimately, the tailings facility will be closed as a saturated deposit with a combined cover of 1.5 m over the tailings consisting of the approximately 1.0 m of material and 0.5 m of water.

The potential effects of climate change have been evaluated and duly considered in the design and closure management of the tailings impoundment. Wide-ranging precipitation conditions (e.g. 100-year dry and 100-year wet) were evaluated in water balance modeling for the facility in support of this plan. Continued monitoring of site

meteorological conditions (e.g. precipitation and evaporation) will also provide the necessary supporting data to ensure all tailings design criteria for extreme events, including climate change, are adequate and comprehensive.

During closure, the dominant inflow to the tailings impoundment water balance is runoff originating from the tailings area as a result of decommissioning of diversion ditches and the absence of tailings discharging into the facility. During the initial three years following closure, excess tailings water will be treated through the water treatment plant and water levels in the impoundment will be maintained below the spillway elevation of 1,311.5 m.

At cessation of operations, approximately 80,000 m³ of tailings water will be held in the impoundment. Upon removal of the diversion ditches associated with the tailings facility, annual inflows of runoff and precipitation to the facility are expected to be on the order of 170,000 m³. Accordingly, roughly two tailings water volume replacements are expected to occur each year. By the end of Year 3 closure, approximately five to six complete volume replacements would have occurred with clean runoff water. The impoundment water quality should return to near 'baseline' conditions following treatment and removal of the excess water at closure and upon replenishing the pond with fresh water from annual inflows of clean runoff. As such, the anticipated three-year closure treatment phase is dictated by the time required to flush tailings influenced water from the impoundment to achieve overflow water quality below discharge limits. The tailings impoundment will be closed as a 'wet' facility with a water cover over the tailings and a permanent spillway to manage discharge as well as flood flows through the impoundment.

Because the tailings are to be covered with approximately 1 m of cover material, diffusion of tailings porewater to the overlying water column will be attenuated.

Long-term groundwater quality beneath the tailings impoundment following closure is expected to remain at baseline conditions owing to the fact that the tailings facility is completely lined. Monitoring of surface and ground water quality will continue for at least 10 years following the cessation of mining activities to ensure maintenance of acceptable water chemistry.

20.5 SOCIAL AND COMMUNITY CONSIDERATIONS

The Wolverine Project is located within the traditional territory of the Kaska nation. Consultation on the Wolverine Mine with the Kaska Nation began in 1995 with Atna and Westmin in the form of meetings through advisory committees, technical working groups, and public open houses. In 1997-8, an ALO was appointed in Ross River to liaise between the Kaska Nation and Yukon Zinc. The position was reinstated during 2000-1 by Expatriate Resources (now named Yukon Zinc Corp.) and again in 2005 by Yukon Zinc, and is a currently active role.

Yukon Zinc has contacted the Ross River Dena Council (RRDC) (who acts on behalf of the Kaska Nation, its bands and their respective members for the Wolverine Mine), regularly through 2005 and 2006 during the development of two agreements:

In July 2005, Yukon Zinc signed a Socio-Economic Participation Agreement on July 12, 2005 with the RRDC that provides a basis for participation by all Kaska in project exploration and development activities, including the review of environmental, social and economic matters related to these activities and the environmental assessment and permitting of the Wolverine Mine. As long-term partners, the Kaska agree to support all activities to develop, operate and reclaim the Wolverine Mine.

Yukon Zinc entered into a Traditional Knowledge (TK) Protocol Agreement with the RRDC on September 28, 2005 to ensure that traditional knowledge was not compromised in any way by ongoing or future project activities. Elder interviews were conducted by the RRDC TK Coordinator from September to December, 2005 and a helicopter overview flight was conducted on October 5, 2005. The TK Report was submitted to Yukon Zinc in February 2006. Partial ground truthing by land stewards representing the RRDC and Liard First Nation was conducted along proposed road route June 2006.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

The Wolverine Mine is currently in production, and therefore has no pre-production capital.

The total capital cost required for the Wolverine Mine from year 2012 for the LOM is estimated at \$63.0 million and includes the following:

- development costs – \$18.1 million
- full extraction of the Wolverine LOM reserves – \$19.2 million
- sustaining capital for the LOM – \$23.0 million
- closure and reclamation costs – \$2.6 million.

21.1.1 MINE DEVELOPMENT COSTS

Mine development costs that have been capitalized are all infrastructure related development including: ramps, level access, remucks, sumps, fresh air and return air raises. Capital development for the Wolverine LOM totals 2,404 m for a total cost of \$18.1 million. The LOM capital development schedule, meters and cost, is shown in Table 21.1.

All costs are in Canadian dollars unless otherwise stated.

Table 21.1 Capital Development Schedule

Mine Development	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Metres										
Ramp – Waste	511	165	170	78	-	-	-	-	-	924
Ramp – Ore	231	67	80	82	-	-	-	-	-	460
Level Access	88	30	21	23	-	-	-	-	-	162
Remucks	78	38	38	14	-	-	-	-	-	168
Sumps	9	9	9	9	-	-	-	-	-	36
Return Air Raise	95	34			-	-	-	-	-	129
Fresh Air Raise	89	0	99	35	-	-	-	-	-	223
Escapeways	150	34	83	35	-	-	-	-	-	302
Total	1,251	377	500	276	-	-	-	-	-	2,404
Cost (\$000 000)										
Ramp – Waste	4.888	1.577	1.625	0.746	-	-	-	-	-	8.835

table continues...

Mine Development	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Ramp – Ore	2.209	0.640	0.765	0.784	-	-	-	-	-	4.398
Level Access	0.543	0.185	0.130	0.142	-	-	-	-	-	0.999
Remucks	0.480	0.234	0.234	0.086	-	-	-	-	-	1.035
Sumps	0.056	0.056	0.056	0.056	-	-	-	-	-	0.222
Return Air Raise	0.386	0.138	0.000	0.000	-	-	-	-	-	0.524
Fresh Air Raise	0.361	0.000	0.402	0.142	-	-	-	-	-	0.905
Escapeways	0.609	0.138	0.337	0.142	-	-	-	-	-	1.226
Total	9.531	2.969	3.548	2.097	-	-	-	-	-	18.145

21.1.2 OTHER CAPITAL COST

To ensure full extraction of the Wolverine LOM reserves the following capital expenditures are required and shown in Table 21.2.

- DMS: required to increase milling efficiency and metal recoveries (\$4.2 million) (subject to ongoing test work).
- Raise Tailings Dam: required to be raised to full height to allow excess water storage prior to completion of the water treatment plant in 2013 (\$5.0 million).
- Warehouse Facility: required for storage of mine and mill operating spare parts which are currently stored outside in sea cans (\$4.0 million).
- Water Treatment Plant: scheduled to be constructed in 2013 and is required to reduce selenium in the discharge water (\$3.0 million).
- Process Control Improvements: required to improve mill efficiencies and metal recoveries (\$0.8 million).
- Generator Upgrade: Current power output is sufficient for current levels of production. Additional generators are being added to allow removal of smaller units and accommodate increased mill capacity in winter months (\$2.3 million).

Table 21.2 Other Capital Expenditures (\$000 000)

Item	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
DMS and Installation	-	4.160	-	-	-	-	-	-	-	4.160
Raise Tailings Dam	5.000	-	-	-	-	-	-	-	-	5.000
Warehouse Facility	-	4.000	-	-	-	-	-	-	-	4.000
Water Treatment Plant	-	3.000	-	-	-	-	-	-	-	3.000
Process Controls	0.750	-	-	-	-	-	-	-	-	0.750
Generator Upgrade	2.330	-	-	-	-	-	-	-	-	2.330
Total	8.080	11.160	-	-	-	-	-	-	-	19.240

21.1.3 SUSTAINING CAPITAL COST

Sustaining mine capital cost was determined by Yukon Zinc to be nil in 2012 and \$1.0 million per year from 2013 to 2018 totaling \$6.0 million. Mine sustaining costs are estimated to cover mine equipment replacement and mine equipment major repairs to sustain continuous mine production levels.

Sustaining mill process and plant infrastructure capital cost was determined by Yukon Zinc to be \$17.0 million from 2012 to 2018. This capital is required to ensure milling levels are maintained to meet production levels. The LOM sustaining capital expenditures are shown in Table 21.3.

Table 21.3 Sustaining Capital Expenditures (\$000 000)

Area of Expense	2013	2014	2015	2016	2017	2018	2019	2020	Total
Mine	1.000	1.000	1.000	1.000	1.000	1.000	-	-	6.000
Process & Infrastructure	2.000	3.000	3.000	3.000	3.000	3.000	-	-	17.000
Total	3.000	4.000	4.000	4.000	4.000	4.000	-	-	23.000

21.1.4 RECLAMATION CAPITAL COST

A reclamation (closure) capital cost was supplied by Yukon Zinc at \$2.6 million and a deposit of this amount will be made in 2012.

21.2 OPERATING COST ESTIMATE

21.2.1 MINING OPERATING COSTS

Mining costs were developed by Yukon Zinc on a first principle basis. Stope extraction cycles, based on large drift and fill excavation (70% of production) and small drift and fill excavation (30% of production), were determined to develop the costs. Costs include contractor labour, Yukon Zinc supervision, mining consumables and contractor equipment required to maintain the planned mine production of 1,700 t/d. Mining costs include operating development in waste required in the ore mining cycle. Equipment is assumed to be contractor owned and operated, new or well maintained. Ramp and raise development are capitalized and therefore identified as a cost under capital development.

Yukon Zinc has determined that they will be able to attain productivity improvements in their mine operating cost equating to 5% in 2013 and 2014, 2% for 2015 through 2018 and 1% in 2019 and 2020.

The LOM average mine unit cost was determined to be \$87.81/t of ore mined and delivered to the mill.

The mill processing cost, including labour, reagents, consumables, and power was determined by Yukon Zinc to be \$68.83/t of ore milled in 2012 and will be reduced by \$8.83/t to \$60.00/t of ore milled in 2013. Subject to ongoing test work, a DMS plant is planned to be installed in 2013 and to be in full operation in 2014. This will further reduce the mill process cost by \$3.84/t milled. Therefore, the mill process cost was assumed to be \$56.16/t ore milled, 2014 through 2020.

Yukon Zinc has determined G&A cost at \$15 million per year when in full production. The G&A unit cost in 2013 when full production of 621,000 t is attained is \$24.15/t. As the production will vary toward the end of the LOM the power of 0.6 rule was applied to determine a reduced G&A costs when production levels off. The drop in G&A costs, as production levels off, is not linear as the G&A costs are mainly administrative manpower. Therefore, the power of 0.6 rule is applied which allows for factoring of the G&A costs on a non-linear basis. The formula is as follows: (base tonnes/new tonnes)^{0.6} x (new tonnes/base tonnes) where base tonnes represents full capacity rate.

21.2.2 LOM OPERATING COST

LOM operating costs, based on first principle calculations, were supplied by Yukon Zinc. Total LOM operating costs and unit cost per tonne of ore mined, are shown in Table 21.4.

Table 21.4 LOM Operating Costs

Cost Area	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Cost (\$000 000)										
Mine	50.6	56.8	53.8	52.6	51.4	33.2	18.6	7.5	4.3	328.8
Mill	31.8	37.3	34.9	34.9	34.9	23.0	13.2	5.4	3.1	218.5
G&A	19.0	15.0	15.0	15.0	15.0	12.7	10.2	7.1	5.7	114.7
Total	101.4	109.0	103.7	102.5	101.3	68.9	42.1	20.1	13.0	662.0
Cost (\$/t)										
Mine	109.45	91.45	86.64	84.71	82.79	80.86	78.94	77.98	77.01	87.81
Mill	68.83	60.0	56.16	56.16	56.16	56.16	56.16	56.16	56.16	58.36
G&A	41.13	24.15	24.14	24.15	24.15	30.96	43.17	73.70	103.18	30.63
Total	219.40	175.60	166.95	165.03	163.10	167.99	178.27	207.84	236.36	176.81

22.0 ECONOMIC ANALYSIS

22.1 INTRODUCTION

An economic evaluation of the Wolverine Mine was prepared by Tetra Tech based on a pre-tax financial model. For the nine-year mine life and 3.8 Mt reserve, excluding Inferred Resources, the pre-tax financial model included the following assumptions:

- Tetra Tech adopted the Energy and Metal Consensus Forecast (EMCF) metal prices, in effect Q4 2011. The metal prices used in determining the NPVs of Wolverine Mine were recommended by the appropriate internal group of the independent technical expert which surveys the leading international financial institution forecasts. Such metal prices were based on the EMCF, a consensus projection of commodities prices by a group of over 20 senior analysts from major financial institutions, published by Consensus Economics, a macroeconomic survey firm.
- EMCF CAD\$1.00/US\$1.00 exchange rate was used based on a two-year currency exchange forecast, in effect Q4 2011.
- The discount rate of 8% was established for use in determining the NPVs of Wolverine Mine. The discount rate of 8% is consistent with that used in other comparable base metals producing projects at similar stages of production.
- Smelting and refining terms, transportation and marketing costs supplied by Yukon Zinc representing their current budget inputs.
- Fuel costs have a significant impact on operating costs and the current model incorporates \$1.07/L for diesel fuel (\$0.32/kWh).
- No remedial measures were assumed completed or required to meet proposed production rates and recoveries used in the NPV calculations and the economic analysis.
- The mill recovery rates and grades were provided by Yukon Zinc and reviewed by the technical expert. The projected recovery rates were based on overall metal recovery rates and test results released in independent test labs' reports available at the time of writing the report. The projected recovery rates were based on overall metal recovery rates as indicated in prior metallurgical test work completed by Central South University, Hunan, China, as discussed in Section 13.1.

The actual recovery rates realized through to August 2011 were not used in the NPV calculation as these actual recovery rates reflect the ramp-up

phase of mining and milling operations and are not considered to be indicative of metal recovery rates to be realized once full commercial production is achieved. Metallurgical test results are a better indicator of long-term performance, as required for the NPV calculation and economic analysis, than the recoveries realized during the ramp-up phase.

- The mine operating unit cost varies based on expected productivity improvements provided, by Yukon Zinc from 2013 to 2020.
- As a result of the planned installation of the DMS circuit in 2013 the milling unit cost is expected to change, when compared to 2012, as follows:
 - decrease by \$8.83/t of ore milled in 2013
 - an additional reduction of \$3.84/t in 2014 from 2013
 - remain constant from 2015 through to 2020

The total annual mill operating costs will vary based on the number of tonnes of ore milled.

- General and administrative expenses are expected to remain constant upon the expected achievement of full production in 2013 and through to 2016, at which time such expenses are expected to decrease as production levels at Wolverine Mine are expected to decrease from 2017 to 2020. The independent technical expert determined the expected decrease in general and administrative expenses using the “power of 0.6 rule”, which is a formula commonly used in the mining industry to calculate the effects of changes in production levels on related manpower requirements by accounting for the non-linear relationship between manpower requirements and production levels.

The financial model can be used to evaluate the sensitivity of the Wolverine Mine 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 Wolverine Mine determined.

22.2 PRE-TAX MODEL

The mine production schedule has been incorporated into the 100% equity pre-tax financial model to develop annual recovered metal production from the relationships of tonnage milled, head grades and recoveries.

NSR revenue was determined by metal grades and metal prices and adjusted for smelting, refining, transportation and marketing costs. An estimated marketing cost of 0.5%, as determined by Yukon Zinc, was applied to the result of gross revenue less smelting, refining and transportation costs. The LOM NSR revenue was determined to be \$1,331.3 million or \$355.54/t ore.

LOM net revenue, determined as NSR revenue less royalties, was \$1,271.6 million or \$339.60/t ore.

Unit operating costs for mining, milling and G&A areas were applied to annual milled tonnages to determine the overall mine site operating cost, which has been deducted from the net revenue to derive annual operating margin. The LOM operating margin was determined to be \$609.5 million or \$162.79 /t ore.

Capital development, other capital requirements and sustaining capital were considered in the year spent. Capital development consisting 2,404 m of ramps, level accesses, sumps, remucks, and ventilation raises were determined to be \$18.1 million over the LOM. Other capital requirements of \$19.2 million consist of a dense media separation plant, raising the tailings dam, warehouse facility, water treatment plant, process control improvements, and generator upgrades. Sustaining capital for mining and process and infrastructure were determined to be \$23.0 million for the LOM.

A reclamation or closure cost of \$2.6 million was incorporated and assumed deposited in 2012.

Pre-tax cash flow (EBITDA), determined as net revenue less operating, capital and reclamation costs, was estimated as \$546.6 million.

The pre-tax NPV of the Wolverine Mine is \$443.8 million determined at a real discount rate of 8%.

The Wolverine Mine is in production in 2012 and as such does not have a negative cash flow, therefore, no IRR or payback are realized.

A summary of the LOM financials for Yukon Zinc is shown in Table 22.1.

The undiscounted annual cash flows are illustrated in Figure 22.1.

Figure 22.1 Annual Undiscounted Pre-tax Cash Flow

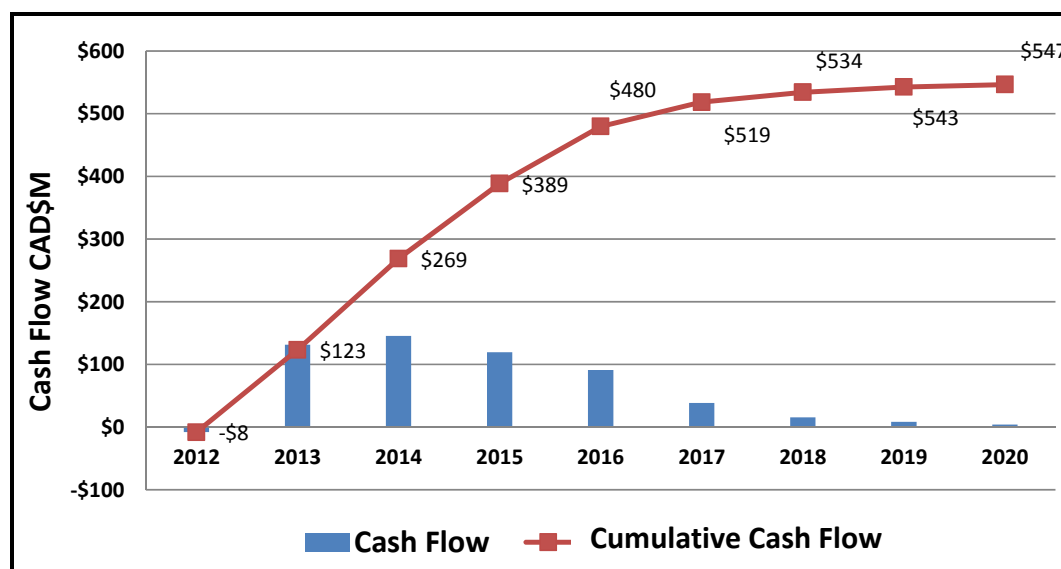


Table 22.1 Yukon Zinc LOM Pre-tax Financials

	Unit	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Metal Prices											
Zn	US\$/lb	0.97	1.13	1.11	1.15	1.10	1.04	1.04	1.04	1.04	1.07
Cu	US\$/lb	3.77	3.98	3.70	3.58	3.41	2.96	2.96	2.96	2.96	3.36
Pb	US\$/lb	1.00	1.16	1.14	1.15	1.09	0.99	0.99	0.99	0.99	1.06
Ag	US\$/oz	33.91	30.85	28.39	25.35	22.65	19.12	19.12	19.12	19.12	24.18
Au	US\$/oz	1,726	1,561	1,438	1,348	1,257	1,200	1,200	1,200	1,200	1347.78
Exchange Rate	CAD:US	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Production	tonnes	462,000	621,000	621,253	621,018	620,871	410,405	235,893	94,714	55,204	3,744,358
Recovered Zn	tonnes	35,278	63,784	65,706	64,333	61,704	39,400	21,979	8,102	4,108	364,394
Recovered Cu	tonnes	1,577	4,598	5,536	5,734	5,622	3,604	1,960	861	624	30,117
Recovered Pb	tonnes	2,411	4,359	4,299	3,880	3,594	2,239	1,228	587	350	22,948
Recovered Ag	oz	2,981,053	5,431,680	5,021,092	4,439,253	4,006,917	2,596,894	1,419,612	794,700	501,783	27,192,984
Recovered Au	oz	6,082	17,744	19,195	15,949	13,062	8,547	4,729	2,766	1,654	89,727
Payable Zn	tonnes	29,463	53,578	55,193	54,039	51,831	33,096	18,463	6,805	3,450	305,920
Payable Cu	tonnes	1,482	4,356	5,260	5,447	5,340	3,424	1,862	818	593	28,583
Payable Pb	tonnes	2,025	3,705	3,654	3,240	3,002	1,870	1,026	491	292	19,305
Payable Ag	oz	2,412,826	4,557,405	4,287,756	3,764,689	3,382,775	2,194,865	1,197,988	688,262	438,823	22,925,410
Payable Au	oz	5,239	15,860	17,169	13,977	11,228	7,378	4,089	2,459	1,454	78,853
Payable Zn	\$M	63.0	133.5	135.1	137.0	125.7	75.9	42.3	15.6	7.9	736.0
Payable Cu	\$M	12.3	38.2	42.9	43.0	40.1	22.3	12.2	5.3	3.9	220.3
Payable Pb	\$M	4.5	9.5	9.2	8.2	7.2	4.1	2.2	1.1	0.6	46.6
Payable Ag	\$M	81.8	140.6	121.7	95.4	76.6	42.0	22.9	13.2	8.4	602.6
Payable Au	\$M	9.0	24.8	24.7	18.8	14.1	8.9	4.9	3.0	1.7	109.9
Gross Revenue	\$M	170.7	346.5	333.6	302.5	263.8	153.1	84.5	38.1	22.6	1,715.4
Gross Revenue	\$/t ore	369.38	558.01	536.93	487.09	424.87	373.11	358.36	394.21	408.54	458.12
Zn Smelting	\$M	(12.1)	(23.4)	(23.3)	(24.5)	(21.4)	(13.0)	(7.2)	(2.7)	(1.4)	(129.0)
Cu Smelt/ Refining	\$M	(8.5)	(11.9)	(4.3)	(4.2)	(4.0)	(2.3)	(1.4)	(0.7)	(0.5)	(38.0)
Pb Smelt/ Refining	\$M	(7.0)	(5.6)	(4.4)	(4.3)	(3.9)	(2.5)	(1.4)	(0.7)	(0.4)	(30.0)
Zn Transport	\$M	(15.6)	(23.9)	(22.3)	(21.9)	(21.0)	(13.4)	(7.5)	(2.8)	(1.4)	(129.7)
Cu Transport	\$M	(2.5)	(5.0)	(4.7)	(4.9)	(4.8)	(3.1)	(1.7)	(0.7)	(0.5)	(27.8)
Pb Transport	\$M	(2.8)	(4.1)	(4.0)	(4.0)	(3.7)	(2.3)	(1.2)	(0.6)	(0.4)	(22.9)
Marketing	\$M	(0.6)	(1.4)	(1.4)	(1.2)	(1.0)	(0.6)	(0.3)	(0.2)	(0.1)	(6.7)
Process Cost	\$M	(49.1)	(75.2)	(64.3)	(64.8)	(59.8)	(37.3)	(20.7)	(8.2)	(4.6)	(384.1)
	\$/t ore	(106.34)	(121.03)	(103.47)	(104.41)	(96.37)	(90.97)	(87.76)	(85.08)	(83.02)	(102.58)
Net NSR	\$M	121.5	271.4	269.3	237.7	204.0	115.8	63.8	29.9	18.0	1,331.3
	\$/t ore	263.04	436.99	433.46	382.68	328.51	282.14	270.60	309.13	325.53	355.54
Equity Royalty	\$M	(0.5)	0	0	0	0	0	0	0	0	(0.5)
Kink Royalty	\$M	(0.5)	(0.5)	(0.5)	(0.5)	(0.4)	(0.2)	(0.1)	(0.1)	(0.0)	(2.9)
Atna Royalty	\$M	(6.9)	(13.3)	(11.9)	(9.0)	(7.1)	(3.9)	(2.1)	(1.3)	(1.0)	(56.3)
Royalty Cost	\$M	(7.9)	(13.9)	(12.4)	(9.5)	(7.5)	(4.1)	(2.2)	(1.3)	(1.0)	(59.7)
	\$/t ore	(17.02)	(22.32)	(20.03)	(15.33)	(12.04)	(10.01)	(9.52)	(13.82)	(15.44)	(15.95)
Net Revenue	\$M	113.7	257.5	256.8	228.1	195.5	111.7	61.6	28.6	17.1	1,271.6
	\$/t ore	246.02	414.66	413.42	367.35	316.46	272.13	261.07	295.31	310.09	339.60
Mine	\$M	(50.6)	(56.8)	(53.8)	(52.6)	(51.4)	(33.2)	(18.6)	(7.5)	(4.3)	(328.8)
Mill	\$M	(31.8)	(37.3)	(34.9)	(34.9)	(34.9)	(23.0)	(13.2)	(5.4)	(3.1)	(218.5)
G&A	\$M	(19.0)	(15.0)	(15.0)	(15.0)	(15.0)	(12.7)	(10.2)	(7.1)	(5.7)	(114.7)
Operating Cost	\$M	(101.4)	(109.0)	(103.7)	(102.5)	(101.3)	(68.9)	(42.1)	(20.1)	(13.0)	(662.0)
	\$/t ore	(219.40)	(175.60)	(166.95)	(165.03)	(163.10)	(167.99)	(178.27)	(207.84)	(226.36)	(176.81)
Operating Margin	\$M	12.3	148.5	153.1	125.6	95.2	42.7	19.5	8.5	4.1	609.5
	\$/t ore	26.62	239.06	246.48	202.33	153.36	104.14	82.81	87.47	73.73	162.79
Capital Development	\$M	(9.5)	(3.0)	(3.5)	(2.1)	0	0	0	0	0	(18.1)
Other CAPEX	\$M	(8.1)	(11.2)	0	0	0	0	0	0	0	(19.2)
Sustaining CAPEX	\$M	(0.0)	(3.0)	(4.0)	(4.0)	(4.0)	(4.0)	(4.0)	0	0	(23.0)
Reclamation	\$M	(2.6)	0	0	0	0	0	0	0	0	(2.6)
Capital	\$M	(20.2)	(17.1)	(7.5)	(6.1)	(4.0)	(4.0)	(4.0)	0	0	(63.0)
	\$/t ore	(43.71)	(27.58)	(12.15)	(9.82)	(6.44)	(9.75)	(16.96)	0	0	(16.82)
Pre Tax Cash Flow	\$M	7.9	131.3	145.6	119.6	91.2	38.7	15.5	8.5	4.1	546.6
Pre Tax Net Margin	\$/t ore	(17.09)	211.48	234.33	192.51	146.92	94.40	65.85	87.47	73.73	145.97
NPV @ 8%	\$M	443.8	-	-	-	-	-	-	-	-	-

Note: CAPEX – Capital Expense

22.2.1 METAL PRICE SCENARIOS

The LOM average metal prices for the three cases were identified with their corresponding NPVs at discount rates of 8%, 5% and 0%. The results are shown in Table 22.2.

Table 22.2 Pre-tax NPV by Metal Price Scenario

Case	LOM Average Metal Prices					NPV (CAD\$000 000)		
	Zn (US\$/lb)	Cu (US\$/lb)	Pb (US\$/lb)	Ag (US\$/oz)	Au (US\$/oz)	@ 8%	@ 5%	@ 0%
Base +10%	1.18	3.70	1.16	26.60	1,483	564	607	694
Base	1.07	3.36	1.06	24.18	1,348	444	475	543
Base -10%	0.96	3.02	0.95	21.76	1,213	317	342	392

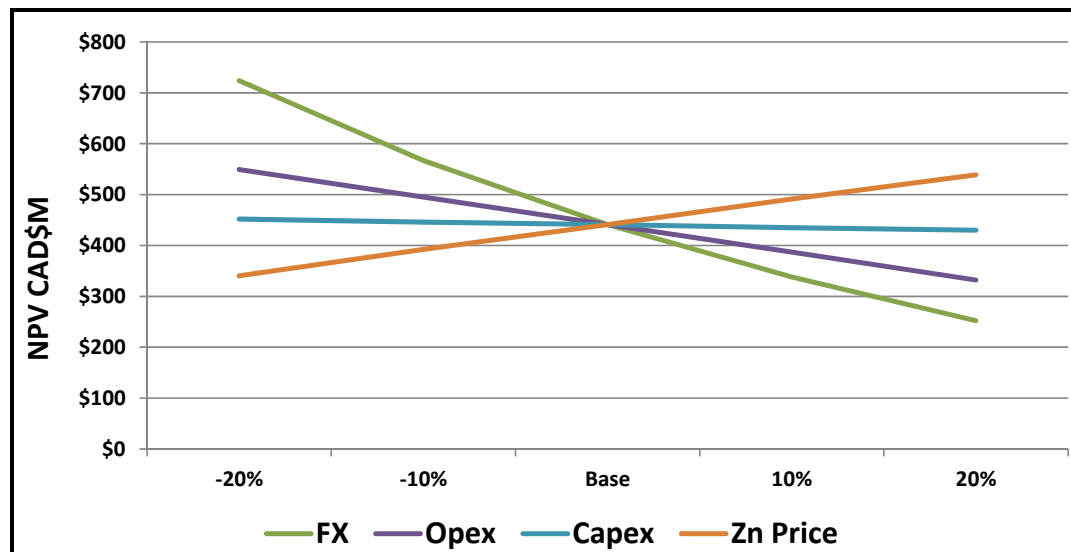
22.2.2 SENSITIVITY ANALYSIS

Sensitivity analyses were carried out on the following parameters:

- exchange rate
- operating cost
- capital cost
- zinc price.

The analyses are presented graphically as financial outcomes in terms of the pre-tax NPV. Figure 22.2 illustrates that the Wolverine Project NPV (8% discount) is most sensitive to the exchange rate, and in decreasing order, zinc price, operating cost, and capital cost.

Figure 22.2 Sensitivity Analysis – Pre-tax NPV at 8%



22.3 POST-TAX MODEL

PwC provided a tax model that accurately predicts the post-tax cash flow.

The economic evaluation indicated a post-tax NPV of \$414 million determined at a discount rate of 8%.

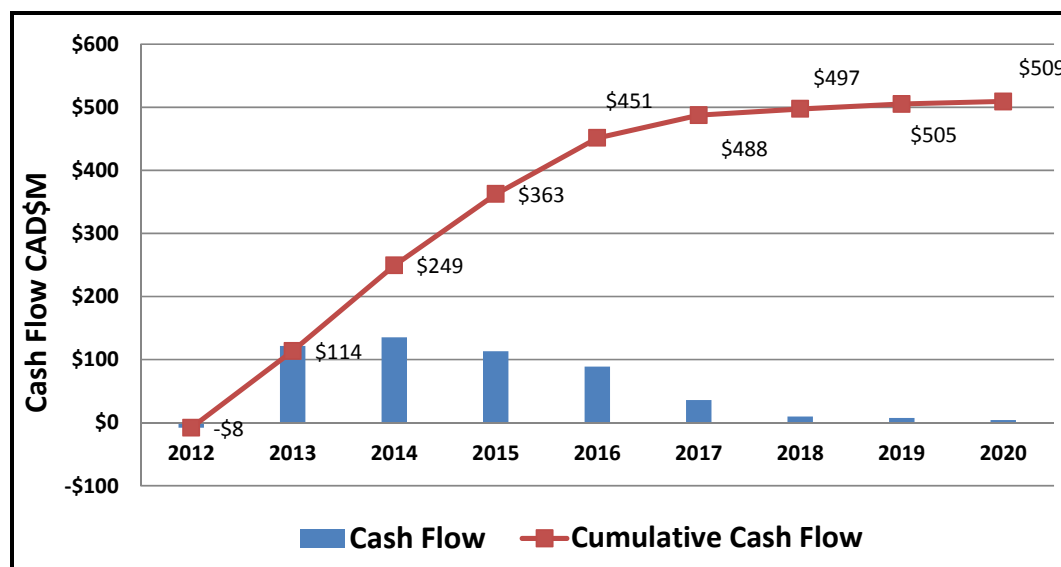
The post-tax financial model used the same inputs as the pre-tax economic evaluation with the following key taxation rates:

- Federal income tax rate at 15%
- Yukon income tax rate at 14.54%
- Yukon mining tax rate at 12.16%.

A mine owner, manager, holder, tenant, occupier or operator in Yukon must pay a royalty to the Government of Yukon under the Quartz Mining Act (QMA). The royalty is a share of profits from a mine in Yukon that is reserved and paid to the Government of Yukon as owner of the mineral rights for permitting extraction of mineral resources. The calculation of this royalty has been included under “Yukon mining tax rate at 12.16%”.

The undiscounted post-tax annual cash flows are illustrated in Figure 22.3.

Figure 22.3 Annual Undiscounted Post-tax Cash Flow



22.3.1 METAL PRICE SCENARIOS

The same metal price and exchange rate scenarios that were used for the pre-tax model were used for the post-tax model to evaluate the sensitivity on NPVs at discount rates of 8%, 5% and 0%, as shown in Table 22.3.

Table 22.3 Post-tax NPV by Metal Price Scenario

Case	LOM Average Metal Prices					NPV (CAD\$000 000)		
	Zn (US\$/lb)	Cu (US\$/lb)	Pb (US\$/lb)	Ag (US\$/oz)	Au (US\$/oz)	@ 8%	@ 5%	@ 0%
Base +10%	1.18	3.70	1.16	26.60	1,483	494	531	603
Base	1.07	3.36	1.06	24.18	1,348	414	444	507
Base -10%	0.96	3.02	0.95	21.76	1,213	304	329	377

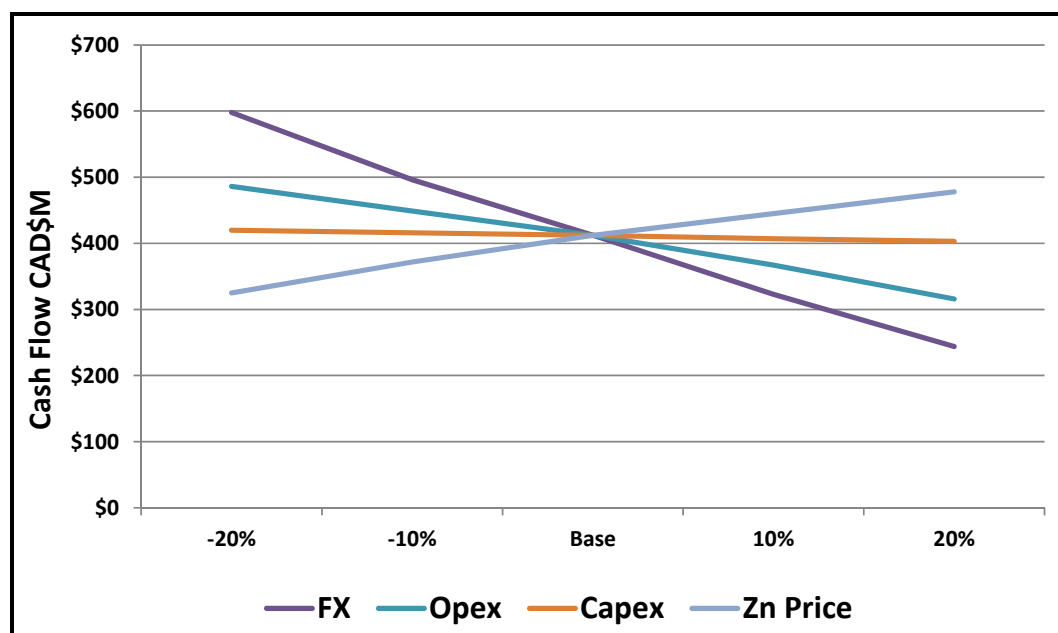
22.3.2 SENSITIVITY ANALYSIS

As in the pre-tax model sensitivity analyses were carried out on the following parameters:

- exchange rate
- operating cost
- capital cost
- zinc price.

The analyses are presented graphically as financial outcomes in terms of the post-tax NPV. Figure 22.4 illustrates that the Wolverine Mine NPV (8% discount) is most sensitive to the exchange rate, and in decreasing order, zinc price, operating cost and capital cost.

Figure 22.4 Sensitivity Analysis – Post-tax NPV at 8%

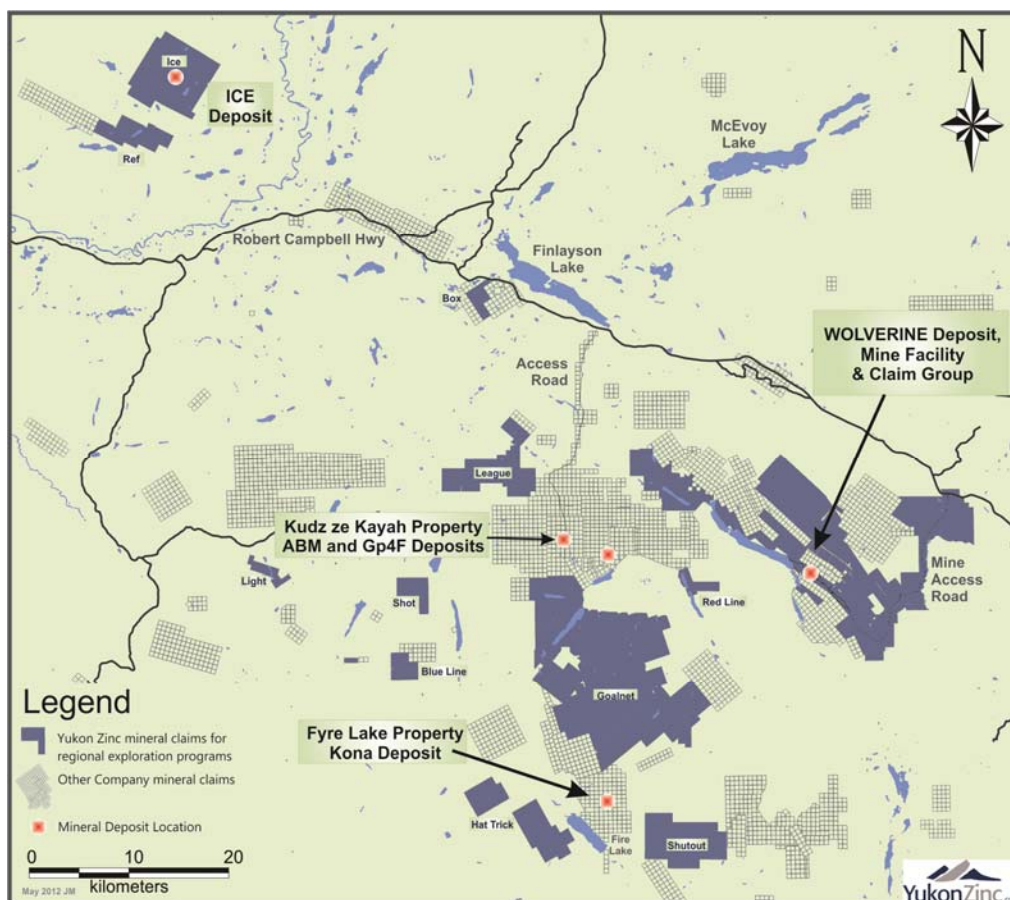


23.0 ADJACENT PROPERTIES

23.1 INTRODUCTION

Significant VMS deposits were discovered from 1994 to 1998 in the Finlayson District. Five of the largest and best-known VMS deposits in this region include Kona, ABM, GP4F, Wolverine and Ice deposits. In this discussion, the ABM and GP4F deposits are included as “adjacent properties” as they are located approximately 30 km to the west of the Wolverine deposit. Three “zones” of mineralization (Fisher, Sable and Puck) are also discussed as they occur within the Wolverine Belt; a geologically contiguous stratigraphic horizon with that of the Wolverine deposit. Other VMS deposits within the FD (e.g. Ice) are not included as they are distal to the Wolverine deposit, or are owned by Yukon Zinc, and do not meet the definition of “Adjacent Property” under NI 43-101.

Figure 23.1 Adjacent Property Map



23.2 ABM AND GP4F DEPOSITS

The ABM deposit on Teck Resources Ltd.'s Kudzu Kayah Property is located 30 km west of the Wolverine Mine (Figure 23.1). 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 Teck 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 Teck 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 drillholes 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 drillholes totalling 16,178 m were completed in 1995 at the ABM Deposit and on nearby regional targets including GP4F. A total of 17 diamond drillholes totalling 3,566 m were completed in 1997; seven holes tested the down dip/plunge and fault offset margins of the ABM deposit; two holes tested regional targets in the area and nine holes were completed in the Fault Creek area. Drilling has identified a historic resource of 13,720,000 t grading 6.0% Zn, 1.6% Pb, 0.90% Cu, 139.2 g/t Ag, and 1.38 g/t Au in the ABM deposit and 1,500,000 t 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).

23.3 WOLVERINE LAKE GROUP

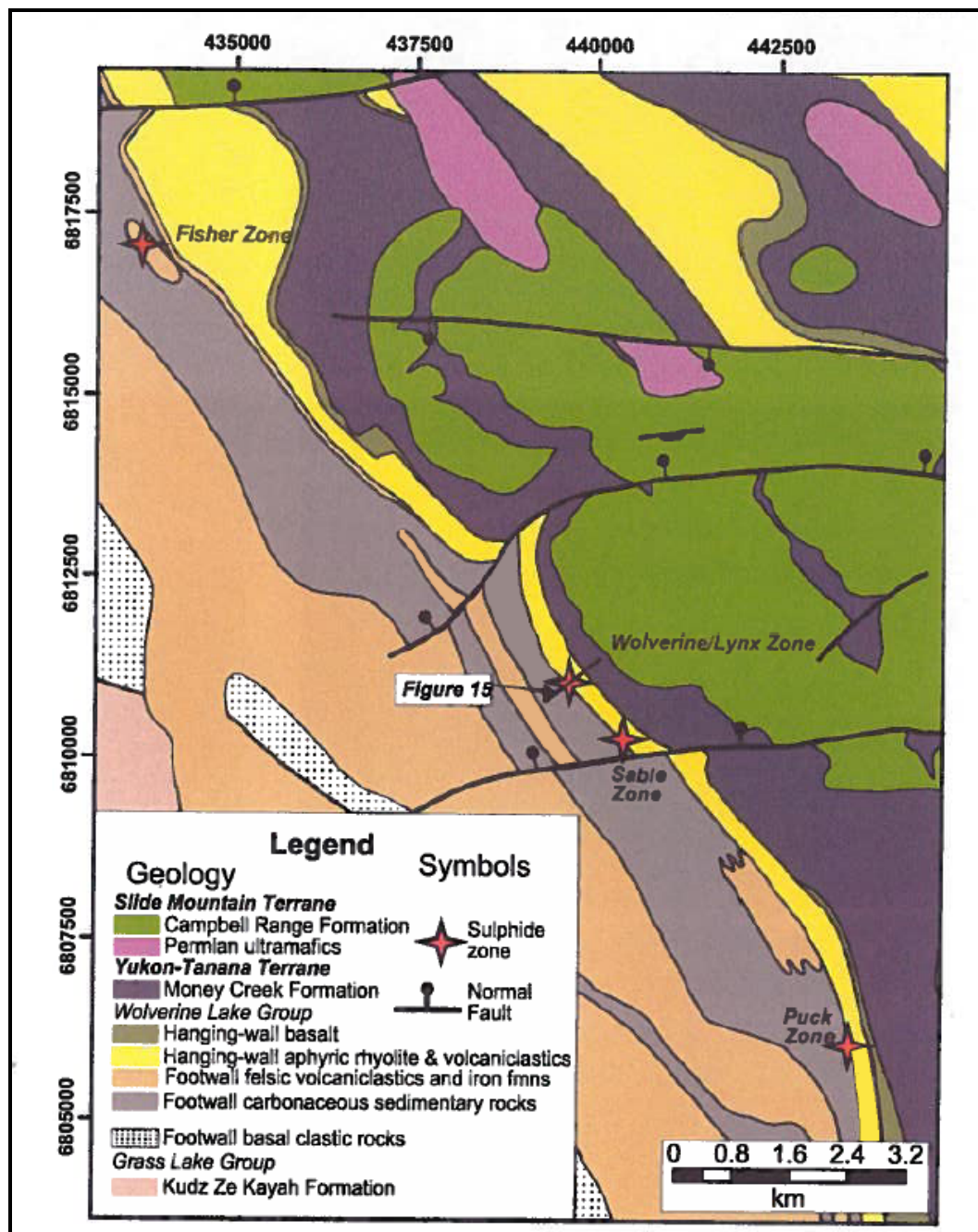
The Wolverine deposit is hosted by rocks of the Wolverine Lake Group; dominated by felsic volcanoclastic and carbonate sedimentary rocks, capped by mafic volcanic rocks. The mineralized stratigraphic contact within the stratigraphic sequence of the Wolverine Lake Group is referred to as the Wolverine Belt. Along this belt there are a series of zones of mineralization. These are located on strike northwest and southeast of the Wolverine deposit and named the Fisher, the Sable and the Puck zones (Figure 23.2) (Peter et al. 2007).

The Fisher zone, discovered in 1995 by Westmin Resources Ltd., is located 8 km northwest of the Wolverine deposit, and on the same apparent stratigraphic horizon, referred to as the Wolverine Group (Figure 23.2). This zone consists of numerous bands or lenses of sphalerite-pyrite-galena within altered felsic volcanic and volcanoclastic rocks. The footwall comprises felsic porphyry similar in appearance, composition and geochronology (346.0 ± 2.2 Ma) to a felsic porphyry in the Wolverine and Lynx deposit zones (347.8 ± 1.3 Ma) (Peter et al. 2007).

The Sable zone, located 1.6 km southeast and along strike of the Wolverine deposit, consists of thin intersections of high-grade massive sulphides and strongly silica-pyrite-altered and iron carbonate-altered rhyolite (Figure 23.2) (Peter et al. 2007). In the Sable zone area, exposures of strongly silica-pyrite altered rhyolite, quartz-feldspar phyric pyroclastic strata and the "breaching" of the lower iron formation by porphyritic rhyolite or tuff, suggests an area of felsic volcanic focus and/or a portion of a flow-dome complex. Iron carbonate alteration is also stronger developed in this area than elsewhere and may be related to focused hydrothermal activity. This unit is best developed toward the interpreted proximal environment to the east of Wolverine, with a thickness axis that contrasts with that of the other units (Tucker 1999). The previously mapped exhalite and rhyolite formation in this area suggests the strong continuity of the Wolverine Group rocks immediately in the hanging wall and footwall of the Wolverine deposit.

The Puck zone is immediately south of the Sable zone (Figure 23.2). It is composed of zinc-lead-copper sulphide stringer veinlets in sericitized quartz-phyric felsic volcanic rocks. Notable to this deposit is exhalative pyrite, magnetite-bearing iron formation and barite. Like the Fisher zone, the Puck zone also has footwall felsic porphyry rocks of similar geochronology (356.9 ± 0.5 Ma) (Peter et al. 2007). This unit, along with the overlying iron formations, is evidence of at least one exhalative system that perhaps had a relatively long, if episodic, history.

Figure 23.2 Surface Geology of the Wolverine Belt Showing Sulphide Zones with Respect to the Wolverine Deposit (from Peter et al. 2007)



24.0 OTHER RELEVANT DATA AND INFORMATION

This section is not applicable to this report.

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 INTERPRETATIONS AND CONCLUSIONS

25.1.1 GEOLOGY

The Wolverine deposit is a VHMS deposit located in the FD, Yukon. The FD forms a crescent-shaped 300 km x 50 km area of Palaeozoic volcanic, volcanoclastic and sedimentary rocks that extends from Watson Lake (south) to Ross River (north). The middle succession within the FD, called the Wolverine Succession, consists of carbonaceous argillite, felsic volcanics and high level intrusions, and as well as exhalative carbonate and/or iron oxides. The Wolverine polymetallic VHMS deposit occurs near the base of Wolverine Succession.

25.1.2 MINERAL RESOURCE

The geological interpretation and resource modelling were carried out using 3D geological modelling software provided by Gemcom Software Version 6.2.3. Grades were interpolated within the block model using the OK interpolation method with weighting parameters based on the semi-variogram data.

Tetra Tech estimated that the Wolverine deposit contains 4.140 Mt of Indicated resources averaging 12.91% Zn, 374 g/t Ag, 1.3% Cu, 1.83 g/t Au, and 1.7% Pb, at a US\$150 RMV cut-off. In addition, Tetra Tech also estimated that the Wolverine deposit contains 1.635 Mt of Inferred Resources averaging 11.8% Zn, 368 g/t Ag, 1.2% Cu, 1.56 g/t Au, and 1.6% Pb (Table 25.1).

Table 25.1 Wolverine Mineral Inventory

Classification	Cut-off (US\$)	Tonnes ('000 t)	Volume (m ³)	Zn (%)	Ag (g/t)	Cu (%)	Au (g/t)	Pb (%)
Indicated	150	4.140	1,045	12.9	374	1.3	1.83	1.7
Inferred	150	1.635	413	11.8	368	1.2	1.57	1.6
Contained Metals				Zn ('000 t)	Ag ('000 g)	Cu ('000 t)	Au ('000 g)	Pb ('000 t)
Indicated	-	-	-	534	1,548,360	54	7,576	70
Inferred	-	-	-	193	601,680	20	2,567	26

25.1.3 MINING

The Wolverine Mine possesses an economically minable resource with the potential for an increase in resource with additional diamond drilling. The current mine plan is

consistent with the production rates to meet the required feed to the processing plant over the life of the operation. Current mining practices will provide continuous supply of ore to the mill, but strict adherence to the GCMP is required to prevent ground failures that could lower or halt production. Several opportunities exist for decreasing operating cost such as relocation of the main access ramp, increasing the heading size where possible and reducing over break with perimeter control blasting. More detail on the cost savings is provided in Section 26.0. The proposed equipment fleet is expected to require 323,280 cfm. This sum is based on equipment utilization percentages and the Yukon Occupational Health and Safety Act regulations. The current fan is expected to provide 326,000 cfm when the mine has reached the 1070 Level. The actual airflow produced by the current fan will most likely be greater due to additional power from the auxiliary fans. If additional airflow is required above the current capacity of 326,000 cfm due to an increase in mine equipment, the current fan motor could be replaced by a larger model.

25.1.4 *ECONOMIC EVALUATION*

The economic evaluation of the project has a pre-tax cash flow of \$547 million and a NPV at 8% of \$444 million. The Wolverine Mine is currently in production, and therefore has no pre-production capital.

Table 25.2 shows a summary of payable metal in relation to the recovered metal in concentrates over the LOM.

Table 25.2 Payable Metals by Concentrate

Metal	Units	Metal Recovered to Concentrates (LOM)	Payable Metal (LOM)
Zinc	t	364,394	305,920
Copper	t	30,117	28,583
Lead	t	22,948	19,305
Silver	oz	27,192,984	22,925,410
Gold	oz	89,727	78,853

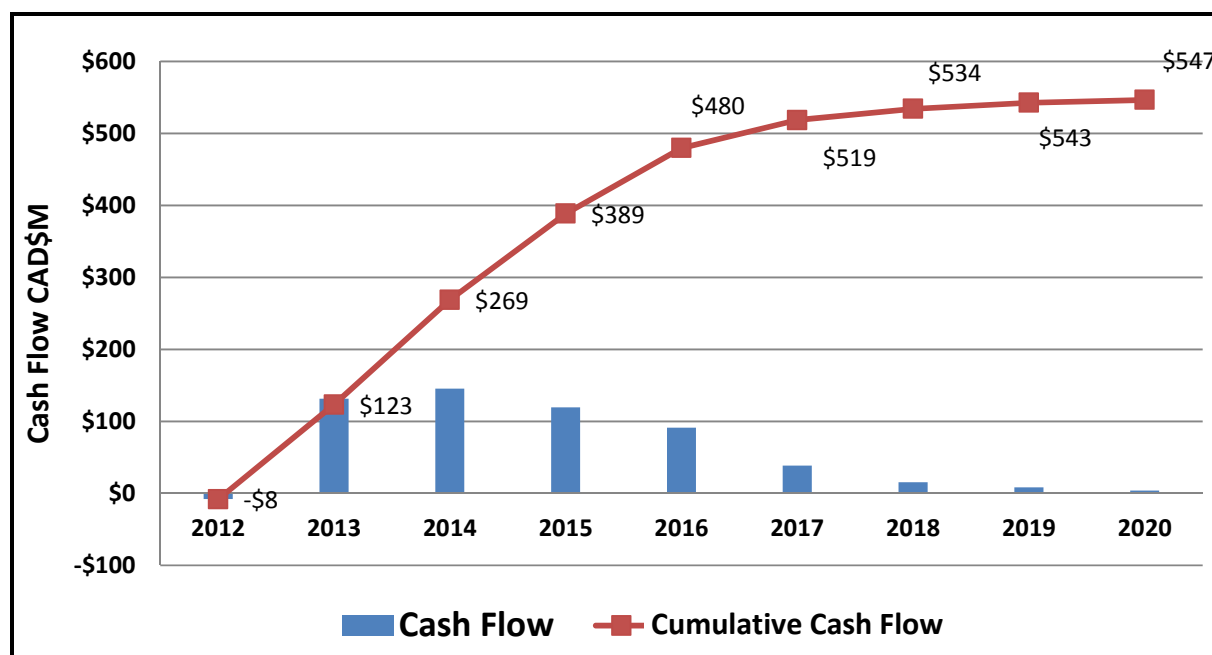
Table 25.3 shows a summary of varying metal prices versus varying NPV's.

Table 25.3 Pre-Tax NPV by Metal Price Scenario

Case	LOM Average Metal Prices					NPV (CAD\$000 000)		
	Zn (US\$/lb)	Cu (US\$/lb)	Pb (US\$/lb)	Ag (US\$/oz)	Au (US\$/oz)	@ 8%	@ 5%	@ 0%
Base +10%	1.18	3.370	1.16	26.60	1,483	564	607	694
Base	1.07	3.36	1.06	24.18	1,348	444	475	543
Base -10%	0.96	3.02	0.95	21.76	1,213	317	342	392

Figure 25.1 shows the mine annual pre-tax cash flow and the cumulative cash flow over the nine year LOM.

Figure 25.1 Annual Undiscounted Pre-tax Cash Flow



It can be seen from the tables above that the Wolverine Mine is economically viable. Tetra Tech's conclusion is that the Wolverine Mine is not only economically viable but is technically viable as well.

25.2 RISKS

The consequences of an event were classified according to the following categories:

- major
- moderate
- minor.

The likelihood of the occurrence of an event was rated according to the following categories:

- likely to occur
- possible to occur
- unlikely to occur.

Combining the category of consequence and likelihood of occurrence will result in the following risk ratings:

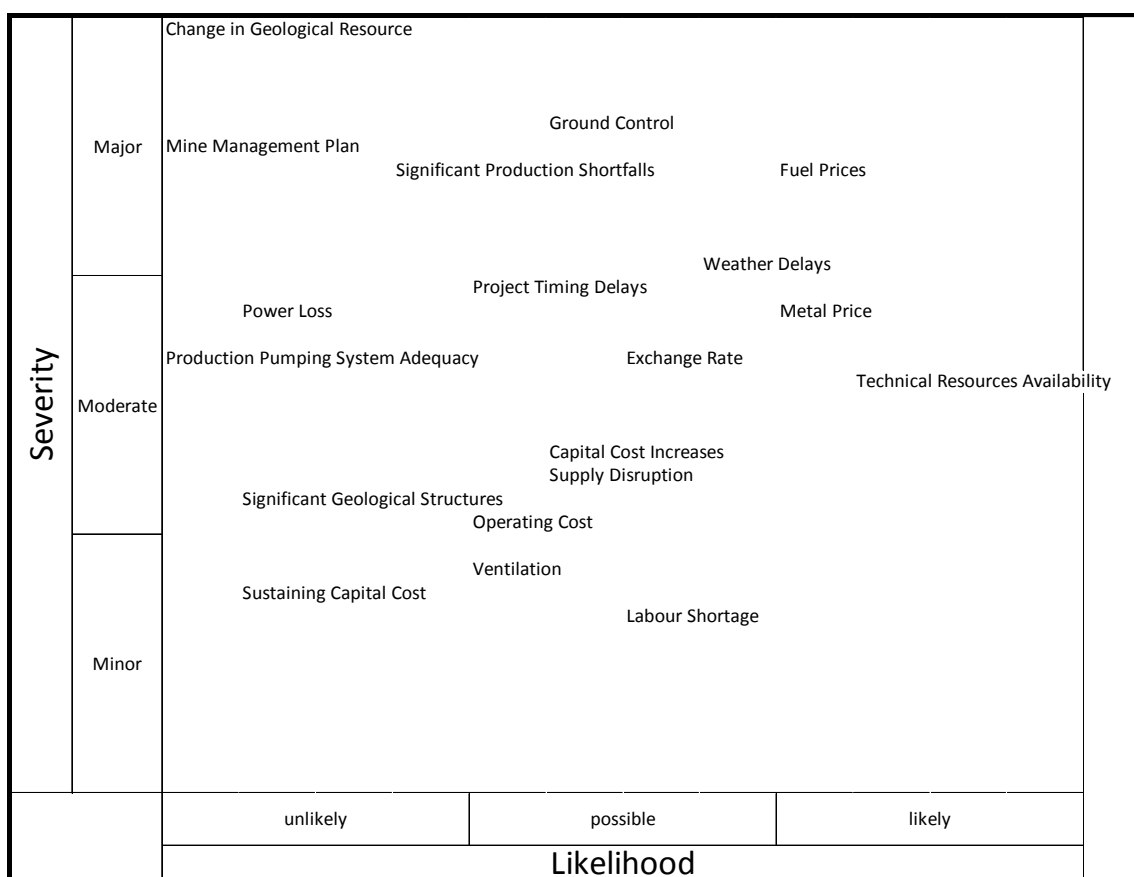
- low
- medium
- high.

The Figure 25.2 shows the events that were assessed in a risk assessment chart. Tetra Tech believes that very few events impose risk that would shut the operations down. For events rated with a medium to high risk (red triangle) a mitigation assessment was completed. The following list shows the events Tetra Tech rated as having the highest risks along with suggestions to mitigate against them.

- **Fuel Prices:** A significant increase in the fuel price would impose the highest risk to the Wolverine Mine. Since fuel prices are very volatile and operations at Wolverine depend to a high degree on the use of diesel fuel for power and equipment usage, an increase in fuel price can greatly affect the economics of the operation. To mitigate the risk of the fuel prices increasing Yukon Zinc could hedge their fuel supply on an annual basis. Another mitigation could be to evaluate the use of alternative sources of fuel such as ethanol or liquid natural gas (LNG). There is also potential for Yukon Zinc to consolidate fuel purchases with other Yukon based mining companies to increase volume to obtain better pricing and reduce transportation costs.
- **Ground Control:** Ground control failures, if severe, can shut down the mine and therefore impose a high risk to the operation. Tetra Tech strongly advises development advance should not exceed 3.5 m to prevent excessive spans of unsupported ground with potential for unraveling. To mitigate this risk Yukon Zinc has in place a well-designed GCMP. Close monitoring and inspections are performed daily by engineers and mining operators. Tetra Tech advises strict compliance to Yukon Zinc's GCMP.
- **Weather Delays:** Due to the relative remoteness and climate of the Wolverine Mine location weather delays are possible, with the consequence that resources or consumables are not able to reach the operation. To mitigate this risk, Tetra Tech advises maintenance of sufficient stockpiles of critical resources on site. To ensure that personnel currently flown to site can reach the mine site, Yukon Zinc should establish a contingency for labour transport via road.

- **Metal Prices:** Metal prices are very volatile, however due to the favorable economics of the Wolverine Mine a decrease in metal price will not impose a severe risk (shut down) to the mine. To further decrease the risk of unfavorable project economics due to a decrease in metal price, Tetra Tech proposes that Yukon Zinc should closely monitor and analyze operating cost for savings as well as maintain adequate liquidity to mitigate short price downturns or supply increases.
- **Technical Resource Availability:** Due to the shortage of technical resources in the mining industry it is possible that it will be a challenge to get skilled labour and technical resources to work at the mine. Good labour and technical resource pools can be found in foreign countries and since Canada's immigration policies facilitate the sponsorship of skilled foreign workers this would be a good option for Yukon Zinc. Tetra Tech also recommends that Yukon Zinc evaluate the effectiveness of different retention programs and implement the most optimal program at the Wolverine Mine.

Figure 25.2 Risk Assessment Chart



25.3 OPPORTUNITIES

Several opportunities for improving the economics of the Wolverine Mine exist. Since the majority have been discussed in detail in the recommendations section, the following list provides a summary of the opportunities Tetra Tech has identified.

- Additional exploration diamond drilling from underground could add to the overall resource and also convert much of the current 1.64 Mt of Inferred Resources into Indicated Resources, adding it to the available tonnage for milling.
- Additional exploration diamond drilling from both underground and surface, local to the deposit could add additional Inferred and Indicated tonnage to the resource. The resource model overall is considered open at depth and also towards the fringes of the ore body (see Section 7).
- Current test work underway at G & T Metallurgical Services indicates significant improvements may be achievable to the metallurgical response of the ore as being fed to the mill. This revision to the flotation regime, while unproven in practice, indicates the potential to produce better concentrates at higher metallurgical recoveries. While this is early in this set of test results, this change in regime, when added to the installation of a DMS plant could represent significant upside to the operation.
- There are a number of changes which could be made underground to improve productivity. Some of these possible changes are:
 - increasing the heading size where possible to increase the mining production rate
 - reducing over break with perimeter control blasting to reduce dilution of the ore and decrease ground support costs
 - using one equipment manufacturer to match equipment sizes and therefore increase productivity and reduce equipment maintenance costs
 - relocating the main ramp to allow more efficient hauling and increase productivity
 - instead of having one ramp in the Saddle deposit, creating two separate ramps from the existing central main ramp will create the possibility of multiple production faces from one level and lower the requirement of available levels to achieve the proposed mining rate
 - the continuous use and adjustment of the VFD on the primary fan will provide significant energy savings (this requires regular ventilation surveys and effective communication with operations staff).
- To significantly reduce the costs of production Yukon Zinc should change from using a mining contractor to owner operation underground. Tetra Tech has not had the opportunity to calculate potential costs savings and capital cost expenditures required to support this transition.

- Yukon Zinc can increase total resources from the Saddle deposit by changing the design of the ramp. Moving the ramp out of the Saddle mineralized zone will increase recovery of Saddle Zone ore, because ramp corridor pillars would not be left in the stopes. Such a move also has the potential to improve speed of mining of the ramp as it is not continuously crossing into and out of the ore zone.

26.0 RECOMMENDATIONS

26.1 GEOLOGY

The Wolverine deposit was initially drilled out from surface. The massive sulphide ore horizons were originally modelled by Snowden, Tetra Tech and AMC Mining Consultants to be laterally continuous. However, underground mapping and associated observations have demonstrated significantly more structural complexity. The ore zones have been thrust-faulted and rolled into each other causing stacking of massive sulphide positions, while simultaneously rolling associated hanging wall material into “chimneys” of broken ground. Further complications are noted with respect to some degree of horizontal swivelling in these thrust-thickened ore positions.

The structural complexity of the massive sulphide lenses cannot be confidently modelled and mined using the current surface-drilled sample density.

A detailed underground drill program may help further define ore zones. An estimated budget for the drilling program is shown in Table 26.1.

Table 26.1 Drilling Program Summary Costs

	Cost (\$)
Camp Operations	1,226,550
Drilling and Assay	8,075,000
Analysis and Reporting	571,500
Travel	231,000
Total	10,104,050

26.2 MINING

Tetra Tech has made several recommendations to improve mine design and productivity. To evaluate the validity of some of these recommendations, trade-off studies will have to be completed.

- Relocating the planned main access ramp entirely in the footwall would improve the overall project. This design improvement would eliminate repeatedly crossing the footwall contact with the main ramp infrastructure. Each development cycle across the contact will present geotechnical

challenges that may slow ramp development. Relocating the main ramp would allow the ramp to be more haulage friendly with more straight sections and less curves. Another positive to moving the ramp out of the Saddle mineralized zone is that it would increase recovery of the Saddle zone material because ramp corridor pillars would not be left in the stopes.

- Tetra Tech recommends a trade-off study be completed to examine the value of creating two separate ramps from the existing central main ramp. Each separate ramp would service one deposit. By doing this, it would create the possibility of multiple production faces from one level and lower the requirement of available levels to achieve the proposed mining rate.
- Opportunity exists to advance diamond drilling efforts ahead of the mining horizons being developed. This may improve the resources reserve potential by promoting inferred material into higher categories and possibly extending the mine life.
- When in mineralization and where geotechnically feasible, the heading sizes should be increased. Larger headings would result in more tonnes per blast, thereby decreasing the amount of active headings required to meet production requirements.
- Perimeter control blasting techniques present a strong opportunity at the Wolverine site. This control technique would reduce unplanned dilution by reducing overbreak. A further benefit would be the overall decrease in ground support effort and capital supplies.
- Using one manufacturer for mining equipment is recommended. This will help by standardizing the maintenance program, easing the various types of spare parts needed on site and may lead to a discount from the supplier.
- A low-cost web based CCTV system could be strategically placed during the fill cycle to allow observation of the fill pour. Any abnormalities during the fill pour could be addressed without tying up labour resources underground.
- Numerical modeling of the mining sequence is recommended to aid in assessing if the ground underneath the poured sill mat will pose any geotechnical challenges as it is later mined from below.
- The primary ventilation system is powered by a single fan. It is typical for two smaller fans to be used instead of one large fan. The use of two small primary fans allows the continued operation of the mine in the event of a primary fan failure. Currently, the failure of the primary fan would result in the shutdown of mining activities. The location of the Wolverine Mine also makes the replacement or repair of a primary fan more difficult. The failure of a mine air heater during winter would also cause the suspension of mining activity.
- Ventilation that is non-compliant with regulation 15.61(2)(b) could result in action from Yukon mine inspectors. Wolverine Mine has stated that the Yukon mine inspectors are satisfied with airflows as specified on the equipment permits and the TLVs. The Yukon Zinc Corporation Wolverine

Project Mine Ventilation Study, by W.V. Smyth, has recommended the mine obtain formal confirmation of this as acceptable if they plan to ventilate based on this strategy.

26.3 METALLURGY AND PROCESS

The following process matters require attention:

- confirmation that the plant can treat 1,700 t/d on a continuous basis
- the pre-flotation circuit requires characterization
- installation of a DMS circuit is recommended which has the potential to improve process results
- a review of the regrind product sizes and circuit configurations is required
- the establishing of realistic target concentrate grade values and recoveries is required
- the reclaim water system requires an overall review regarding potential deleterious side-effects to the process responses for the three circuits
- on-going metallurgical testwork by mine staff and outside facilities is recommended to assist in obtaining a detailed understanding of the ore characteristics and responses.

Each of these matters is presently under review by the mine staff.

27.0 REFERENCES

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28.0 CERTIFICATES OF QUALIFIED PERSONS

MARINUS ANDRE DE RUIJTER, P.ENG.

I, Marinus Andre de Ruijter, P.Eng., of Delta, British Columbia, do hereby certify:

- I am a Senior Metallurgical Engineer with Tetra Tech WEI Inc. with a business address at 800–555 West Hastings St., Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled “Technical Report on the Wolverine Mine”, dated May 30, 2012 (the “Technical Report”).
- I am a graduate of the University of Witwatersrand, Johannesburg, South Africa (M.Eng., 1979). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#31031). My relevant experience with respect to this project includes sulphide mineral flotation and gravity concentration and flotation research, and development work on cassiterite, wolframite, and chromite ores. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was September 27, 2011 for three days.
- I am responsible for the preparation of Sections 1.5, 13.0, 17.0, 18.5, 18.6, 26.3, 27.0, 28.0 of the Technical Report.
- I am independent of Yukon Zinc Corporation as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 8th day of August, 2012 at Vancouver, British Columbia.

*“Original document signed and sealed
by Marinus Andre de Ruijter, P.Eng.”*

Marinus Andre de Ruijter, P.Eng.
Senior Metallurgical Engineer
Tetra Tech WEI Inc.

PHILIP BRIDSON, P.ENG.

I, Philip Bridson, P.Eng., of Lively, Ontario, do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech WEI Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled "Technical Report on the Wolverine Mine", dated May 30, 2012 (the "Technical Report").
- I am a graduate of the Michigan Technological University (B.Sc., Mining Engineering, 1972 and B.Sc., Engineering Administration, 1972). I am a member in good standing of the Association of Professional Engineers of Ontario (#5181011). My relevant experience includes expertise in underground long range and short range mine planning and scheduling, financial evaluations, life of mine development and analysis, project evaluations and strategic planning. From 1972 to 2008 I worked at Canadian and US underground mines holding positions as a miner, Blasthole Engineer, First Line Supervisor, Design Engineer, Sr. Development Engineer, Sr. Project Engineer, Sr. Long Range Planning Engineer, Sr. Planning Supervisor and Sr. Business Planner. Since 2009 I have worked as a consultant mining engineer. As a Mining Engineer I have completed a variety of projects for mining companies involving scoping studies, pre-feasibility studies, feasibility studies, project evaluations, and life-of-mines for projects in Canada, US and Australia. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not visited the Property that is the subject of this Technical Report.
- I am responsible for the preparation of Sections 1.8, 1.9, 19.0, 21.0, 22.0, 25.1.4, 28.0 of the Technical Report.
- I am independent of Yukon Zinc Corporation as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 8th day of August, 2012 at Toronto, Ontario.

*"Original document signed and sealed by
Philip Bridson, P.Eng."*

Philip Bridson, P.Eng.
Senior Mining Engineer
Tetra Tech WEI Inc.

ROY LEVESQUE, P.ENG.

I, Roy Levesque, of Sudbury, Ontario do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech WEI Inc. with a business address at 101-957 Cambrian Heights, Sudbury, Ontario, P3C 5M6
- This certificate applies to the technical report entitled "Technical Report on the Wolverine Mine", dated May 30, 2012 (the "Technical Report").
- I am a graduate of Laurentian University, (B.Eng., Mining, 1999). I am a member in good standing of the Association of Professional Engineers of Ontario (#100072522). My relevant experience includes mine design, planning and scheduling. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was September 27, 2011 for three days.
- I am responsible for Sections 1.1, 1.4, 1.6, 1.10, 1.11, 2.0, 3.0, 15.0, 16.0, 18.1, 18.2, 18.3, 18.4, 18.9, 24.0, 25.1.3, 25.2, 25.3, 26.2, 27.0, 28.0 of the Technical Report.
- I am independent of Yukon Zinc Corporation as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 8th day of August, 2012 at Sudbury, Ontario.

*"Original document signed and sealed by
Roy R. Levesque, P.Eng."*

Roy R. Levesque, P.Eng.
Senior Mining Engineer
Tetra Tech WEI Inc.

ROBERT SINCLAIR MORRISON, PH.D., MAUSIMM (CP), P.GEO.

I, Robert Sinclair Morrison, Ph.D., MAusIMM (CP), P.Geo., of Toronto, Ontario, do hereby certify:

- I am a Lead Resource Geologist with Tetra Tech WEI Inc., with a business address at 900-330 Bay Street, Toronto, Ontario, Canada, M5H 2S8.
- This certificate applies to the technical report entitled "Technical Report on the Wolverine Mine", dated May 30, 2012 (the "Technical Report").
- I am a graduate of Acadia University, (B.Sc. 1981) and University of Adelaide (Ph.D. 1990). I am a Member in good standing of the Australasian Institute of Mining and Metallurgy (#11212), and I am registered as a Chartered Professional in Geology with the Australasian Institute of Mining and Metallurgy since 2004. I am a Member in good standing of the Association of Professional Geoscientists of Ontario (#1839) since 2010. My relevant experience with respect to deposit geology, ore body modelling and resource estimation includes 10 years with WMC Resources and Gold Fields Ltd as an Extensional Exploration Geologist, Senior Project Geologist, Resource Evaluation Geologist and Senior Resource Evaluation Geologist at the St Ives Gold Mine. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was September 27, 2011 for three days.
- I am responsible for Sections 1.2, 1.3, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 14.0, 23.0, 25.1.1, 25.1.2, 26.1, 27.0, 28.0 of the Technical Report.
- I am independent of Yukon Zinc Corporation as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 8th day of August, 2012 at Toronto, Ontario.

*"Original document signed and sealed by
Robert Sinclair Morrison, Ph.D.,
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Robert Sinclair Morrison
Ph.D., MAusIMM (CP), P.Geo.
Lead Resource Geologist
Tetra Tech WEI Inc.

MONICA DANON-SCHAFER, PH.D., P.ENG.

I, Monica Danon-Schaffer, Ph.D., P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am Manager, Environment and Sustainable Development with Tetra Tech Inc., with a business address at 800-555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled "Technical Report on the Wolverine Mine" dated May 30, 2012 (the "Technical Report").
- I am a graduate of the University of British Columbia (Ph.D, Chemical and Biological Engineering, 2010), University of British Columbia (M.Eng. Civil Engineering, 1993), and the Universidad Iberoamericana (B.Sc. Chemical Engineering, 1988). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#22768), Ontario (#100045213), Saskatchewan (#20307), Yukon (#1655), Northwest Territories/Nunavut (#L2018). My relevant experience includes preparation of environmental risk assessments, evaluation of Equator Principles, environmental health and safety compliance audits and due diligence at mines, mine waste and management. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 1.7, 18.7, 18.8, 20.0, 26.4, 28.0 of the Technical Report.
- I am independent of Yukon Zinc Corporation as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 8th day of August, 2012 at Vancouver, British Columbia.

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