Report to:

NORTH AMERICAN TUNGSTEN CORPORATION LTD

Amended Technical Report on the Mactung Property

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Report to:

AMENDED TECHNICAL REPORT ON THE MACTURN PROPERTY

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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	COV
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)	Cdn$
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

Amax Northwest Mining Co. Ltd.	Amax
Ammonium Nitrate and Fuel Oil	AN/FO
Ammonium Paratungstate	APT
Anvil Range Mining Corporation	Anvil Range
Barton's Tunnelling Quality Index	Q
Best Management Practices	BMPs
Build, Own, and Operate	BOO
Build, Own, Operate, and Transfer	BOOT
Canada Tungsten Mining Corporation	CTMC
Canadian Council of Ministers of the Environment	CCME
Canadian Dam Association	CDA
Capital Cost Estimate	CAPEX
China Tungsten Industry Association	CTIA
Climax Molybdenum	Climax
Closed Side Setting	CSS
Closed-circuit Television	CCTV
Coefficient of Permeability	K
Colorado School of Mines and Research Institute	CSMRI
Colorado School of Mines	CSM
Construction Management	CM
Distributed Control System	DCS
Down-the-hole	DTH
Dry-stacked Tailings Facility	DSTF
EBA Engineering Inc.	EBA
Effective Shear Strength Parameter	ϕ'
Elemental Tungsten	W
Engineering and Procurement	EP

Engineering, Procurement, and Construction Management	EPCM
Engineering, Procurement, and Construction	EPC
Environmental Effects Monitoring	EEM
Environmental Management System	EMS
Equivalent Dimension	De
Estimate at Completion	EAC
Excavation Support Ratio	ESR
Excavation Width	B
Factor of Safety	FOS
Ferrotungsten	FeW
Fixed Exchange Rates	FXR
Friction Factor	k-factor
Global Discovery Laboratories	Global Discovery
Goodall Business and Resource Management Pty Ltd	GBRM
Gränges Mineral Processes	Gränges
Gridded Seam Model	GSM
Hazard and Operability Analysis	HASOP
Health, Safety, and Environmental	HS&E
High-density Polyethylene	HDPE
Hunan Non-ferrous Metals	HNC
Impact-Benefits Agreements	IBA
Input/Output	I/O
Internal Rate of Return	IRR
In-The-Hole	ITH
Inverse Distance Cubed	ID^3
Inverse Distance Squared	ID^2
Inverse Distance	ID^1
King Island Scheelite	KIS
Lakefield Research of Canada Ltd.	Lakefield
Liard First Nation	LFN
Load-haul-dump	LHD
Local Study Area	LSA
Long-hole	LH
Mechanized Cut-and-Fill	MCF
Memorandum of Understanding	MoU
Metal Mining Effluent Regulations	MMER
Modified Stability Number	N'
Motor Control Centres	MCC
Na-Cho Nyak Dun	NND
National Air Quality Objectives	NAQOs
National Air Quality Surveillance	NAQS
National Instrument 43-101	NI 43-101
Net Present Value	NPV
North American Tungsten Corporation Ltd.	NATC
Northwest Territories	NWT
Operator Interface Station	OIS

Personal Computer Local/Wide Area Network	PC LAN/WAN
Piping and Instrumentation Diagrams	P&IDs
Playfair Mining	PLY
Points West Heritage Consulting Ltd.	Points West
Potassium Amyl Xanthate	KAX
Potentially Acid-generating	PAG
Process and Utility Flow Diagrams	PFDs
Process Design Criteria	PDC
Project Execution Plan	PEP
Proponent's Guide to Information Requirements for Executive Committee Project Proposals	Proponents Guide
Quartz Mining License	QML
Queensland Ores	QOL
Resistance Temperature Determination	RTD
Rock Quality Designation	RQD
Ross River Dena Council	RRDC
Rotating Biological Contactor	RBC
Run-of-Mine	ROM
Scott Wilson Roscoe Postle Associates Inc.	Scott Wilson RPA
Sewage Treatment Plant	STP
Shape Factor	S
Single Line Diagrams	SLDs
Socio-economic Participation Agreements	SEPAs
Specific Gravity	SG
Standard Penetration Resistance	N
Tailings Storage Facility	TSF
Three-dimensional	3D
Tungsten Tri-oxide	WO_3
Two-dimensional	2D
Valued Socio-Economic Components	VSECs
Vegetation Local Study Area	VLSA
Visual Flight Rules	VFR
Voice-over-internet Protocol	VoIP
Wardrop Engineering Inc.	Wardrop
Work Breakdown Structure	WBS
X-ray Diffraction	XRD
Yukon Environmental and Socio-economic Assessment Act	YESAA
Yukon Environmental and Socio-economic Assessment Board	YESAB

1.0 SUMMARY

This Technical Report describes the scope, design features, and economic viability of the Mactung project. The property, owned by North American Tungsten Corporation Ltd. (NATC), is located in southeast Yukon, Canada and is adjacent to the Northwest Territories (NWT).

The project is an underground mine utilizing a combination of long-hole blast and mechanized cut-and-fill mining methods. The ore is delivered to the mill by conveyor and subsequently processed through gravity and flotation concentration circuits at the rate of 2,000 tonnes per day (t/d). Construction will occur over a 27 month period at a capital cost of $402.1 million; commercial production is scheduled to commence during the first quarter of 2013, and will continue for 11.0 years. All currencies in this report are expressed in Canadian dollars (CDN$) unless otherwise noted.

For the expected 11.0 year mine life and 8 million tonne (Mt) reserve, Wardrop Engineering Inc. (Wardrop) used the Goodall Business and Resource Management Pty Ltd. (GBRM) preliminary market review that was commissioned by NATC as source for tungsten metal price for the economic evaluation of the Mactung project.

The project is expected to produce an average of 640,000 metric tonne units (mtu) (14.0 million pounds (lb)) of tungsten for each year of the expected mine life. Based on GBRM's market forecast of US$255/mtu for gravity concentrate, US$300/mtu for Ammonium Paratungstate (APT) concentrate, and an exchange rate of 0.88 (US$/CDN$), the project has a pre-tax Internal Rate Return (IRR) of 23.5% and a pre-tax Net Present Value (NPV) of CDN$276.8 million at an 8.0% discount rate, with recovery of capital within 2.9 years.

The Mactung project will draw skilled workers from the operating Cantung mine, which is owned and operated by NATC. On-site diesel-powered generators will generate electrical power for operations.

The Mactung scheelite skarn deposit contains mineral reserves of 8.5 Mt, at an average grade of 1.082% WO_3. Long-hole stoping enables mining of the majority of the ore at a high production rate, while mechanized cut-and-fill stoping will mine the thinner and steeper ore.

The design of the process plant is based on a series of metallurgical tests conducted in 1985, along with processing experience gained from the existing Cantung Mine. On average, the plant is designed to recover 82% of the metal in the form of a scheelite concentrate with a gravity concentrate grade of 67% WO_3 and a flotation concentrate grade of 55% WO_3.

The dry-stack nature of the tailings provides backfill for the underground mine operation, eliminates the need for a permanent tailings dam and impoundment, and allows for the re-establishment of the natural drainage paths at the end of the mine life. An application for assessment has been filed through the *Yukon Environmental and Socio-economic Assessment Act* (YESAA) process during the fourth quarter of 2008. Extensive local community and First Nations consultation programs are ongoing.

1.1 GEOLOGY

NATC retained Scott Wilson Roscoe Postle Associates Inc. (Scott Wilson RPA) in 2005 to prepare an updated mineral resource estimate and a National Instrument 43-101 (NI 43-101)-compliant Technical Report on the Mactung tungsten deposit.

Scott Wilson RPA visited the property on August 19, 2005 and August 8, 2007. The following sub-sections 1.1.1 (Mineralization) and 1.1.2 (Mineral Resource Estimate) were extracted from Scott Wilson RPA's Technical Report.

1.1.1 MINERALIZATION

The Mactung mineralization can be characterized as a metasomatic skarn deposit formed by magmatic hydrothermal fluids originating from a Cretaceous granitic stock. The deposit comprises an Upper and Lower mineralized skarn zone separated by 100 m of hornfelsed pelitic sediments. The Lower zone, while dipping in the same general direction as the Upper zone, contains a "Z" fold, with an amplitude of about 90 m.

Scheelite is the economic mineral of interest at Mactung; wolframite is reported only occasionally. Scheelite occurs predominantly with pyrrhotite in the pyroxene-pyrrhotite facies, wherein the scheelite content increases and grain size decreases with pyrrhotite content. Minor scheelite also occurs in the garnet facies, and is coarser grained than that of the pyrrhotite facies.

1.1.2 MINERAL RESOURCE ESTIMATE

The kriged estimate contains an indicated mineral resource of 33.0 Mt grading 0.88% WO_3, or 290 kilotonnes (kt) of contained WO_3. An additional resource of 11.9 Mt grading 0.78% WO_3, or 92 kt WO_3, has been estimated for the inferred category. These estimates, which are based on assays capped at unique levels for each zone, are reported at a block cut-off of 0.5% WO_3, which Scott Wilson RPA considers appropriate for the location and cost profile that can be expected for Mactung.

Table 1.1 Indicated and Inferred Mineral Resource Estimate

Classification	kt	WO_3 (%)	WO_3 (kt)	mtu's (millions)
Indicated	33,029	0.88	290	29.0
Inferred	11.857	0.78	92	9.2

Notes:
- CIM definitions were followed for mineral resources.
- Mineral resources are estimated at a block cut-off grade of 0.5% WO_3.
- An mtu is 10 kg WO_3.
- Differences in totals due to round-off.
- There are no measured mineral resources in the estimates.

1.1.3 MINERAL RESERVE ESTIMATE

Wardrop calculated the mining cut-off grade at 0.616% WO_3 based on the cost estimates obtained from Wardrop's economic study of the Mactung Project in October 2007. The calculation of the mineral reserves was limited in extent to the Yukon-NWT border.

Table 1.2 Mineral Reserve Estimate

	Upper 2B		Lower 2B		Total	
Classification	kt	WO_3 (%)	kt	WO_3 (%)	kt	WO_3 (%)
Probable	8,588	1.1268	2,202	1.4213	10,790	1.1869

1.2 MINING

Two underground stoping methods will be used: long-hole (LH) stoping and mechanized cut-and-fill (MCF). The use of these methods is determined by the geometry, dip, thickness, and strength of rock. LH will mine 89% of the ore and the remaining 11% will be extracted by MCF. Table 1.3 shows the annual production rate, ore grade and tungsten concentrate.

Table 1.3 Production Rates and Grades

Duration	Ore Mined (tonnes)	Head Grade (% WO_3)	Recovery (%)	Dry Tungsten Concentrate (000 mtu)
Years 1 - 5	3,650,000	1.25	81.7	3,744
Years 6 - 12	4,525,000	0.95	78.5	3,371
LOM (Total)	8,175,000	1.09	80.0	7,115

The key design criteria included in the mining plan are:

- The underground mine design is based on the indicated mineral resources inside Yukon.
- A cut-off grade of 0.616% WO_3 has been used in the mine design.
- LH stoping will be the primary mining method, and will achieve high productivity and low operating costs.
- LH stopes are designed at 17 m wide and 60 m long, with 4-m rib pillars between the primary and secondary stopes. There will be a 4-m transverse pillar between stopes on the same stope line.
- MCF stopes are designed at 17 m wide, with 3-m permanent rib pillars along the strike of the orebody in areas 12 m thick or less, and/or in areas dipping more than 20%.
- Several stopes will be mined simultaneously to meet the planned production rate of 2,000 t/d and to improve blending of ore for grade control.
- LH and MCF mined out stopes will be backfilled using dewatered mill tailings and waste from the waste developments to optimize ore recovery and stabilize the ground.
- The underground mine will be operated in marginal permafrost conditions.

Ore will be hauled and dumped through an orepass to an underground crusher station, where it will be crushed by a primary jaw crusher. Crushed ore will be transported by a 512-m long conveyor belt up an incline to an ore bin at the surface. Dewatered tailings will be transported underground by a conveyor belt running in parallel with the ore conveyor in the same decline. The tailings will be deposited in a stockpile at the crusher station, where it will be loaded into haul trucks and hauled to the mined-out stopes as backfill.

The mine will operate without heating of ventilation air flow to preserve the marginal permafrost conditions. In the event permafrost conditions are not sustained during the summer, pumps will be installed to dewater the underground workings.

Mining operations will be carried out with an equipment fleet consisting of two long-hole down-the-hole (DTH) production drills, three 5-m^3 load-haul-dump (LHD) units, and five 30-t haul trucks. The primary equipment will be supported by development equipment, rock bolters, graders, ANFO loaders, dozers, and personnel carriers.

1.3 METALLURGY AND MILL

The Mactung project process plant will treat scheelite ore at the rate of 2,000 t/d, or 730,000 tonnes per year (t/a). The plant design is based primarily on metallurgical testwork conducted by Lakefield Research in 1985, and processing experience from the existing Cantung Mine in the NWT.

The feed to the process plant will have a nominal tungsten grade of 1.30% WO_3, which could vary from as low as 1.00% WO_3 to as high as 1.75% WO_3. The total scheelite concentrate recovery will be 82%. The process will consist of a gravity circuit, which will recover about 55% of the total scheelite concentrate at a grade of 67% WO_3, while the flotation circuit will recover about 27% of the total scheelite concentrate with a grade of 55% WO_3. Head grades below 1.30% will reduce the recovery to less than 82% recovery, while higher grades will conversely result in higher recoveries.

The treatment processes as designed will involve comminution followed by the rejection of sulphide minerals and the concentration of scheelite. Sulphide minerals will be rejected via wet magnetic separation and several sulphide flotation stages. The scheelite concentrate from the gravity circuit will be further upgraded through sulphide flotation and dry magnetic separation methods to reach the target tungsten grade and impurity specification limits. Scheelite flotation concentrate will receive no further upgrading at the mine.

All plant tailings will be thickened, filtered, and then either transported to the underground workings as back-fill, or to the surface tailings disposal location. Process water from the tailings thickener overflow will be directed to the ageing pond, where it will remain for 30 days before re-use so as to mitigate the possible influence of residual reagents and chemicals on the process.

The processing facilities will include the following unit operations:

- three-stage crushing processes with the primary crushing stage situated underground and the secondary and tertiary crushing stages situated at the processing plant on the surface
- a two-stage grinding circuit
- thickening process
- a two-stage wet magnetic separation process
- bulk sulphide flotation circuit, which includes the sulphide concentrate regrind stage and the cleaner sulphide flotation circuit
- bulk sulphide flotation tailings/gravity plant feed classification step
- coarse gravity separation circuit including a two-stage spiral separation stage, followed by a two-stage table cleaning/upgrading tabling stage
- fine gravity separation circuit including a two-stage spiral separation process, followed by a two-stage table cleaning/upgrading step
- regrind circuit for the tailings and middlings from the coarse gravity separation circuit, and the table middlings only from the fine gravity separation circuits, followed by the sulphide flotation in the gravity regrind sulphide flotation circuit
- gravity circuit concentrate sulphide flotation circuit

- gravity circuit concentrate dewatering circuit, including wet magnetic separation, concentrate drying, a dry magnetic separation process and the concentrate packaging
- thickening of scheelite flotation feed
- scheelite flotation circuit, including the rougher, scavenger, and three stages of cleaner flotation circuit
- scheelite flotation concentrate dewatering, drying, and packaging
- final tailings thickening, filtering for backfill and/or for dry stack disposal.

1.4 TAILINGS DISPOSAL

Mactung tailings will be thickened and filtered to facilitate their disposal either as backfill material underground, or as dry stacked tailings. Half of the tailings material will be returned underground via a conveyor belt system and used as backfill in mined-out stopes. The remaining 50% of tailings material will be trucked to a Dry-stacked Tailings Facility (DSTF).

Mactung's filtered tailings will be placed as backfill material in the mined out stope in an unconsolidated form. The tailings have to be free draining and testing has to be performed to confirm the percolation rate. The tailings material is recommended to have a minimum percolation rate greater than 10 cm/h (Hassani & Archibald 1998).

The DSTF will be constructed as a sidehill fill structure with a 25% grade. The facility will be constructed through a series of 600 mm lifts, which will then be compacted to 95% of the maximum dry density as determined by ASTM D698 using a vibratory compactor to ensure maximum stability. To further ensure structure stability an upstream diversion berm will be constructed to divert surface run-off water around the structure.

A dam will be constructed downstream of the mine's footprint in order to collect all runoff from the tailings facility, and to create a catchment to age the process water before reuse. The ravine dam was designed to a 1:100 year flood event and a 1:1000 year seismic event, as suggested by the Canadian Dam Association for structures of significant consequence. Material for dam construction will be a combination of locally-available materials and geosynthetics. Geosynthetic materials must be used due to a lack of high-quality aggregates and low-permeability core material. These materials will be primarily used to construct drains and the dam's core, and for erosion protection.

1.5 INFRASTRUCTURE

The infrastructure planned at Mactung to support the mining and processing operations includes:

- Forty-eight km of access roads to provide access from the existing North Canol Road to the project site. Based on fieldwork conducted in May 2008, it was recommended to undertake a follow up assessment of snow melt in the area and to develop a hazard management plan for the access road and mine site. As no avalanche protection was incorporated into the road design, subsequent engineering work will plan to re-align the road route to minimize potential avalanche hazards.
- An airstrip, which is owned and maintained by the Government of Yukon. NATC will upgrade the airstrip to 1,375 m long by 30 m wide to accommodate a Beechcraft 1900 or similar aircraft.
- Fresh water that will be pumped from the Hess River tributary to the plant site by a pipeline approximately 10-km long that is insulated and heat traced. The fresh water and fire water will be stored in one 10 m by 11 m diameter fresh/fire water tank. Potable water will be stored in a 4.2 m by 4.2 m diameter tank. The fresh water and process tanks will be insulated and heated. A hypochlorinator will be provided for water treatment.
- On-site waste disposal to minimize or eliminate the requirement for off-site waste removal services. Grey water will be piped through the heated utilidors to a sewage treatment plant. Solid waste will be transported to the dry stack tailings area and deposited along with the tailings.
- Ancillary facilities consisting of the truckshop/warehouse, administration building/mine dry, bunkhouse complex, and mess hall.
- A power plant consisting of five diesel generators with heat recovery modules located adjacent to the process plant to minimize power distribution losses and connection costs. There will be sufficient recovered heat to heat surface facilities in the winter. Portable propane heaters will provide heat for surface facilities when waste heat is not available from the power plant.
- Electrical power distributed throughout the mine site at 4.16 kilovolts (kV). Power will be delivered to the underground operations through a single 4.16 kV underground tunnel power cable.
- A telecommunication system will have adequate data, voice, and other communication channels available. The system will include:
  - a voice-over-internet protocol (VoIP) telephone system
  - satellite communications for voice and data
  - ethernet cabling and wireless internet access
  - leaky feeder communication system in the underground mine
  - satellite TV.

- Fuel storage and distribution.
- Propane storage.

1.6 YUKON'S ASSESSMENT AND PERMITTING REGIME

The Mactung project will be reviewed under YESAA, a federal act that sets the process and conditions for environmental and socio-economic effects assessments. The Mactung project will trigger an Executive Committee Screening to be conducted by the Yukon Environmental and Socio-economic Assessment Board (YESAB).

The YESAA process has specified timelines associated with work conducted by the Board. Efficiencies in the timing of the assessment may be achieved through:

- completeness of the project proposal
- responsiveness of the proponent
- clear description of the project
- accessibility of the proposal's content
- active communication with groups to be consulted.

NATC has collected an extensive amount of environmental baseline data from the site to date and continues to collect additional data to ensure adequacy of the project proposal for the assessment process.

The project will require two main permits: a Type A Water Licence and a Quartz Mining License (QML). The Type A Water Licence, is issued through the Yukon Water Board in accordance with the Yukon's *Water Act*. The process to obtain this permit is not expected to pose a significant hurdle for the project.

The QML adheres to the Government of Yukon's *Quartz Mining Act*, and is administered by Government of Yukon's Department of Energy, Minerals, and Resources. The QML is issued in two parts to advance non-water related works associated with the project. No significant obstacles are anticipated through this process.

Other permits are required at different stages of the project; a comprehensive list of required licenses can be compiled once more detailed engineering has been completed.

1.7 RECLAMATION, DECOMMISSIONING, AND CLOSURE

NATC understands and respects the importance of minimizing the environmental impact of the mining project. All practices of mining, processing, and auxiliary operations have been designed in accordance with best practices and an end goal of returning the land as close to its natural state as practical. The closure plan and environmental management system being developed by EBA Engineering Inc. (EBA) will provide for the site to be remediated to an acceptable standard.

Progressive reclamation activities will be carried out throughout the mine life, but the major reclamation work will take place once mining is completed. Mine closure reclamation will leave the site in a stable state with all measures taken to mitigate possible contamination and negative effects on the land.

At the close of production all on-site structures will be removed, with all salvageable parts sold or recycled and the remainder disposed of appropriately, some non-salvageable waste and structures may be placed in the underground workings. All hazardous material will be removed from site and appropriately disposed of.

Responsibility for the resource access road is expected to be handed over to the Government of Yukon if the road is deemed an asset.

Closure will require that the dry-stacked tailings pile will be properly capped in accordance with industry best practices. The process water aging and runoff storage pond will be recontoured, and the natural runoff channels will be re-established by decommissioning the Ravine Dam. A directed program will be put in place to ensure that the natural drainage pattern is returned to the site, including making the existing stream continuous once again from its headwaters to the confluence of Tributary A.

Periodic inspection and monitoring of the site post after closure will be required pursuant to the Metal Mining Effluent Regulations (MMER) for a period of three years following closure.

1.8 PROJECT EXECUTION

The Project Execution schedule includes the following key milestones:

- Receipt of environmental approvals and permits
  - YESAA application submission – Q4 2008
  - Yukon YESAA Decision Documents issued – Q1 2010
  - QML permit issued – Q2 2010
  - Detailed Engineering completed – Q2 2010
  - Water Licence issued – Q3 2010
- Procurement of long lead items:
  - Order crushers/grinding mills – Q3 2009
  - Order other process equipment – Q1 2010
  - Order major mining equipment – Q1 2010

- Construction activities:
  - Mobilize to site – Q2 2010
  - Complete airstrip upgrade – Q3 2010
  - Complete access road – Q4 2010
  - Complete camp – Q4 2010
  - Commence pre-production mine development – Q1 2011
  - Complete site development – Q1 2012
  - Complete power plant/power lines – Q2 2012
  - Complete underground conveyor – Q2 2012
  - Complete tailings area/dam – Q3 2012
  - Plant start-up/commissioning – Q4 2012
  - Commercial production – Q1 2013.

1.9 CAPITAL COST ESTIMATE

The capital cost for the project is estimated to be CDN$402.1 million in Q3 2008 dollars as shown in Table 1.4. Costs in this report have been converted using a fixed currency exchange rate of CDN$1.00:US$0.88. The expected accuracy range of the capital cost estimate is ±15%.

Table 1.4 Project Capital Costs (Q3 2008 $)

Description	Cost (Million $)
Direct Costs	
Overall Site	32
Mining	40
Processing	91
Tailings	23
Site Services and Utilities	15
Ancillary Buildings and Equipment	35
Temporary Services	16
Off-site Infrastructure and Facilities	22
Direct Works Subtotal	**274**
Indirects	
EPCM and other Indirects	61
Owners Costs	21
Contingencies (%)	46
Indirects Subtotal	**128**
Total Project	**402**

The capital costs exclude allowances for escalation for the duration of the project.

1.10 Operating Cost Estimate

On site operating costs were estimated at $103.65/t of ore milled as shown in Table 1.5.

Table 1.5 Operating Cost

Area	Unit Cost ($/t milled)
Mining	38.14
Processing	36.39
General and Administrative	29.13
Total Operating Cost	**103.65**

1.11 Preliminary Market Review

Wardrop used the GBRM market study that was commissioned by NATC as the source for tungsten metal price for the economic evaluation of the Mactung project. This summary was taken from the GBRM's market study entitled "A Preliminary Review of Tungsten".

1.11.1 Production and Supply

Tungsten prices (both concentrate and APT) remained extremely low during the 1990s and in fact well below the true cost of production. As a direct result, exploration and mine development programs were almost totally abandoned.

However, commencing in the early part of this decade, the global market structure for tungsten began to change, particularly driven by a rapid increase in domestic demand of tungsten products in China. This signalled to the global market, two key issues:

- The rapid increase in demand in China would tighten raw material availability to other markets, particularly given the Chinese Government policy of curtailing mining programs to maintain reserves for future domestic requirements.
- The Chinese program of developing downstream processing would place increased pressure on processing companies outside China. In order to remain competitive and secure sufficient base concentrates, these companies urgently needed to develop alternative supplies outside China.

Recognition of these circumstances has since led to a significant increase in exploration and mine development activities in Vietnam, Australia, and in North and South America. However despite this increased activity, no new major production has actually been realized and this is unlikely to occur until 2010 at the earliest.

The lack of new production has been recognized by Chinese processors which have recently entered into a number of agreements with companies such as King Island Scheelite and NATC in order to assist development through input of capital and off-take agreements.

1.11.2 CONSUMPTION

The consumption of tungsten is reasonably broad based, both in industrial and geographical terms. Major applications are metal-cutting tools, drill bits, light bulb filaments, high temperature alloys, x-ray shielding, military use, and chemical applications. Regionally, the largest consuming area is China, followed by Europe and North America. China also has by far the fastest growth; however India is also showing signs of rapid growth in demand albeit from a low base.

Over the next five years to the end of 2012, global consumption of tungsten is expected to increase from its current level of approximately 85,000 t W (64,500 t W virgin tungsten) tonnes per annum to 110,000 t W (82,000 t W virgin tungsten) tonnes. Mature markets such as Europe and North America are only expected to grow by an average of 2% per annum.

Chinese domestic consumption is forecast to continue growing in excess of 10% per annum, largely driven by the increase in requirements for cutting and drilling tools. However, it should also be noted that China's growth over the past five years has averaged 15% and if these levels were to be maintained world tungsten demand will exceed 122,000 t W (91,500 t W virgin demand) by 2012.

1.11.3 PRICE STRUCTURES

As a result of the constant over-supply situation through the 1990s and early 2000s, prices fell to low levels of US$45/mtu for concentrates and only marginally higher for APT. However, a number of important events have since occurred, which has greatly influenced current and forward price structures.

China has not only curtailed domestic mining programs, but has now become a significant importer of tungsten concentrates and tungsten scrap. Also the Chinese Government has moved from a position of providing export incentives for tungsten exports to now introducing production and export quotas, and consistently increasing export tariffs.

The net result of these changes coupled with the continuing growth in global demand has resulted in strong price increases commencing in 2004, to a current level of approximately US$220/mtu for concentrates and US$250-$260/mtu for APT (quoted price by Asian Metal News May 2008).

With the Chinese Government strongly encouraging an even higher level of downstream processing, with a large percentage of these downstream products being consumed domestically, plus expected further increases in export tariffs for semi processed products, the availability of tungsten units to markets outside China, in all forms, will continue to tighten. Furthermore, China has now become a major importer of tungsten concentrates and scrap.

The net result of these changing circumstances, continued strong growth in China's consumption, the recognition of increasing costs to develop and operate mines outside China, plus the fact that tungsten has a 'high value in use' in most applications and substitution is unlikely, result in the conclusion that a continuing escalation in price structures is highly likely. During the next five years, it is forecast that global prices for APT will reach or even exceed US$300/mtu.

1.11.4 *Opportunity for New Producers*

The projected strong growth in tungsten demand over the period 2007-2012 will require the development of between 33-45 kt W of new production.

Analysis of current projects that are in the feasibility study stage indicates that even if the majority come on stream, the market will continue in tight supply through to 2012 and onwards. Many of the tungsten projects that are in the early stage of assessment that could be brought into production from 2012 onwards, are based on much lower WO_3 grades than the projects currently under feasibility and will require stable tungsten prices at high levels to be feasible.

Consequently there is a definite requirement for the development of larger (+4,000 t/a W) mines with 15-20 year mine life in order to support long-term market demand. These types of world-class deposits are rare and of the projects under feasibility study only. NATC's Mactung deposit is capable of achieving these production rates over 10 years plus.

This project is currently scheduled to come on line in 2013, with production of 30,000 t W over its initial 5 years of operation. Projections are that the market will clearly require this additional production by 2013 and could in fact accommodate an earlier start-up.

In addition to Mactung there are only four other projects under feasibility: Vital Metals Watershed Project (Australia), King Island Scheelite's King Island Project (Australia), the Nui Phao deposit in Vietnam, and the Hemerdon deposit in the UK that are projecting production levels of between 2,000-4,000 t/a W.

Barriers of entry for a new producer outside China with a good grade tungsten deposit are expected to be minimal. In the projected strong market, long term letters of intent to purchase either concentrates or APT should be readily achievable and these would assist to underwrite project finance.

1.11.5 PRELIMINARY MARKET REVIEW CONCLUSIONS

The global market for tungsten is forecast to maintain strong growth over the next five years and beyond. While China continues to dominate world mining and primary processing, the availability of tungsten units to non Chinese markets will continue to decline. A strong escalation in prices has already occurred over the past three years. However with producers struggling to meet demand, global mining costs continuing to increase, mining grades dropping, and the Chinese Government likely to impose tighter production quotas and higher export tariffs to maintain reserves, further global price escalation appears certain.

Barriers of entry for new producers are relatively low apart from actual development costs, and new mine projects will continue to receive strong encouragement from processors both in and outside China.

While there are a number of other projects at various stages of development, from exploration to bankable feasibility study, the timing to bring these projects into production is proving to be considerably longer than originally projected.

Consequently, for the long term stability of supply, consumers are keen to form relationships that will assist in the development of the larger, longer term producers with good grade tungsten deposits.

1.12 FINANCIAL ANALYSIS

Wardrop based the economic evaluation of the Mactung project on a pre-tax and post-tax financial model. For the expected 11.2-year mine life and 8.0 Mt reserves, Wardrop used the GBRM Preliminary Market Review that was commissioned by NATC as the source for tungsten metal price for the economic evaluation. Wardrop calculated the following base case pre-tax financial results based on GBRM's forecast of tungsten.

- 23.5% IRR
- 2.9 years payback on CDN$402.1 million capital
- CDN$276.8 million NPV at 8% discount value.

The base case is calculated using an APT price of US$300/mtu. Refer to section 26.0 Market Review for more detailed commodity pricing information.

The post-tax model was provided by NATC and audited by W.H. Taylor Inc. calculating the following:

- 17.6 % IRR
- 3.2 year payback on US$402.1 M initial capital
- CDN$ 147.7 M NPV at an 8.0% discount rate.

Sensitivity analyses were carried out to evaluate the project economics. These analyses are presented graphically as financial outcomes in terms of NPV and IRR. The project NPV is most sensitive to the foreign exchange rate and in decreasing order: head grade, price, operating costs, and capital costs.

Similarly, the project IRR is most sensitive to foreign exchange rate, head grade, and initial capital costs.

Figure 1.1 Pre-Tax NPV Sensitivity Analysis

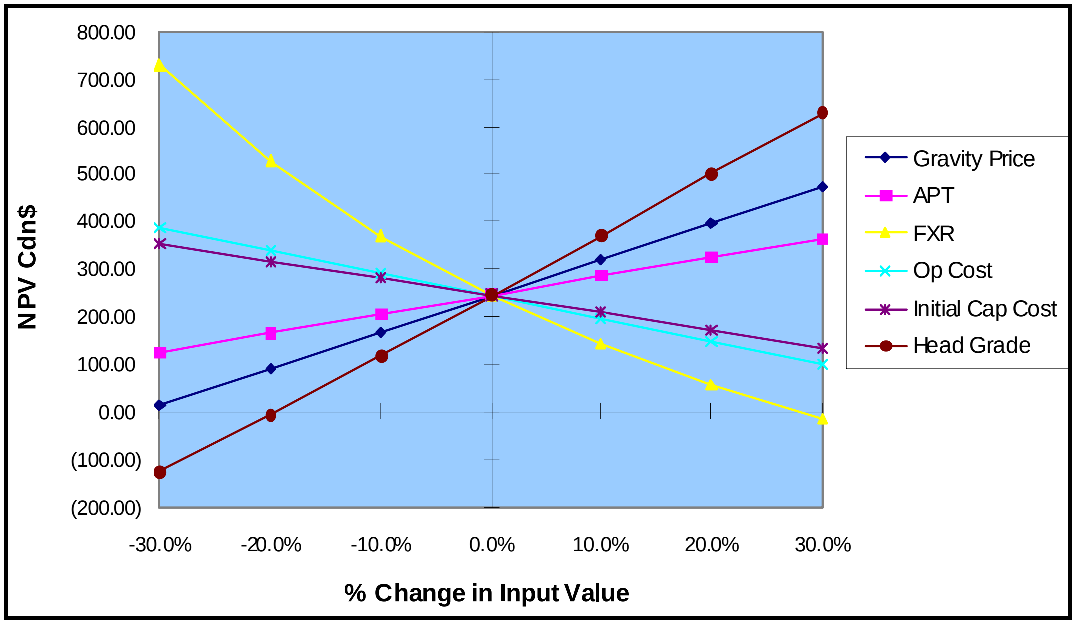
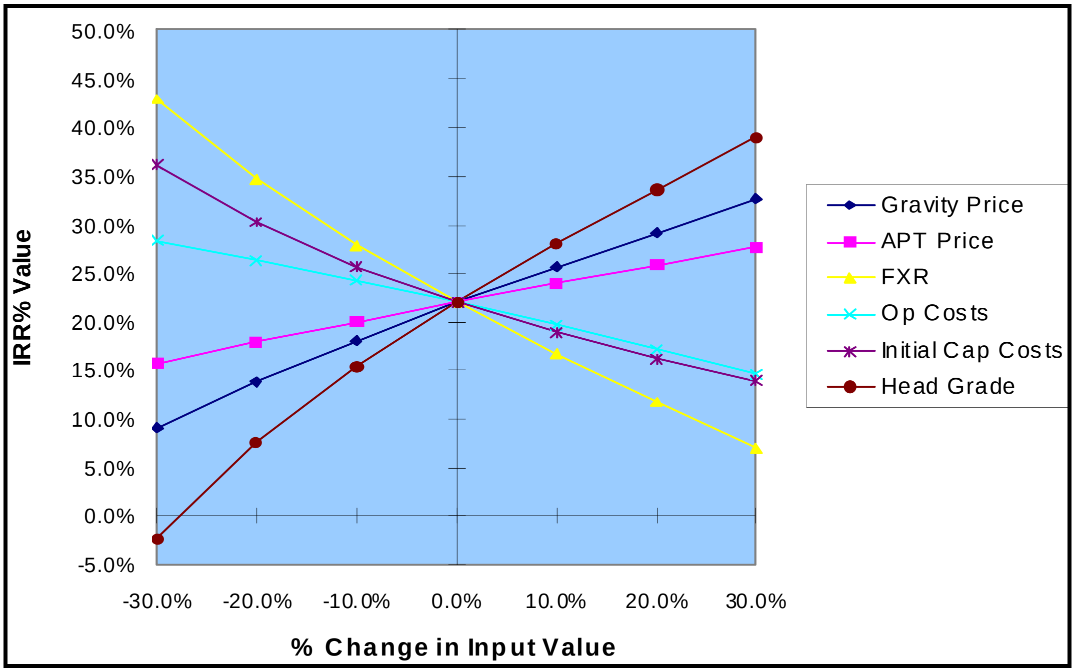


Figure 1.2 Pre-Tax IRR Sensitivity Analysis



1.13 Risk and Mitigation

The identification of risks early on and the timely mitigation of those risks will enhance the project's continuous operation.

The following outlines the major risks and mitigation of these risks:

- Delays in project schedule result from equipment delivery delays: mitigation of equipment delays is done through timely procurement of long-lead items as identified in advanced engineering.
- Non-availability of key construction personnel: plan a smooth transition of personnel from the Cantung Mine, early placement of contracts and initiate prompt and effective personnel recruitment.
- Seasonal river crossing: increase inventory rate of essential materials to ensure continuity of construction and the subsequent operations.
- Initial production: monitor and advance detail engineering work to ensure timely pre-production mine development, assess mining extraction sequences, and milling operational delays during start-up.
- Fuel price sensitivity: monitor and control fuel consumption, optimize the heat recovery from the diesel generators and advance the evaluation of wind energy as a supplementary power source.

- The marginal permafrost impacts on working conditions in the underground mine. Perform permafrost and hydrogeological investigation to understand permafrost temperature sensitivity and rockmass permeability.
- Backfill operations: establish equipment and personnel safety through enhancements of work procedures and the application of remote equipment operation during backfill of mined-out stopes.
- Unconsolidated backfill: investigate marginal permafrost and hydrogeological conditions to determine the impact on backfill stability and the requirement for backfill bulkheads.
- Advance numerical modelling to predict induced mining stress on excavation openings and pillar configuration for optimization.
- Dry stack tailings slope failure: facility is designed to the standards and placed upstream of the run-off dam.
- Ravine dam: dam is designed to safe slope standards and appropriately lined to prevent leakage.
- Avalanche hazard: protection from avalanche run-out to the tailings and site facilities such as the mill and power plant will be considered in detailed design. Road access will require active avalanche management to maximize the construction season as well as safety.

1.14 PROJECT OPPORTUNITIES

A number of potential opportunities have been identified.

- Geology:
  - Improve the reliability of the mineral resource estimates in the upper zones and in the periphery and northerly portions of all zones to identify additional higher grade ore.
  - Convert the inferred mineral resources to indicated category to possibly complement the higher grade ore and add to the life of the mine hence, fostering positive economic benefit to the project.
- Mining:
  - The feasibility of decreasing the mine breakeven cut-off grade based on the feasibility study economics will potentially increase the mineral reserves and lengthen the life of underground resulting in increased value for the project.
  - A detailed investigation of the permafrost condition will provide a better understanding of the ground conditions resulting in potential cost savings in ground support requirement for the underground mine.

- A better understanding of the backfill behaviour will promote confidence on the mining method and achieve optimal extraction sequences thus improving productivity and ore grades in the early years of mining.
- Investigate the potential for an open pit to mine shallow grade ore to further enhance project economics. This could potentially add significantly more years to the mine life.
- Investigate underground mining of the shallow high grade ore to complement the open pit and therefore add more value to the project economics.

- Metallurgy:
  - There are some opportunities for possible cost reductions with regards to the process equipment by confirming the 1985 pilot plant tests results.
- Permit Requirements:
  - The centralized permit process would be beneficial to the project in obtaining timely permits for the early start up of construction in 2010.
- Tailings Disposal:
  - Further optimization of the dry stacked tailings disposal and the method of handling and placing the dry tailings will potentially reduce costs and enhance confidence in the design.
- Sustainability and Environmental Matters:
  - The development of an Environmental Management System (EMS) for the project will make available detailed information on spill and disaster response, waste materials handling and emergency contact information for the project.
  - The underground mining methods to be used at the Mactung mine are favourable to the effective management of waste materials to reduce potential long-term risks at the site.
- Infrastructure:
  - As more site data is made available, re-design of buildings would focus on minimizing excavations, structures, and concrete quantities resulting in the reduction of site construction costs.
  - Further optimization of the power plant configuration would be an area under consideration during detailed engineering to reduce initial project capital.
  - Pre-fabrication of buildings off-site is an opportunity to be investigated to reduce the high cost of site work.
  - Conduct further optimization conveyor layouts for the ore and tailings handling systems.
- Alternative Energy:

- Evaluate wind power as a renewable energy resource to supplement the diesel electric generation. This could substantially reduce project operating cost over the long term due to projected high diesel fuel costs.

1.15 CONCLUSIONS AND RECOMMENDATIONS

Wardrop developed three scenarios with varying economic parameters to evaluate the Mactung project on a pre-tax basis. Based on the base case scenario for the project, an NPV of CDN$277 M at an 8% discount rate and an IRR of 23.5% were obtained using the forecast metal prices as recommended by GBRM.

Sensitivity of the project was evaluated based on inputs such as metal price, exchange rate, grades, operating costs, and capital costs. The project NPV is most sensitive to the foreign exchange rate and, in decreasing order, head grade, price, operating, and capital costs.

On the basis of this study for the base case scenario with GBRM market projections, it is recommended to proceed with basic to detailed engineering, procurement, construction, and commissioning to target full production in 2013 Q1.

Conclusions and recommendations for Geology, Mining, Metallurgy and Mill, Environmental Baseline Considerations and Risks, Tailings Disposal, Infrastructure, Capital Costs, Operating Costs, and Project Execution Plan are outlined in Section 17.0 – Conclusions and Recommendations.

2.0 INTRODUCTION

This NI 43-101-compliant report has been prepared by Wardrop for NATC based on work performed by the following companies:

- Scott Wilson RPA: geology and mineral resource estimate
- Wardrop: mining, processing, infrastructure, capital (excluding Construction Management costs), and operating costs, and financial analysis (based on GBRM metal price forecasts)
- Wardrop: NATC provided basic information of their drilling database on length, core recovery, and geological description in terms of Mactung geological rock codes. Wardrop determined the geotechnical parameters for the geotechnical evaluation contained in Section 18.6 from a visual assessment of core photographs from 22 boreholes from the MS-series diamond drill holes. This series was selected because its rock quality designation (RQD), one of the parameters for rock mass quality, had been previously determined.
- Merit Consultants International Inc. (Merit): construction management costs
- GBRM: preliminary market review and metal prices
- EBA: tailings disposal, environmental, socio-economic assessment and permitting regime reclamation, decommissioning, and closure.

In compliance with NI 43-101, NATC filed with SEDAR the “Technical Report on the Mactung Tungsten Deposit, MacMillan Pass, Yukon” as prepared by Scott Wilson RPA on April 18, 2007. This report, including a 3D block model of the deposit, was used in the mining study.

Wardrop has prepared this Technical Report to define the scope, design features, and the economics of the project to an accuracy of ±15%. The project is based on mining an underground tungsten ore deposit using a combination of long-hole blast and mechanized cut-and-fill mining methods. The project is expected to produce about 7 million mtu of tungsten over an 11.0 year mine life.

NATC is a Canadian mining company listed on TSX Venture Exchange. It owns two tungsten deposits: Mactung in Yukon, and Cantung in the NWT. The Mactung deposit has been characterized as one of the world’s largest tungsten deposits and ranked as the largest undeveloped skarn-type deposit (USGS, 1998).

Many of the documents reviewed in the preparation of this report come from a library of Amax drawings and reports, accumulated between 1963 and 1985, and are now housed in NATC’s Whitehorse office. Copies of these reports were provided to

Wardrop as references for the preparation of the Feasibility Study and Technical Report.

The documentation reviewed and other sources of information are listed at the end of this report in Section 32.0 – References.

The Qualified Persons (QPs) responsible for each section of this report are outlined in Table 2.1. The Certificates of QPs are included in Appendix A.

Table 2.1 Report Preparation and QPs

	QP	
Report Section	**Company**	**Name**
1.0 – Summary	Wardrop	Nory Narciso
2.0 – Introduction	Wardrop	Nory Narciso
3.0 – Reliance on Other Experts	Wardrop	Nory Narciso
4.0 – Property Description and Location	Scott Wilson RPA	Peter Lacroix
5.0 – Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Scott Wilson RPA	Peter Lacroix
6.0 – History	Scott Wilson RPA	Peter Lacroix
7.0 – Geological Setting	Scott Wilson RPA	Peter Lacroix
8.0 – Deposit Types	Scott Wilson RPA	Peter Lacroix
9.0 – Mineralization	Scott Wilson RPA	Peter Lacroix
10.0 – Exploration	Scott Wilson RPA	Peter Lacroix
11.0 – Drilling	Scott Wilson RPA	Peter Lacroix
12.0 – Sampling Method and Approach	Scott Wilson RPA	Peter Lacroix
13.0 – Sample Preparation, Analysis, and Security	Scott Wilson RPA	Peter Lacroix
14.0 – Data Verification	Scott Wilson RPA	Peter Lacroix
15.0 – Adjacent Properties	Scott Wilson RPA	Peter Lacroix
16.0 – Mineral Processing and Metallurgical Testing	Wardrop	Andre de Ruijter
17.0 – Mineral Resource and Mineral Reserve Estimates	Scott Wilson RPA	Peter Lacroix
18.0 – Mining		
Mining and Ventilation	Wardrop	Andrew Nichols
Geotechnical Evaluation	Wardrop	Nory Narciso
Mine Equipment Operating Parameters and Productivities, Mining Capital and Operating Costs	Wardrop	Iouri Iakovlev
19.0 – Tailings Disposal	EBA	Richard Trimble
20.0 – Infrastructure		
Access Roads	Wardrop	Adrian Tanase
Power Supply and Distribution	Wardrop	Guy Impey
Other Infrastructure	Wardrop	Nory Narciso
21.0 – Yukon Assessment and Permitting Regime	EBA	Richard Trimble

Report Section	QP	
	Company	Name
22.0 – Reclamation, Decommissioning, and Closure	EBA	Richard Trimble
23.0 – Project Execution	Wardrop	Nory Narciso
24.0 – Capital Cost Estimate		
Construction Management	Merit	Jay Collins
Other Costs (Mining Capital Costs taken from Section 18.13)	Wardrop	Nory Narciso
25.0 – Operating Cost Estimate		
Processing	Wardrop	Andre de Ruijter
Other Costs (Mining Operating Costs given in Section 18.14)	Wardrop	Nory Narciso
26.0 – Preliminary Market Review	GBRM	Nigel Goodall
27.0 – Financial Analysis		
Financial Model (Metal Forecasts given in Section 26.0)	Wardrop	Scott Cowie
28.0 – Risk and Mitigation	Wardrop	Nory Narciso
29.0 – Project Opportunities	Wardrop	Nory Narciso
30.0 – Interpretation and Conclusions	Wardrop	Nory Narciso
31.0 – Recommendations	Wardrop	Nory Narciso
32.0 – References	Wardrop	Nory Narciso

table continues...

Peter Lacroix (P.Eng.), Associate Mining Engineer with Scott Wilson RPA, prepared the geology, mineral resource, and related sections in conjunction with Barry Cook, (P.Eng.), also from Scott Wilson RPA. Barry Cook visited the Mactung property on August 19, 2005 and Peter Lacroix visited the site on August 8, 2007.

Andy Nichols (P. Eng.), Associate Mining Engineer with Wardrop, visited the site on August 8, 2007 and is the QP for matters relating to mining and ventilation.

The personal inspections undertaken by Peter Lacroix and Andy Nichols are still relevant to support the NI 43-101 Technical Report. The subsequent work programs conducted in the summer of 2008 are in support of the surface observations and site selections for the mill and other facilities made during the site visit. Discussion with Peter Lacroix and Andy Nichols confirmed that there is no material change to the technical information of the surface observed during the site visit.

Richard Trimble (P.Eng.), Project Director with EBA, visited the Mactung property in the summer of 1983 to complete a detailed geotechnical drilling program. Richard Trimble subsequently visited the site in 2005 and 2006 to complete aerial reconnaissance of the proposed infrastructure.

EBA's Whitehorse office engineering staff, under the direction of Richard Trimble, completed detailed site geotechnical investigation programs during the summers of 2007 and 2008.

Bengt Pettersson of EBA visited the Macmillan Pass area in 2003. Under his direction, 10 professional and technical staff from EBA's Whitehorse Environment group visited the Mactung project site and associated locations during 2006 to 2008 to conduct environmental and hydrogeological studies associated with the Mactung project.

In addition, EBA staff from other offices, such as Vancouver and Yellowknife had also participated in these studies and visited the site.

3.0 RELIANCE ON OTHER EXPERTS

Wardrop has followed standard professional procedures in preparing the contents of this Technical Report. Data used in this report have been verified where possible and Wardrop has no reason to believe that the data were not collected in a professional manner.

Wardrop relied on the technical work and reports of other experts in the preparation of this Technical Report. Wardrop has no reason to believe that the data used for these reports were not collected in a professional manner.

3.1 GEOLOGY, MINERAL RESOURCES, AND RELATED INFORMATION

These sections have been taken from the "Technical Report on the Mactung Tungsten Deposit, MacMillan Pass, Yukon" prepared by Scott Wilson RPA for NATC (April/May 2007).

Scott Wilson RPA has relied on ownership information provided by NATC. Opinions on property title have been provided by NATC's legal counsel, Fraser Milner Casgrain LLP (FMC, 2007).

Scott Wilson RPA provided a 3D block model to Wardrop. The mineral resources as contained in this model provided the basis for the Technical Report.

3.2 MARKET REVIEW AND METAL PRICE FORECASTS

GBRM was commissioned by NATC to undertake a market review for tungsten in 2008. GBRM submitted the report "A Preliminary Market Review of Tungsten" in June 2008 and this report was the source for the tungsten metal price in the economic evaluation of the Mactung project.

GBRM has prepared this review for the purpose of providing an overview of the global tungsten industry, and to examine both the activity of existing producers and current and forward consumption patterns.

3.3 CONSTRUCTION MANAGEMENT COST

As agreed with NATC, Merit was requested to provide construction management services for the Mactung project. Wardrop has relied on the information supplied by Merit in the preparation of this portion of the capital cost estimate. This estimated cost includes (for the duration of the project construction) head office expenses, indirect costs, field office facilities and services, field staff personnel, and quality control.

3.4 Tailings Disposal

EBA was commissioned by NATC in 2008 to undertake site and material investigations, placement specifications, and generate material quantities for the proposed Dry Stack Tailings Facility, dam, and diversions structures for the Mactung project. Wardrop has relied on information provided by EBA as a basis for the capital cost estimate of construction material quantities for the tailings area, dam, and diversion structures. Wardrop has no reason to believe that the data used for these reports were not collected in a professional manner.

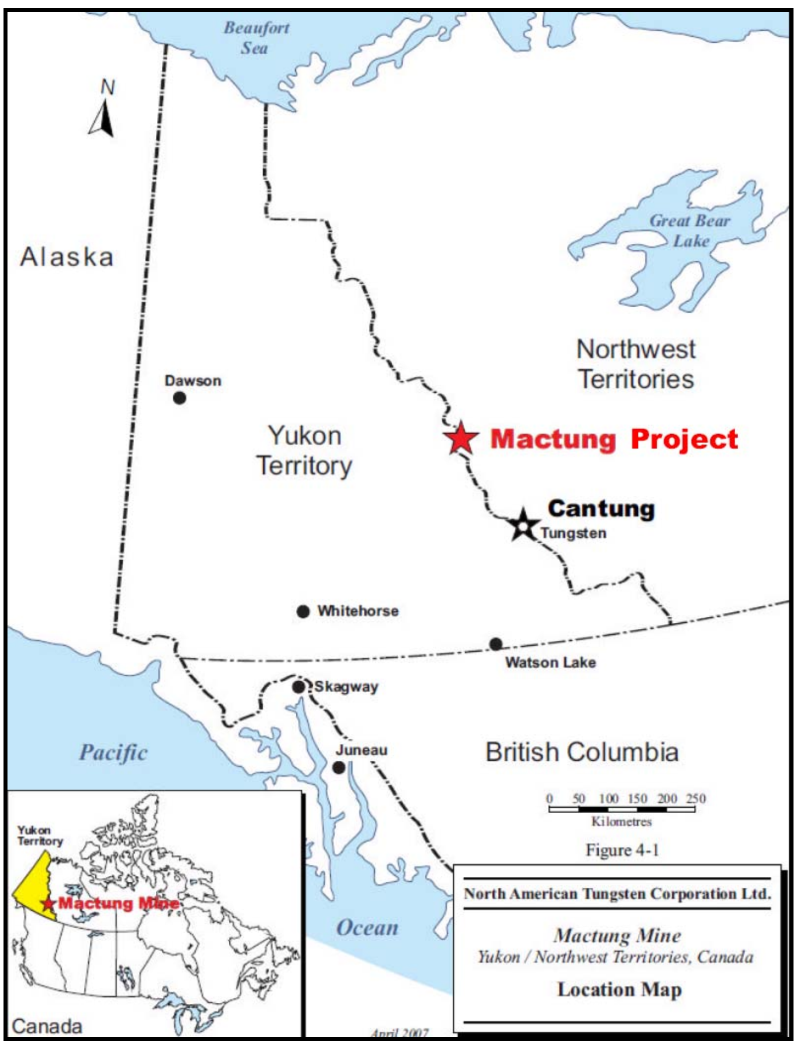
3.5 Yukon Assessment & Permitting Regime/Reclamation, Decommissioning & Closure

EBA was commissioned by NATC in 2008 to undertake environmental and hydrogeological studies associated with the Mactung project. Wardrop has relied on the information provided by EBA in regards to the environmental and hydrogeological components of the Technical Report. Wardrop has no reason to believe that the data used for these reports were not collected in a professional manner.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Mactung property is located in the Selwyn Mountain Range and covers the area around Mt. Allan on the Yukon/NWT border, approximately 8 km northwest of MacMillan Pass (Figure 4.1). The nearest settlement accessible by road, Ross River, is 250 km away to the southwest along the Canol Road, a drive that takes about 6 hours. A ferry at Ross River and the Canol Road are maintained by the Yukon Territorial Government in the summer only. The property is located at latitude 63°17'N and longitude 130°10'W, and the Cantung Mine is approximately 160 km to the south.

Figure 4.1 Property Location Map



4.1 Claim Status

The Mactung property comprises 113 mineral claims and 38 mining leases in Yukon and eight mining leases in the NWT, with a total area of 4,541.6 ha, or 11,217.7 ac (Figure 4.2). Opinions on property title have been provided by NATC's legal counsel, Fraser Milner Casgrain LLP (FMC) (2007). FMC reports that all mineral claims and leases pertaining to the Mactung property are in good standing as at May 14, 2007.

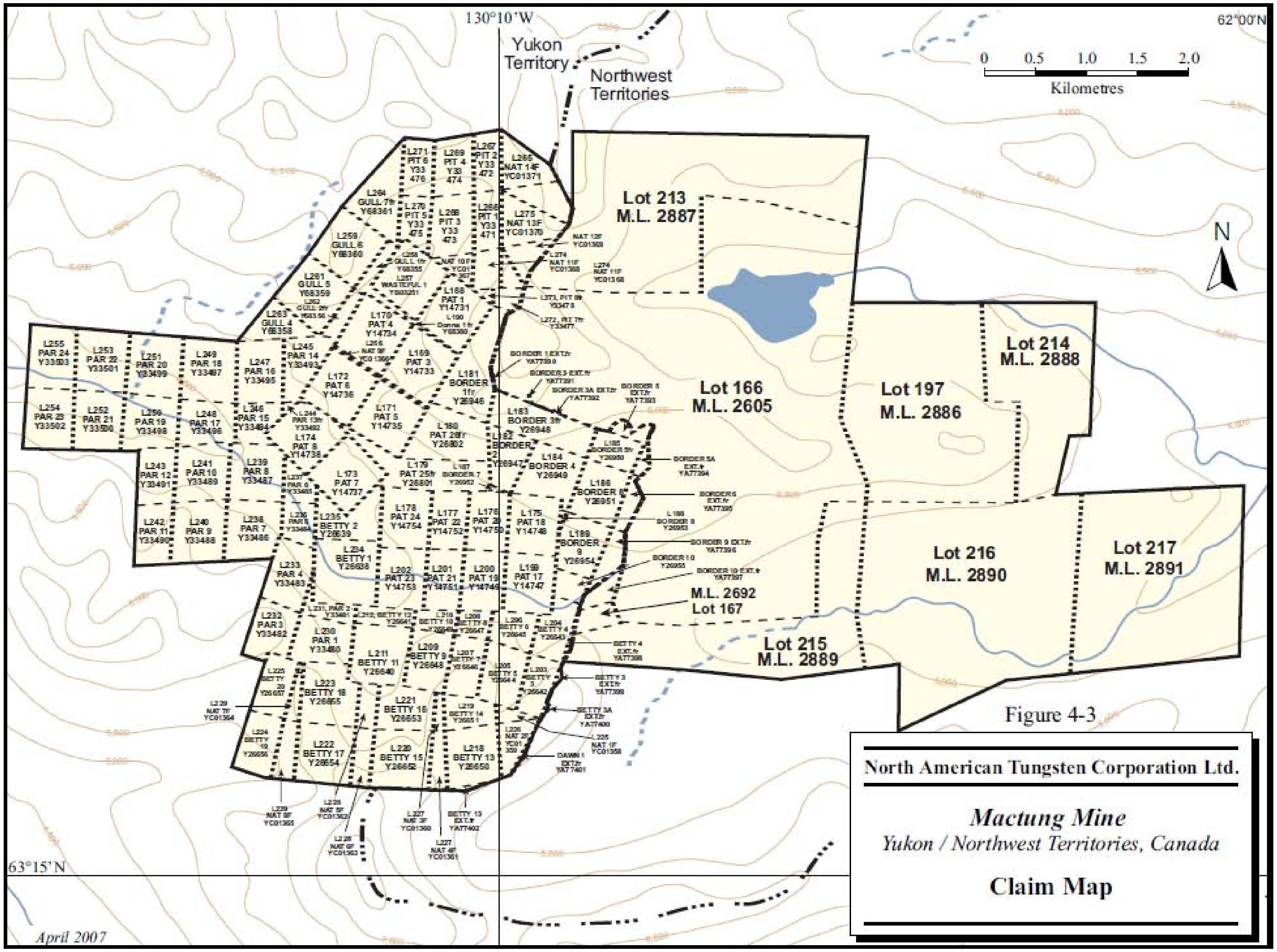
In the 1970s, Amax had the then existing claims and leases surveyed by Underhill & Underhill (Now Underhill Geomatics) of Whitehorse. The territorial border was surveyed by Paul S. Dixon C.L.S. in the period July 28 to August 5, 2003. The 36 grind claims staked in 2005 using a handheld GPS unit have not been surveyed. The claims and leases, none of which are patented, are contiguous and cover a single block of ground that straddles the Yukon/NWT border. The Mactung mineral resource is located on mineral leases, mainly on the Yukon side of the border. All leases and claims belong to NATC.

4.2 Royalties

Royalties on production are payable to the Yukon and NWT Governments under their respective mining legislation.

There is also a reduced royalty of 1% payable to Teck Cominco Ltd. (Teck Cominco) upon payment of $100,000 in September 2005. NATC has the option to pay $1,000,000 by the earlier of March 30, 2015 or 60 days after the receipt of a water licence issued in connection with any proposed mineral production on the property. If NATC has not exercised the option by March 30, 2010, it must pay an additional $200,000 to Teck Cominco on or before this date to extend the option. The option to buy down the royalty will terminate if NATC misses any of the option payments to Teck Cominco.

Figure 4.2 Claim Map



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 ACCESS

The Mactung property is accessible from Ross River, a settlement 250 km southwest along the Canol Road. This road is maintained by the Yukon Territorial Government as far as the NWT border in the summer only, and is only accessible while the ferry at Ross River is in operation. The short section of the Canol Road in the NWT, between the border and the mine access road, is not maintained and is in poor driving condition. The 11 km long access road from the Canol Road to the property itself is currently in fair driving condition.

There are two airstrips locally, one near Tom Creek at MacMillan Pass, which is maintained by the Yukon Territorial Government and can accommodate light aircraft, and another at Tsichu River, the condition of which is not known but probably poor as it is not maintained by the NWT Government.

5.2 CLIMATE

The area has a continental climate modified by the mountain setting. The Tsichu River meteorological station was operated by Amax with assistance from the Atmospheric Environment Service from October 1974 to August 1982 (Kershaw and Kershaw, 1983). The mean annual temperature for this period was -7.7°C, with a mean monthly minimum ranging from -30°C in December and January up to about +4°C in July. For the same periods, the average monthly maximum temperature varied from -18°C to +15°C. Temperature extremes for the Mactung property range from -42°C to +24°C, with a mean of -8.5°C. The average annual precipitation is 490 mm and snowfall 294 cm. Midwinter snow pack varies from thin discontinuous on windswept sites to greater than 2 m in drifted areas. Frosts occur in all the growing season months, but mean daily temperatures are above freezing from late May to mid-September. Winds are most commonly from the west.

5.3 Local Resources

Labour and services are available from the adjacent communities of Yukon, Ross River, Faro, Mayo, and Watson Lake. These communities have had considerable experience with the mining industry in the past.

5.4 Infrastructure

The infrastructure is represented by an adit and associated underground workings that were driven into the Lower Zone (2B) ore horizon in 1973. They are located on the Yukon side of the border at an elevation of 1,900 m. Fresh water is available from the Hess River tributary downstream from the proposed plant site. Diesel power generation is proposed due to the remote location of the project. The main permits required for a producing tungsten mine include a Quartz Mining Licence and a Type A Water Licence. A complete list cannot be provided at this time, as permitting requirements will be determined upon completion of project design and planning.

5.5 Physiography

The topography is rugged and the area has been glaciated. Landforms include small glacier remnants, rock glaciers, glaciated surfaces, moraines, and fluvio-glacial deposits (Kershaw, 1976). Rock talus slopes are common especially on Mount Allan. The valleys on the Yukon side of the border are locally relatively narrow and steep sided, while those on the NWT side are broader and have shallower gradients. The valley floor lies at an elevation of about 1400 m, while the peak of Mount Allan is at 2200 m.

The region is above the tree line and can be classified as arctic/alpine tundra. The vegetation is better developed in valleys and is limited mainly to grasses, small shrubs, moss, and lichen (Kershaw and Kershaw, 1983). Mountains, especially at higher elevations, are extensively covered with talus and intermittently with grasses, moss, and lichen.

The predominant wildlife species in the area are moose, caribou, grizzly bears, wolves, and Dall's sheep, with smaller mammals such as voles, lemmings, chipmunks, shrews, ground squirrels, hares, and foxes. Over 50 species of birds have been reported in the Mactung area, with 40 species using the area for breeding purposes. No fish were discovered by surveys in 2005, but some were identified in earlier surveys by Amax.

6.0 HISTORY

The historical description of the Mactung property has been taken from the "Technical Report on the Mactung Tungsten Deposit, MacMillan Pass, Yukon" prepared by Scott Wilson RPA for NATC (April/May 2007).

6.1 1962 – 1972

The Mactung deposit was discovered in 1962 by James Allan, an Amax geologist, probably as a result of follow-up prospecting to a regional stream sediment survey carried out as part of the Ogilvy Reconnaissance Project (Allan, 1963). The deposit was originally known as MacMillan Pass Tungsten and then as MacMillan Tungsten before it became known as Mactung.

During the years 1963 to 1967, Amax completed geological mapping, rock geochemical sampling, magnetometer surveying, and grid geochemical soil sampling on the property. The 5 surface diamond drill holes completed in 1968 (1,513 m) were followed by 11 km of access road construction from the Canol Road to the property in 1970, and an additional 21 surface diamond drill holes (2,313 m) in 1971 and 48 holes (6,956 m) in 1972.

6.2 1973 – 1980

In 1973, an adit was collared at the 1,890 m elevation and 726 m of lateral development and 27 m of raising were completed in the Lower Zone. A 295-t bulk sample was excavated and shipped to an Amax facility in Colorado for metallurgical testing. Every second round taken in the adit was crushed and then sampled using a Jones Riffle (Splitter). A total of 43 underground holes (1,653 m) were drilled from the adit to better define the mineralization in the Lower Zone, stratigraphically known as the "2B" horizon. Further surface diamond drilling was done in 1979 (Table 6.1), and another 49 m of underground development was done in 1979, with nine 45-gallon barrels of mineralized skarn blasted for metallurgical test purposes. The last surface drilling conducted by Amax was in 1980.

6.3 1981 – 1985

Ongoing environmental and feasibility studies, including an examination of local flora and fauna, archaeology, geomorphology, air quality, water quality, and soil studies that commenced in the early 1970s continued until 1985 when falling tungsten prices caused work on the project to stop.

Nearly all the diamond drill core was relogged during the period 1982 to 1985. This work was undertaken by D. Atkinson (1982, 1983); D. Baker (1982); J. MacMillan (1984, 1985); J. Mustard (1985) and L. Erdman (1985). As most drill holes have been logged on several different occasions at differing levels of detail, it will be a challenge to summarize the information to produce a single diamond drill hole record for each drill hole. The drill core that remains, which is no longer a complete record of the drilling because it has been extensively sampled and resampled, is stored at the Cantung Mine.

6.4 1986 – 1996

Amax sold the Mactung property to Canada Tungsten Mining Corporation (CTMC) in 1986 as part of a larger sale that also included the Cantung mine. CTMC merged with Canamax Resources and Minerex Resources in 1993, to become Canada Tungsten Inc. In August 1994, Aur purchased a 48% interest in Canada Tungsten Inc. and subsequently, in January 1997, the two companies merged.

6.5 1997 – 2007

In October 1997, the property, along with the Cantung Mine and other Aur assets, was sold to NATC, the present owner.

In 2005, NATC drilled 25 surface diamond drill holes (6,639 m) to better define the west end of the deposit and to upgrade the resource classification of some mineral resource blocks from the "Inferred" to "Indicated" category. Also one old drill hole was "twinned". The adit was rehabilitated and a bulk sample of 79 tonnes in size taken for metallurgical test purposes.

During 2005 and 2006, EBA restarted environmental studies partly to confirm previous work done by Amax, and partly to prepare for new environmental and mining permit applications; this work is ongoing.

In compliance with NI 43-101, NATC filed with SEDAR the "Technical Report on the Mactung Tungsten Deposit, MacMillan Pass, Yukon" as prepared by Scott Wilson RPA on April 18, 2007.

Table 6.1 Mactung Deposit History of Exploration and Development

Year	Works	Company
1962	During work on the Selwyn Project, James Allan, a geologist working for Amax, discovered and staked the Mactung Deposit.	Amax
1963	Geological mapping and surface sampling	
1964	Geological mapping and surface sampling	
1967	Geological mapping and surface sampling	
1968	1,513 m in 5 surface diamond drill holes	Cameron McCutcheon Diamond Drilling
1969	Canol road reopened from Ross River to MacMillan Pass	
1970	Construction of 11 km access road to property	
1971	2,313 m surface diamond drilling in 21 holes	Canadian Mine Services
1972	6,956 m surface diamond drilling in 48 drill holes	Canadian Mine Services
1973	Excavated 9 m adit	Cameron McMynn Ltd.
	747 m lateral development underground and 27m of raising	Cameron McMynn Ltd.
	Every second round in skarn crushed in gravel plant and sampled using a Jones Riffle	
	300 ton bulk sample sent to Colorado	
	1,653 m underground diamond drilling in 48 drill holes	Canadian Mine Services
	Reserve estimate 44,517,000 tons	Amax Northwest Mining Co. Ltd.
1979	1,113 m surface diamond drilling in 7 holes.	
	668 m underground diamond drilling in 8 holes	Canadian Mine Services (?)
1980	2,305 m surface diamond drilling in 10 holes.	Amity Diamond Drilling
1981	Capital costs, scheduling, project design	Wright Engineers Ltd.
1982	Geological mapping and relogging of diamond drill core	Amax Northwest Mining Co. Ltd.
	Surface bulk samples Units 3D, 3E and 3F	
	Ore Reserve Study for the Mactung Project	Strathcona Mineral Services
	Initial Environmental Evaluation	Amax Northwest Mining Co. Ltd.
1983	Relogging diamond drill core	Amax Northwest Mining Co. Ltd..
	Adit reopened and two bulk samples totalling 720 t taken	Redpath Ltd.
1984	Relogging diamond drill core	Amax Northwest Mining Co. Ltd.
	Mactung Project Scope Book, volumes 1, 2, and 3	Amax Northwest Mining Co. Ltd.
1985	Canada Tungsten Mining Co. Ltd. purchased the Mactung deposit from Amax	
1993	Canada Tungsten Inc. becomes the owner of the property through company mergers	

table continues…

Year	Works	Company
1994	Aur Resources purchases the property	
1997	NATC purchased the Mactung deposit from Aur Resources	
2005	6,639 m surface diamond drilling in 25 holes	DJ Diamond Drilling for North American Tungsten
	Environmental studies resumed	EBA Engineering
2006	Ongoing environmental studies	EBA Engineering
2007	NI 43-101 compliant resource estimate	Scott Wilson RPA
	Scoping study started	Strathcona Mineral Services

6.6 HISTORICAL MINERAL RESOURCE ESTIMATES

The historic mineral resource estimates referred to in Table 6.2 predate current NI 43-101 regulations and do not comply with current requirements as set out in CIM Definition Standards on Mineral Resources and Mineral Reserves. The terms “resources”, “measured”, “indicated”, and “inferred” used in the original documents should not be construed to infer compliance with present CIM classifications and current NI 43-101 regulations and therefore should not be relied upon.

Table 6.2 Historical Mineral Resource Estimates

Measured & Indicated		Inferred		Total		
Tons	%WO_3	Tons	%WO_3	Tons	%WO_3	Comments
31,917,000	0.96	12,600,000	0.94	44,517,000	0.95	P.Cain, F.Harris, W.Lodder, Amax Exploration1973
31,917,000	0.96	31,000,000	0.92	62,917,000	0.94	R.Steininger, Climax Mine Evaluation Group, 1976
36,091,000	0.95	27,181,000	0.95	63,272,000	0.95	R.Steininger, Climax Mine Evaluation Group, 1979
9,369,800	1.17	22,400,300	0.93	31,769,100	1.00	R.Steininger, Climax Mine Evaluation Group, 1980
16,159,000	1.01	13,785,000	0.84	29,944,000	0.93	Strathcona Mineral Services, 1982
-	-	-	-	22,055,700	0.80	A.Noble 1982
-	-	-	-	31,991,700	0.92	Atkinson & McNeil (?), 1983
-	-	-	-	31,991,700	0.92	Amax Scoping report 1984 “Geologic”
-	-	-	-	25,351,700	0.88	Amax Scoping report 1984 “Mineable”
13,669,000	0.95	-	-	13,669,000	0.95	Roscoe Postle Associates Inc. 2001. “Mineable”

Note: Mineral resource estimates are not NI 43-101 compliant.

These estimates, which, except for the Scott Wilson RPA estimate, are summarized by Atkinson and McNeil (1983), were made from 1973 to 2001 using a cut-off grade of 0.4% WO_3 and minimum mining widths from 3.0 m to 4.5 m (10 ft to 15 ft). The Mineral Resource estimated by Strathcona (1982) forms the basis for the potentially mineable reserves reported in the Mactung Project Scope Book (1984).

In its 2001 detailed review, RPA used the Strathcona resource classifications to estimate the amount of proven and probable reserves and noted that, since the Mactung deposit had not yet been demonstrated to be economic, the potential reserves should be classified as Measured and Indicated Resources. These resources, along with some prior estimates, are listed in Table 6.2. The Strathcona report also included an additional Inferred Resource of 13,785,000 t grading 0.94% WO_3 which is not compliant with the CIM classifications and should not be relied upon. Part of the objective for the surface drilling carried out in the summer of 2005 was to upgrade that part of these “inferred reserves” lying at the west end of the deposit to indicated resources. The drill results from this work are included in the updated mineral resource described in this section.

7.0 GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

The Mactung deposit is located in the eastern Selwyn Basin, an outer miogeoclinal basin that formed on the then western margin of the North American continent. Selwyn Basin refers to a region of deep-water, off shelf sedimentation that persisted from late Precambrian to Middle Devonian time (Gordey and Anderson, 1993). The dominantly thin-bedded siliciclastic rocks (shale, chert, and basinal limestone) grade to the northeast into the thick-bedded carbonate sediments of the variably subsiding Mackenzie Platform. Local stratigraphy of importance may include the Late Cambrian to Early-Middle Ordovician Gull Lake Formation (dolomitic siltstone and mudstone, slate, limestone conglomerate) and Rabbitkettle Formation (basinal silty limestone) and the Ordovician to Lower Devonian Road River Group which includes the Duo Lake Formation (black graphtolitic shale, laminated chert, and minor limestone) and the overlying Steel Formation (pyritic, locally wispy laminated, siliceous, locally dolomitic mudstone to siltstone). Facies-changes between deep-water clastic rocks (shale basin) and shallow water carbonate rocks (platform) are transitional.

Mid-Devonian rifting and/or wrench faulting resulted in a regional marine transgression that abruptly terminated the Selwyn Basin phase of passive margin sedimentation. An influx of marine, turbiditic, chert-rich clastic rocks (Earn Group) spread to the south and east from an uplifted source in northern Yukon and to the east from uplifted western portions of Selwyn Basin. These clastics rocks, locally accompanied by mafic and less abundant felsic volcanism, blanketed all previous facies, covering Selwyn Basin sediments and onlapping onto the western Mackenzie platform. The Selwyn Basin, as a distinct topographic entity, no longer existed.

In Jurassic and Early Cretaceous time, the miogeocline was deformed by northeast-directed compression caused by plate convergence and the accretion of pericratonic terranes onto North America. The rocks of Selwyn Basin, relatively incompetent when compared to the carbonate rocks of the platforms, responded by thrust faulting and the development of open to tight similar folds. Structural trends generally parallel the arcuate Paleozoic shale-carbonate facies boundary.

Widespread Early to Late Cretaceous granitic magmatism intruded the deformed rocks of the miogeocline. Five main intrusive suites are recognized, one of which, the Tungsten (97 Ma – 92 Ma), is responsible for a string of tungsten skarn deposits along the eastern flank of the former Selwyn Basin.

7.2 LOCAL GEOLOGY

The rocks in the Mactung area are part of the west-trending Macmillan Fold Belt, which is discordant to the regional northwest structural grain. This fold belt is interpreted to reflect a deep-seated Devonian fault zone that localized facies changes within the Earn Group and also responded differently to Mesozoic deformation. Folding is tight and a narrow imbricate fault zone of southerly directed east-west trending thrust faults repeats Lower Cambrian to Devonian stratigraphy. South of the imbricate belt, open to closed folds and steep faults are the dominant structures.

Stratigraphy in the general area of Mactung trends generally E-W, and dips from 10° to 40° to the south. The axes of large folds also trend E-W and may have a shallow westerly plunge. Several ages of high-angle normal faulting, of various orientations, are known in the area. Strong slaty to fracture cleavage can be developed. At least in the Paleozoic rocks, the grade of regional metamorphism is very low. The area has been glaciated.

Mactung is the most northerly of a group of W-Cu (Zn) skarn deposits strung out in a 200 km long, northwesterly trending belt which roughly follows the NWT-Yukon boundary. The Cantung deposit is about 160 km to the southeast of Mactung. These deposits are localized within thermal aureoles, typically above the altered apical zones of a suite of Late Cretaceous quartz-monzonite stocks. At Mactung, the apparently related intrusive has been referred to variously as the Cirque Lake stock (Harris, 1977) or the Mactung pluton (Anderson, 1982 and 1983).

7.3 PROPERTY GEOLOGY

The property geology has been described by Dick and Hodgson (1982), Harris and Godfrey (1975), and Atkinson and Baker (1979).

The Mactung mineralization occurs within a bedded sequence of altered limestones, shales and siltstones of Cambrian to Silurian age up to 230 m in thickness. The deposit consists of scheelite-bearing skarns developed near the south contact of a granite intrusion, the Cirque Lake stock. The main outcrop occurs in the NWT along a steep northerly sloping cliff on the north side of Mount Allan. The watershed at the top of the cliff marks the border between the Yukon and the NWT in this region. The main sedimentary sequence dips at low angles to the south. The deposit comprises an Upper and a Lower Skarn Zone, with associated calcareous and pelitic sediments and their metamorphic equivalents, separated by 100 m of pelitic sediments, now largely metamorphosed to hornfels. The hornfels is a light to dark brown or black and represents metamorphosed shales and siltstones with various amounts of muscovite, biotite and graphite. Numerous thin veinlets of quartz or quartz carbonate containing pyrite, pyrrhotite, scheelite, and molybdenite cut the unit.

A stratigraphic sequence has been established on the property, with nine mappable units distinguished and designated from oldest to youngest numerically I, 2B, 3C, 3D, 3E, 3F, 3G, 3H, and 4 (Figure 7.1 and Figure 7.2). The following descriptions are taken from Atkinson and Baker (1979).

Unit 1, the lowermost unit exposed on the property, is a heterogeneous brown to grey, thinly to moderately bedded clastic unit composed of interbedded mudstone, shale, siltstone and greywacke. The unit is considered to be of lowermost Cambrian age; however, confirmation of this age must await more definitive work on the Hadrynian-Cambrian contact.

Unit 2B, host to the Lower ore zone, is highly variable in thickness and composition. The unit is characterized by the presence of limestone slump breccias which appear to have formed as a series of coalescing debris fans at this stratigraphic level. The unit has been correlated with the Lower Cambrian Sekwi Formation. In outcrops on the North Face of Mount Allen, 20 m of dominantly well-bedded, fine-grained limestones and clastics with interbedded slump breccias are interpreted to represent the upslope extension of 35 m of chaotic, medium to light grey limestone slump breccia exposed in underground workings. Down dip, these slump breccias abruptly thin and fragment size decreases as the slumps grade into a few centimetres of calcareous pelite as seen in southern drill holes. South of these drill intersections, additional slump breccias also outcrop. Slumps are chiefly lime or locally mud hosted. Fragments include: limestone clasts, which may be fossiliferous containing Archaeocyathids, well-bedded or breccias; calcareous pisolites and ooids; phosphatic nodules; and various siliciclastic rocks including fragments of Unit 1. Clasts are generally elongate and range from a few millimetres up to 10 m in diameter. Slumps rest locally with erosional unconformity on Unit 1, although, in southern drill intersections, the calcareous pelites of Unit 2B appear to conformably overlie shales of Unit 1.

Unit 3C is in gradational contact with Unit 2B. It consists of 100 m of black, pyritic, carbonaceous, fine-grained clastic rocks and rare thin limestone beds. Numerous elongate clasts of mudstone, shale, siltstone, and collophane occur as separate distinct clasts, as intraformational conglomerates, and as boudinaged beds presumably disrupted by soft sediment deformation. This unit separates the Lower and Upper ore zones. Siliceous sponge spicules found in Unit 3C have been identified as Protospongia of broad Early to Middle Cambrian age.

Unit 3D consists of 20 m of repetitively intercalated 2 cm to 1 m thick beds of calcic and phosphatic limestone slump breccias, mudstone, shale and siltstone that conformably overlie Unit 3C. Slump breccias contrast with Unit 2B in that the breccia beds are characteristically thin and contain smaller, compositionally less variable, well sorted and bedded fragments. Fragments include limestone clasts, black phosphatic nodules, and siliciclastic rocks. Metasomatized calcic limestones within Unit 3D form the basal unit of the Upper ore zone.

Unit 3E is in gradational contact with Unit 3D, as slump breccias die out and the sequence becomes dominantly pelitic. The unit consists of 60 m of finely interbanded black to brown mudstones, shales, and siltstones, with limestone beds scattered throughout. The central portion of Unit 3E, with up to 20% limestone beds, hosts the middle part of the Upper ore zone.

Unit 3F is similar to Unit 3E, consisting of 30 m of intercalated compositionally distinct layers commonly less than 10 cm in thickness. The central part of Unit 3F contains up to 35% limestone beds which are host to the upper part of the Upper ore zone.

Unit 3G, a 20 m thick cliff forming unit of light coloured talc-tremolite dolomite with thin shale interbeds, conformably caps the Upper ore zone.

Unit 3H consists of 90 m of black, carbonaceous, pyritic, fissile shale which is characterized by strong limonite staining on surface exposures.

Unit 4 consists of at least 50 m of black, carbonaceous, fossiliferous flagstones and shale. Abundant graptolite fossils include late Ordovician species (all the above from Atkinson and Baker, 1979).

The Lower zone, while dipping in the same general direction as the Upper zone, contains a “Z” fold (viewed down plunge to the west), with an amplitude of about 90 m. It has been suggested that in fact the fold is a fault, as there is significant fault material associated with it. The Lower zone unconformably overlies the phyllite unit which comprises a substantial thickness of folded schistose micaceous phyllite of Cambrian age. This rock forms the local base of the geological succession.

The entire sequence is overthrust to the north, producing a recumbent isoclinal fold with an axis that plunges at a shallow angle (about 16°) to the west in the west, and at a shallow angle to the east at the eastern end of the deposit. This gives the upper limb of the 2B horizon a slightly domed appearance.

The Cirque Lake quartz monzonite stock cuts across the skarn succession on the north face of Mount Allan, penetrating the succession in large apophyses, dikes, and sills. Samples taken from the granite are reported to contain from 4 ppm to 20 ppm tungsten, and variable amounts of molybdenum, beryllium, and tin.

The deposit is cut and offset by numerous steeply dipping northerly trending faults. Some of the faults have displacements of up to 30 m or more, as interpreted by Strathcona (1982), and up to 45 m as recorded in the Mactung Project Scope Book (1984). The faults are generally characterized by up to one metre of clay and sand gouge, with breccia zones of quartz, calcite, and ice-filled pore space.

Figure 7.1 Generalized N-S Cross Section (Looking West)

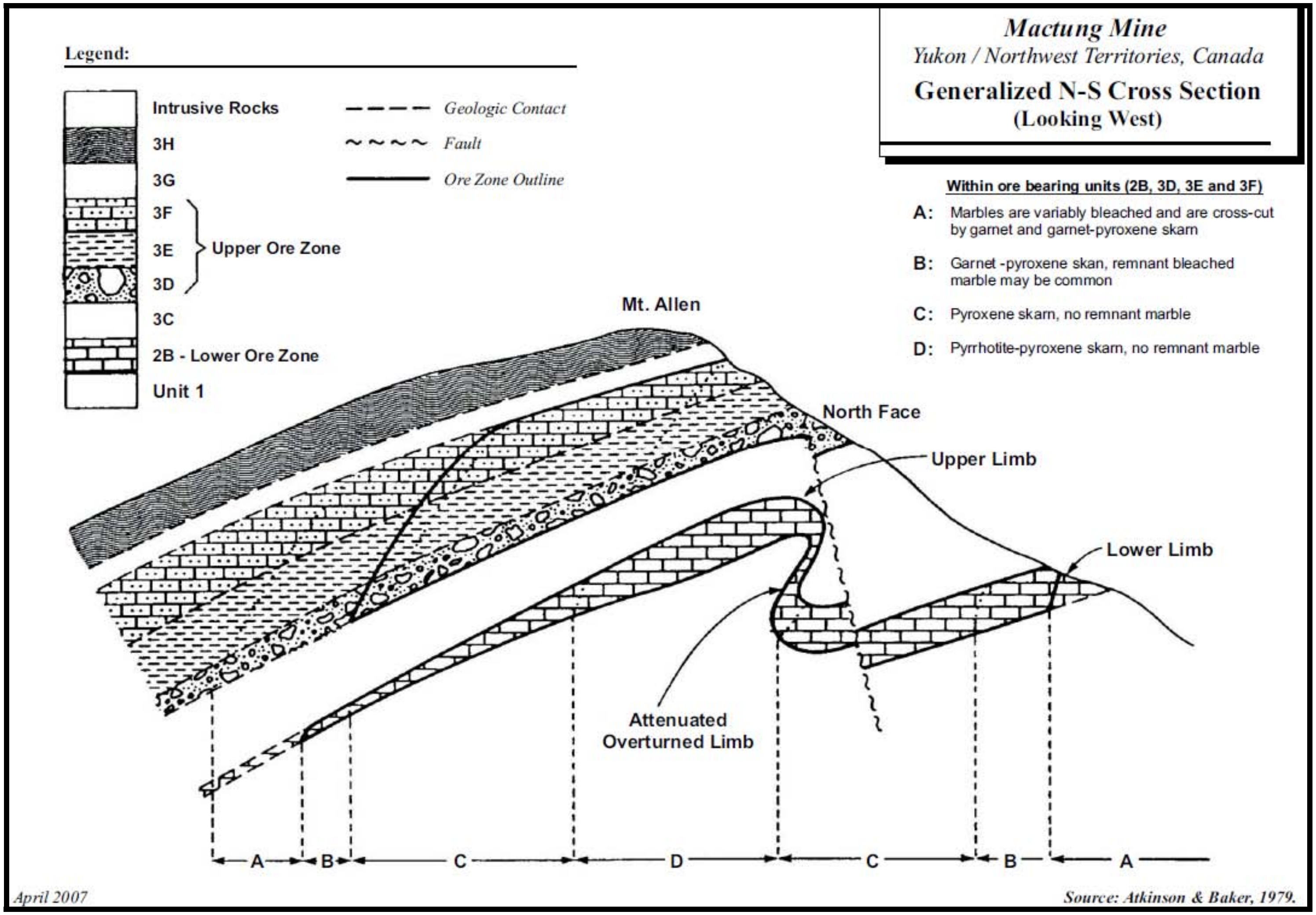
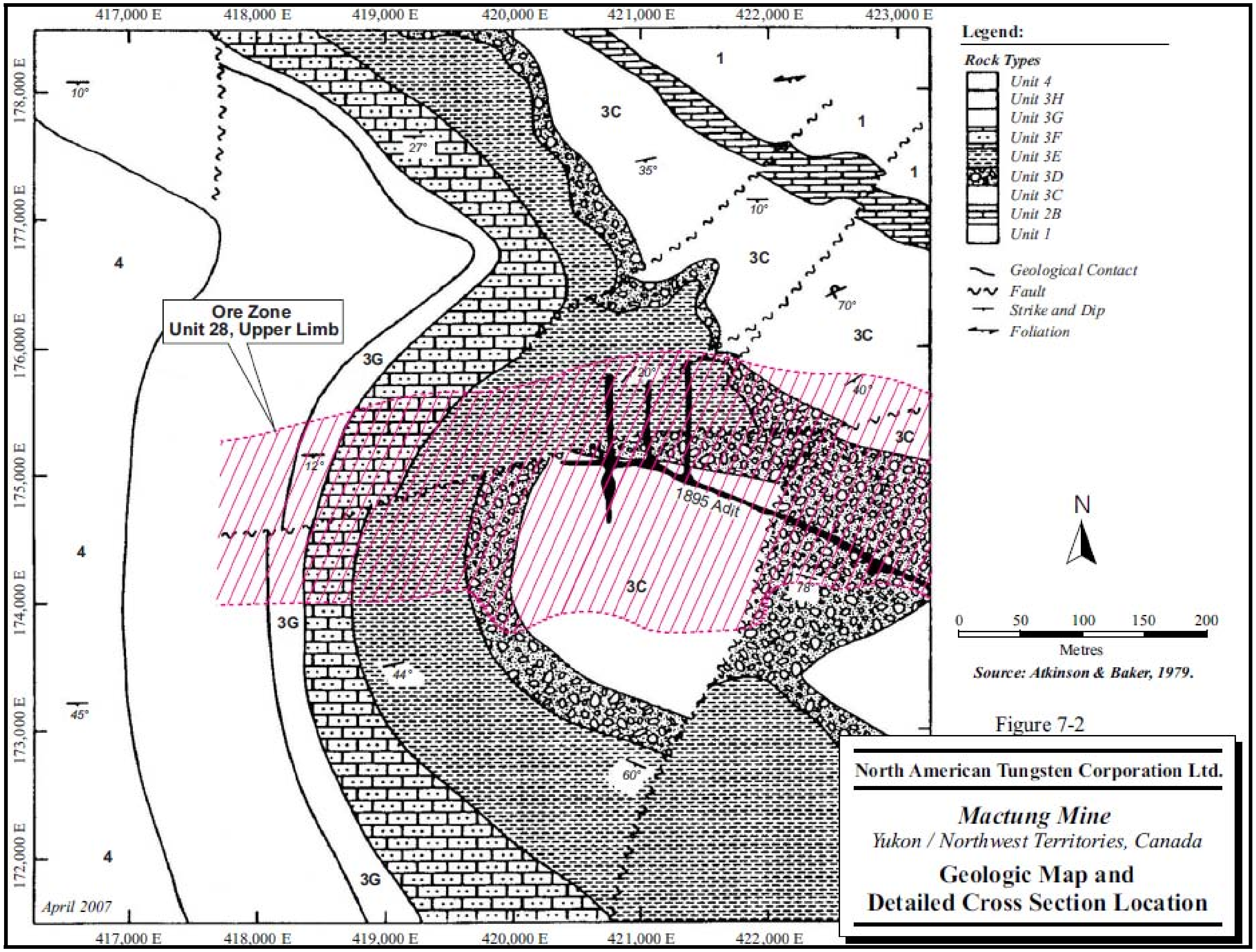


Figure 7.2 Geologic Map and Detailed Cross Section Location



8.0 DEPOSIT TYPES

The ore genesis at Mactung is characterized as a contact metasomatic skarn formed by magmatic hydrothermal fluids originating from a Cretaceous granitic stock. The fluids migrated via channel ways to react with permeable limestone strata of Lower Palaeozoic age depositing scheelite mineralization.

9.0 MINERALIZATION

The mineralogy of the Mactung deposit was described in detail by Dick and Hodgson (1982) and the alteration facies by the Mactung Project Scope Book (1984).

The skarns associated with the metamorphosed limestone units may be divided into two main facies: garnet-pyroxene and pyroxene-pyrrhotite. Scheelite occurs predominantly with pyrrhotite in the pyroxene-pyrrhotite facies. In this facies, the scheelite content increases and grain size decreases with pyrrhotite content. Minor scheelite also occurs in the garnet facies, and is coarser grained than that of the pyrrhotite facies.

Scheelite occurs in five separate skarn horizons formed from lime rich layers in a 300m thick sequence of Lower Cambrian metasediments that lie near the margin of a Cretaceous stock. Locally, the rocks include phyllite (Unit 1), fragmental limestone (Unit 2B), hornfels (Unit 3C), calc silicate and pyrrhotite skarns, limestone, phosphatic limestone (Units 3D, 3E, 3F) and black shale (Unit 4).

Pyrite is widely disseminated in some of the phyllite layers of the lower unit, with galena and sphalerite occurring in small quartz veinlets. Scheelite is the economic mineral of interest at Mactung, with wolframite reported only occasionally in biotite skarn. Chalcopyrite is the main base metal found in the deposit, but grades of economic value have not been demonstrated.

The Upper Skarn unit, which is approximately 30 m thick, is composed of interbedded shale and white limestone that is often phosphatic because it contains fine grained apatite. Scheelite and pyrrhotite occur in veins, fractures, and disseminations. The limestone is generally recrystallized to white marble and varieties of skarn variably described as greenish, fine-grained diopside-hedenbergite, with local red-brown garnet with various amounts of pyrrhotite. Tremolitic amphibole and biotite-rich skarns are described as occurring in patches, derived from pyroxene skarn. The retrograde skarns are enriched in scheelite compared to the disseminated forms found elsewhere. An argillite, siltstone, and hornfels unit of indeterminate thickness, containing interbedded limestone and marble, overlies the Upper Skarn and outcrops mainly to the west of the deposit.

The main hornfels unit, which separates the Upper and Lower Skarn Zones, is approximately 100 m thick. The hornfels is a light to dark brown or black and represents metamorphosed shales and siltstones with various amounts of muscovite, biotite and graphite. Numerous thin veinlets of quartz or quartz carbonate cut the unit, containing pyrite, pyrrhotite, scheelite, and molybdenite.

The close spatial association with the tungsten deposit, the presence of abundant accessory garnet and quartz-tourmaline veins within the Cirque Lake stock led previous workers to suggest that hydrothermal fluids originated from this stock. However, Atkinson and Baker (1979) state that this simple interpretation was not compatible with critical geological observations, notably that no ore is developed in contact with the stock. They concluded that the Mactung deposit is only coincidentally located near the contact with the Cirque Lake stock and that the source of mineralizing fluids is probably a blind stock located immediately south of Mactung. Support for this theory is found in the work of Selby et al. (2003) who indicate that the U-Pb and Re-Os age data suggest that the exposed Mactung (Cirque Lake) stock is not the source of the ore-fluid for the tungsten skarns and that the progenitor pluton for the ore-fluid is unknown.

10.0 EXPLORATION

Between 1962, when the Mactung deposit was discovered and staked, and 1985, a total of about $26 million was spent by Amax on exploration and development, and a further $1.6 million was spent by NATC in 2005.

Geological mapping, geochemical sampling, and magnetometer surveys were carried out in 1963, 1964, and 1967. The deposit gave rise to patchy magnetic readings. Surface diamond drill programs followed in 1968, 1971 and 1972, and an adit into the Lower Zone (2B) and an underground drill programme (43 drill holes totalling 1,653 m) were completed in 1973. During the years 1974 to 1978, Amax changed their emphasis away from geology to environmental, metallurgical, and engineering studies. In 1979, the adit was reopened and more bulk samples of the mineralization taken for metallurgical testing. Another eight holes (668 m) were drilled from the underground workings, and another seven holes (1,113 m) on surface. The year 1980 saw another ten surface holes completed (2,305 m). From 1982 to 1985, drill core was relogged, and extensive environmental and engineering studies completed, culminating in the Initial Environmental Evaluation in 1983 (International Environmental Consultants, 1983) and the Mactung Project Scope Book (1984). The property was dormant from 1985 until 2005, when DJ Drilling of Watson Lake, under contract to NATC, carried out a 25 hole (6,639 m) surface diamond drilling programme on the west and deeper end of the deposit. At the same time, the adit was reopened and a metallurgical test bulk sample of approximately 79 tonnes blasted by Mainstreet Mining of Whitehorse. This sample is being held for possible future testing. Environmental and permitting studies started the same year by EBA are ongoing.

11.0 DRILLING

Between 1968 and 2005, a total of 169 surface and underground diamond drill holes, with an aggregate depth of 23,158 m, were completed on the property. Fifty-one of these holes (7,614 m) were drilled underground from the adit. Strathcona states that 93 of the Amax surface drill hole collars were surveyed, as well as 11 of the 51 underground drill holes (Strathcona, 1982). The remaining underground hole locations were determined from the underground development survey performed in 1980 and the original diamond drill log. Four surface holes drilled in 1980 were for testing of the mill site and tailings impoundment areas. All of the 2005 drill collars were surveyed by Underhill Geomatics of Whitehorse using a differential GPS system. In 1981 and 1982, the project site was resurveyed and the local mine grid, which exists in both imperial and metric forms, was reconciled to the UTM NAD27 grid. This work was updated in 2005 by Underhill Geomatics of Whitehorse who converted the NAD 27 collars to a NAD 83 datum that is currently in use.

Most of the drill holes that intersected the deposit were collared on the south facing slopes of Mount Allan, and drilled at an angle of about seventy degrees to the north, which is approximately perpendicular to the dip of the sedimentary bedding in most of the deposit. In the earlier drilling north-south drill hole section lines were spaced at intervals of 30 m (100 ft), but this was increased to 60 m (200 ft) in 2005 owing to the good continuity of the mineralization along strike from east to west. Holes were generally placed from 40 m to 60 m apart up and down the dip of the mineralized horizons. The closer spacing was indicated because there was more variability of both tungsten grade and of thickness of mineralization in this direction.

12.0 SAMPLING METHOD AND APPROACH

12.1 1968 – 1983

In plan, the Mactung deposit extends approximately 700 m from east to west with a maximum width of about 500 m from north to south. Nearly all of the diamond drilling in the current computerized diamond drill hole database is within this area. The original drilling by Amax from 1968 to 1980 was done on an imperial grid that had north-south drill section lines at 30 m (100 ft) intervals, with holes generally spaced along the lines at 30 m to 60 m intervals. Drill core sample lengths were routinely 1.5 m (5 ft), but varied from 0.3 m to 3.05 m with very minor exceptions.

The drill core and samples have not been examined by Scott Wilson RPA; however, sample data in the form of drill logs, assay sheets, and the computerized diamond drill hole database have been reviewed and used in the current resource estimate.

12.2 2005

The 2005 drilling program was designed to test the Lower Zone (2B) at the west end of the deposit where it was open on strike to the west. All holes also incidentally intersected the Upper Zone. The objective was to upgrade a large block of mineralization designated as "inferred reserves" by Strathcona, to the Indicated Resource category, and extend the known mineralization as far to the west as possible.

The NQ size core recovered by the drillers was logged for geology and core recovery, and RQD measurements made. Recoveries were at, or close to, 100% for the most part. The core was also examined with a short wavelength ultraviolet lamp, which causes any scheelite on the surface of the drill core to fluoresce a bright bluish white, and visual estimates of the scheelite content made. These estimates were typically overstated, but did provide a relative estimate of the grade that could be used as a check against the final assay results. Because of the good continuity of the mineralization along strike from east to west, holes were drilled on north-south section lines that were 60 m apart on a grid. This was done by extending the existing Amax grid to the west. Holes were placed at 40 m to 60 m intervals up and down the dip of the mineralized horizons, because there was more variability of both tungsten grade and thickness of mineralization in this direction, which justified the closer interval. Hole dips and bearings were measured with a "Flexit" down hole magnetic survey instrument, but only the dip was used, because the effects of any local pyrrhotite, some of which is magnetic, could not be predicted. Some acid tests were also done.

Surface diamond drill hole MS 156 was collared beside old hole MT72071 and in the same direction and at the same dip. The width and grade encountered in the "2B" horizon by these two holes were very similar (35.3 m grading 1.55% WO_3 in hole MS156 and 32.0 m grading 1.66% WO_3 in hole MT72071).

12.3 BULK SAMPLES

A 300 ton underground bulk sampling was sent to Colorado for metallurgical test purposes in 1973 (Amax Exploration, 1973). The average calculated grade of the 55 muck samples that were composited for this work was 1.66% WO_3, while the average grade for all the muck samples in the bulk sampling area was 1.46% WO_3. The averaged grade for a set of underground chip samples taken during the same summer was 1.62% WO_3. Grades are higher than the average grades for the Lower Skarn unit because the underground development passed through a higher grade portion of it. Average calculated underground diamond drill hole grades were calculated by Strathcona to be highest closest to the walls of the adit at 1.73% WO_3 (Strathcona, 1982). The average grade of the 28 channel samples taken in the adit in 2005 was also 1.73% WO_3.

Metallurgical test bulk samples were taken from both surface and underground in 1979, and most recently a 79-t bulk sample was taken from underground in 2005. The grade and potential metallurgical recoveries of this last sample have not yet been determined.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

13.1 1968-1980

Scott Wilson RPA has not reviewed the sampling procedures for the drill holes or bulk sampling; however, based on the Strathcona report, no irregularities were found in the drilling or sampling procedures.

13.2 2005

Diamond drill core selected for assaying was marked off in the core box using a red crayon, and a metal tag with the sample number inscribed on it, nailed to the core box at the start of the sample run. A pre-numbered paper sample tag was placed with it. A record of the sample “from” and “to” was made in the sample book on the appropriate sample ticket stub. This information was also recorded on the drill log along with the sample number and the recovered length of core, which was usually 100%. Diamond drill core, which was mainly sampled in lengths of 1.5 m, was split with a hydraulic core splitter set up in a room attached to the core storage shed on the Mactung property. Some core was split with a diamond saw. Once the sample was split, it was placed in a large polyethylene bag, which also had the sample number marked on it in black felt marker. This bag was then placed inside a second identical bag and the paper sample tag placed between the two bags, which were then sealed with a single plastic tie. The samples were transported in rice bags, each rice bag containing about five samples. The rice bags were sealed with a numbered plastic security tie and shipped by commercial carrier from Whitehorse or Watson Lake to Global Discovery Laboratories (Global Discovery) in Vancouver. Sample pulps were shipped by Global Discovery to ALS Chemex of Vancouver and Becquerel Laboratories of Toronto for further assaying.

13.3 DUPLICATES

All duplicate testing for Mactung was performed on splits from the same pulps used for the original assays. The duplicate program did not include analyses of separate splits from the core. Consequently, the results from the various check assay programs at Mactung are primarily a measure of laboratory precision and accuracy rather than sample variability and/or bias. Future programs should include analysis of separate splits of core to assess the variability in sampling. Check assays were performed by a number of laboratories over the years, including Bondar Clegg of

Vancouver, Chemex of North Vancouver, Warnock Hersey of Vancouver, and Crest Laboratories of Vancouver.

Scott Wilson RPA reviewed analytical results and found that differences in the means for original and duplicate analytical results were statistically significant at a 95% confidence interval for two sets of paired original and check assays. In both cases, the original assays were done at Amax's own laboratories, while the checks were performed at Chemex and Bondar Clegg respectively. A scatter chart of original versus duplicate samples for the largest data set (Chemex) is depicted in Figure 13.1, while Table 13.1 summarizes the results of t tests for paired duplicate samples from various labs, including the paired Amax/Chemex assays.

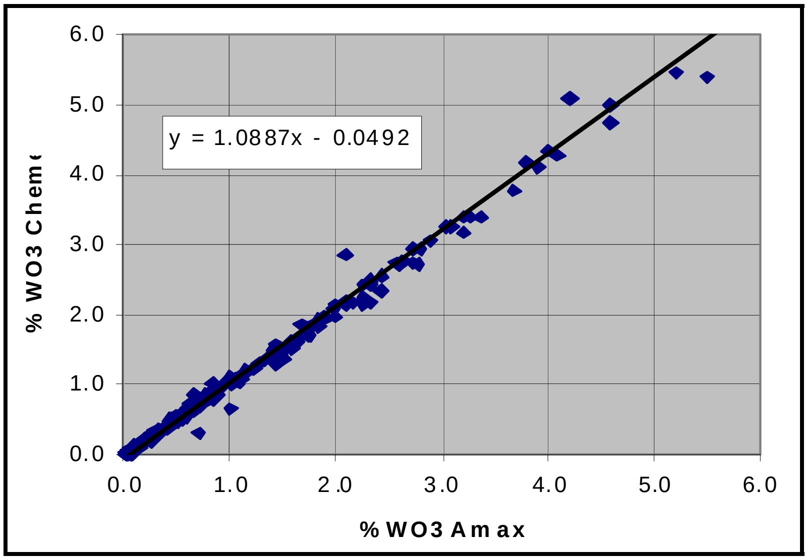
Table 13.1 Duplicate Statistics

	Duplicate					
	Amax-Colour	**Crest Colour Gravity**	**Chemex Colour**	**Bondar Clegg Colour**	**Amax Golden Colour**	**Becquerel Neutron Activation**
Original Lab	Warnock Hersey	Amax	Amax	Amax	Amax	GDL
Original Method	Gravity	Colour	Colour	Colour	Colour	Fusion/XRF
Observations	42	47	190	39	26	48
Original % WO_3	0.234	0.793	1.362	0.739	0.719	1.036
Duplicate % WO_3	0.249	0.766	1.434	0.682	0.765	1.026
% Difference	6.3%	-3.5%	5.3%	-7.8%	6.3%	-1.0%
t Statistic (T)	-0.955	1.481	-3.950	4.319	-0.975	0.704
P(T<=t) 2-tail	34.5%	14.5%	0.0%	0.0%	33.9%	48.5%
t Critical 2-tail	2.020	2.013	1.973	2.024	2.060	2.012

Where the absolute value of the t statistic exceeds the critical value for a 5% level of significance, the difference is said to be statistically significant at that level (i.e. there is at least a 95% probability that the difference is significant). Those with high t statistics indicate a higher probability of statistically different means.

While the analysis indicated that differences in means are statistically significant at a 5% level of significance for two sets of paired data, the magnitude of the difference for the largest data set (Chemex colour) is only 5.3%. Some laboratory bias likely exists in both the Chemex and Bondar Clegg colour assays when compared to the original Amax assays, although the bias is in opposite directions. Irrespective of the potential for bias, the differences are not large in Scott Wilson RPA's opinion, and not material to the mineral resource estimates contained in this section. In the 2005 duplicate program (Becquerel), the difference between the means of the original and check assays is only 1%.

Figure 13.1 Scatter Chart of Original vs. Duplicate Samples for Amax/Chemex WO_3 Assays



14.0 DATA VERIFICATION

14.1 2005 DATA VERIFICATION

On August 19, 2005, R.B. Cook visited the Mactung property. Andy Hureau provided an overview of the property geology and showed the writer cores of four drill holes from the surface drill program then in progress. Under Cook's direction, the half core from three previously sawn samples was quarter sawn. These samples were bagged, tagged, and sealed in a larger plastic bag by Cook and they remained in his possession for the trip back to Toronto. The bag of samples was dispatched by courier to the SGS laboratory in Don Mills. Analysis for tungsten by ICP and Au by fire assay/flame AA finish was requested and the results are listed in Table 14.1.

SGS is accredited to the ISO 17025 Standard by Certificate Number 456.

Table 14.1 Assays of Quarter Sawn Drill Core, DDH MS-157

DDH	Sample Location	Sample No.	Sample Description	SGS Assay Au (ppb)	SGS Assay WO_3 (%)	Original Mactung Assay WO_3 (%)
MS-157	215.7 m – 216.6 m	70962	Sawn Core	51	5.27	4.34
MS-157	218.8 m – 220.5 m	70963	Sawn Core	10	8.57	6.27
MS-157	222.4 m – 223.9 m	70964	Sawn Core	25	4.71	3.24
	Duplicate	70962		51	5.43	

Clearly there is a significant amount of tungsten in the material sampled. The difference between the original assays and the SGS assay values could easily be explained by the very coarse nature of the mineralization. The mineralization assayed should not be taken as representative of the grade of the mineralized zones.

14.2 SCOTT WILSON RPA DATA VERIFICATION

Assay data provided to Scott Wilson RPA was in the form of Excel™ spreadsheets. Scott Wilson RPA independently verified a portion of the database by randomly selecting a hole on each drill section and comparing the WO_3 values in the provided data with the assay certificates and/or assay sheets from the various labs. Data from the 2005 drilling program was compared against Excel™ spreadsheets provided by the labs while data from earlier drilling programs was verified by using scanned handwritten or typed assay sheets. Much of the scanned data did not identify the lab. In total, assay results for the mineralized portions of 31 holes drilled within the four interpreted zones were verified.

Other than a small transcription error in one interval, no errors were found in the data; however, the data needs to be better organized and consolidated. The database originally provided to Scott Wilson RPA did not contain the check assays and survey information was not well organized. Collar locations and azimuths were reported on a number of different grids and a few discrepancies were present in the initial data provided. The files containing the assay certificates were not labelled logically and Scott Wilson RPA did not receive some of the certificates until after the data verification exercise was complete. Consequently, some sequences of data could not be verified from the original certificates. While not material to the mineral resource estimates contained in this section, it is recommended that these deficiencies be addressed forthwith. The use of database management software such as Access™ should be considered.

15.0 ADJACENT PROPERTIES

There are no adjacent mineral properties to the Mactung deposit. Approximately 8 km to the southeast, near MacMillan Pass, there are two Sedex Zn-Pb prospects, the Tom and Jason.

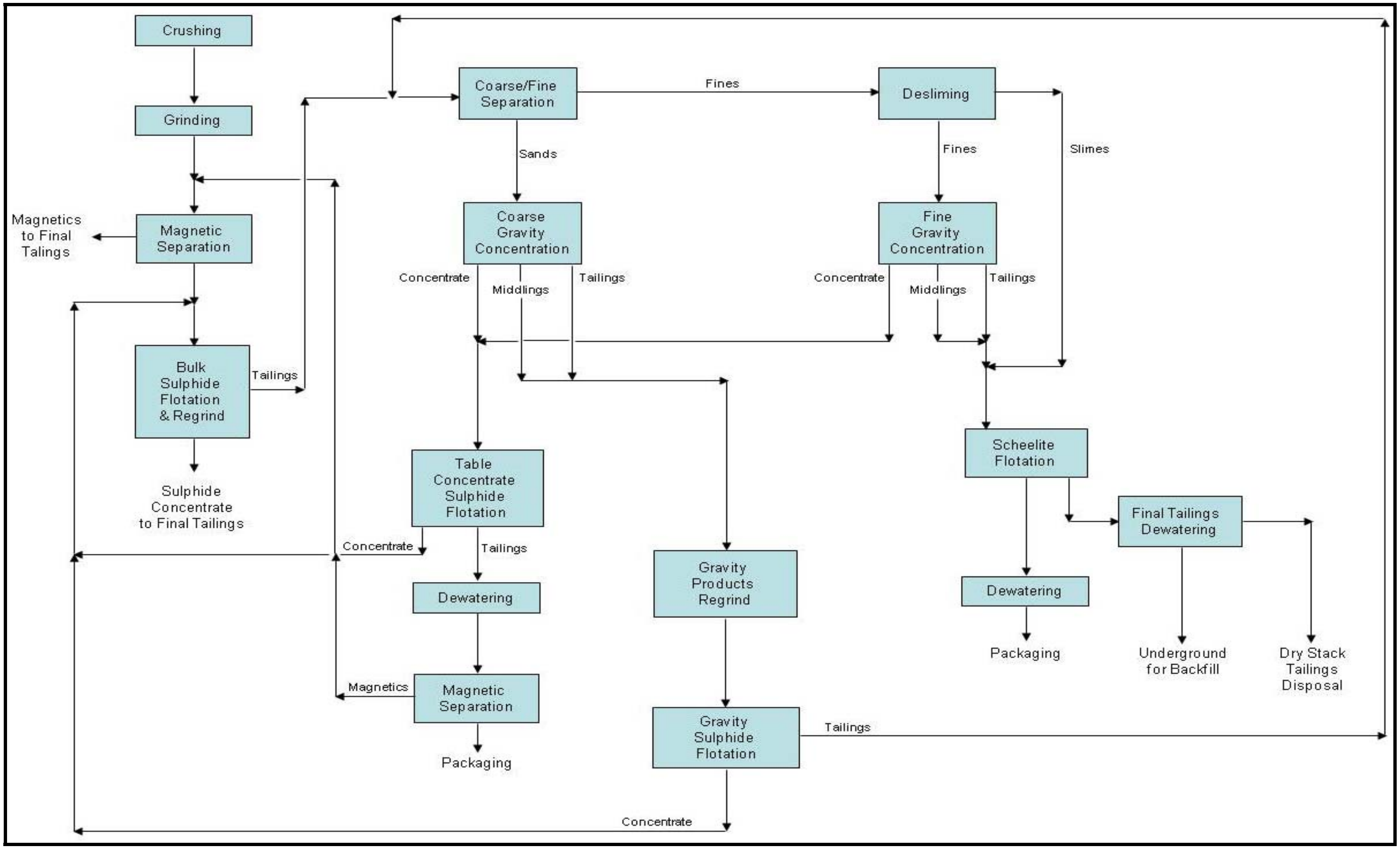
16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 INTRODUCTION

Wardrop designed the Mactung project process plant to treat scheelite ore that will be mined from underground at the rate of 730,000 t/a. The design is primarily based on metallurgical testwork conducted by Lakefield Research Limited of Canada (Lakefield) in 1985, together with processing experience from the existing Cantung Mine, NWT, although various testwork results from as early as 1974 have been used to define the recovery process.

The treatment processes as designed will involve comminution followed by the rejection of sulphide minerals and the concentration of scheelite. As shown in the simplified flowsheet (Figure 16.1), sulphide minerals will be rejected via the wet and dry magnetic separation process and several sulphide flotation stages. High grade tungsten concentrates will be produced through coarse and fine gravity separation methods, as well as a lower grade tungsten concentrate through the scheelite flotation process.

Figure 16.1 Simplified Flowsheet



16.2 METALLURGICAL TESTWORK REVIEW

Various historical metallurgical testwork programs investigated crushing, grinding, sulphur rejection, scheelite gravity concentration, and flotation over the period from 1974 to 1985.

16.2.1 HEAD ANALYSIS

Head analyses have been determined on all test programs carried out since 1974, and specific gravity determinations were obtained from selected samples. The head assay and specific gravities test results considered to be the most representative were obtained from the Lakefield pilot plant testing program in 1985 and these results are listed in Table 16.1. These values correspond closely with the anticipated grade of the plant feed ore for the first four years of production with an average grade of 1.29% WO_3 according the mine plan.

Table 16.1 Calculated Feed Grade of Pilot-plant Samples – Lakefield, 1985

Sample	WO_3 (%)	Cu (%)	Fe (%)	S (%)	SG
Pilot-Plant Feed (calc.)	**1.50**	**0.23**	**18.2**	**8.05**	**3.32**

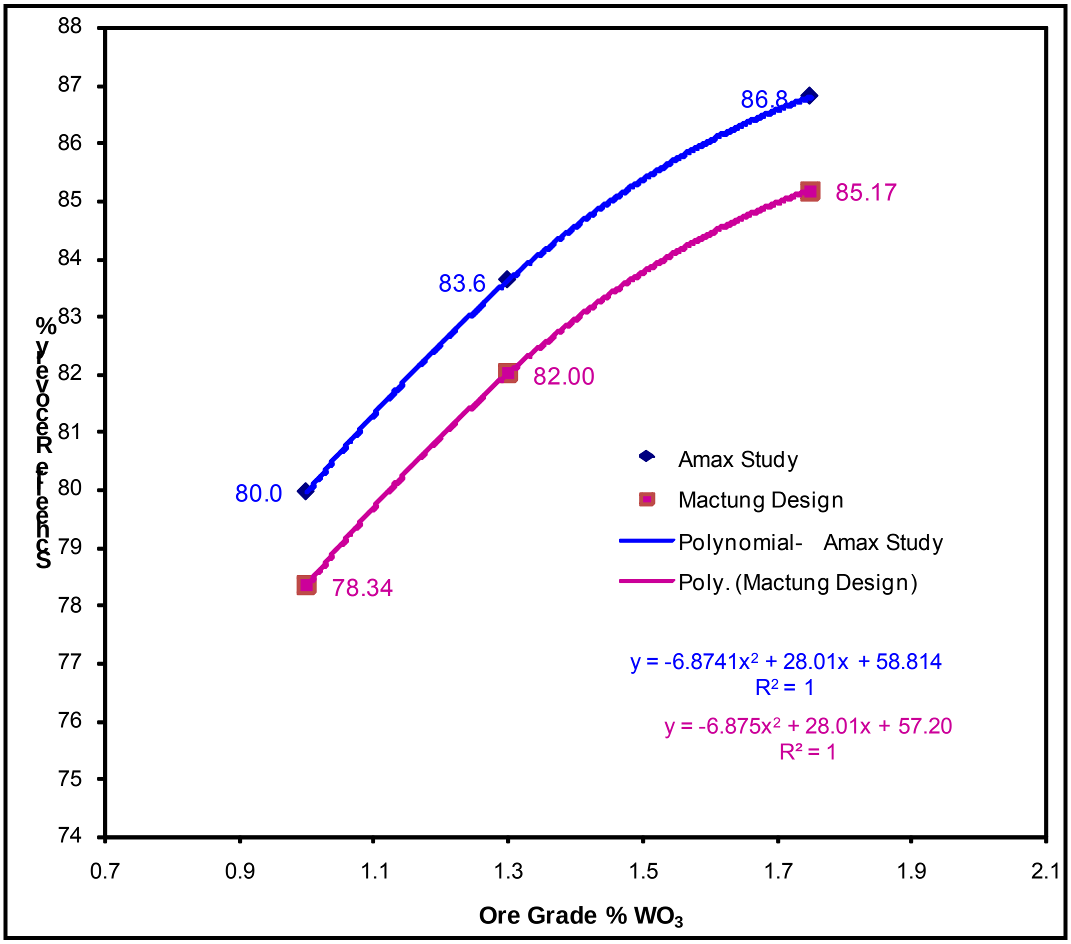
Although the ore feed grade of 1.30% WO_3 has been used in the design of the process plant, it has been established that the recovery of scheelite is sensitive to ore grade variations. In a scoping study report published in 1984 by Amax Extractive Metallurgy Laboratory (Amax), the effect of variations in ore grade on the recovery was characterised in Table 16.2 and is presented in Figure 16.2.

Although the Amax study was conducted on the basis of a proposed treatment plant feed rate of 907 t/d and producing 4 scheelite products, the sensitivity of the grade and recovery is apparent. The graph also shows the designed ore grade and scheelite recovery obtained from the Mactung deposit as used in the present report, namely 82% recovery based on a feed grade of 1.30% WO_3. It also shows the recovery variations resulting from changes in the scheelite grade of the ore to the Mactung plant, as based on the same relationship developed in the Amax study.

Table 16.2 Ore Grade and Tungsten Recovery – Amax, 1984

Sample No.	Ore Grade (% WO_3)	Recovery (%)
1	1.00	79.95
2	1.30	83.60
3	1.75	86.78

Figure 16.2 Relationship between Ore Grade and Scheelite Recovery



16.2.2 MINERALOGICAL STUDIES

The minerals and mineral groups identified from the geological reports, and the maximum mineral content of the samples investigated, is provided in Table 16.3. In addition to the major mineral species, other minerals identified include scheelite, epidote, ilmenite, apatite, collophane, zircon, tourmaline, chalcopyrite, and sphalerite.

Table 16.3 Mineral Abundance

Major Mineral	Content (Maximum Reported, %)	Major Mineral	Content (Maximum Reported, %)
Plagioclase	74.5	Garnet	7.5
Biotite	47.5	Quartz	47.5
Diopside	91.2	Calcite	91.5
Chlorite	2.0	Ankerite	9.0
Muscovite	6.0	Pyrrhotite	30.5
Tremolite	62.5	Pyrite	8.5
Sericite	42.5		

Note: The term plagioclase includes feldspar, anorthite, microcline, andesite and albite. Diopside includes pyroxene and amphihole, while tremolite includes actinolite. Both monoclinic and hexagonal pyrrhotite have been reported in the geological reports.

16.2.3 *Comminution Testwork*

The Colorado School of Mines Research Institute (CSMRI) determined the Bond and rod mill work indices during the 1974 pilot-plant test program requested by Amax. The maximum particle size of the rod and ball mill product was 1.7 mm and 0.3 mm. The weighted average ball and rod mill work indices of all the samples are 15.2 and 15.0 kWh/t, respectively.

In 1985 Lakefield conducted grinding tests on a pilot-plant scale. The test results are shown in Table 16.4. The rod mill work index for Tests PP-A4 to PP-A10 was 16.0 kWh/t, which was close to the results from Tests PP-A11 to PP-A13 of 16.7 kWh/t. The average rod mill work index for the other two series of tests was higher, at 20.1 and 20.5 kWh/t. The average ball mill work index was low at 13.0 kWh/t.

Table 16.4 Average Rod and Ball Mill Grinding Results – Lakefield, 1985

Test No.	Average Feed Rate (kg/h)		Average P_{80} (µm)			Average Work Index (kWh/t)	
	Coarse	Fine	Combined Feed	Combined Product – Rod Mill	Combined Product – Ball Mill	Rod Mill	Ball Mill
PP-A 4 to PP-A 10	722	266	6,064	641	122	16.0	11.7
PP-A 11 to PP-A 13	729	271	5,933	637	174	16.7	12.9
PP-A 14 to PP-A 16	759	144	6,067	710	123	20.1	14.1
PP-A 17 to PP-A 19	753	119	6,017	723	126	20.5	14.8
Average – Test PP-A4 to 10 and 11 to 13	**726**	**269**	**5,999**	**639**	**148**	**16.4**	**12.3**
Overall Average	**736**	**216**	**6,031**	**669**	**133**	**17.7**	**13.0**

Considering the Mactung design for the grinding circuit, the rod mill work index of 16.0 kWh/t was selected, and the ball mill work index value of 13.0 kWh/t was used for the gravity regrind circuit.

16.2.4 SULPHIDE MINERAL REMOVAL

In the testwork reviewed, the removal of the sulphide minerals was investigated by using either sulphide flotation, or magnetic separation followed by sulphide flotation, or sulphide flotation followed by magnetic separation, or using both processes at various stages in the upgrading of the ore.

SULPHIDE FLOTATION

Using the sulphide flotation as the single treatment for sulphide minerals removal was first examined by Amax in 1973. With a two-stage bulk sulphide flotation, the scheelite loss was still high about 3.3% at a grade of 0.62%WO_3. In 1974, Amax conducted further differential sulphide flotation tests on a pilot-plant scale which consisted of a copper mineral flotation step followed by an iron flotation step; each included multiple cleaner flotation stages. The reported scheelite loss on average was 0.12% to the copper concentrate, and 0.41% to the iron concentrate. The sulphur distribution in the final scheelite flotation concentrate was about 0.07 to 0.51% grading at 0.23 to 3.56% S.

MAGNETIC SEPARATION FOLLOWED BY SULPHIDE FLOTATION

In the Amax 1974 pilot-plant testwork, magnetic separation followed by the differential sulphide flotation, was examined. The results indicated that the loss of scheelite in the iron sulphide flotation concentrate was reduced by 2% compared with the situation where no magnetic separation was employed. The sulphur distribution in the final scheelite flotation concentrate was about 0.03 to 0.08% grading at 0.37 to 1.40% S, which was an improvement on the flotation-only process.

In 1985, Lakefield employed magnetic separation, primary sulphide mineral flotation, and secondary sulphide flotation in gravity circuit in order to reject sulphide minerals prior to the scheelite flotation. The magnetic separation process consisted of a rougher with or without a cleaner stage. The non-magnetic material produced was the feed material of the primary sulphide flotation, which was performed prior to the gravity concentration circuit. The primary, or bulk, sulphide flotation included the following steps: rougher/scavenger sulphide flotation, a regrinding of the combined rougher and scavenger sulphide concentrate, followed by cleaner sulphide flotation stages. The secondary sulphide flotation in the gravity circuit removed sulphide minerals from the fine gravity circuit tailings.

The test results showed that magnetic separation treatment removed between 35 and 53% of the sulphur content. The scheelite loss to the magnetic concentrate was from 0.7 to 1.2% when a cleaner magnetic separation stage was included in the test.

The primary sulphide flotation stage removed about 59% sulphide minerals on average from selected tests at a grade of 36% S. About 5% additional sulphide minerals were removed from the secondary sulphide flotation circuit having a grade of 19% S. On average, the combined sulphur removal was 94% with the scheelite loss to the sulphide mineral concentrate being 2.3% with a tungsten grade of 0.15% WO_3.

Sulphide Flotation Followed by Magnetic Separation

The laboratory amenability tests of the1974 Amax pilot-plant testing used the process of the differential sulphide flotation followed by a magnetic separation of the gravity concentrate to remove sulphide minerals. The test results showed that although the cleaned gravity concentrate had a 77% WO_3 grade at a recovery of 39% on average, the magnetic concentrate still contained relatively high scheelite content, about 3.5% of the total scheelite with a grade of 25% WO_3.

16.2.5 Gravity Concentration Testwork

Gravity concentration, including the traditional spiral and tabling processes, were mainly examined in the 1974 Amax and 1985 Lakefield pilot-plant testwork programs.

1974 – Amax

Laboratory scale tabling tests were carried out on sulphide flotation tailings in three particle size groups of 45 to 75 µm , 75 to 150 µm , and plus 150 µm. The concentrate produced had an average grade of 77% WO_3 at an average scheelite recovery of 39%, with a range from 33 to 54%. The gravity concentrate obtained was further purified through a roasting stage on selected samples with a temperature above 650°C, followed by a magnetic separation. However, the tungsten grade in the final concentrate was only increased marginally from 77% to 78.9% WO_3. Since no sulphur analyses were performed on the products, the sulphur removal efficiency cannot be calculated.

The pilot-plant tabling tests used the modified flowsheet developed from the laboratory-scale test. The sulphide flotation tailings were classified at 75 and 180 µm and each of the size group was subjected to a tabling test. The scheelite concentrate product had a lower average grade of about 35% WO_3 at a recovery of 27%. The highest grade of gravity concentrate obtained had a value of 56% WO_3 at a recovery of 35%.

1985 – Lakefield

The 1985 Lakefield test program included a primary and a secondary gravity concentration test stages. The primary stage was performed on pilot-plant scale, and the secondary concentration test was conducted on a laboratory-scale. The test results are discussed below.

Primary Gravity Concentration Circuit

The feed material for the primary gravity circuit test was obtained from the previous generated sulphide flotation tailings. The basic gravity circuit included the coarse and fine spiral separation which was employed in Tests PP-A4 to PP-A7. This produced a combined gravity concentrate with a tungsten grade of between 13.3 to 28.3% WO_3 and with the respective recoveries varying from 50.5 to 58.2%.

Various flowsheet changes were tested to improve the performance of the primary gravity concentration circuit. Table 16.5 summarizes the flowsheet modifications and the respective test results obtained. From the table, it can be seen that the optimum metallurgical performance from the primary gravity concentration is obtained from Tests PP-A17 to A18. These tests were based on the Flowsheet 7 as shown in Figure 16.3, which also includes sulphide mineral rejection and the scheelite flotation processes. The combined primary gravity concentrates recovered for scheelite was between 68.9 to 75.9% at a tungsten grade of 15.5 to 19.8% WO_3.

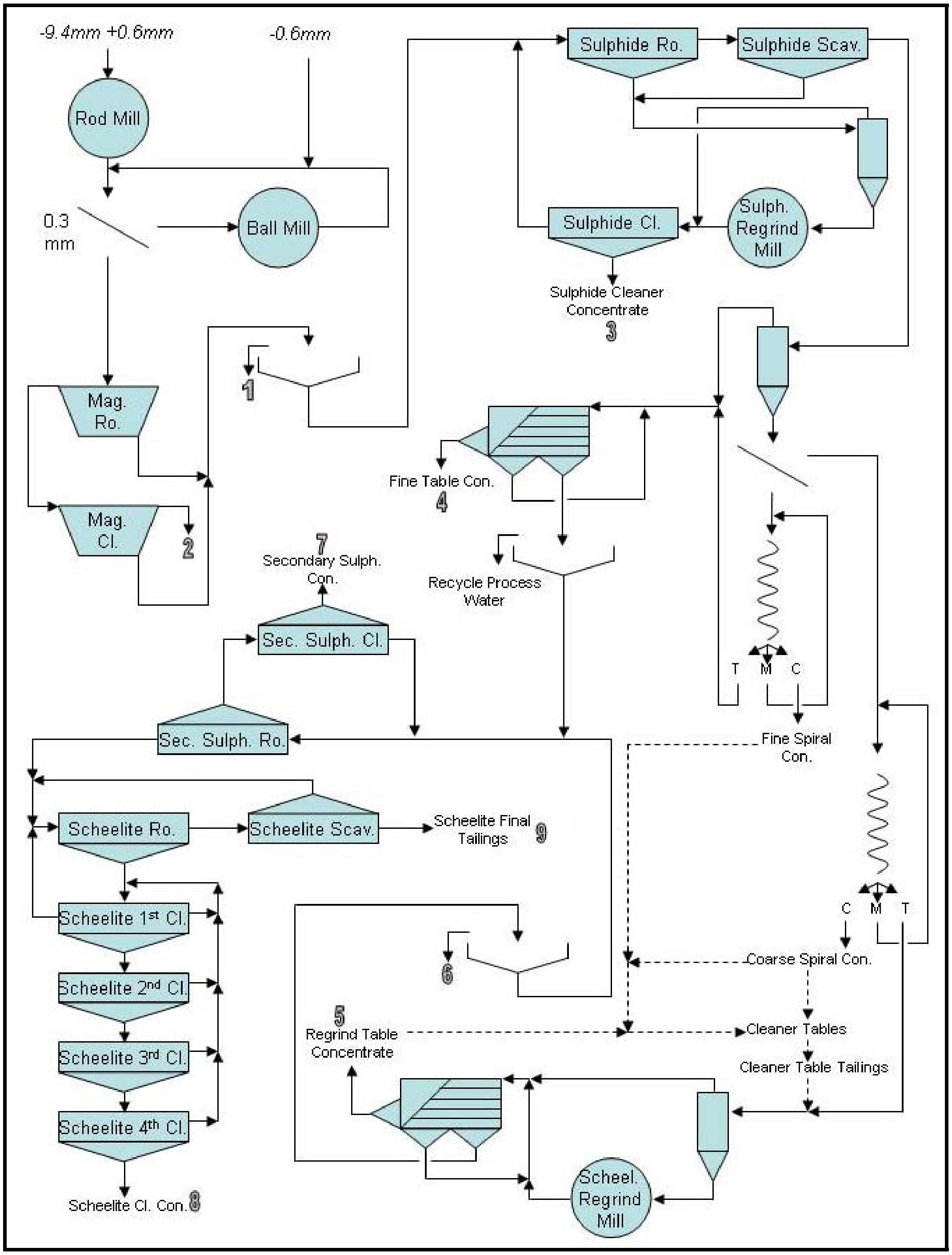
Secondary Gravity Concentration Circuit

The coarse and fine gravity concentrate obtained from the primary tabling concentration stage was used for the secondary gravity circuit using laboratory scale test equipment. A two-stage and a single stage cleaner tabling circuits were employed to upgrade the coarse and fine concentrate, respectively. The final concentrates obtained from this testwork was combined as the final gravity circuit concentrate. The coarse and fine tailings were combined as well and reground before being processed in a regrind cleaner table. The tailings and concentrate produced from the regrind cleaner table were discharged as the final products while the middlings were recycled. The test results showed that the highest tungsten grade of 78% WO_3 of the final gravity concentrate was obtained from Tests PP-A11 and PP-A13 at a recovery of 85%. The highest recovery of scheelite was obtained from Tests PP-A17 to A18 with a value of 88% at a grade of 71% WO_3.

Table 16.5 Flowsheet for Primary Gravity Separation – Lakefield, 1985

Test No.	Flowsheet Features	Slime Table Concentrate		Regrind Table Concentrate		Combined Gravity Coarse & Fine Concentrate	
		Grade (% WO_3)	Dist. (%)	Grade (% WO_3)	Dist. (%)	Grade (% WO_3)	Dist. (%)
PP-A4	**Flowsheet 1:** Basic With Coarse and Fine Spirals	-	-	-	-	13.3	50.5
PP-A5		-	-	-	-	24.4	55.3
PP-A6		-	-	-	-	28.3	58.2
PP-A7		-	-	-	-	22.3	60.2
PP-A8	**Flowsheet 2:** Add Slime Table and Regrinding Mill to Flowsheet 1	36.1	5.7	-	-	23.1	60.9
PP-A9		24.1	8.2	-	-	24.2	55.0
PP-A11		32.5	4.4	-	-	17.7	61.0
PP-A13		30.6	8.2	-	-	20.8	54.3
PP-A10	**Flowsheet 3:** Remove Deslime Cyclone in Flowsheet 2	22.7	11.8	-	-	27.9	51.0
PP-A12	**Flowsheet 4:** Remove Classification Screen & Parallel Spirals in Flowsheet 3; Add Two Sequential Spirals	9.4	2.7	-	-	18.8	52.6
PP-A14	**Flowsheet 5:** Replace Two Sequential Spirals with a Coarse Table in Flowsheet 4	-	-	-	-	19.1	51.9
PP-A15		-	-	-	-	22.5	46.2
PP-A16	**Flowsheet 6:** Replace Coarse Table in Flowsheet 5 with Coarse Spiral; Add Regrind Table	-	-	10.8	0.5	15.7	67.7
PP-A17	**Flowsheet 7:** Add Regrind Table in Flowsheet 2	17.9	12.2	17.9	11.1	14.5	52.6
PP-A18		27.9	4.3	21.7	8.4	18.9	56.2
PP-A19	**Flowsheet 8:** Replace Regrind Table in Flowsheet 7 with a Regrind Spiral	16.1	7.0	4.9	2.6	17.5	53.0

Figure 16.3 Final Pilot Plant Flowsheet – Lakefield, 1985



16.2.6 Settling Tests of Scheelite Flotation Feed

A settling test of the scheelite flotation feed material was carried out as a laboratory scale test as part of the 1985 Lakefield pilot-plant test program. The samples were obtained from the pilot-plant Test PP-A15. Table 16.6 lists the test conditions and the result obtained.

Table 16.6 Setting Test Results of Scheelite Flotation Feed – Lakefield, 1985

Test PP-A15 Product	Feed Concentrate Zone		
	Solids (%)		Thickener Area ($m^2/t/24$ h)
	Initial	Final	
Scheelite Flotation Feed	23	71	0.19

16.2.7 SCHEELITE FLOTATION

The scheelite flotation process was extensively studied and reported in the various documents reviewed.

Preliminary flotation tests were conducted by Lakefield in 1973 and 1974 with limited success. Further flotation test work was carried in 1974 by Amax both on a laboratory and pilot-plant scale. The laboratory scale tests were open-cycle tests that produced a tungsten concentrate with an average grade of 64% WO_3 at a recovery of 27%. In the pilot-plant testwork, the optimum results were from the locked-cycle test configuration grading at 65% WO_3 at a recovery of 26%, and with the first cleaner tailings discarded as tailings.

In 1978, Climax Molybdenum Division Mine Evaluation conducted a pilot-plant scale testwork program to evaluate scheelite flotation ahead of the gravity concentration circuit. The scheelite flotation performance was poor with a concentrate grade of 14.1% WO_3 at a recovery of 38.5%. It was reported that the presence of -38 µm material was the reason for the poor scheelite flotation response.

In 1979, Amax evaluated the all-flotation scheelite circuit in comparison to the traditional flowsheet consisting of a gravity circuit followed by the scheelite flotation step. The all-flotation circuit was an open-cycle test and included two rougher flotation stages, two cleaner flotation stages, and one scavenger flotation stage.

A high grade scheelite concentrate and a low grade scheelite concentrate could be produced from the all-flotation circuit. A typical high grade concentrate had a tungsten grade of 71% WO_3 at a recovery of 80%. However, a high concentration of impurities was reported typically with sulphur in the range of 1.05 to 1.34% S, and phosphorous between 0.136 and 0.321% P. These values were significantly higher than the market product specifications of 0.3% S and 0.03% P, respectively. A magnetic separation and a chemical leaching process were subsequently used to successfully lower impurity contents to below the product specification limits.

The optimum reagent conditioning order was found to be very important with the sequential additions of first the soda ash, then sodium hydroxide/sodium cyanide, and then followed by sodium silicate. The degree of agitation during conditioning also positively affected the rougher and cleaner flotation concentrate grade in the range tested. The optimum pulp temperature was found to be about 20°C in the temperature range studied which varied from 0 to 50°C.

The conventional gravity-flotation flowsheet was tested as a comparison. For a grind of about 12% plus 150 µm, the best results obtained had a gravity concentrate grade of 50% WO_3 at a recovery of 82%, and a flotation concentrate grade of 74% WO_3 at a recovery of 80%.

A semi-continuous pilot-plant test was performed in 1979 by Amax to further evaluate the all-flotation circuit to recover scheelite concentrate based on the promising laboratory test results which had been obtained. Two products were produced. The best metallurgical results produced a high grade flotation concentrate of more than 65% WO_3 at a recovery of 70%, although the lowest grades obtained resulted in a flotation concentrate of 20% WO_3 at a recovery of 10%. However, high concentrations of impurity elements were found in the high grade scheelite concentrate and this would require further treatment for their removal.

Lakefield evaluated the locked-cycle scheelite flotation circuit and tested different flowsheet options and configurations on a pilot-plant scale in the 1985 test program. Promising metallurgical performances were observed from the results of Tests PP-A14 to PP-A19 when using the flowsheet which consisted of four cleaner flotation stages with all the cleaner stage flotation tailings being recycled back to the first cleaner flotation stage, and with the first cleaner tailings and the scavenger concentrate combined and returned to the rougher flotation stage. The results of the six tests conducted are presented in Table 16.7. The best results were from Test PP-A15 which produced a flotation concentrate grade nearly at 75% WO_3 with a recovery of 46%. The lowest grade scheelite flotation concentrate was obtained in Test PP-A19 which gave a grade of 40% WO_3 at a recovery of 11%. Lakefield explained the reason for the reduced grades and recoveries as being the decrease of the tungsten grade in the feed material when the secondary tabling and regrind spiral products were added to the gravity circuit ahead of the flotation circuit.

Table 16.7 Scheelite Flotation Test Results, Four-stage Cleaner Flotation and One-stage Scavenger Flotation Procedure – Lakefield, 1985

Test No.	Product	Mass of Feed to Pilot-Plant (%)	Assay (% WO_3)	Distribution (% WO_3)
PP-A14	Head (Calculated)	76.14	1.07	58.1
	Concentrate	1.04	58.10	43.1
	Final Tailings	75.10	0.28	15.0
PP-A15	Head (Calculated)	74.09	1.05	59.9
	Concentrate	0.80	74.70	45.8
	Final Tailings	73.29	0.25	14.1

table continues…

Test No.	Product	Mass of Feed to Pilot-Plant (%)	Assay (% WO_3)	Distribution (% WO_3)
PP-A16	Head (Calculated)	70.65	0.64	30.0
	Concentrate	0.69	43.00	19.8
	Final Tailings	69.96	0.22	10.2
PP-A17	Head (Calculated)	73.32	0.74	38.2
	Concentrate	0.48	71.00	23.8
	Final Tailings	72.84	0.28	14.4
PP-A18	Head (Calculated)	79.27	0.82	40.5
	Concentrate	0.81	55.80	28.3
	Final Tailings	78.46	0.25	12.2
PP-A19	Head (Calculated)	70.31	0.65	34.9
	Concentrate	0.37	39.90	11.4
	Final Tailings	69.94	0.44	23.5

16.2.8 CONCLUSIONS

The following are the salient conclusions reached from the metallurgical testwork history.

SULPHIDE MINERALS REJECTION

The sulphide rejection process in the plant design will be performed via the combination of the magnetic separation, the bulk sulphide mineral flotation steps, as well as in process sulphide mineral flotation. A similar flowsheet was tested in the 1985 pilot-plant test program by Lakefield and the results were successful.

SCHEELITE FLOTATION

The scheelite flotation circuit should consist of a rougher, a scavenger, and multiple cleaner stages in order to produce a high grade tungsten concentrate. The cleaner tailings from each stage should be returned as feed to the first cleaner stage and be combined with the rougher concentrate.

The alternative process of using an all-flotation circuit produced very promising results which could be particularly suited to the treatment of oxidized feed material. However, some factors hindered its application, which mainly included the high cost to remove the impurities in the flotation concentrate, the application of depressant reagents in the scheelite flotation process such as sodium cyanide, and the potential loss of scheelite to the -10 µm fraction.

The sequence of addition of reagents to the flotation process was also determined to be very important, as was the degree of agitation and the pulp temperature.

GRAVITY CIRCUIT CONCENTRATE ROASTING

The laboratory tests by Amax in 1974 indicated that the roasting of the gravity concentrate in order to remove sulphur can only marginally improve the concentrate grade, and that the sulphur gases produced will result in environmental problems requiring further treatment. Therefore this operation should not be employed in the design of the Mactung process plant. High intensity dry magnetic separation will be used instead to further upgrade the gravity concentrate.

FLOWSHEET DEVELOPMENT

The above testwork constitutes the basis of the design for the Mactung process plant, together with the operating plant experience obtained from the Cantung Mine. The design adopted for the Mactung process plant follows the generalized flowsheet from the 1985 Lakefield pilot-plant testwork program, which incorporated:

- crushing and grinding
- sulphide mineral removal using flotation and magnetic separation at various stages during the ore processing
- using coarse and fine gravity concentration to produce a high grade tungsten concentrate
- using scheelite flotation to produce a lower grade concentrate.

16.2.9 *RECOMMENDED TESTWORK*

Some additional metallurgical and environmental testwork will be required for the detailed engineering design stage. This will include:

- the determination of the crushing index
- rod and ball mill work indices confirmation tests in relation to the feed and product particle size
- characterization tests of the monoclinic and hexagonal forms of pyrrhotite
- confirmation of the density of the magnetic separation feed pulp
- settling tests of the various thickener feed streams, namely, the grinding thickener, the scheelite flotation thickener, the scheelite flotation concentrate thickener, and the final tailings thickener.

In addition, the particle size distribution, specific gravity, and Uniaxial Compressive Strength tests of the final tailings, and the environmental tests will be recommended for the future testwork program.

16.3 Process Description

The process plant will treat the ROM ore from underground at a nominal feed rate of 2,000 t/d. The ROM ore will be a skarn-type scheelite bearing ore with a nominal grade of 1.30% WO_3 and 8.0% S.

Two scheelite concentrate products will be obtained: one from the gravity concentration circuit grading at 67% WO_3 at a recovery of 55%, and the other from the flotation circuit grading at 55% WO_3 at a recovery of 27% for an overall scheelite recovery of 82%.

The process plant will produce about 13,315 t/a scheelite concentrate on average, including the gravity circuit concentrate of 8,322 t/a and the flotation circuit concentrate of 4,993 t/a. A simplified flowsheet is show in Figure 16.1.

16.3.1 Primary Crushing

The ROM ore will be dumped into the ore pass through a static grizzly which will have square openings of 750 mm x 750 mm. The ore pass will have a 400-t live capacity, allowing ROM ore to pass to the coarse ore vibrating grizzly feeder. The vibrating grizzly feeder will classify the ROM ore at 125 mm and will feed the primary crusher with material larger than 125 mm. The primary jaw crusher will have a feed opening of 1,044 mm x 838 mm and a closed side setting (CSS) of 100 mm. Together with the undersize product from the vibrating grizzly feeder, the primary crusher product will be conveyed from underground to the coarse ore bin on surface.

The coarse ore bin will have a live capacity of 500 t and will be equipped with two vibrating pan feeders. The conveyor belt will be equipped with a belt magnet, which will remove the tramp iron from the crushed ore and deposit this material into a tote bin for periodic removal and disposal.

An air-operated diverter gate at the coarse ore bin will allow for the dumping of ore onto a temporary coarse ore stockpile. Ore will be recovered from the stockpile by front-end loaders which will tip the ore into a feed chute and onto the coarse ore recovery conveyor.

16.3.2 Secondary and Tertiary Crushing Circuit

The ore reclaimed from the coarse ore bin will report to the double-deck vibrating scalping screen. The top deck of the scalping screen will have an aperture size of 75 mm, and the bottom deck opening will be 50 mm. The scalping screen undersize product will join the discharge from both crushers on a collection conveyor for transportation to the double deck vibrating sizing screen. The oversize product from both decks will range from 50 mm to 125 mm and will be crushed to a particle size P_{80} of 38 mm by the secondary crusher. The crushed ore from the secondary cone crusher and the scalping screen undersize will be conveyed to the sizing screen.

The sizing screen will classify the feed material at 20 mm. The top deck aperture will be 50 mm and the bottom deck will have the slotted openings of 20 mm x 100 mm. The sizing screen undersize will be conveyed to the fine ore bin for further processing. The screen oversize from the two decks will pass to the tertiary crushing feed hopper which will have a surge capacity of about 10 minutes and will then be fed to the tertiary cone crusher. The tertiary cone crusher will have a CSS of 16 mm. The tertiary cone crusher product will also report to the sizing screen. The secondary crusher circuit circulating load has been estimated to be 110%.

An air-operated plow will divert the crushed ore to a fine ore stockpile should this be required operationally. The material will be reclaimed using front-end loaders.

The fine ore bin will have a live capacity of 2,000 t and will be equipped with two belt feeders. A conventional baghouse will be used to collect and control the dust generated in the secondary and tertiary crushing circuits.

16.3.3 PRIMARY AND SECONDARY GRINDING CIRCUIT

The crushed ore from the fine ore bin will be conveyed to the primary rod mill together with process water and the flotation reagent potassium amyl xanthate (KAX). The diameter of the primary rod mill will be 3,400 mm and the length will be 5,500 mm. The milling density has been designed to be 80% solids. Rotating at 72% of the critical speed, the primary rod mill will grind the feed material from a particle size P_{80} of 16 mm to a P_{80} of 505 µm. A motor of 745 kW will be installed for this operation. The ground material will be diluted with process water to 45% solids for classification by the grinding screens.

The grinding classification screens will be the 5-tray stack vibrating type with each deck having a slotted aperture of 230 µm x 3,000 µm. There will be four sizing screens installed and three will be in operation at any time and one on standby. The feed slurry to the screens will be classified at 230 µm. The screen undersize will report to the grinding thickener for the next stage of processing. The screen oversize obtained will feed the secondary grinding rod mill.

The secondary rod mill will have the same dimensions, motor size, and rotation speed as the primary rod mill abut with a lower milling density of 75% solids. The secondary rod mill discharge will join the primary rod mill discharge and feed the grinding classification screens. The circulating load of 200% has been designed for secondary grinding.

A sampler will be installed in the grinding screen undersize stream.

16.3.4 SULPHIDE MAGNETIC SEPARATION STAGE

The milled ore slurry with a P_{80} of 150 µm will report to a 2-tray/3-compartment stacked conventional thickener with a diameter of 18.3 m. The thickener underflow will feed the three rougher sulphide magnetic separators at a density of 50% solids.

The thickener overflow solution will be returned to the process water tank for recycling. No flocculant will be used in the thickening process in order to prevent any potential interference in the downstream gravity concentration circuits, and the subsequent sulphide and scheelite flotation unit processes.

The rougher magnetic separators will process both the grinding thickener underflow and a small amount of magnetic concentrate collected from the magnetic separation stage used for the cleaning of the scheelite gravity concentrate. The feed density to the rougher magnetic separators will be 50% solids. About 75% by weight of the total feed in the slurry will be discharged as the non-magnetic material for further treatment. The 25% by weight magnetic product collected will be diluted to 50% solids with process water and will be treated in the cleaner magnetic separation circuit.

The cleaner magnetic separation will be performed using one magnetic separator at a feed density of about 50% solids. The cleaner magnetic concentrate product will be diluted with process water to 30% solids and will then be discharged to the final tailings pumpbox. About 8.4 t/h cleaner magnetic concentrate will be produced. The non-magnetic material from the cleaner and rougher magnetic separators will be combined together and will be the feed to the bulk sulphide flotation circuit.

The rougher and cleaner magnetic separators have been identically sized at 1.20 m diameter and 1.83 m long, and operating at a magnetic field strength of 850 gauss.

A sampler will be installed in the bulk magnetic sulphide cleaner concentrate line.

16.3.5 BULK SULPHIDE FLOTATION CIRCUIT

The non-magnetic material from the sulphide magnetic separation circuit will feed the bulk sulphide flotation conditioning tank together with three other slurry streams. These slurry streams include the bulk sulphide cleaner flotation stage tailings, the regrind sulphide flotation concentrate from the gravity concentration circuit, and the sulphide flotation concentrate from the flotation stage of the table concentrate from the gravity concentration circuits. The flotation reagents used will be the collector KAX, the frother DF250, and the activator copper sulphate that will be added to the conditioning tank and will be mixed with the feed slurries. The design residence time for the conditioning of the feed slurry will be five minutes.

The overflow from the conditioning tank will report to the bulk sulphide rougher flotation bank of cells at a density of 42% solids. The bulk sulphide rougher flotation process will be conducted in 6 mechanical flotation tank cells each with a volume of 20 m^3. The design assumes a 30-minute residence time and a slurry pH value of 7.0 (neutral). About 25% by weight of the flotation feed solids will report to the froth concentrate, which will mainly be composed of sulphide minerals. This sulphide mineral concentrate, after being diluted with process water, will feed the sulphide regrind cyclone via the regrind cyclone feed pumpbox. The sulphide rougher flotation stage tailings will be discharged to the gravity hydrosizer for further treatment, together with the tailings from the regrind sulphide flotation circuit, the overflow from the gravity concentrate dewatering auger, and process water as required.

Two cyclones with a diameter of 250 mm will be used as classifiers in the bulk sulphide regrind circuit. One unit will be operational and the other will be acting as a spare. The regrind cyclone will classify the reground sulphide rougher concentrate stream and will produce a cyclone overflow with a particle size P_{80} of 60 µm. The cyclone overflow will be pumped to the sulphide cleaner flotation stage. The regrind cyclone underflow at a density of 70% solids by weight will be discharged to the sulphide regrind ball mill and will be reground to liberate locked sulphide and associated scheelite minerals.

The sulphide regrind ball mill will be 1,600 mm in diameter and 2,750 mm in length. The feed particle size F_{80} of the ball mill will be about 160 µm and the milling density has been designed to be 70% solids. The sulphide regrind ball mill will operate in open circuit to minimize the over-grinding of the scheelite present in the feed material. The regrind ball mill will be equipped with a 56 kW motor for this operation. The mill discharge will be combined with the regrind cyclone overflow to constitute the feed to the bulk sulphide cleaner flotation circuit.

The bulk sulphide cleaner flotation will be carried out at neutral pH with no further flotation reagents added. There will be five flotation cells each with a volume of 10 m^3. This will give a residence time of 30 minutes. The cleaner flotation process will produce 9.6 t/h of sulphide concentrate at a pulp density of 32% solids which will be diluted with process water and discharged as the final tailings. The cleaner flotation tailings will be pumped back to the bulk sulphide flotation stage.

Samplers will be installed to collect concentrate and tailings samples from the bulk sulphide rougher flotation circuit, and concentrate samples from the bulk sulphide cleaner flotation circuit, for process control and metallurgical accounting.

16.3.6 Gravity Concentration Circuit

Hydrosizer Classification

The bulk sulphide rougher flotation tailings, the gravity circuit regrind sulphide flotation tailings, and the gravity concentrate dewatering auger overflow will be combined and classified in a hydrosizer. Process water will be added to dilute the feed slurry to 25% solids and to fluidize the feed material during the classification process. The diameter of the hydrosizer will be 3,048 mm. The hydrosizer overflow will have a particle size P_{80} of 110 µm and will be further processed in the fine gravity concentration circuit. The hydrosizer underflow will be processed in the coarse gravity concentration circuit.

COARSE GRAVITY CONCENTRATION CIRCUIT

Two-stage Spiral Separation

The hydrosizer underflow will be diluted with process water to 25% solids density in the spiral feed pumpbox and will then be pumped to the rougher spiral concentrator distributor. The two-start type of spiral concentrators will be used for both the rougher and scavenger spiral concentration stages.

The rougher spiral concentrate will be further purified in the tabling stage, together with the scavenger spiral concentrate. The rougher middlings, after dilution with process water to a pulp density of 25% solids, will be the feed reporting to the scavenger spiral units. The middlings and tailings produced will be combined with the rougher spiral tailings and will be reground in next stage.

Two-stage Table Separation

The two-stage tabling process will include a cleaner tabling step followed by the one-stage tabling of the middlings from the cleaner tables. Two double-deck tables will be used in the cleaner tabling step and both tables will be in operation at any time. The feed to the coarse cleaner table will consist of the coarse spiral concentrates, and the returned coarse gravity middling cleaner table middlings, which will have a density of 25% solids after dilution with process water. The coarse cleaner table concentrate will be diluted to about 17% solids and will be further upgraded in the tables concentrate sulphide flotation circuit. The coarse cleaner table middlings will feed the coarse middlings cleaner table. The coarse cleaner table tailings will be discharged for regrinding.

The coarse middlings cleaner tabling stage will use one double-deck table. The feed density will be adjusted to 20% solids with the addition of process water. The concentrate will also be diluted and combined with the coarse cleaner table concentrate for further upgrading. The middlings will be returned to feed the coarse cleaner table. The tailings will also be discharged for regrinding.

FINE GRAVITY CONCENTRATION CIRCUIT

Fine Gravity Deslime Cyclone Classification

There will be 6 deslime cyclones, each with a diameter of 250 mm, and 4 cyclones in operation at any time. The feed material from the hydrosizer overflow will be sized at a D_{50} size of 30 µm. The deslime cyclone overflow will consist of slime material with a particle size P_{80} of 40 µm, which will be further processed in the scheelite flotation circuit. The deslime cyclone underflow at a density of 25% solids will be discharged to the fine gravity circuit.

Two-stage Spiral Separation

The deslime cyclone underflow will feed the fine rougher spirals at a rate of 49 t/h solids via a distributor. The two-start type of spirals will be used and this will produce three products. The fine rougher spiral concentrate will be upgraded in the subsequent fine tabling stage. The diluted fine rougher spiral middlings will feed the fine scavenger spiral. The rougher fine spiral tailings will be discharged to the scheelite flotation circuit.

In the fine scavenger spiral concentration stage, the fine rougher middlings will be concentrated in the two-start spiral concentrators. The fine scavenger concentrate will be combined with the rougher spiral concentrate for further tabling. The fine scavenger spiral middlings and tailings will be combined with the fine rougher spiral tailings and report to the scheelite flotation circuit.

Two-stage Table Separation

The combined concentrates from the fine rougher and scavenger spirals will feed the fine cleaner tables at a rate of 4.7 t/h solids. With the addition of process water, the feed pulp density will be reduced to 25% solids. Two triple-deck tables will be used in the first fine cleaner tabling stage and both tables will be operational. The fine cleaner table concentrate will be diluted to about 17% solids and will be further upgraded in the table concentrate sulphide flotation circuit. The fine cleaner table middlings will feed the fine middlings cleaner table. The fine cleaner table tailings will be discharged to the scheelite flotation circuit.

In the second fine middlings cleaner tabling stage, one triple-deck table will be used. The fine middlings cleaner table concentrate will be diluted to about 17% and will be further processed in the tables concentrate sulphide flotation circuit. The fine middlings cleaner table middlings will be discharged to the gravity regrind circuit. The fine middlings cleaner table tailings will report to the scheelite flotation circuit.

16.3.7 GRAVITY REGRIND CIRCUIT

The gravity regrind circuit will facilitate the recovery of scheelite by liberating interlocked scheelite from gangue minerals. The feed material for the regrinding circuit will include the coarse rougher spiral tailings, the coarse scavenger spiral middlings, and tailings, the coarse cleaner table tailings, the coarse middlings cleaner table tailings, and the fine gravity middlings cleaner table middlings.

The 6 slurry streams will be diluted to 20% solids and feed the gravity regrind cyclone cluster. There will be 6 regrind cyclones, each with a diameter of 150 mm with 4 operational at any time and 2 on standby. The cyclone overflow will have a particle size P_{80} of 30 µm that will be discharged to the scheelite flotation circuit. The cyclone underflow will have a particle size P_{80} of 180 µm and will be reground in the gravity regrind ball mill. The flotation collector KAX and frother DF250 reagents will also be fed to the regrind mill. The gravity regrind ball mill will be 1,900 mm in diameter and 4,000 mm in length powered with a 130 kW motor. Rotating at 76% of the critical speed, the ball mill will grind the feed material to a particle size P_{80} of 100 µm. The mill discharge, at 60% solids, will be report to the gravity regrind sulphide flotation circuit at a diluted density of 35% solids.

16.3.8 GRAVITY REGRIND SULPHIDE FLOTATION CIRCUIT

There will be 5 conventional mechanical flotation cells in the gravity regrind sulphide flotation circuit, each with a volume of 10 m^3. The sulphide flotation will proceed at pH 7.0. The concentrate obtained will be diluted to 20% solids by adding process water, and will be returned to the bulk sulphide flotation circuit. The gravity regrind sulphide flotation tailings will report to the bulk rougher flotation tailings pumpbox for further processing.

Samplers will be installed in the gravity regrind sulphide flotation tailings and concentrate lines.

16.3.9 TABLE CONCENTRATE SULPHIDE FLOTATION CIRCUIT

The combined coarse and fine gravity concentrate will be diluted to 16% solids and will then feed the dewatering cyclone cluster. There will be 4 dewatering cyclones each with a diameter of 40 mm. Two of the four dewatering cyclones will be operational at any time with two spare units. The dewatering cyclone overflow at a particle size P_{80} of 10 µm will report to the gravity concentrate dewatering circuit. The dewatering cyclone underflow at a density of 35% solids will feed the table concentrate sulphide flotation circuit.

The dewatering cyclone underflow and the flotation reagents (KAX and DF250) will be mixed at a slurry pH 7.0 for approximately 5 minutes in the conditioning tank. The conditioned feed slurry will feed the table concentrate sulphide flotation circuit.

The table concentrate sulphide flotation can be performed either in the flash flotation type cells or in conventional mechanical cells. The designed flotation cell volume will be 8 m^3 to allow 30 minutes residence time. Sulphide minerals will be concentrated in the froth product which will be diluted with process water and will be returned to the bulk sulphide flotation circuit. The table concentrate flotation tailings containing the high tungsten grades will report to the gravity concentrate dewatering circuit.

A sampler will be installed in the feed line of the tables concentrate dewatering circuit.

16.3.10 GRAVITY CONCENTRATE DEWATERING AND MAGNETIC SEPARATION CIRCUIT

The combined dewatering cyclone overflow and tailings from the table concentrate sulphide flotation stage will first be processed in a wet, low-intensity magnetic separator to remove the remaining magnetic sulphide minerals. The dimensions of the magnetic separator will be the same as the magnetic separators in the sulphide removal circuit. The magnetic material will be discharged to the magnetic separation rejects pumpbox. The non-magnetic material will feed the dewatering auger.

The dewatering auger used will be a spiral classifier which will reduce the feed moisture content from 87% to 20%. The auger overflow will be returned to the bulk sulphide rougher tailings pumpbox. The dewatered concentrate will be further dried in a holoflite screw dryer.

The screw dryer will dry the feed material to a moisture content that will be less than 0.5%. The drying temperature, lower than 120°C, will be low enough to avoid any sulphur dioxide formation. The off-gases from the dryer will be discharged to the atmosphere via the dryer exhaust fan. The dried gravity concentrate will be discharged to the water-jacketed dryer discharge screw conveyor for cooling and transportation to feed the high intensity magnetic separation circuit. The water-jacketed screw conveyer cooling water will be returned to the warm process water tank for re-use in the scheelite flotation circuit.

The cooled gravity concentrate will first report to the dry gravity concentrate oversize screen to remove any tramp material bigger than 2 mm. The oversize material will be intermittently discharged to a tote bin, which will be periodically transported to the primary rod mill for recycling. The screen undersize will feed to the dry high-intensity magnetic separation stage via a feed hopper, which will act as a surge bin.

The dry high-intensity magnetic separation circuit will consist of a rougher magnetic separation step and a two-stage cleaner magnetic separation step. The magnetic separators will be rare-earth permanent high-intensity magnetic separators. About 6% by weight of the feed to the rougher magnetic separator will be separated into the magnetic concentrate. This material will report to the magnetic separation rejects pumpbox and will be combined with the wet auger feed magnetic separator magnetic product. With the addition of process water, the combined rejects will be pumped back to the wet magnetic separation circuit for recycling.

The non-magnetic product from the rougher dry high-intensity magnetic separation will subsequently be treated in the two-stage cleaner high-intensity magnetic separation circuit. The resulting magnetic concentrate will be classified at 74 µm via a vibrating screen. The screen oversize will be collected in a tote bin and will be returned to the primary rod mill for recycling. The screen undersize will be combined with the non-magnetic product from the cleaner magnetic separation step. The combined material will be the final gravity circuit concentrate that will be discharged to a hopper and then be packaged.

A dust collector baghouse will be installed to collect dust generated in the areas of the dryer discharge point, the dry gravity concentrate bucket elevator transfer points, the dry high-intensity magnetic concentration oversize screen, the dry high-intensity magnetic separation cleaner magnetic undersize screen, the gravity concentrate hopper, and the gravity concentrate packaging area.

16.3.11 Scheelite Flotation Circuit

Flotation Feed Thickening

The feed to the scheelite flotation circuit will be a combination of the following products:

- overflow from the fine gravity concentration deslime cyclone
- tailings from the fine gravity circuit rougher spirals
- middlings and tailings from the fine gravity concentration scavenger spirals
- tailings from the fine gravity concentrate tables
- tailings from the fine gravity middlings table
- overflow from the gravity regrind cyclone.

The combined streams will be equally distributed to two scheelite flotation feed thickeners at a pulp density of about 15% solids.

Each scheelite flotation feed thickener will be a 3-tray/4-compartment unit with a diameter of 18.3 m. No flocculant will be added to the feed in order to prevent the potential interference with the scheelite flotation process. The overflow from the two thickeners will be discharged to the process water tank for recycling. The thickener underflows, at a density of 60% solids, will be combined to feed the scheelite flotation conditioning tanks.

Flotation Feed Conditioning

A series of four conditioning tanks will be used. In the first conditioning tank, the designed pH value will be adjusted to 10.5 by adding a mixture of caustic soda and soda ash solution. Warm process water will also be introduced to increase the slurry temperature to the required 22°C and thereby also diluting the flotation feed slurry. The slurry pH value will be about 10.4 in the second conditioning tank and the reagents Quebracho and Sodium Silicate will be added to depress silicate minerals as well as any remaining sulphide minerals.

In the third conditioning tank, the reagent Pamak, which is the trade name for a fatty acid, will be added as a scheelite flotation collector. Two more feed slurry streams, including the scheelite flotation first cleaner tailings and the scheelite flotation scavenger concentrate, will also join this conditioning tank. The combined slurry with a pH value of 10.3 will overflow to the last conditioning tank.

The reagent Emcol, the trade name for another type of fatty acid that also acts as a scheelite collector, will be added to the fourth conditioning tank. The slurry pH value will be about 10.2. The conditioned flotation feed slurry will have a density of 40%

solids and will overflow the final conditioning tank to enter the scheelite rougher flotation cells at a feed rate of 76 t/h solids.

SCHEELITE ROUGHER FLOTATION

The rougher scheelite flotation will be performed in the first four cells of a bank of eight mechanical flotation cells. The volume of the rougher flotation cells will be 10 m^3 and this set of cells will be equipped with double launders. When the rougher concentrate grade is high, the high grade concentrate will be directed to the outer launder and will then be treated in the first cleaner scheelite flotation circuit. During the period when the rougher concentrate tungsten grade is very low, the concentrate will be collected in the inner launder and will be combined with the scheelite scavenger concentrate. The combined concentrate will then be returned to the third scheelite flotation conditioning tank as long as required in order to build up the feed grade. In this process description, it will be assumed that the rougher concentrate will report to the outer launder for upgrading in the cleaner circuit.

The designed rougher flotation stage residence time will be 15 minutes. The rougher concentrate, with a lip density of 25% solids, will be joined with warm process water will be pumped to the cleaner scheelite flotation circuit. The rougher tailings will feed the scheelite scavenger flotation circuit.

SCHEELITE SCAVENGER FLOTATION

The scavenger flotation will be conducted in the last four cells of the bank of eight mechanical flotation cells. Each cell will have a unit volume of 10 m^3 but will be equipped with single launders. The flotation retention time will be about 15 minutes.

Warm process water will be introduced to the scavenger concentrate launder to maintain the flotation temperature and for dilution of the concentrate. The scavenger concentrate will report to the third conditioning tank. The scavenger flotation tailings will be discharged as the final tailings.

SCHEELITE CLEANER FLOTATION

The scheelite cleaner flotation circuit will consist of three cleaner flotation stages. The feed material will be a combination of the scheelite rougher concentrate, the second cleaner stage tailings, and the third cleaner stage tailings streams. The combined streams will be pumped together with Sodium Silicate and Quebracho to the first cleaner flotation circuit at a rate of 3.4 t/h solids.

The scheelite first cleaner flotation stage will be performed in a bank of 4 conventional mechanical cells each with a volume of 3 m^3. The feed slurry will have a pulp density of 11% solids. After a 10-minute residence time, the first cleaner tailings will be discharged to the third scheelite flotation conditioning tank. The first cleaner concentrate with a lip density of 25% solids, will be diluted with warm process water and will be pumped to the scheelite second cleaner flotation stage.

The scheelite second cleaner flotation stage will be conducted in a column flotation type cell at a feed slurry density of 15% solids. The second cleaner flotation column will be 4 m^3, with dimensions of 0.91 m in diameter and 6.0 m in height. The second cleaner concentrate will constitute about 85% by weight of the feed material. Warm process water will be sprayed onto the froth bed to assist in draining the impurities off, and to maintain the scheelite flotation temperature. The diluted second cleaner concentrate will report to the third cleaner stage. The second cleaner flotation tailings will be recycled to the first cleaner flotation circuit.

The third scheelite cleaner flotation stage will also be performed in a column cell with a feed slurry density of 15% solids. The volume and dimension of the third cleaner stage column cell will be the same as the second cleaner flotation column. The third cleaner tailings will be re-processed in the first cleaner flotation circuit. Again, warm process water will be used to spray the froth bed. The third cleaner flotation concentrate will be the final flotation scheelite concentrate produced at a solid rate of 0.6 t/h.

Samplers will be installed in the lines of the scheelite feed thickener underflow, the conditioned scheelite flotation feed slurry, the scheelite scavenger flotation tailings, the final tailings pumpbox discharge, the first scheelite cleaner flotation concentrate, the second scheelite cleaner flotation concentrate pumpbox discharge, and the final scheelite flotation concentrate.

16.3.12 SCHEELITE FLOTATION CONCENTRATE DEWATERING STAGE

A high capacity thickener with a diameter of 8 m will be employed to initially process the concentrate feed slurry. Flocculant will be added to assist with the settling of the concentrate. The thickener underflow will have a density of 60% solids and will be filtered with a vacuum belt filter. The thickener overflow will be returned to the process water tank for recycling.

The thickened scheelite flotation concentrate will be pumped to the concentrate filter feed tank. The retention time of the feed material in the filter tank has been designed to be 8 hours to allow sufficient surge capacity for possible operational interruptions. A filter aid will be employed to facilitate the filtration process by adding this to the filter feed tank and mixing with the flotation concentrate. The slurry will then be pumped to the scheelite flotation concentrate filter.

The vacuum belt filter will reduce the flotation concentrate moisture content from 40% to 10%. The filter cake will be discharged into the scheelite flotation concentrate dryer. The filtrate will be pumped back to the scheelite flotation concentrate thickener. The flotation concentrate dryer will dry the flotation concentrate to the designed moisture content of less than 0.5%. The dryer used will be a holoflite screw type. The dryer will have a diameter of 914 mm, a length of

6,400 mm, and will be powered by a motor of 7.5 kW. The dried flotation concentrate will be sampled and packaged.

Off-gases generated from the dryer will be released to atmosphere through the exhaust fan. The off-gases and dusts generated from the dryer discharge end will be collected in a dust collection system. The solids will be returned to the dried flotation concentrate. The gases from the cyclone will be released to the atmosphere.

16.3.13 Tailings Treatment

The final tailings will consist of the wet cleaner magnetic concentrate, the sulphide cleaner flotation concentrate, and the scheelite scavenger flotation tailings. The final tailings will have a density of 35% solids and will first be pumped to the tailings thickener.

The tailings thickener will be the high capacity type with a diameter of 22 m; flocculant will be added. The clear thickener overflow will be discharged to the ageing pond before recycling to the process water tank. The ageing pond has been designed to provide 30 days of retention time which will allow the flotation reagents, especially the fatty acids including Pamak and Emcol, to decompose so that there will be no interference in the sulphide and scheelite flotation processes.

The reclaimed water will be returned to the process water tank via two stages of pumping, which will consist of vertical turbine pumps mounted on a floating barge, and the land-based horizontal solution/water pumps. All the pumps will be inside an insulated, lighted, and heated enclosure.

The tailings thickener underflow, with a density of 60% solids, will be pumped to the agitated tailings filter feed tank and will then be filtered in the automatic plate and frame pressure filters. The tailings filter feed tank has a nominal retention time of four hours. The moisture content of the filter cake will be an estimated 12 to 15%. The filter cake will be discharged by the variable speed belt feeders and transported to the belt conveyor. A flop gate plough will either direct the filtered tailings to the underground workings conveying system as backfill or to the conveying system for the surface tailings disposal location.

The filtrate from the pressure filters and the filter cloth wash water will report to the air and water separator. The water will be returned to the tailings thickener, while the air portion will be released to atmosphere.

16.3.14 Reagents

Various reagents will be used in the metallurgical processes. The flotation collector reagent KAX will be added in the primary rod mill feed and in the bulk rougher sulphide flotation conditioning tank as the sulphide mineral collector. Copper Sulphate will also be used in the bulk rougher sulphide flotation stage as the sulphide mineral activator.

DF250 will be added as the frother in all the flotation circuits.

A combination of reagents will be employed for the scheelite flotation circuit. In the first scheelite conditioning tank, a mixture of the reagents Caustic Soda and Soda Ash will be used to adjust the slurry pH value and to act as a dispersant. Quebracho, the depressant for calcite, apatite, any remaining sulphide minerals, and silicate minerals, will be added into the second conditioning tank. Sodium Silicate, a dispersant, will also be introduced into the same tank. Pamak and Emcol, which are the scheelite mineral fatty acid collectors, will be added in the third and fourth conditioning tanks, respectively.

Flocculant will be used to facilitate the settling of scheelite flotation concentrate and the final tailings in their respective thickeners.

Filter aid will be added to assist in the filtration of the scheelite flotation concentrate.

The reagents Quebracho, Copper Sulphate, Caustic Soda, Soda Ash, and flocculant will be processed in bulk bags as solids, and will initially be mixed with fresh water to reach the designed solution strength, and will then be stored in the respective holding tanks. Magnetically coupled pumps will be used to transport the prepared reagent solutions to the various addition points. Liquid reagents will consist of Emcol, Pamak, DF250, KAX, Sodium Silicate, and filter aid. Of these, Emcol, Pamak, DF250, and filter aid will be pumped in the as-received concentration to the addition points by metering pumps. KAX and Sodium Silicate will be diluted with fresh water in the mixing tanks and then will be stored in holding tanks. The diluted reagent solutions will be delivered to the respective addition points by magnetically coupled pumps.

16.3.15 WATER SYSTEM

Fresh water usage will be required for reagent preparation, slurry pump gland seals, the mill lube cooling system, and potable water consumption. Fresh water will be supplied from the South Tributary of the Hess River by staged pumping. The fresh and fire water tank will have the top section limited by an internal standpipe. The bottom of the tank will be reserved for fire protection water. The fresh water tank will have a diameter of 12.0 m and will be 11.5 m high. The tank will be located above the residential camp to provide fresh water service by gravity. The potable water treatment system will be located at the camp, with the mill and other buildings at the mill site being supplied by a gravity line from the camp.

Process water will consist of overflows from the grinding thickener, the scheelite flotation feed thickeners, the scheelite flotation concentrate thickener, and the tailings thickener plus reclaim water from the ageing pond with provision for fresh water make-up to the process water tank, if required. The process water will be added to the points of application throughout the plant as required.

Excess water from the ageing pond will be discharged by pumping to the river after testing and ensuring that this water complies with the environmental discharge specifications. It is recommended that ageing tests be conducted at the same time that the supernatant toxicity testwork is undertaken for environmental purposes. A warm process water tank will be installed to provide warm process water to the scheelite flotation circuit. The input streams of the warm process water tank will be composed of the cooling water return from the gravity concentrate dryer discharge screw conveyor, and process water heated in the powerhouse with waste heat from the generators and transferred via plate and frame heat exchangers. There will be two heat exchangers; one will be in service while the second exchanger will be off-line for cleaning.

16.3.16 Air Services

The primary crushing area air will be supplied by an underground compressor with no standby unit. An air receiver will be equipped for the compressor.

The plant air will be produced by one operating compressor with one standby unit. It will be used in the secondary and tertiary crushing area, process and power plant, rod and regrind mill clutches, and in preparation of the instrument air via filtration and drying stages.

High-pressure flotation air will be supplied by a single compressor with no stand-by unit. Plant air can be used to provide the high pressure air in the event the high-pressure flotation compressor is off-line. This unit will be equipped with an air receiver with a volume of 2 m^3.

Low-pressure flotation air will be supplied by low-pressure centrifugal blowers without air receivers. Two blowers will be installed with one operating and one unit on standby.

16.4 Process Design Criteria

Table 16.8 contains a summary of the process design criteria (PDC) for the Mactung plant design with the selected design parameters based on metallurgical testwork results that were previously discussed in this section.

Table 16.8 Process Design Criteria Summary

Description	Units	Design
Operating Schedule		
Process Plant Shifts per Day		2
Process Plant Hours per Shift	h	12
Availability/Utilization		
Annual Overall Processing Rate	t/a	730,000
Daily Processing Rate, Nominal	t/d	2,000
Crushing Plant Operation Availability	%	67
Processing Plant Operation Availability	%	94
ROM Ore Characteristics		
Head Grade		
Tungsten, Nominal	% WO_3	1.3
Sulphur, Nominal	% S	8.0
Specific Gravity, Design		3.4
Moisture, Nominal	%	2.0
Bulk Density, Design	t/m^3	1.8
Primary and Secondary Grinding Rod Mill Work Index	kWh/t	16.0
Sulphide Flotation Circuit Regrind Ball Mill Work Index	kWh/t	10.0
Gravity Circuit Regrind Ball Mill Bond Work Index	kWh/t	13.0
Production Criteria		
Scheelite Recovery to Gravity Concentrate	%	55.0
Scheelite Gravity Concentrate Tungsten Grade	% WO_3	67.0
Scheelite Recovery to Flotation Concentrate	%	27.0
Scheelite Flotation Concentrate Tungsten Grade	% WO_3	55.0
Overall Scheelite Recovery	%	82.0
Overall Scheelite Concentrate Tungsten Grade	% WO_3	63.0

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 MINERAL RESOURCE ESTIMATE

17.1.1 SUMMARY

Scott Wilson RPA has completed a 3D solid model, or wireframe, and 2D block model for the Mactung property. The 2D model is a gridded seam model (GSM) and contains two separate estimates of grade (kriged and polygonal) and a single estimate of thickness for each block. Scott Wilson RPA considers the kriged estimate more appropriate for this deposit. Table 17.1 summarizes the Kriged estimates.

Table 17.1 Indicated and Inferred Mineral Resources Estimate

Classification	kt	WO_3 (%)	WO_3 (kt)	mtu's (millions)
Indicated	33,029	0.88	290	29.0
Inferred	11.857	0.78	92	9.2

The kriged estimates, which are based on assays capped at unique levels for each zone, are reported at a block cut-off of 0.5% WO_3, which Scott Wilson RPA considers appropriate for the location and cost profile that can be expected for Mactung. CIM definitions (December 2005) were followed for the classification of the mineral resources. Scott Wilson RPA estimates an average drill spacing of 50 m based on the average distance between each composite and its four nearest neighbours. Scott Wilson RPA considers the spacing close enough to classify approximately 76% of the resources as indicated.

Further drilling is required to improve the reliability of the mineral resource estimates for the upper 3 zones (3D, 3E, and 3F) as well as on the periphery and northern portions of all four zones. Initial efforts should focus on the areas of the deposit where higher grade intercepts have not been entirely closed off by peripheral drilling. Assessments of grade variability indicate a maximum drill spacing in the range of 120 m to classify any portion of the resource as indicated category, although this must be confirmed by further drilling in the upper 3 zones.

17.1.2 DATABASE – GENERAL DESCRIPTION

The mineral resource estimates for the Mactung project are based on information from surface and underground drilling supplemented in part by surface and underground mapping. The collar database provided to Scott Wilson RPA contains 168 drill holes, of which 161 were used for grade interpolation. Those drill holes within the modeled area cover a 1,000 m (E-W) by 700 m (N-S) area. Holes vary in length from 5 m to 450 m and the average spacing in plan is estimated at 50 m based on the average distance between each composite and its four nearest neighbours. Most surface holes are drilled toward the north at -50° to -90° (vertical), with the majority drilled at -70°. Underground holes are drilled at various horizontal and vertical angles (up and down) from the underground drifting in the upper 2B horizon.

17.1.3 ASSAYS

The assay database provided to Scott Wilson RPA for the Mactung Project contains 6,311 assay intervals, of which all but 414 contain assay values of 0.01% WO_3 or greater. A total of 3,738 intervals are located within the interpreted mineralized zones. Assay intervals vary from 0.3 m to over 8 m in length, although most are 1.5 m. The data approximate a normal distribution when transformed to natural log values (lognormal). A brief statistical summary of WO_3 assays within the interpreted mineralized zones is provided in Table 17.2.

Table 17.2 WO_3 Assay Statistics

		Grade (% WO_3)	
Zone	**Count**	**Average**	**Standard Deviation**
2B	1,904	1.238	1.111
3D	823	0.692	0.589
3E	529	0.604	0.602
3F	482	0.862	0.829
Total	**3,738**	**0.980**	**0.958**

Note: Only those assays within the interpreted mineralized zones are reported.

17.1.4 GEOLOGICAL MODEL

For the purpose of resource estimates, the Mactung deposit has been modeled as four sheet-like mineralized envelopes dipping roughly S30°W at -20°. Thickness averages 18 m and varies from less than 1 m to over 50 m. The four main zones have been further subdivided by three main faults and a fold structure to produce 12 individual lenses. Scott Wilson RPA based its interpretation primarily on the previous historic estimate (Strathcona, 1982), although subsequent drilling has led to some changes, particularly around the fault structures. Scott Wilson RPA modeled the hanging wall and footwall surfaces for each lens using an external cut-off of 0.4% WO_3, creating 3D solids, or wireframes, representing the 12 mineralized envelopes.

These envelopes were used to control compositing and block selection in subsequent interpolation runs.

17.1.5 *Assay Capping (Cutting)*

In order to reduce the influence of statistically anomalous sample data on resource estimations, a number of higher-grade assay values are often capped prior to compositing at levels determined by various means, including examination of probability distribution data. Scott Wilson RPA produced plots of the WO_3 distribution for each of the four zones using the assay data provided. The distribution curves exhibit obvious breaks or inflection points in the upper parts of the curves and a general tailing off beyond these points. These breaks often indicate the existence of several distinct populations within the grouped data, with upper values representing a very small fraction of the total population.

Scott Wilson RPA selected the upper break in the distribution curve as caps for assay data. In total, 37 assay intervals were capped. These intervals represent less than 1% of the total number of assays greater than or equal to 0.01. The net impact of the capping was to reduce the average assay grade within the interpreted zones by 0.7% of the uncapped mean grade. Table 17.3 provides a summary of capping statistics for the Mactung data. All data above the stated capping levels were set back to these levels prior to compositing.

Table 17.3 Assay Capping Levels

Grade (% WO_3)						
Zone	**Cap**	**No. SD's**	**Population Maximum**	**No. Capped**	**Average Before**	**Average After**
2B	6.2	4.5	8.40	9	1.238	1.234
3D	2.8	3.6	4.30	8	0.692	0.689
3E	3.0	4.0	4.78	7	0.604	0.594
3F	3.2	2.8	7.53	13	0.862	0.840
Total			**8.40**	**37**	**0.980**	**0.973**

Notes: Values within the interpreted mineralized zone 0.01% WO3 only.
No. SD's is the number of standard deviations that the capping level is from the mean.

17.1.6 *Composites*

Composites, which were produced by Scott Wilson RPA for a "gridded seam" or 2D model, are based on single intercepts for each drill hole that pierces a lens. Assay intervals for each drill hole were composited down the hole from the top of the interpreted mineralized envelope to the point of exit, producing a single length-weighted average grade for each intercept. The composites were tagged with a unique code for each lens to control composite selection in subsequent grade interpolation runs. In Scott Wilson RPA's opinion, the 2D method is ideally suited to the Mactung deposit because of the limited thickness of each lens. A summary of composite statistics is provided in Table 17.4.

Table 17.4 Assay Composite Statistics

Zone	Count	Grade (% WO_3)				Average Length (m)
		Uncapped		Capped		
		Grade	Standard Deviation	Grade	Standard Deviation	
2B	164	1.124	0.653	1.121	0.648	17.1
3D	84	0.622	0.322	0.619	0.318	15.3
3E	61	0.503	0.317	0.492	0.287	14.6
3F	42	0.705	0.408	0.688	0.376	20.0
Total	**351**	**0.846**	**0.575**	**0.84**	**0.568**	**16.6**

17.1.7 Block Model and Grade Estimation Procedures

Scott Wilson RPA built a GSM based on blocks with fixed EW and NS dimensions and variable vertical thickness. Individual block dimensions are 10 m EW by 10 m NS in plan. In the Mactung GSM model, each block carries an interpolated "vertical thickness" value based on the difference between the elevations of the modeled hanging wall and footwall surfaces at block centre. Often, the thickness is interpolated along with grade, using the composite thicknesses. In the case of Mactung, some of the holes were drilled from development within the upper 2B horizon, resulting in a number of composites which are shorter than the actual thickness of the lens. As well, some surface holes did not exit the mineralized horizons. Use of these composites to interpolate thickness would have inevitably resulted in an underestimate of block thickness. As an alternative, the 3D location of the hanging wall and footwall pierce points were used to model the two surfaces. For drilling that did not completely pierce a particular lens (both underground and surface holes), only the point of contact with the hanging wall or footwall was used.

Each block with its centre located within the interpreted zone was assigned a zone code that matched the composites. A separate "seam" is reserved for each lens, resulting in a total of 12 seams, or 2D matrices, representing the various components or fault/fold blocks of the four mineralized horizons (2B, 3D, 3E, 3F). Each seam is assigned a unique code and grades estimated using only those composites with matching codes. For reporting purposes, the estimated grades for each block are weighted by their corresponding thickness values and assigned densities.

Ordinary kriging and polygonal methods were utilized to estimate grades for each block. The polygonal method assigns each block within a particular seam the grade of the closest matching composite while the kriged grade is based on a weighted average of the surrounding matching composites. Grade estimates were based on 2D searches with the distance in the XY plane only considered. Scott Wilson RPA reviewed the variography for the deposit and a reasonable model was developed, however, the nugget value is relatively high, indicating high grade variability between closely spaced holes. Small-scale faulting may be, in part, responsible for the variability. More close-spaced drilling, particularly in the three upper horizons, would help in providing a more definitive assessment of the spatial continuity for the grades and structures within this deposit. Table 17.5 summarizes the parameters for the variogram model used for Mactung. Models for 3D and 2B are depicted graphically in Figure 17.1 and Figure 17.2.

Table 17.5 WO_3 Variogram Model

Parameter	Value
Model Type	Spherical
Nugget C_0	0.55
Sill $C_0 + C_1$	1.05
Range (m) – Major (Y)	120
Range (m) – Minor (X)	120
Range (m) – Vertical (Z)	NA

Figure 17.1 Mactung Variogram Model – 3D

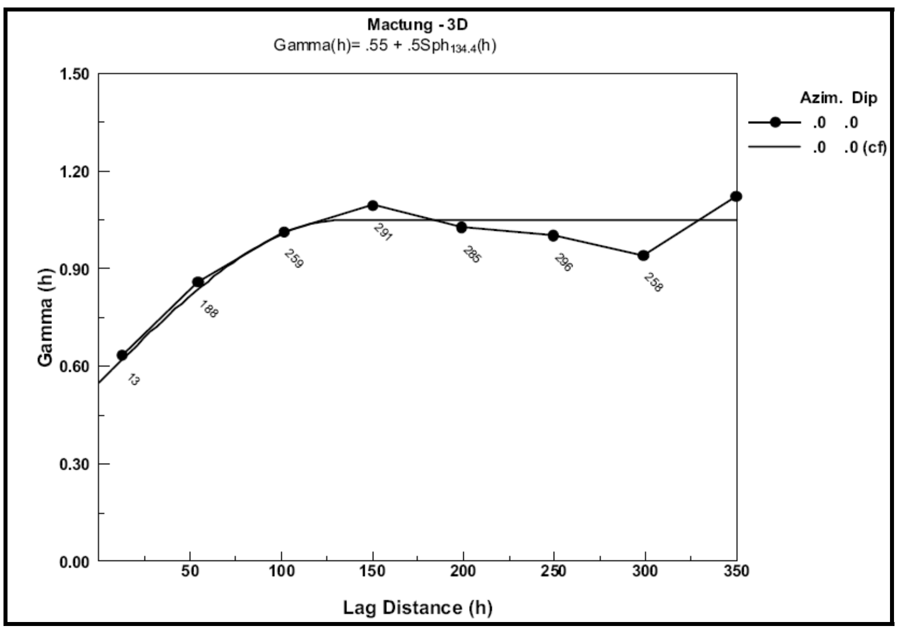
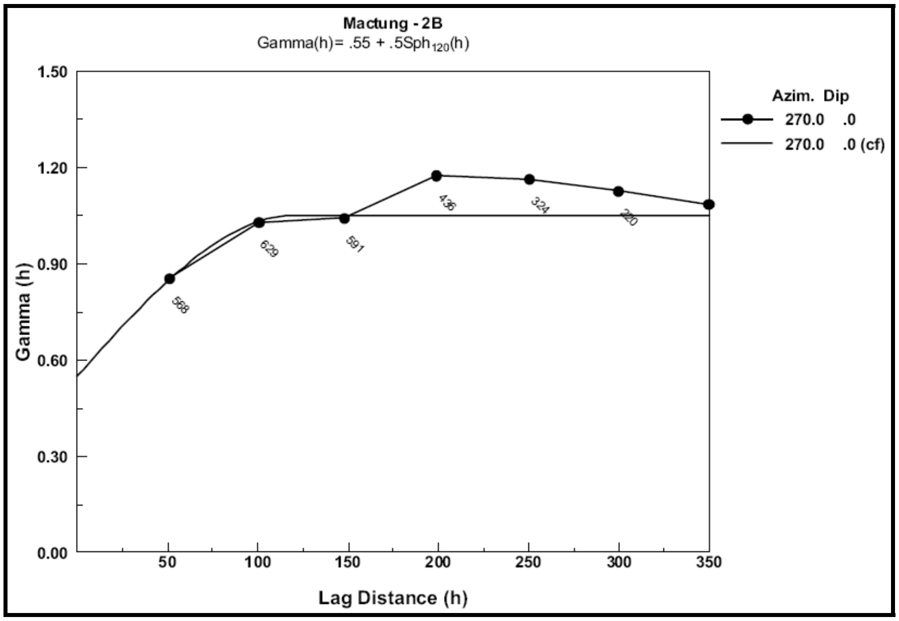


Figure 17.2 Mactung Variogram Model – 2B



The minimum and maximum numbers of composites used in kriging interpolations were set at 2 and 4, respectively. This allows a grade to be estimated for any block with two or more holes within the search area but prevents over-smoothing the estimate by using too many composites. The search strategy employed by Scott Wilson RPA was based on two passes with a maximum search distance of 120 m (major) x 120 m (minor) on the first pass and 80 m (major) x 80 m (minor) on the second pass. The second pass also required that the closest composite be no more than 60 m from the block centre. Those blocks that meet the search criteria in the second pass are overwritten while leaving other blocks unchanged from the first pass. The two-pass strategy allows most blocks within the interpreted envelopes to receive grade estimates on the first pass, while using a tighter search on the second pass to reduce the influence of distant composites on grade estimates close to the drill holes. Grade estimates were made using only the composite values derived from the capped assays.

Finally, after completing the grade interpolations, overall grades were calculated by weighting the block grades by the thickness of each block within the interpreted mineralized envelopes. A minimum vertical thickness of 4.5 m was applied to each block, diluting the grade accordingly, if required. The tonnage is estimated by multiplying their respective volumes by the SG. For the purposes of the estimates, an SG of 2.99 was used for 3D, 3E, and 3F, while a value of 3.14 was used for 2B. The latter is based on densities established by the underground bulk sample taken from 2B. Scott Wilson RPA has not reviewed any data or calculations related to the SG determination. Strathcona used identical values in their estimates while the preceding study (Steininger, 1980) used an SG of 3.08 for the upper horizons. Application of the methodology described above resulted in the estimates of mineral resources summarized in Table 17.6 and Table 17.7.

Table 17.6 Indicated Mineral Resource Estimates

	Kriged				Polygonal			
Zone/Lens	kt	WO_3 (%)	Vertical Thickness (m)	kt WO_3	kt	WO_3 (%)	Vertical Thickness (m)	kt WO_3
3F South	4,053	0.77	23.7	31.2	3,757	0.84	23.9	31.4
3F North	2,299	0.69	26.3	15.8	2,422	0.73	27.7	17.7
3E South	3,364	0.64	22.2	21.5	2,875	0.74	23.7	21.3
3E North	1,705	0.65	26.1	11.0	1,485	0.74	25.9	11.0
3D SE	16	0.60	13.5	0.1	165	0.63	13.2	1.0
3D S Wedge	177	0.74	6.8	1.3	143	0.90	6.6	1.3
3D South	6,037	0.75	23.3	45.3	5,781	0.79	23.8	45.9
3D North	2,576	0.75	20.9	19.3	1,764	0.93	18.7	16.4
2B Upper	9,174	1.09	26.9	100.2	8,400	1.14	27.8	96.2
2B Middle	-	-	-	-	-	-	-	-
2B Lower South	2,310	1.38	21.8	32.0	2,277	1.51	22.3	34.3
2B Lower North	1,318	0.93	20.0	12.2	725	1.22	18.1	8.8
Totals	**33,029**	**0.88**	**24.1**	**289.9**	**29,794**	**0.96**	**24.6**	**285.4**

Notes: - CIM definitions were followed for mineral resources.
- Mineral resources are estimated at a block cut-off grade of 0.5% WO3.
- Polygonal estimates shown for comparison purposes only.
- Differences in totals due to round-off.

Table 17.7 Inferred Mineral Resource Estimates

	Kriged				Polygonal			
Zone/Lens	kt	WO3 (%)	Vertical Thickness (m)	kt WO3	kt	WO3 (%)	Vertical Thickness (m)	kt WO3
3F South	1,398	0.72	17.3	10.1	1,228	0.94	17.7	11.5
3F North	2,316	0.77	25.6	17.8	2,366	0.98	26.5	23.1
3E South	333	0.54	25.3	1.8	835	0.63	23.9	5.2
3E North	1,163	0.62	25.7	7.3	1,339	0.69	25.0	9.2
3D SE	648	0.79	11.0	5.1	636	0.81	11.0	5.2
3D S Wedge	-	-	-	-	-	-	-	-
3D South	2,436	0.77	26.8	18.8	2,393	0.81	26.9	19.4
3D North	905	0.66	21.1	6.0	315	0.94	18.9	3.0
2B Upper	743	0.61	23.6	4.5	218	0.79	25.9	1.7
2B Middle	230	1.06	6.6	2.4	185	1.26	6.4	2.3
2B Lower South	719	1.16	18.4	8.3	719	0.94	18.4	6.7
2B Lower North	965	1.02	23.3	9.8	544	1.58	24.1	8.6
Totals	**11,857**	**0.78**	**22.6**	**92.0**	**10,778**	**0.89**	**23.0**	**96.0**

Notes: - CIM definitions were followed for mineral resources.
- Mineral resources are estimated at a block cut-off grade of 0.5% WO3.
- Polygonal estimates shown for comparison purposes only.
- Differences in totals due to round-off.

In Scott Wilson RPA's opinion, a cut-off of 0.5% WO_3 would be appropriate for reporting purposes. The current cut-off for NATC's operating mine is 0.75% WO_3; however, should the Mactung property be put into production, it is likely that its larger resource would allow significant gains in economy of scale. Based on current prices, a case could certainly be made for a lower cut-off, however, from a long-term price and cost perspective, 0.5% WO_3 is reasonable. Table 17.8 and Table 17.9 provide estimates at increasing block cut-off grades; however, caution is advised in the use of higher cut-offs because the resource breaks up into smaller clusters of blocks as the cut-off increases and may be difficult to mine without encompassing some of the surrounding lower grade material reported at lower cut-off grades.

Table 17.8 Indicated Mineral Resource Estimates Shown at Increasing Cut-offs

	Kriged				Polygonal			
Cut-off (WO_3 %)	kt	WO_3 (%)	Vertical Thickness (m)	kt WO_3	kt	WO_3 (%)	Vertical Thickness (m)	kt WO_3
0.5	**33,029**	**0.88**	**24.1**	**290.0**	**29,794**	**0.96**	**24.6**	**285.4**
0.6	27,927	0.94	24.2	262.2	26,246	1.01	24.5	266.0
0.7	22,198	1.01	24.4	224.8	22,042	1.08	24.4	239.0
0.8	15,571	1.13	24.7	175.6	17,268	1.18	24.2	203.1
0.9	10,423	1.27	25.7	132.2	12,521	1.30	24.3	163.1
1.0	8,245	1.36	26.7	111.8	9,321	1.42	24.9	132.5

Notes: - CIM definitions were followed for mineral resources.
- Mineral resources are estimated at a block cut-off grade of 0.5% WO_3.
- Polygonal estimates shown for comparison purposes only.
- Differences in totals due to round-off.

Table 17.9 Inferred Mineral Resource Estimates Shown at Increasing Cut-offs

	Kriged				Polygonal			
Cut-off (WO_3 %)	kt	WO_3 (%)	Vertical Thickness (m)	kt WO_3	kt	WO_3 (%)	Vertical Thickness (m)	kt WO_3
0.5	**11,857**	**0.78**	**22.6**	**92.0**	**10,778**	**0.89**	**23.0**	**96.0**
0.6	9,260	0.84	23.0	78.0	9,115	0.95	23.3	86.5
0.7	6,614	0.92	22.8	60.9	7,447	1.01	22.9	75.6
0.8	4,631	1.00	21.4	46.1	5,433	1.11	23.7	60.3
0.9	2,929	1.08	22.1	31.7	3,099	1.30	22.5	40.3
1.0	1,658	1.20	21.1	19.9	2,356	1.41	22.8	33.3

Notes: - CIM definitions were followed for mineral resources.
- Mineral resources are estimated at a block cut-off grade of 0.5% WO_3.
- Polygonal estimates shown for comparison purposes only.
- Differences in totals due to round-off.

17.1.8 MODEL VALIDATION

As part of the block model validation process, polygonal grade estimates were also produced and are provided in Table 17.6 through Table 17.9 for comparison purposes. The polygonal estimates are generally lower in tonnage and higher in grade at lower cut-offs; however, metal content is virtually identical at the selected cut-off of 0.5% WO_3.

In addition to producing parallel estimates of grade by polygonal methods, Scott Wilson RPA conducted a series of point validation exercises where the grade at each composite location is estimated from the surrounding composite data by kriging (using the variogram model produced by Scott Wilson RPA) and inverse distance methods and compared to the actual composite values at those locations. Results are shown in Table 17.10. Only those composites that had a minimum of four surrounding composites within the variogram range (120 m) were used in the analysis. In all, 309 points were estimated from the surrounding data. While kriging did not model the extremes, as well due to the high nugget value, the mean and median values are closer to the actual values when compared to inverse distance weighting.

Table 17.10 Point Validation, Composites

		WO_3 (%)				
Item	**IDW Power**	**Mean**	**Standard Deviation**	**Minimum**	**Maximum**	**Median**
ACTUAL	N/A	0.893	0.559	0.000	2.810	0.810
KRIGING	N/A	0.893	0.420	0.120	2.120	0.800
1ST IDW	1	0.890	0.437	0.070	2.500	0.790
2ND IDW	2	0.887	0.462	0.050	2.690	0.780
3RD IDW	3	0.886	0.480	0.050	2.710	0.780
4TH IDW	4	0.886	0.493	0.030	2.710	0.790
5TH IDW	5	0.887	0.504	0.010	2.710	0.790

Inverse distance cubed (ID^3) is often used as an alternative to kriging because grade estimates for blocks that are very close to a composite generally show good agreement with the composite values, while block grade estimates between composites are not overly smoothed like those for inverse distance (ID^1) and inverse distance squared (ID^2). A power of 4 or higher generally produces grade estimates similar to polygonal models, as can be observed in the reported minimums and maximums in Table 17.10. Scott Wilson RPA also analyzed the point validation results for ID^3 and kriging by linear regression. As can be seen in Table 17.11, kriging shows marginally better results with a higher correlation coefficient, a lower intercept value, and a slope closer to 1. A perfect linear correlation would have a 0 Y intercept and a slope of 1.

Table 17.11 Point Validation, Regression Analysis

Item	IDW Power	Intercept A	Slope B	Correlation Coefficient
3RD IDW	3	0.2890	0.6559	0.5902
KRIGING	N/A	0.2174	0.7253	0.5957

Note: Y=A+BX where Y is Actual, X is ID3 or Kriging.

Based on the comparisons of means and medians for actual versus predicted composite grades as well as the results of the regression analysis discussed above, it is Scott Wilson RPA's opinion that the kriged estimates provide superior assessments of grade variability and distribution compared to inverse distance weighting and polygonal interpolation.

17.1.9 CLASSIFICATION

CIM definitions (December 2005) were followed for the classification of the mineral resources. Scott Wilson RPA analyzed the drill spacing within the modeled area by estimating the average distance between each composite and its four closest neighbours. For Mactung, this was determined to be 50 m (this is skewed by the higher density of drilling in 2B). A common approach is to use a threshold for the maximum spacing of 4/3 the variogram range in order to classify a particular area as indicated and 2/3 as measured. Because of the numerous faults and high nugget values observed in the variography, it is Scott Wilson RPA's opinion that the maximum drill spacing for indicated should be no more than the variogram range for indicated. Given a maximum variogram range of 120 m, which should be confirmed by more drilling for the upper 3 zones, the maximum distance from the block centre to the closest composite used in the block grade estimate was set at 60 m (i.e. 120 m spacing) for indicated. No blocks were classified as measured. A small portion of the upper 2B lens where the bulk sample was taken could be justifiably upgraded to measured; however, Scott Wilson RPA has not reviewed the bulk sample data. The 2B middle zone, which is interpreted as the middle limb of a z-fold, has been classified entirely as inferred due to the limited number of intercepts and the presence of abundant fault material that may suggest the limb is actually a fault zone rather than the limb of a fold.

17.1.10 MINERAL RESOURCE ESTIMATE

The indicated and inferred mineral resource estimate is shown in Table 17.12.

Table 17.12 Indicated and Inferred Mineral Resource Estimate

Classification	Kt	WO_3 (%)	WO_3 (kt)	mtu's (millions)
Indicated	33,029	0.88	290	29.0
Inferred	11,857	0.78	92	9.2

Notes:
- CIM definitions were followed for mineral resources.
- Mineral resources are estimated at a block cut-off grade of 0.5% WO_3.
- An mtu is 10 kg WO_3.
- Differences in totals due to round-off.

17.2 MINERAL RESERVE ESTIMATE

17.2.1 MINE CUT-OFF GRADE

Total indicated resources at a 0.5% WO_3 cut-off grade, reported by Scott Wilson RPA in the April 2007 Technical Report, were used by Wardrop to design the mine. Wardrop loaded the data files provided by Scott Wilson RPA into the Gemcom Surpac™ 3D block model.

The mine cut-off grade had to be calculated in order to estimate the mineral reserves. The mine cut-off grade is calculated to be the grade at which the Net Present Value (NPV) of the project is equal to zero and/or when the annual net revenue is equal to the annual operating costs.

The economic parameters as shown in Table 17.13 were used to calculate the mine cut-off grade. The operating costs were obtain from Wardrop's economic study of the Mactung Project in October 2007.

Table 17.13 Economic Parameters for Mine Cut-off Grade Calculation

Items		Units	Value
Metal Price	Gravity Concentrate	US$/mtu	$200.00
	Flotation Concentrate	US$/mtu	$175.00
Exchange Rate	USD to CAD	Cdn$/US$	$1.139
Discount Rate		%	8.0%
Production Rate	Tonnes Milled	Tonnes/day	2,000
Mill Recovery		%	84.0
Concentrate Grade	Gravity Concentrate	%	65%
	Flotation Concentrate	%	45%
Concentrate Recovery	Gravity	%	64%

	Flotation	%	20%
Moisture	Flotation	%	7%
Operating Costs	Mining	Cdn$/t ore	$36.17
	Mill	Cdn$/t ore	$29.87
	Maintenance/surface	Cdn$/t ore	$22.76
	G&A	Cdn$/t ore	$15.09
	Vancouver Office	Cdn$/t ore	$3.43
	Exploration	Cdn$/t ore	$1.00
Third Party Royalty	Estimate	%	1.0%
Net Smelter Return	Gravity Conc transport, Vancouver	Cdn$/t ore	$2.55
	Insurance & Marketing	Cdn$/t ore	$0.51
	Flotation Conc transport, Minnesota	Cdn$/t ore	$1.74
	Insurance & Marketing	Cdn$/t ore	$0.16

An allowance for sustaining capital estimated at about $3.5 million per year was allowed for in the mine cut-off grade analysis.

The mine cut-off grade of 0.616% WO_3 was determined to be the estimated head grade that equates the NPV of the project to zero.

17.2.2 MINERAL RESERVES BASED ON 0.616% WO_3 CUT-OFF GRADE

Using the 0.616% WO_3 mining cut-off grade, Wardrop re-calculated the probable mineral reserves using Gemcom Surpac™ Version 6.02 software. The results are shown in Table 17.144.

Table 17.14 Mineral Reserve Estimate

	Upper 2B		Lower 2B		Total	
Classification	**Tonnes**	**WO_3 (%)**	**Tonnes**	**WO_3 (%)**	**Tonnes**	**WO_3 (%)**
Probable	8,587,951	1.1268	2,201,891	1.4213	10,789,842	1.1869

18.0 MINING

18.1 PROJECT OVERVIEW

18.1.1 PROPERTY GEOLOGY

The deposit consists of scheelite-bearing skarns developed near the south contact of a granite intrusion, known as the Cirque Lake stock.

The main sedimentary sequence dips at low angles to the south. The deposit comprises an Upper and a Lower Skarn Zone. A statigraphic sequence of the property established nine mappable units, distinguished and numerically designated from oldest to youngest: 1, 2B, 3C, 3D, 3E, 3F, 3G, and 4 (Figure 7.1).

Unit 2B, host to the Lower ore zone, is highly variable in thickness and composition. The 2B Upper ore zone provides the main source of ore for the underground mine; this zone is relatively flat along the strike and dips approximately 30% southwest, with some footwall and hanging wall undulations. The zone's thicknesses vary between 10 m and 40 m and averages 26 m. Its strike length varies from 90 to 300 m and averages 180 m. The 2B Lower zone is well-defined, occurring some 20 m below 2B Upper zone. It averages 21 m thick and 74 m long along strike, and is located closer to the northeast striking fault.

A minimum thickness of 4.5 m was applied to the model and grades have been diluted to the minimum thickness where necessary.

18.1.2 MINERAL RESOURCES

The Mineral Resources report, authored by Scott Wilson RPA April 18, 2007 and filed as the Mactung Tungsten Deposit Technical Report, complies with NI 43-101 regulations and provides the basis of this section and the entire technical report.

The kriged estimate contains an indicated mineral resource of 33.0 Mt grading 0.88% $WO_{3,}$ or 290 kt of contained WO_3. An additional resource of 11.9 Mt grading 0.78% WO_3 or 92 kt WO_3 was estimated for the inferred category. These estimates are reported at a block cut-off 0.5% WO_3 (see Section 17.0).

18.2 Mineral Reserves

18.2.1 Mine Break-even Cut-off Grade

Total Indicated resources at a 0.5% WO_3 cut-off grade, reported by RPA in the April 2007 Technical Report, were used by Wardrop to design the mine. Wardrop loaded the data files provided by RPA into the Surpac 3D block model. Wardrop calculated the mining cut-off grade at 0.616% WO_3, based on operating cost estimates developed in the October 2007 Internal Economic Update.

18.2.2 Mineral Reserves Based on 0.616% Cut-off Grade

Using the 0.616% WO_3 mining cut-off grade, Wardrop re-calculated the probable mineral reserve using Surpac Version 6.02 software. The results are shown in Table 18.1.

Table 18.1 Mineral Reserve Estimate

Classification	Upper 2B		Lower 2B		Total	
	Tonnes	**WO_3 (%)**	**Tonnes**	**WO_3 (%)**	**Tonnes**	**WO_3 (%)**
Probable	8,587,951	1.1268	2,201,891	1.4213	10,789,842	1.1869

18.2.3 Underground Mineral Reserve Estimate

NI 43-101 specifications define a mineral reserve as "the economically mineable part of a measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study". Based on this definition, the mineral reserves contained in the Yukon side of the Mactung deposit were calculated for long-hole (LH) stope mining using a total recovery factor of 73.2% and 12% dilution, grading 0.1% WO_3 For mechanized cut-and-fill (MCF) mining, mineral reserves were calculated at 83.5% recovery and 9% dilution, grading 0.1% WO_3 (Table 18.2).

Table 18.2 Mineable Reserve Estimate – Yukon

	Upper 2B		Lower 2B	
	Tonnes	**WO_3 (%)**	**Tonnes**	**WO_3 (%)**
Long-hole Stoping	5,862,483	1.058	1,544,276	1.292
Mechanized Cut-and-Fill	1,136,182	0.938		
Total per Zone	6,998,665	1.038	1,544,276	1.292
Grand Total	**8,542,941**	**1.083**		

18.3 DESIGN PARAMETERS

Scott Wilson RPA completed a 3D block model consisting of 10 m x 10 m x 10 m blocks. The block model includes the following parameters:

- northing, easting, and elevation coordinates
- zone identification (Upper or Lower 2B zone, 3C, 3D and 3E zones)
- resource category (indicated or inferred)
- WO_3 percentage
- zone percentage (portion of the block that lies within the zone).

The Mactung project will mine only the measured and indicated mineral resources located within Yukon. LH stoping will be the primary mining method; MCF stoping will be used where the orebody thickness is less than 12 m thick, or in areas where the orebody dips steeply, such as the southeast end of the deposit. Several stopes will be producing simultaneously at any given time to provide the number of working areas to produce 2,000 t/d.

LH stopes will be 17 m wide and 60 m long on the strike, with 4 m rib pillars in between the primary and secondary stopes, and 4 m transverse pillars between stopes on same stope line. Transverse and rib pillars will be permanent. The back or roof of the drilling sill of the stopes will be reinforced with 2.1 m long friction stabilizer type bolts at 1.2 m x 1.2 m spacing and with 8 m long cable bolts on a 1.8 m grid staggered with bolting grid. The walls of the drilling drift will be reinforced with 2.1 m long friction stabilizer at 1.2 m x 1.2 m staggered grid.

Mineralized areas 12 m thick or less and/or areas dipping over 20% will be mined using MCF with stope widths of up to 17 m and 3-m wide permanent rib pillars along the strike of the orebody. The permanent pillars will prevent backfill material from contaminating the adjacent mining stope. Mining lifts will be 4 m high, staggered along the foot wall and hanging wall contacts.

LH and MCF mined out stopes will be backfilled using dewatered mill tailings and waste rock from mine developments.

The mine equipment will be trackless, 10.0 t LHD units and 30 t mine trucks with diesel motors. Lateral and long-hole drilling will be carried out with electrical hydraulic drilling rigs. Personnel and materials will be transported in diesel-powered man carriers.

Mine operations will be conducted without using water for drilling, to avoid freeze-ups that the marginal permafrost conditions may cause. Dust from drilling operations will be controlled with drilling equipment that carries heated water in tanks to provide air/water mist flush. Water will be delivered underground in 1,000 L intermediate bulk containers by a flat bed truck.

The mine layout includes access and haulage ramps as well as 5 m wide x 4 m high crosscuts, which will be bolted on a 1.8 m grid. The ramps and crusher station will have galvanized steel screens bolted on the backs and walls on high areas in the crusher station.

As previously indicated, the cut-off grade used for the design of the mine is 0.616% WO_3.

To allow flexibility in mine output grade control, and to increase the productivity of equipment and personnel, the mine layout enables production from three or four individual stopes at any given time to meet the 2,000 t/d requirement.

Table 18.3 lists basic parameters for mine design including shift schedule, production rate, and net working time per shift.

Table 18.3 Input Data

Operating Factors	Unit	Quantity
Underground Production		
Mine Days	d/month	30.4
Mine Days	d/a	365
Nominal Ore Mining Rate	t/d	2,000
Average Ore Mining Rate	t/a	730,000
Underground Waste Development		
Average Advance Rate	m/month	100
Average Advance Rate	m/d	3.0
Average Cross Section	m^2	20
Waste from Development	t/d	199
Rock Characteristics		
In situ Density Ore	t/m^3	3.14
In situ Density Waste	t/m^3	2.99
Swell Factor	%	60%
Loose Density Ore	t/m^3	1.96
Loose Density Waste	t/m^3	1.87
Ore Average Thickness	m	26.00
Dip	degrees	18
Backfill Requirements		
Average Volume for Backfill	m^3/d	637
Backfill Factor	%	95%
Rock Backfill from Development	m^3/d	107
Backfill Density	t/m^3	1.7
Backfill Requirements	t/d	847

table continues…

Operating Factors	Unit	Quantity
Shift Data		
Working Days per Week	ea	7
Shifts per Day	ea	2
Shift Length	h	10
Shift Change	h	0.75
Lunch/Break/Delays	h	0.75
Equipment inspection	h	0.25
Subtotal Non-productive	h	1.75
Usable Time per Shift	h	8.25
Shift Efficiency	%	82.5%
Usable Minutes per Hour	min	50
Hour Efficiency (50 min in hr)	%	83.3%
Effective Work Time per Shift	h	6.9
Usable Time per Year	h	6,023
Hours Worked per Year	h	7,300
Basic Manyear (52 w x 40 hrs)	h	2,080

18.4 MINE DEVELOPMENT

18.4.1 MINE LAYOUT AND ACCESS

The layout of the underground mine is illustrated in plan view as shown in Figure 18.1 and Figure 18.2. The drawings show the mine infrastructure in relation to the Upper and Lower 2B Ore Zones. The mine is accessed through the existing adit at 1895 m elevation and through the conveyor decline portal at 1845 m elevation. All major mine infrastructures will be located in low grade ore areas of the hanging wall. The rock formation in the hanging wall is classified as geotechnically-competent rock.

The orebody, composed of two superimposed mineralized layers, is approximately 770 m long, 190 m wide and 70 m high including about 20 m of waste thickness between the Upper and Lower zones, dipping approximately -20° to the southwest.

The portal close to the plant is the entrance to the conveyor decline. The decline is a 512 m long section at a grade of -13% that is 5 m x 5 m and ends at the underground crushing station. There will be two conveyor belts installed in parallel along the decline. One conveyor belt transports ore up to the surface from the primary crushing station and the other conveyor transports backfill material from the surface to an underground storage area near the primary crushing station. The decline is also used for personnel access, exhaust ventilation, and for moving equipment in and out of the mine.

Figure 18.1 Mine Infrastructure and Upper 2B Zone

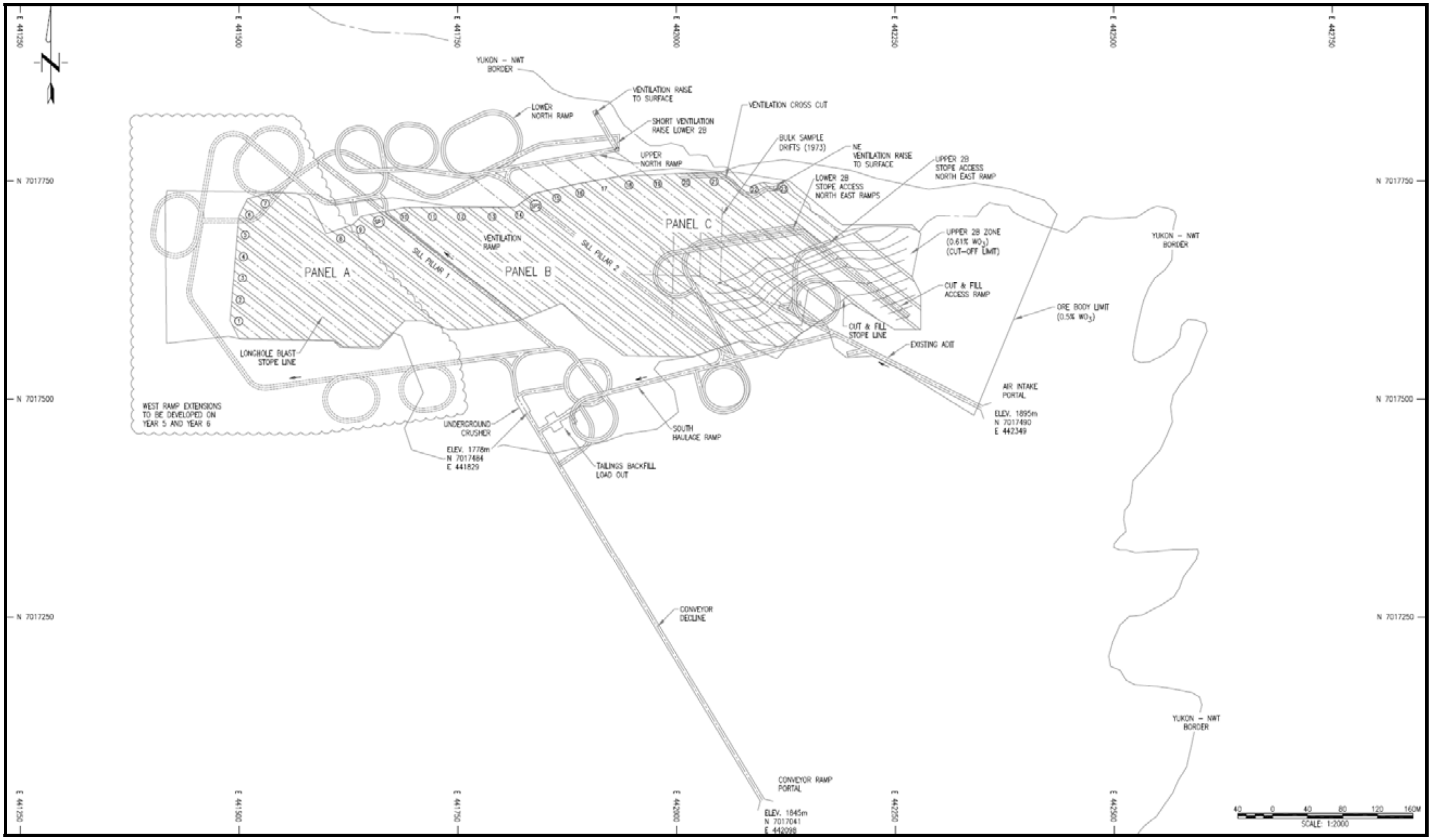
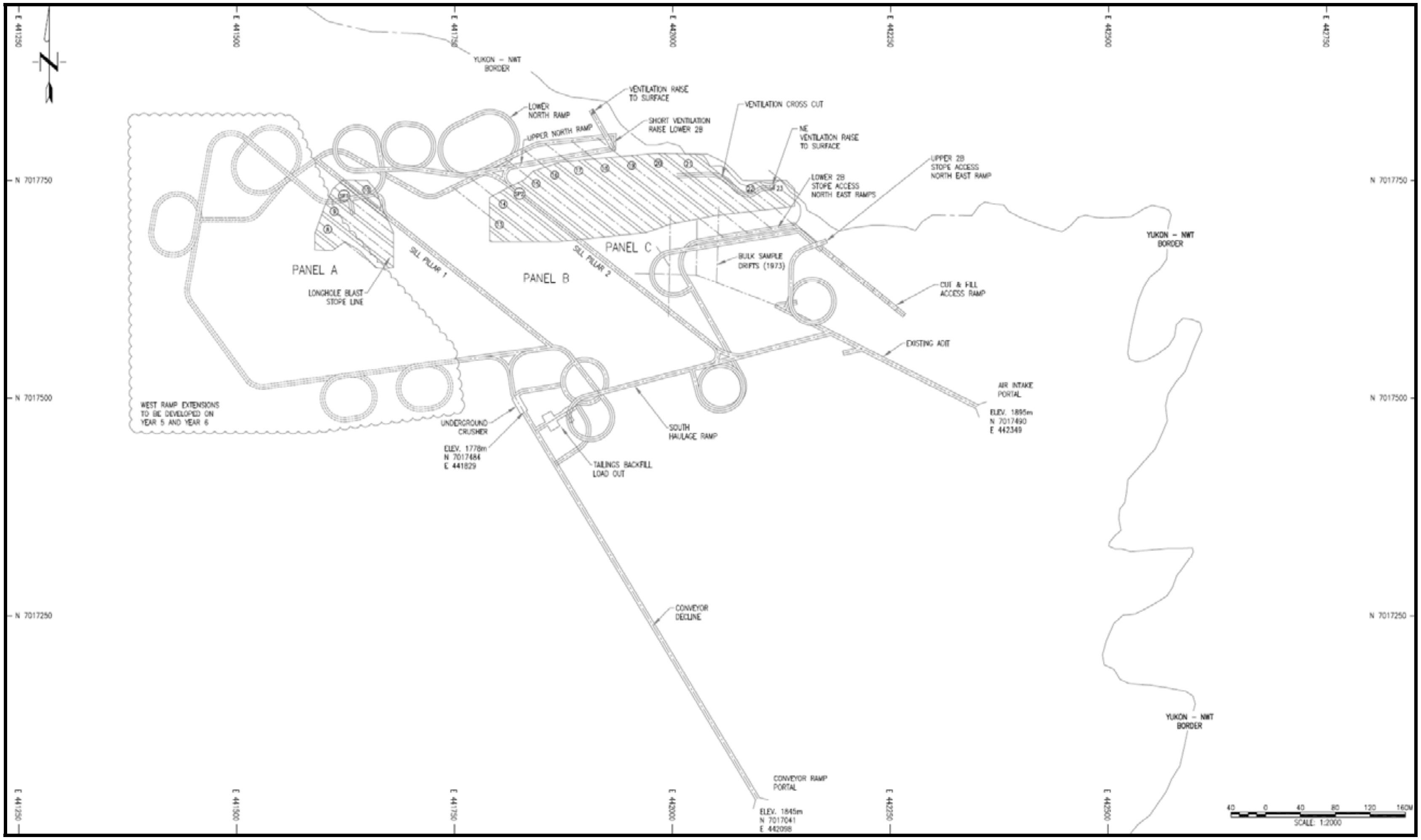


Figure 18.2 Mine Infrastructure and Lower 2B Zone



18.4.2 PRE-PRODUCTION MINE DEVELOPMENT

Pre-production mine development consists of the excavation of underground ramps, drifts, raises and other workings to prepare the mine for production. Pre-production development is estimated to take approximately 20 months.

The ramps, crusher station, raises, and temporary sill pillars are shown in Figure 18.1 and Figure 18.2.

The 1895 m level adit, the upper portion of the South Ramp and the conveyor decline will be developed simultaneously to provide permanent access to existing and future underground mine development.

The crusher station, backfill loading station, and two conveyor belts will be installed concurrent with the excavation of the north side ramps, sill pillar, and raises. Figure 18.3 shows the backfill loading station.

Wardrop estimates that two teams of two shifts of experienced drift miners and one team of mine construction workers can develop a total of 6,358 m of ramps, ventilation raise, crusher station, and other underground excavation in approximately 20 months, at an estimated rate of 10.2 m/d. The total underground excavation will consist of about 4,020 m in mine infrastructure and 2,338 m in stope development.

Table 18.4 summarizes the underground mine infrastructure development during the pre production period.

Figure 18.3 Underground Tailings Backfill Loading Station – Plan and Section View

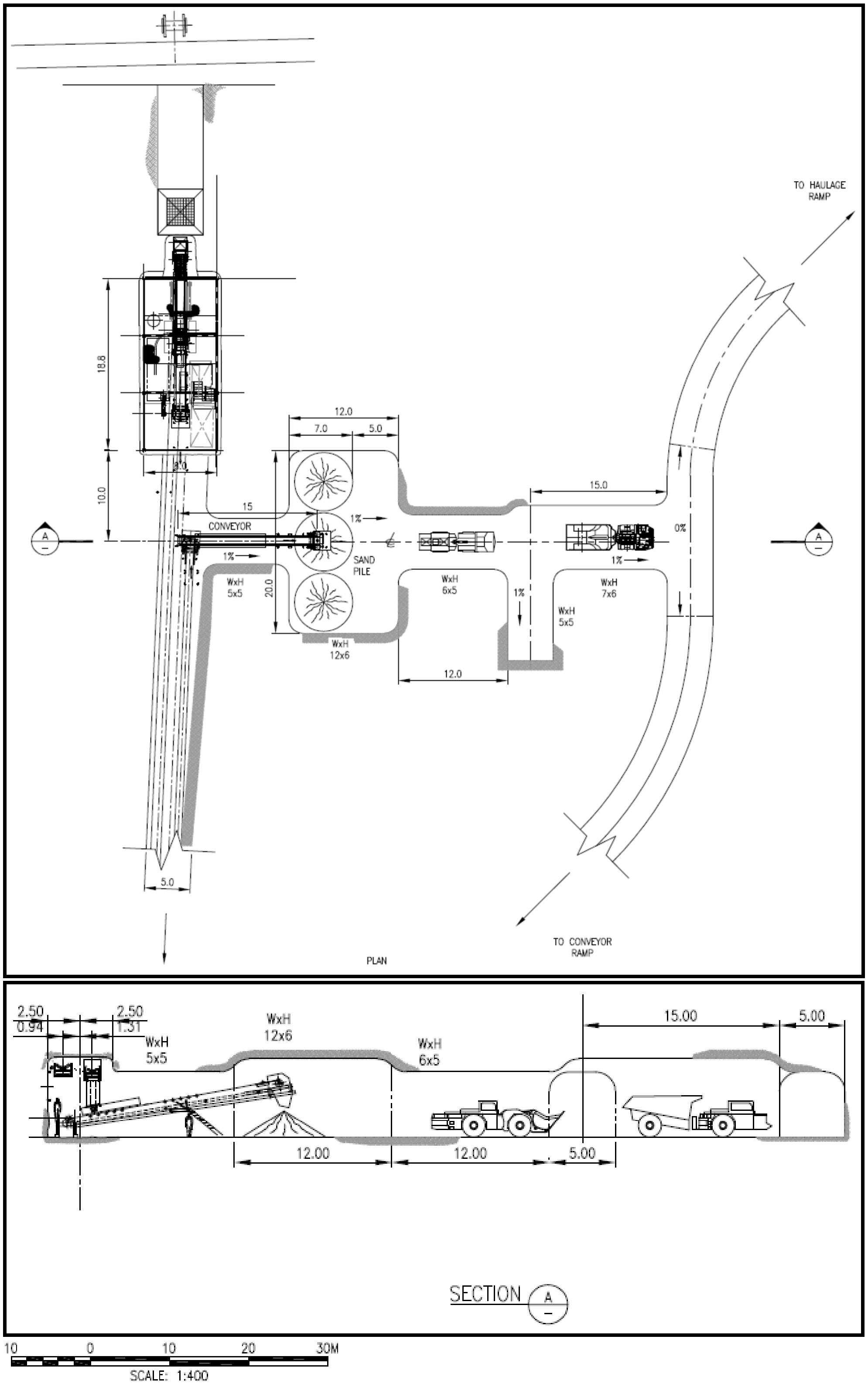


Table 18.4 Mine Infrastructure Development Work

Mine Development Description	Total Development (m)	Pre-production (m)	Development (m)	
			Year 1	Years 5 & 6
Rehabilitate 1895 Level Adit	374	374		
South Ramp Segments	1,206	541		665 (Year 6)
Excavation Crushing Station (10m W x 24m L x 17m H)	4,100 m^3	4,100 m^3		
Crusher Installation	121	121		
Sand Backfill Load Station				
5 m x 5 m at -1% Drift Sand Conveyor Belt	57	57		
20 m L x 12 m W x 6 m H -1% Tailings Loading Station	1,440 m^3	1,440 m^3		
15 m L x 7 m W x 6 m H +1% Tailings Loading Bay	630 m^3	630 m^3		
Ventilation Raises	312	312		
Northeast MCF Mining Area – Developments	984	0	818	166 (Year 6)
Conveyor Decline	1,327	1,327		
Upper North Ramp	1,386	660		726 (Year 5)
Lower North Ramp	1,025	628		397 (Year 5)
Total Mine Infrastructure Development	**6,792**	**4,020**	**818**	**1,954 (Year 5)**
Stope Development	**2,338**	**1,471**	**867**	
Total Development (m/period)	**9,130**	**6,358**	**818**	**1,954**

18.4.3 PRODUCTION DEVELOPMENT

Table 18.5 shows the amount (in metres) of mine development required every year to access and excavate mining stopes in the Upper and Lower 2B Zones. This table includes the ramp developments in years 1, 5, and 6 as shown in Table 18.4.

Table 18.5 Mine Development

	Ore Stope Drifts (m/a)	Waste Stope Drifts (m/a)	Waste Ramps (m/a)	Total Waste (m/a)	Total Development (m/a)
Year 1	1,027	1520	818	2,338	3,365
Year 2	1,204	1,172		1,172	2,196
Year 3	594	901		901	1,495
Year 4	232	612		612	844
Year 5	593	966	1,123		2,082

table continues…

	Ore Stope Drifts (m/a)	Waste Stope Drifts (m/a)	Waste Ramps (m/a)	Total Waste (m/a)	Total Development (m/a)
Year 6	377	443	831	1,274	1,651
Year 7	914	893		893	1,807
Year 8	984	1,122		1,122	2,096
Year 9	937	998		998	1,935
Year 10	213	499		499	712
Total	**6,895**	**9,116**	**2,772**	**11,888**	**18,783**

Note: All drifts and ramps are 5 m wide, 4 m high.

18.4.4 Mine Transport and Crusher Station

All mobile equipment is rubber tired. Mined ore will be transported in 30-t haul trucks from Upper 2B stopes along the South Ramp to an ore pass feeding the underground crusher.

The crusher station will be 10 m wide by 24 m long by 17 m high. It will be located at 1778 m elevation south of the centre of the deposit and in competent ground in the hanging wall. The upper 561 m long segment of the South Ramp will connect the crusher station with the adit at the 1895 level. The South Ramp will be 1,206 m long, and will pass near the crusher station and extend from the adit down to the southwest end of the deposit. Refer to Figure 18.1 and Figure 18.2 showing plan views of the mine developments.

The ore will be crushed underground and conveyed up to the plant on a 1,050 mm (42 in) wide conveyor belt. A second conveyor, installed beside the ore conveyor, will transport dewatered tailings as backfill material, from the plant down to a discharge point close to the crusher station. Both conveyor structures will be suspended from the back by epoxy-grouted, threaded rebar rockbolts, chains, and turnbuckles. The backfill material will be loaded with a LHD on returning haul trucks and transported to mined-out stopes.

A second ramp, the North Ramp, is planned in the competent 3C Zone waste rock, which extends along the ore deposit on the north side.

The North and South Ramps will be connected through both temporary sill pillars to provide haulage flexibility. This drift will be used to truck ore and backfill material and, in later years, as a drilling drift for the mining of the pillar. To reduce pre-production capital expenses, the extension of the North and South ramps will be deferred to about years 5 and 6 of the project.

After the extraction of stopes in this temporary sill pillar in operating year 9 or 10, the connection between the North and South Ramp will be through the west end of the deposit, where both ramps converge.

18.4.5 MINING NEAR THE YUKON BORDER

Due to NATC's decision to mine only within the boundaries of the Yukon, the Upper and Lower North ramps cannot be extended beyond the Yukon border into NWT in order to access the stopes located close to the border from the east. A buffer zone of 5 m will be provided to avoid any development beyond the territorial border.

To access these stopes, a 570 m ramp will be developed northward from the South Ramp, and will connect with the stopes' drilling sill. From there, a 250 m ore-transporting ramp will be excavated to muck the LH stopes, and a 170 m long ramp will be directed south towards the MCF stopes.

Both the LH stopes and MCF stopes will be accessed by crosscut drifts and short ramps from the haulage ramps. The access drifts/ramps will be developed in waste, and will be between 40 and 100 m long.

18.5 MINING METHODS

Two mining methods — LH and MCF stoping — will be used to extract ore from the Upper and Lower 2B mineralized zones.

Both mining methods will use backfill to stabilize excavated stopes. Dewatered mill tailings and waste rock from mine developments will be used as backfilling material. Sill pillars at 4 m wide are designed as skin pillars to confine the fill to the adjacent mining stope.

Ore extracted from stopes and developments will be loaded in 30-t haul trucks by 10.0 t LHD units and will be transported through 5 m wide, 4 m high crosscuts and ramps approximately 750 m up to the ore pass that feeds the jaw crusher. On the return trip, trucks will be loaded with mill tailings at the backfill loading station, located at 450 m, through a 15% ramp, then hauled up to the mined out stope areas over an average distance of 1 km.

18.5.1 LONG-HOLE MINING

LH stopes will be12 to 40 m high, 60 m long and mined along stope lines. They will extend up to 230 m on the strike of the orebody, leaving 4 m wide rib pillars between the primary and secondary stopes.

Transverse pillars, each 4 m wide, will separate stopes at 64 m intervals on the same strike line, producing 60 m long compartments, which will enable independent mucking operations in one stope and backfilling in the preceding stope. Figure 18.4 shows a schematic diagram of LH stope and Figure 18.5 shows a schematic longitudinal section of the upper and lower 2B zones.

Figure 18.4 Schematic Diagram – LH Blast Mining

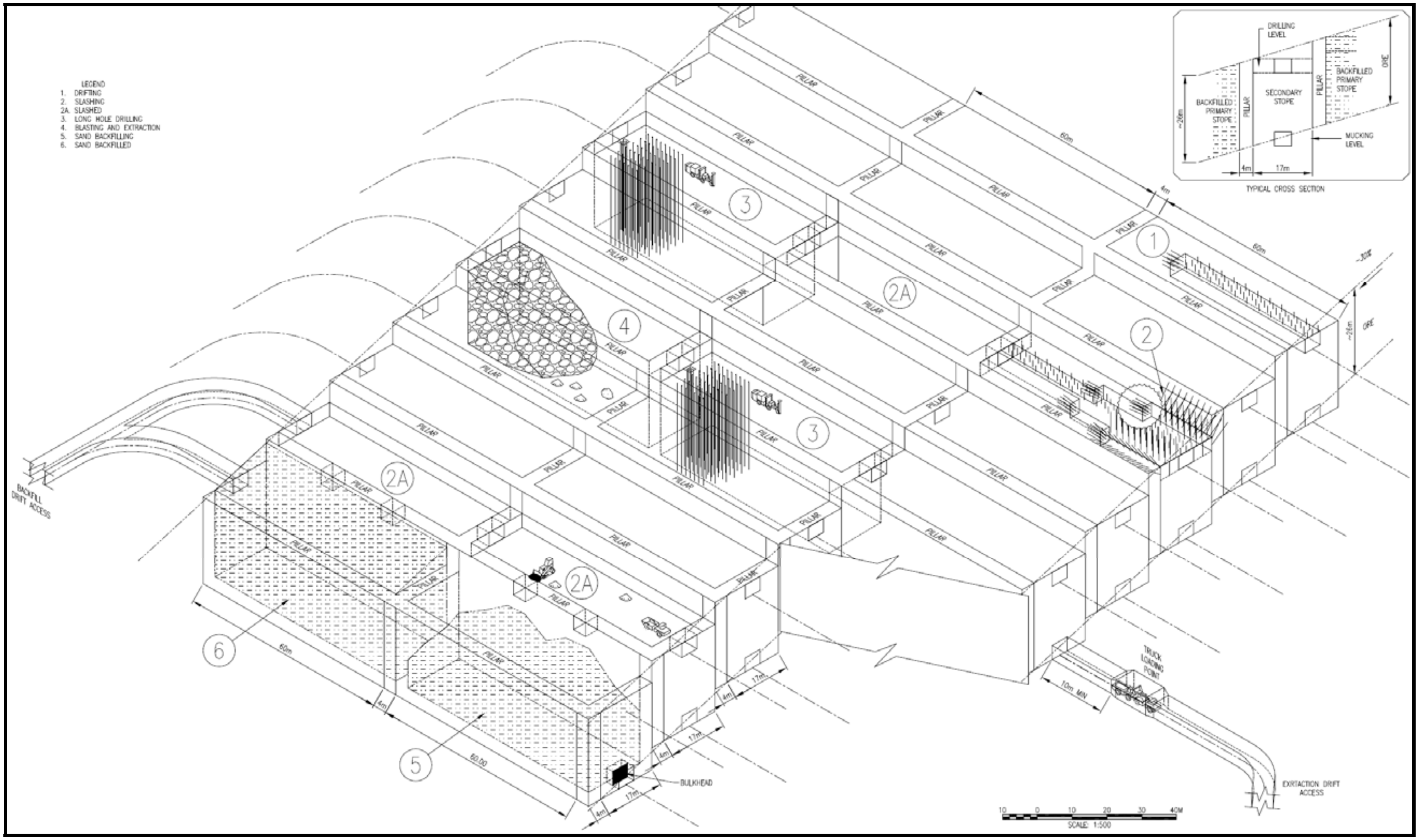
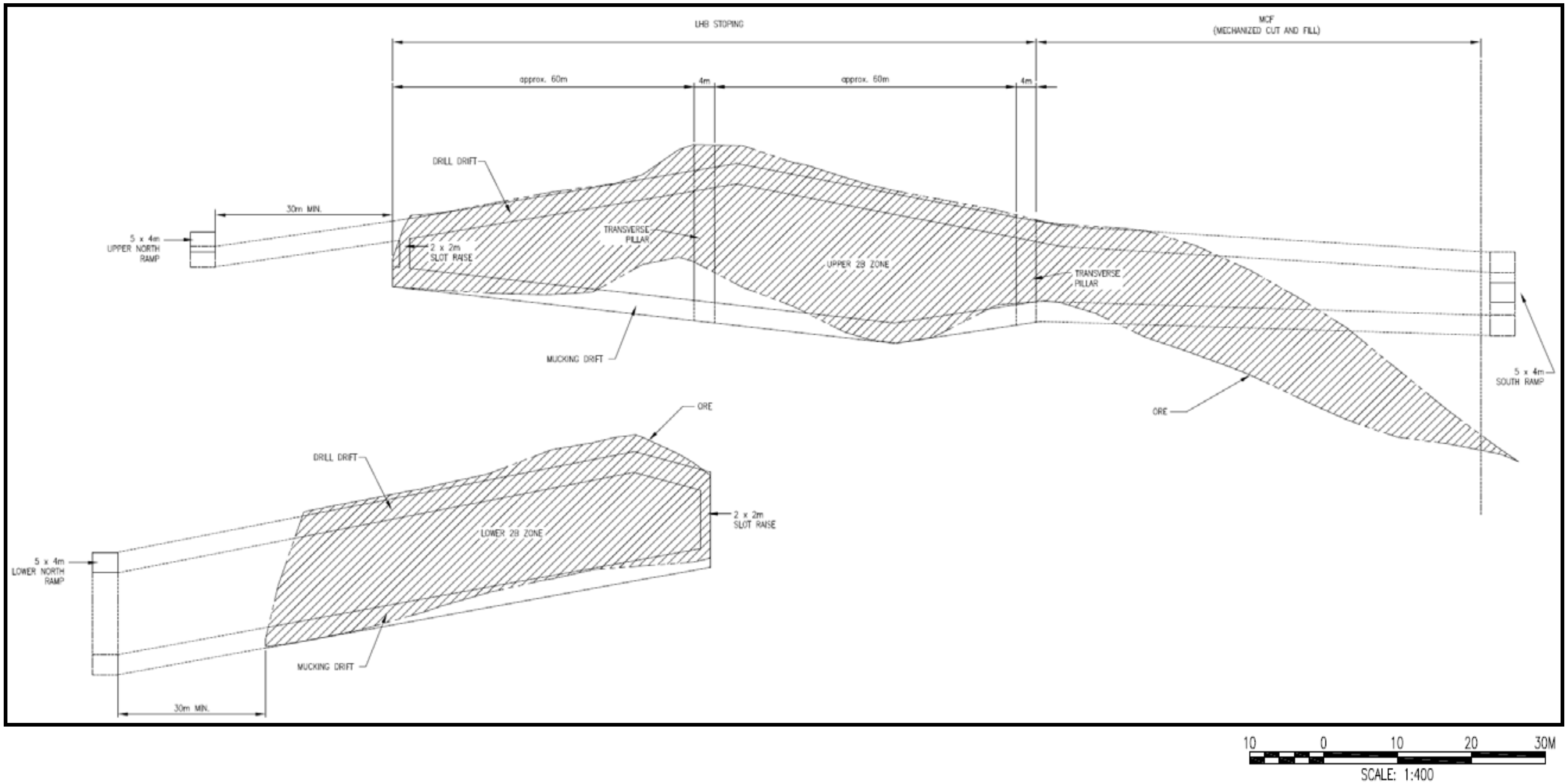


Figure 18.5 Schematic Longitudinal Section – Upper & Lower 2B Zones (Showing Stoping Line 17)



Wardrop divided the mine into three panels by two sequentially-numbered temporary sill pillars (Nos. 1 and 2) located along stoping lines in accordance with ore grade distribution, extraction sequence, and the ore zone dip (Figure 18.1).

Panel A, at the west side of the deposit, contains nine sequentially-numbered stope lines (Nos. 1 through 9). This panel is mined in the final years because of its below-average ore grades.

Panel B is located at the centre of the deposit, closer to the crusher station, and will contain five stope sequentially-numbered lines (No. 10 through 14). This panel will contain average-grade ore and is mined after year five.

Panel C, located at the east side of the deposit, will contain nine sequentially-numbered stope lines (No. 15 through 23). Stope lines No. 15 through No. 18 will contain high- grade ore, and will be mined in the first five years. Stope lines No. 19 through No. 23 present steeply-dipping ore on the south side, requiring MCF mining along contour lines, as shown in. The central and northern areas of these stope lines are mined using the LH method. The southeast area is mined using the MCF method (Figure 18.1).

LH stoping in the Lower 2B Zone will also be carried out in three panels following the same long-term mining sequence as in Upper 2B. The panels contain fewer stope lines and shorter length of stope line than Upper 2B Zone (Figure 18.2).

Based on geotechnical assessment, the stopes are stable enough for stope lines that are parallel and 21 m apart are excavated parallel and in between the primary stope lines, leaving 4 m rib pillars on either side separating them from adjoining primary stopes (Figure 18.5).

Based on the mining sequence, the 60 m long secondary stopes will be mined when the adjoining primary stopes on both sides are completely backfilled.

The bulk of the backfill volume will be hauled through the drilling drift of the active stope, dumped close to the sill of the transverse pillar, and then pushed into the mined-out stope with a small remote-operated bulldozer. Bulldozers will then push the backfill tight against the back of the stope from the stope above, through 4 m wide/4 m high crosscuts driven through the rib pillar (Figure 18.5).

Backfilling of the mined out stopes will be performed by remote-controlled equipment as it is unsafe for personnel to work in the area.

Drill and extraction drifts sized will be driven into the ore as close to the hanging wall and footwall contacts as possible.

Stopes will be aligned along the strike of the deposit with azimuths approximately northwest, and have hanging wall and footwall contacts sloping along the dip of the deposit at approximately 25 to 30% to the southwest.

Drill drifts are driven in the centre line of the stope with the back of the drift 2 m below the contact of the hanging wall. The drift is later slashed to 17 m wide, defining a 60 m long and 17 m wide horizontal sill level needed for parallel long-hole drilling.

Extraction or mucking levels will be driven along the footwall and at the centre line of the stope.

The wedge of ore left on the back of the drill level is blasted at the same time as LH holes with the use of upper blast holes (Figure 18.4).

Ore losses and grade dilution are caused by the blasting of mineralized areas of changing shape along the blasting length or burden, by blasting overbreak and by blasting of areas with layering of waste-ore contacts and possible mini faulting. These conditions are accounted for in the ore reserve and grade dilution calculations using an equivalent 0.6 m waste dilution on hanging wall and footwall contacts. A loss equivalent to 0.2 m thick is included in the estimate for ore left on the stope floor after mining.

In some cases, where footwall contacts present strong undulations, mucking drifts will be driven, crossing the waste rock areas between undulations. LH stoping ore above these levels will be recovered with additional mucking drifts in ore above the undulations to avoid dilution with waste. The ore left between the general transport drift and the short mucking drifts above will be mined by blasting short upper blast holes (back slashing).

The back of drilling sill level will be reinforced with 2.1 m friction stabilizer bolts, using a 1.2 x 1.2 m staggered pattern and 8 m long grouted cable bolting on staggered 1.2 m by 1.2 m grids. The walls of the sill drifts will be reinforced with friction stabilizer bolts, installed on the same grid as the back.

The extraction of the ore will retreat towards the haulage ramp (South Ramp for Upper 2B Zone) via a vertical slot raise, 2 m by 2 m in section, driven in the Upper 2B Zone at the north end of each 60 m long stope and in the Lower 2B Zone at the south end of the stope. An In-The-Hole (ITH) drilling rig, producing 104.6 mm diameter blast holes, will operate on the 17 m wide top sill.

Ore in the stope will be broken by blasting one or two rings of seven drill holes using ANFO, at a total powder factor of 0.36 kg/t (0.8 lb/ton). On average, a total of 132 kg of explosives will be used for each blast hole. The explosive will be delivered to the blasting area using a boom truck with weather treads on tires.

The ore from the Lower 2B Zone is hauled to the crusher ore pass through the Lower North Ramp, passing through Sill Pillar No.1 and reaching the ore pass drift. Ore from the Upper 2B Zone is hauled through mucking crosscuts, the South Ramp, and an ore pass drift. The ore is then dumped onto a grizzly at the ore pass where a rockbreaker breaks oversize rocks. The ore pass, with a storage capacity of 400 tonnes, has a discharge chute with a chain gate and a vibrating grizzly feeder for regulating the feed to the crusher.

Mining will progress, retreating along stope lines from north to south in the Upper Zone and from south to north in the Lower 2B Zone. In the long stoping lines of the Upper 2B Zone, the backfill of stopes located at the north end of the stopes lines will be carried out from the Upper North Ramp without interrupting the extraction upon retreat of the next stope on line. Centre and southern stopes will be backfilled by trucking plant tailings and waste rocks through the South Ramp. In the short stoping lines of the Lower 2B Zone (80 m long), the backfill will be carried from the Upper North Ramp once the stope line is completely mined out.

LH stopes, 60 m long on average, will require a minimum of approximately three months for development, extraction, and backfilling.

In general, the Lower 2B zone stopes have higher grades than the Upper 2B Zone stopes and will be mined first, securing higher grade ore in the first three years of operation. It is estimated that the length of these stopes, averaging 80 m and the waste intercalation of approximately 20 m between the Upper and Lower 2B zones, will have no impact on the stability of the mining operation above the Upper 2B Zone.

18.5.2 *Mechanized Cut-and-Fill Mining*

The total height of the MCF stopes is up to 12 m high (3 cuts), 17 m wide, and 50 to 120 m along strike, with a permanent rib pillar of 3 m between stopes. In areas with wider, steeply dipping ore, the breast mining will advance across the strike of the formation from footwall to hanging wall, approximately 60-80 m apparent width as shown in Figure 18.6. Each cut will be 4 m high along the entire length of the stope. All extraction will be carried out using breast blasting, advancing 4.6 m per blast, and beginning at lower elevation stopes. LHDs with a 10.0 t capacity and 30 t trucks will load the blasted ore and haul it up to the Crusher Ore Pass.

The back of the first 4 m high cut will be reinforced with 2.1 m long friction stabilizer bolts and 16 m long grouted cable bolts. Both systems will be installed on a staggered grid. All stope walls will be reinforced with 2.1 m long friction stabilizer bolts drilled on a 1.2 m grid.

After the first cut is mined out, the empty stope will be backfilled within one metre of the back with either waste rock from underground developments or with tailings backfill material capped with rocks to facilitate movement of wheel equipment. The space without backfill will be used as undercut for the next blasting.

The next 4 m high cut or lift will be by breast-blasted on top of the now-backfilled previous cut, along the whole length of the stope. This sequence will be repeated up to the hanging wall contact, three or four cuts high. The back will be rockbolted, this time without cable bolts. After backfilling the empty stope, the sequence is repeated until the hanging wall is reached. The dangling cables in the stope will be cut with a torch after each lift.

The back of the second cut and each successive cut will be reinforced with 2.1 m long friction stabilizer bolts, installed on a staggered 1.2 m grid between the remaining 16 m long cables that were installed on the first cut. Cable bolts 16 m long were selected to obtain an 8 m thick bolted layer above the upper cut.

MCF mining was selected for ore production from the southeast end of the Upper 2B Zone of steep slopes and limited ore thickness. In this area, some 13,000 t of ore resources are located under the adit at elevation 1895. A crown ore pillar 20 m long, 20 m wide, and 10 m high containing approximately 13,000 t will be left below a 10 m thick waste rock crown pillar.

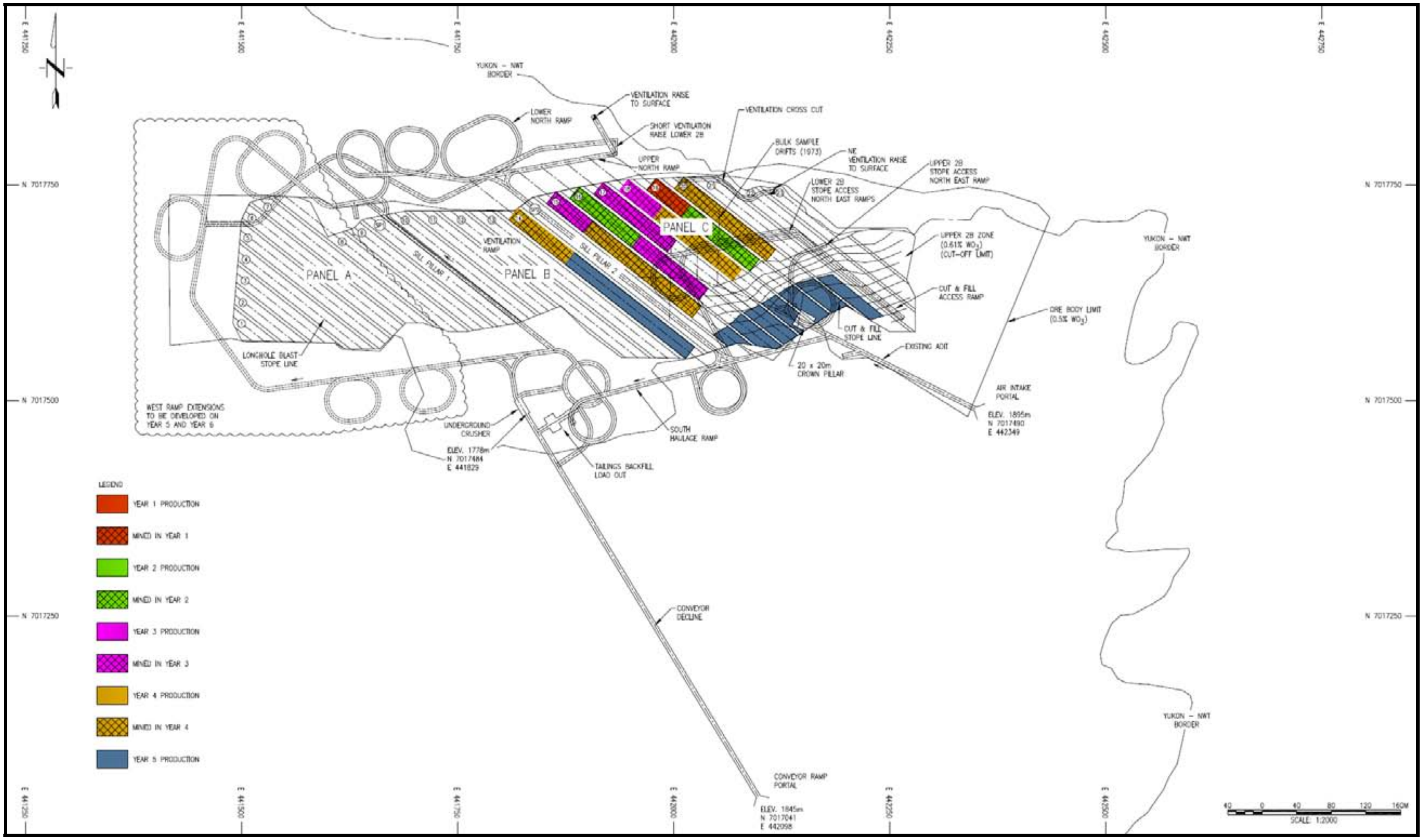
To access the drill sill of these stopes located close to Yukon-NWT border, a 570 m long ramp will be excavated from the South Ramp. From there, a 250 m ore transporting ramp for the LH stopes and a 170 m long ramp accessing the MCF area will be excavated.

Long-hole stopes and MCF stopes will be accessed from the ramps with crosscuts in waste 40 to 100 m long.

During the first five years of operation, higher grade ore located above the crusher station level will be mined so as to reduce the payback period (see Figure 18.6). Ore will be hauled down the ramp by trucks; on the return trip, the trucks will haul backfill material up the ramp. In years 6 to 10, mining will be carried out in lower grade ore areas located below the crusher level.

Ore losses averaging 16.7% and grade dilution averaging 9% will result during MCF mining of an average 12 m high, 17 m wide stope. Main losses are the result of a permanent 3 m wide pillar on a maximum 17 m wide stope (15%) and 1.7% losses after slashing of stope dipping contact walls (hanging wall and footwall). Dilution with waste rock and tailings backfill is the result of steeply dipping hanging wall and footwall contacts and sand fill material dug out with blasted ore from the backfilled floor.

Figure 18.6 Mine Infrastructure and Upper 2B Zone – Year 5



18.6 GEOTECHNICAL EVALUATION

18.6.1 BOREHOLE DESCRIPTION

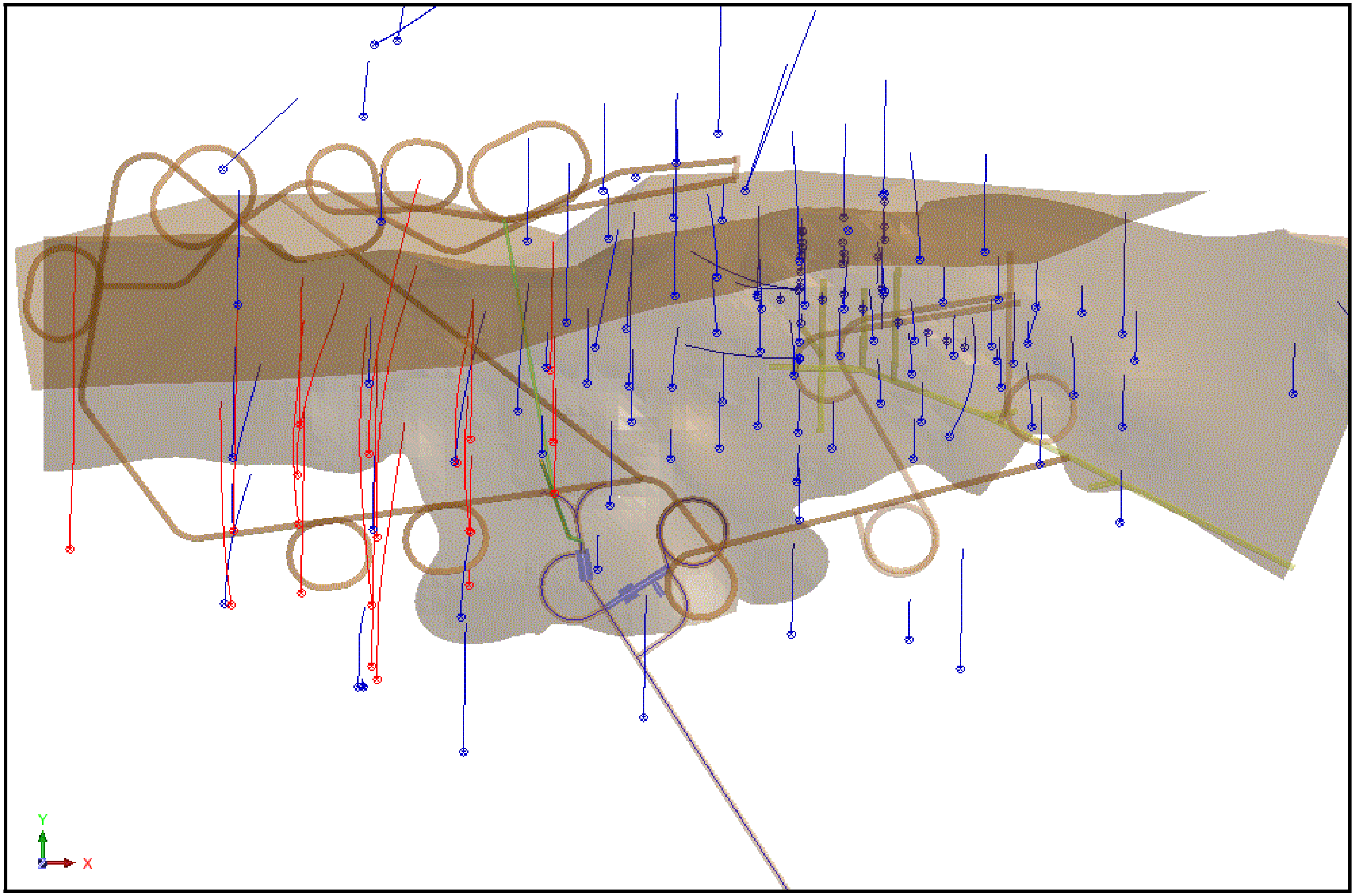
NATC provided basic information of their drilling database on length, core recovery, and geological description in terms of Mactung geological rock codes. Wardrop determined the geotechnical parameters based on a visual assessment of core photographs from 22 boreholes from the MS-series diamond drill holes. This series was selected because its rock quality designation (RQD), one of the parameters for rock mass quality, had been previously determined.

Table 18.6 lists the boreholes evaluated for rock mass classification and the boreholes column information. Figure 18.7 shows the plan for the MS series diamond drill holes (in red).

Table 18.6 Borehole Details

Borehole No.	Northing	Easting	Elevation (m)	Collar Azimuth	Dip (°)	Geotechnical Logging	
						From (m)	To (m)
MS 142	7,017,514.09	441,664.11	2033.84	1.03	-58.5	0.0	365.5
MS 143	7,017,561.41	441,604.07	2066.93	1.03	-70.0	0.0	398.4
MS 144	7,017,600.40	441,606.08	2078.24	1.03	-70.0	0.0	321.0
MS 145	7,017,577.27	441,658.02	2059.68	1.03	-60.0	0.0	377.3
MS 146	7,017,523.93	441,605.32	2054.48	1.03	-70.0	0.0	340.8
MS 147	7,017,462.19	441,660.14	2016.44	1.03	-54.0	0.0	346.9
MS 148	7,017,416.17	441,660.04	1997.64	1.03	-53.0	0.0	340.8
MS 149	7,017,461.90	441,554.99	2035.40	1.03	-66.0	0.0	340.5
MS 150	7,017,416.18	441,659.70	1997.64	1.03	-62.0	0.0	26.8
MS 151	7,017,518.56	441,733.04	2008.73	1.03	-70.0	0.0	261.5
MS 152	7,017,416.16	441,660.00	1997.64	1.03	-62.0	0.0	25.9
MS 155	7,017,399.48	441,653.70	1993.50	1.03	-55.0	0.0	325.6
MS 156	7,017,570.50	441,723.43	2025.30	1.03	-70.0	0.0	164.0
MS 157	7,017,588.76	441,734.07	2024.42	1.03	-70.0	0.0	285.9
MS 158	7,017,477.09	441,732.80	1998.36	1.03	-71.0	0.0	29.9
MS 159	7,017,517.99	441,733.51	2008.94	1.03	-79.0	0.0	261.5
MS 160	7,017,641.64	441,793.79	2013.99	1.03	-70.0	0.0	258.5
MS 161	7,017,471.60	441,607.43	2033.70	1.03	-68.0	0.0	334.7
MS 162	7,017,587.22	441,795.22	1996.20	1.03	-70.0	0.0	191.4
MS 163	7,017,519.09	441,556.53	2055.16	1.03	-62.0	0.0	417.0
MS 164	7,017,547.20	441,796.62	1986.98	1.03	-72.0	0.0	212.1
MS 165	7,017,504.83	441,434.70	2064.16	1.03	-70.0	0.0	447.4

Figure 18.7 Plan of MS Series Diamond Drill Holes



The geotechnical data collected were classified using Barton's Tunnelling Quality Index (Q), which led to a rock mass quality determination and support estimation. The Q classification parameters are based on the block size, shear strength, and active stress.

Barton's Q is defined as:

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

Where:

RQD/Jn = Block Size
Jr/Ja = Inter-block Strength
Jw/SRF = Active Stress
RQD = Rock Quality Designation
Jn = Joint Set Number
Jr = Joint Roughness Number
Ja = Joint Alteration Number
Jw = Joint Water Reduction Factor
SRF = Stress Reduction Factor

18.6.2 SUMMARY OF GEOTECHNICAL OBSERVATIONS

For the purpose of the initial geotechnical analysis at Mactung, the Barton rock mass rating is classified using six categories, ranging from Extremely Good to Very Poor. Table 18.7 provides the Q range associated with each classification.

The factors for Jw and SRF in the Tunnelling Quality Index (Barton et al, 1974) for Mactung's open stope dimension with the use of Stability Graph analysis are set at 1.0. This is assuming that the joints have non-existent to minor inflow of less than 5 L/min for the Jw factor. The SRF factor is estimated based on medium stress environment where joints are moderately clamped but not overly stressed.

Table 18.7 Preliminary Mactung Geotechnical Model

Rock Mass Quality	Range of Q
Very Poor	Less than 1
Poor	1 to 4
Fair	4 to 10
Good	10 to 40
Very Good	40 to 100
Extremely Good	Greater than 100

The criteria set forth by Wardrop for the rock mass classification of Mactung drill cores are as follows:

- RQD for the overburden is set at zero (0) and the remaining reported RQD of zero (0) in very poor ground condition to gouge and fault is being adjusted to 10-20% upon visual inspection of core photographs.
- Jn is set at 1-2 rating, representing Good quality and massive core with no or few joints to one joint set. A rating of 2-3 or "one joint to one set plus random" is set for RQD less than 75% based on observation of the cores. No orientation was measured.
- Ja for unaltered joint walls with surface staining to tightly healed is set at a rating of 0.75-1. The presence of softening or low-friction clay mineral coating such as talc and graphite is set at a rating of 4. A rating of 4, 6 to 10 is classified for shears, gouge and faults, depending on the condition of the core.
- The active stress (parameter Jw and SRF) is set to 1.0 for dry rock mass, subjected to medium stress conditions.

18.6.3 *INTERPRETED GEOTECHNICAL CONDITIONS*

ROCK MASS CLASSIFICATION

The interpretation of the geotechnical condition is classified under the geological units provided by NATC (Table 18.8). Figure 18.8 shows a schematic geological cross section indicating the geological units.

Table 18.8 Summary of the Modified Tunnelling Index (Q')

Geological Units	Q' min		Q' max	
	Average	Standard Deviation	Average	Standard Deviation
1	84	25.6	328	108.1
4	94	11.2	377	44.8
7	88	21.1	352	85.5
9	61	35.5	247	136.2
2B	82	23.2	324	97.3
2BL	98	2.9	400	0.0
3C	80	1.7	397	6.7
3D	77	33.7	303	141.0
3E	75	32.4	294	135.7
3F	81	26.9	317	122.2
3G	78	28.3	305	123.1
3H	63	38.4	250	152.1

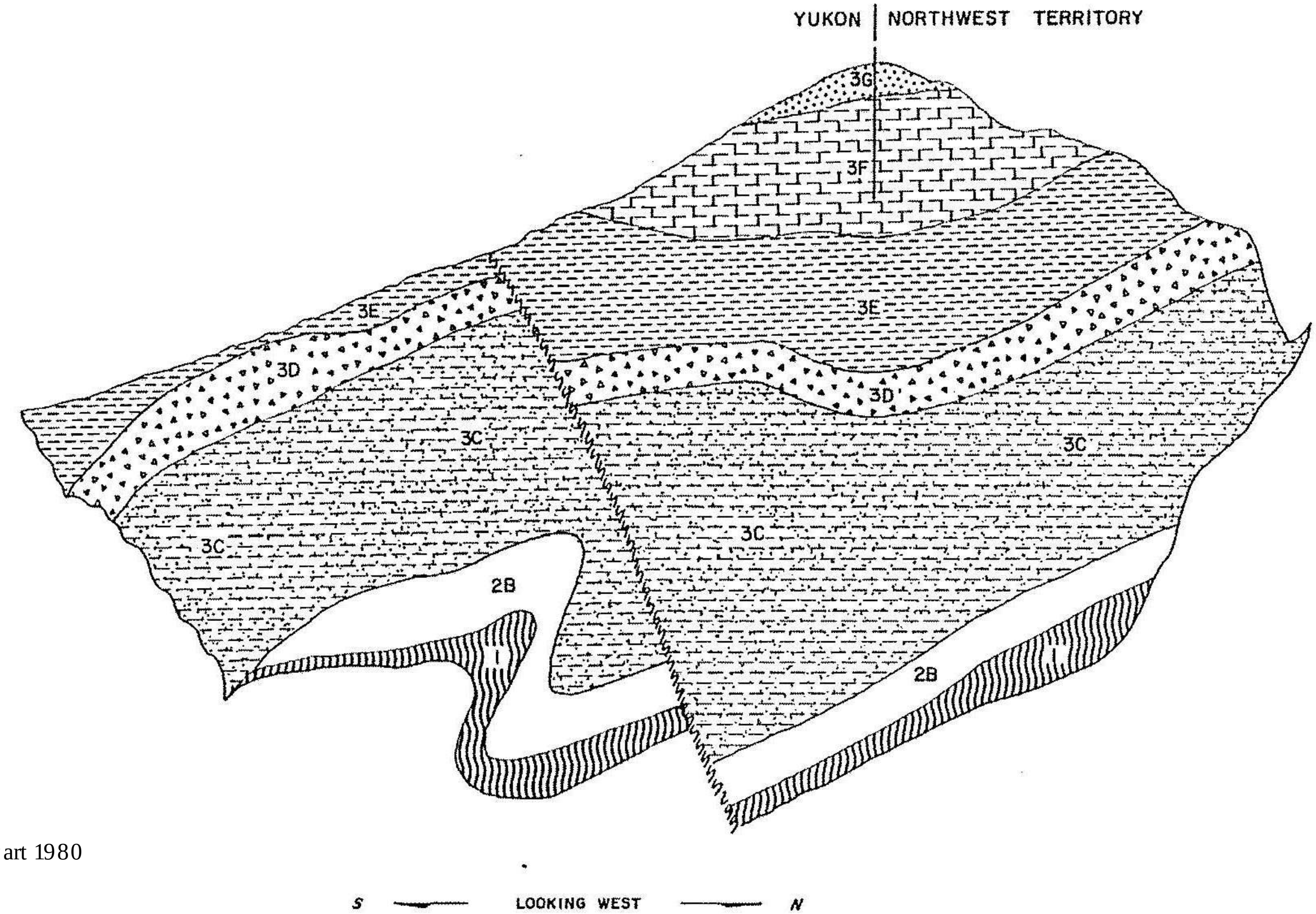
Results from the rock mass rating classified the 2B and 2BL (Upper and lower 2B) as being Very Good to Extremely Good. This classification is consistent with data reported by Strathcona Minerals Limited (Aug. 21, 1982) and Amax (May-October, 1973; revised Oct.1, 1984), which state the following:

- Hanging wall: Rock is argillaceous sediment altered to hornfels which is massive, very hard, and very competent. No difficulty was experienced during bulk sampling underground development in hornfels, except for higher than normal bit wear and steel breakage during bulk sampling (Amax, May-Oct., 1973).
- Ore zone: Generally in good ground with only areas within fault zones likely to cause ground control problems in local areas (Strathcona, Aug. 21, 1982). Mineralized pyroxene-garnet skarn is massive, very hard, abrasive and very competent. No difficulty was encountered during the development of the skarn during the bulk sampling (Amax, May-October, 1973).
- Footwall: Rock is mica schist which is massive with slight foliation, is softer than the skarn and hornfels, but very competent (Amax, May-October, 1973).

Strathcona's 1982 report indicated that the rock types appear to be very tight, with a few vugs with very little moisture contained within the rock.

During the bulk sampling program in 1973, Amax reported that surface drilling cores contained 6.1 to 9.1 m (20 to 30 ft) of gouge, indicating the presence of some areas of extensive faulting in the hornfels. However, the underground development during the bulk sampling program encountered several faults at 38.1 to 53.4 m (125 to 175 ft) interval. The faults ranged from 12.7 to 609.6 mm (0.5 to 24") thick, with frozen gouge material.

Figure 18.8 Schematic of Geological Cross Section Indicating Geological Units (Section 21,300 E)



Source: D.Stewart 1980

JOINT SETS

D. Stewart (1980) identified five joint sets for Mactung (Table 18.9). Joint set 1, or family 1, was identified as the strongest fracture trend and "probably consists of faulting and parallel sympathetic" (D. Stewart, 1980).This set is reported by Stewart to be evident in surface mapping. Flat dipping structures were reported by Stewart to be minor.

Table 18.9 Joint Sets at Mactung

Joint Set or Family	Dip	Dip Standard Deviation	Dip Direction	Dip Direction Standard Deviation	Remarks
1	74.3	9.76	301.6	12.69	Major
2	84.2	7.96	203.7	14.05	
3	51.9	6.74	131.4	13.18	
4	40.7	6.37	36.9	4.11	
5	41.9	6.63	334.7	9.45	

18.6.4 ESTIMATED SUPPORT REQUIREMENTS

SUPPORT CATEGORIES

Wardrop estimated the support categories based on the geotechnical data analysis for the anticipated rock mass quality. The Mactung rock mass categories detailed in Table 18.7 are used to define the support types and classes. Table 18.10 details the ground support classes for each range of the rock mass quality identified at Mactung.

Table 18.10 Ground Support Categories

Support Class	Rock Mass Quality	Ranges of Q'
Type Ia	Good	10 to 40
Type Ib	Fair	4 to 10
Type IIa	Poor	1 to 4
Type IIb	Poor	1 to 4
Type IIIa	Very Poor	0.1 to 1
Type IIIb	Extremely Poor	Less than 0.1

The Type IIa and Type IIb support classes apply to the similar range of Q rating. The distinction between the two support classes is the stand-up time of the excavation, which is described below.

SUPPORT TYPES

Primary Support

Table 18.11 outlines the primary support specification types for each of the ground support categories addressed from the following section. This general guideline can be upgraded to reflect the actual ground conditions encountered once excavation commences.

Type I support is sub-divided into two categories: Type Ia and Type Ib. The only distinction between the two categories is that Type Ia (for Good quality rock) has a bolt spacing of 1.8 m x 1.8 m, while Type Ib (for Fair quality rock) has a bolt spacing of 1.5 m x 1.5 m. A total of five bolts are recommended per pattern, whereby one bolt is placed at each corner and one bolt in the centre of the pattern (staggered pattern). This bolting pattern will provide better confinement and superimposition of support onto the rock mass, thus avoiding loading of the screen with material.

Type II (a and b) support requires that bolting and screening is performed as required to the floor. The distinction between the two support classes is its stand-up time. When the stability of the opening is an issue, the advance rate must be reduced to approximately 70% from the full round of 4.0 m and Type IIb support with 50 mm of shotcrete is recommended.

Type III support is for Very Poor to Extremely Poor ground conditions, where the bolting pattern is reduced to approximately 50%. This support system may only be required in exceptional areas with very poor ground conditions at Mactung. Advance geotechnical data collection and analysis performed in the future will provide an estimated percentage of underground excavation requiring this support system. The Type III support system has not been considered in the cost estimate.

The common approach to Type IIIa support is to lightly scale a majority of the loose material, wash the area, and apply 50 mm of shotcrete. The shotcrete must be allowed to set for a minimum of 8 hours before the installation of rock support and screening, followed by an additional 50 mm of shotcrete over the bolts and screens. When squeezing or highly deformed ground is encountered, the screening and bolting pattern is recommended to continue to floor elevation.

Type IIIb support is similar to Type IIIa, with the addition of lattice girders and spiling. The lattice griders are recommended to be spaced 1 m apart and the spiling are installed over the lattice griders. This allows the lattice griders to support the tail ends of the spiling. The spiling is recommended at least one round ahead (4.0 m) at 25 cm spacing across the back, drilled sub-parallel (inclined +10°) to the back and slightly along the side walls. The spiling will increase stand-up time and reduce the unravelling of the weak material from the back while shotcrete is applied. If ground conditions are weak but not unravelling once open, and the application of shotcrete can be performed successfully, the application of fibre-reinforced shotcrete is recommended followed by appropriate support installation.

Table 18.11 General Ground Support Specification Guidelines

Support Class	Rock Mass Quality	Q' Rating Ranges	Bolting Pattern Bolt Length/Pattern	Mesh	Shotcrete (mm)	Comments
Type Ia	Good	> 10	1.8 m L x1.8 m x 1.8 m	Yes*	No	Systematic bolting to within 1.2 m above floor. Screen, bolts plated and tensioned. Nominal face advance of 4.0 m.
Type Ib	Fair	4 to 10	1.8 m L x 1.5 m x 1.5 m	Yes*	No	
Type IIa	Poor	1 to 4	1.8 m L x 1.2 m x 1.2 m	Yes*	No	Systematic bolting to floor. Screen, bolts plated and tensioned. Nominal face advance of 4.0 m.
Type IIb	Poor	1 to 4	1.8 m L x 1.2 m x 1.2 m	Yes	Yes (50 mm)	Systematic bolting to floor. Screen, bolts plated and tensioned. Nominal face advance of 3 m.
Type IIIa	Very Poor	0.1 to 1	1.8 m L x 1.2 m x 1.2 m	Yes	Yes (100 mm)	Shotcrete sprayed with 50 mm thickness before screening/bolting. Systematic bolting to floor. Screen, bolts plated and tensioned. Nominal face advance of 2.0 m. Re-apply 50 mm of shotcrete onto screen and bolts.
Type IIIb	Extremely Poor	<0.1	1.8 m L x 1.0 m x 1.0 m	Yes	Yes (100 mm)	Shotcrete sprayed with at 50 mm before screening/bolting. Systematic bolting to floor. Screen, bolts plated and tensioned. Nominal face advance of 2.0 m. Spiling of 3.7 long rebars drilled sub-horizontally (+10˚) from the back at 25 cm spacing across the back and sidewalls before advancing next round. Lattice griders may be required to be installed at 1 m spacing. Re-apply 50 mm of shotcrete onto screen, bolts, and lattice girders.

* Screen all permanent openings and as required for Type IIa support. Screen required for shotcreting (Type IIb to IIIb).

When necessary, the application of fibre reinforced shotcrete in selected areas is recommended for the application of the initial layer of shotcrete in Type IIb and III support. Before the application of shotcrete, the corners of the excavation must be clean of debris and blasted muck without jeopardizing the safety of personnel and equipment. All the support had to be installed at least 30 cm from the face with the exception of Type IIIb where the support has to be installed to the face. The application of shotcrete may be required at the face to provide additional confinement if the material from the face has the tendency to cave-in.

All the bolts installed had to be plated and tensioned to provide additional surface confinement to the support system. The torquing of face plate and nut can be achieved with a jackleg with air pressure greater than 90 psi. This provides sufficient pressure on the support system.

The screen can be pre-installed either with the use of a 30 cm length frictional stabilizer bolt or with the assistant of a screen pusher. The screen should overlap for 30 cm.

The ground conditions at Mactung are estimated to be mostly good to fair with some poor ground conditions. In faulted ground, very poor conditions can be encountered. This rock mass quality estimation is made based on visual inspection of the cores.

Estimated Support Types by Geological Units

Generally, each geological unit is in Good to Very Good rock mass where Type I (a and b) support is required based on the estimated Q ratings (Table 18.12). These support design estimates are used as the basis for the support cost estimate (Table 18.13) for the feasibility study. When detailed and advanced geotechnical data and analyses are available, the support systems have to be upgraded to the ground conditions.

The support requirements will be upgraded according to ground conditions during development, based on the recommended support type indicated in Table 18.11. Geotechnical mapping is recommended for every advance to determine the rock mass quality and support requirements. Table 18.13 lists the estimated support length and specifications for excavation types.

Table 18.12 Estimated Support Type per Geological Unit

Geological Units	Q'*		Support Class
	Average	Standard Deviation	
1	84	25.6	Type Ia
2B	82	23.2	Type Ia
2BL	98	2.9	Type Ia
3C	80	1.7	Type Ia

* Based on estimated Q' minimum

Table 18.13 Estimated Support Length and Specification for Excavation Types

Type of Excavations	Width (m)	Height (m)	Estimated Bolt Length	Support Type				
				Pattern** (W x D)		Screen	Back	Side Wall
Fresh Air Intake Adit*	5	4.2	2.1	1.5	1.5	Yes	Resin Rebar	Resin Rebar
Conveyor Decline*	5	5	2.1	1.5	1.5	Yes	Resin Rebar	Resin Rebar
Crosscuts	5	4	2.1	1.2	1.2	Yes	Friction Stabilizer	Friction Stabilizer
Drift	5	4	2.1	1.2	1.2	Yes	Friction Stabilizer	Friction Stabilizer
Haulage Ramp*	5	4	2.1	1.5	1.5	Yes	Resin Rebar	Resin Rebar
Crusher Station*	10	17	2.1 & 5 m Cable bolts	1.5 1.8 (Cable)	1.5 1.8 (Cable)	Yes	Resin Rebar, Shotcrete, Cable bolts	Resin Rebar, Shotcrete, Cable bolts
Ore Chute (Non-man entry)	4	4	Raisebore or Drop Raised					

* To the closest length bolt.
**Four bolts with one centre (staggered pattern) except cable bolts.

Permanent Opening Support Types

Wardrop recommends Type Ib support with 1.5 m x 1.5 m spacing with screening (Table 18.11) permanent openings such as the fresh air intake adit, conveyor decline and haulage ramp, and crusher station. The systematic bolting will be approximately 1.2 m above the floor. Screen and bolts will be plated and tensioned with nominal face advance of 4.0 m. Support types will be upgraded as required based on the rock mass conditions. The suggested support element is forged head resin rebar with screening on the back (roof).

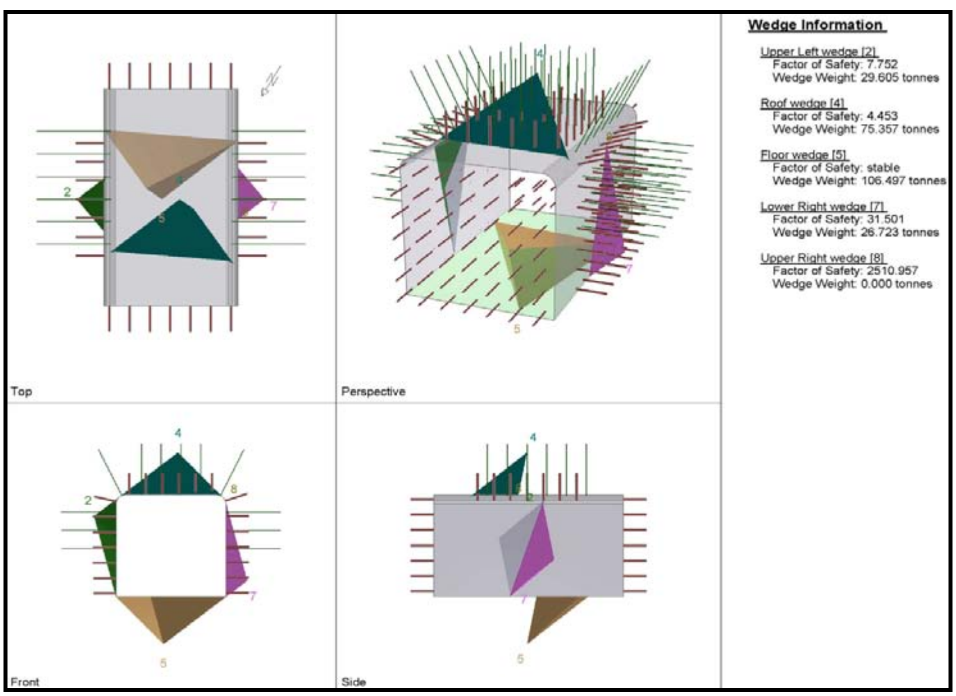
Cable bolts will be required at intersections or when the span of the opening is greater than 5 m.

The crusher station (10 m W x 17 m H x 24 m L) will require complete support to the floor (Table 18.13) with cable bolting spaced 1.8 m x 1.8 m at 5 m length starting at approximately 4.5 m above ground in all excavation faces. This will allow any sliding wedge to be supported and secured before equipment is installed for long term stability of the excavations and safety to both personnel and equipment. A nominal 50 mm unreinforced shotcrete sprayed on top of the screen is required (Figure 18.9). The crusher station is designed for a factor of safety (FOS) greater than 2.

Temporary Opening Support Design

In crosscut and drift, and in temporary opening support design, frictional bolts will be used on the back and walls.

Figure 18.9 Unwedge Analysis for Underground Crusher Chamber



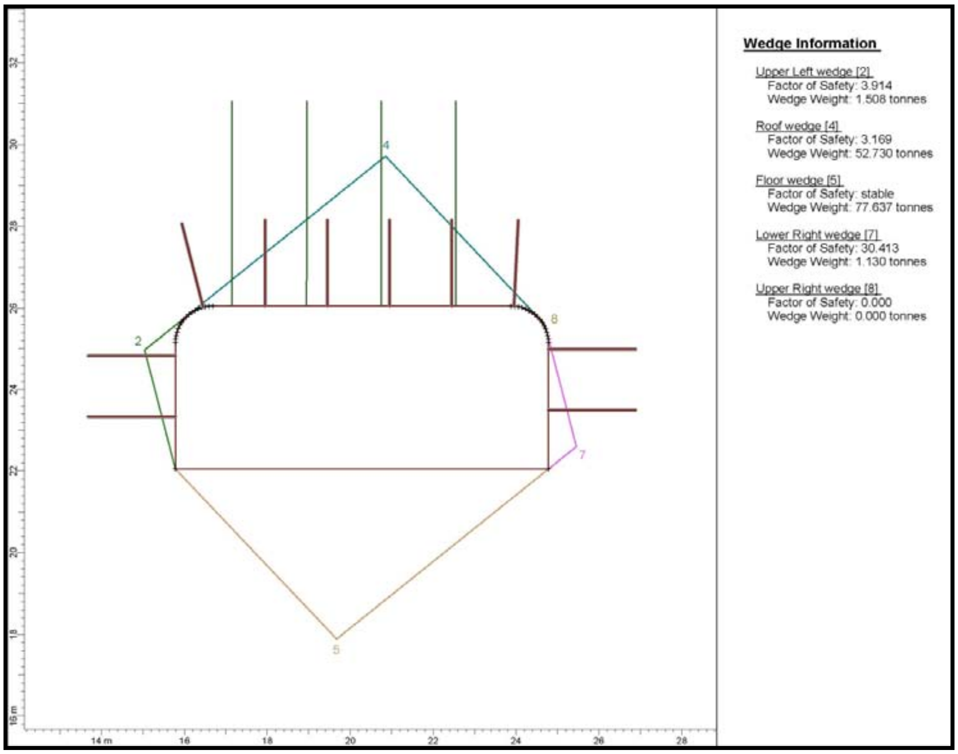
Secondary Support System for Intersection and Sill

Intersection Support

Intersections generate greater spans and expose the back to the wedge formation when the correct joint set angles intercept. All intersections that generate 7.0 m wide span (diagonally) have to be supported by secondary support system (Figure 18.10).

Intersection support is a secondary support system supplementing the primary support system addressed in Table 18.11 and Table 18.13.

Figure 18.10 Intersection Wedge Analysis (7 m Span)



The intersection supports are designed to have an FOS greater than 2. The support must extend at least two to three rows before approaching and after the intersection. This support system provides potential stability issues caused by wedge development, while the primary support addresses the unravelling of material within the secondary support. All intersections in the conveyor decline and permanent openings are recommended to be supported with secondary support.

Wardrop recommends that detailed geotechnical mapping be performed on every intersection, and wedge analysis be performed to identify potential wedge formation. The spacing for the secondary support is recommended to be 1.8 m x 1.8 m and can be more widely spaced, depending on the rock mass quality for the back. These supports must be installed prior to the complete development of the intersecting excavations.

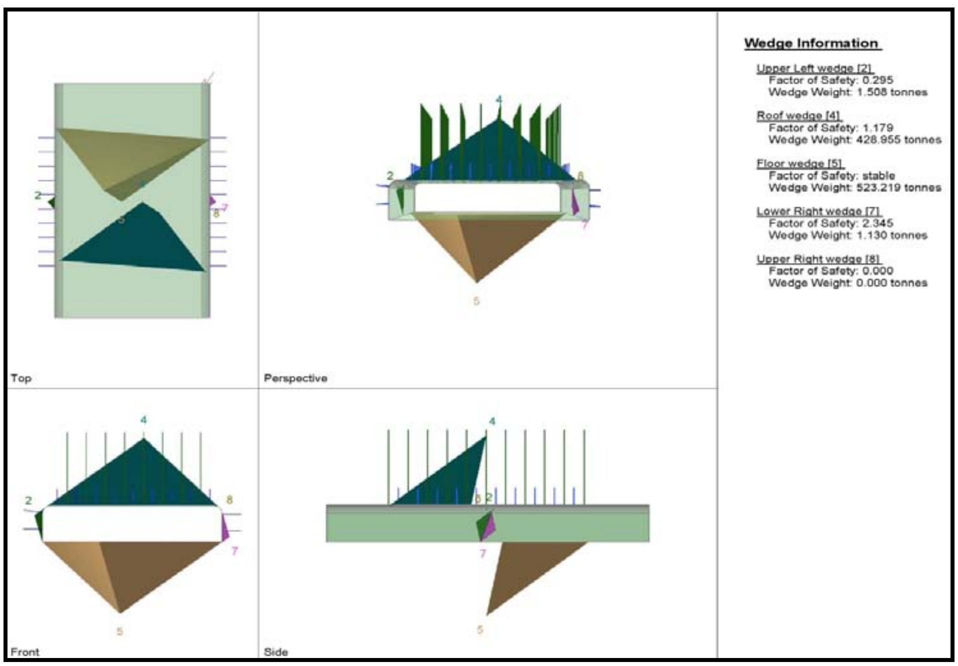
Sill Support in Long-hole and Mechanized Cut-and-Fill Stopes

The drill drive will be driven in the centre of the stope at 5 m W advancing one drill and blast round ahead of the 6.0 m side wall slash. Sill support is a secondary support system supplementing the primary support system, which is required for long-hole stopes. The 17 m wide sill will require plain strand cable bolts of 8 m length spaced at 1.8 m x 1.8 m, plated and tensioned. This provides an FOS of approximately 1.2 for the roof wedge (Figure 18.11). Cable bolting must be performed once the advance reaches 20 m or less when poor ground conditions are encountered.

Supplementary support in MCF depends on the number of lifts per stope. The average total stope height for the MCF is 12 m with mining performed at 4 m high lifts. The recommended cable bolt support length for the MCF is 16 m. The cable bolt pattern is similar to LH stoping.

The cable bolting will be performed while the initial lift is being developed. The 16 m length cable bolt allows the following two lifts to be mined without additional cable bolting. This allows the final lift to be supported by 8 m length cable bolt. The cable bolts are design to yielding an FOS at approximately 1.2. Modified strand cable bolts are recommended for this application.

Figure 18.11 Sill Wedge Analysis (17 m Wide)



18.6.5 STOPE DIMENSIONING

Open stope dimensioning for the Mactung analysis was empirically estimated with the Modified Stability Graph (Potvin, 1988 and Nickson, 1992). The stability graph design procedure is based on the calculation of two factors: the Modified Stability Number (N'), representing the ability of the rock mass to stand up under given conditions, and the Shape Factor (S) or hydraulic radius, which accounts for the stope size.

MODIFIED STABILITY NUMBER

N' is based on Q with alterations to the active stress factor (Jw/SRF) on Q by setting the Jw/SRF factor to 1.0. This alteration yields the Modified Tunnelling Index (Q') where:

$$Q' = \left(\frac{RQD}{Jn}\right) x \left(\frac{Jr}{Ja}\right)$$

Therefore, the Q for this geotechnical investigation is equivalent to the Q'.

N' consisted of:

N' = Q' x A x B x C

Where:

Q' = Modified Tunneling Quality Index
A = Rock Stress Factor
B = Joint Orientation Adjustment Factor
C = Gravity Adjustment Factor

The shape factor, commonly known as the hydraulic radius, for the stope surface under consideration is computed as follows:

$$S = \frac{\text{Area (sq. m)}}{\text{Perimeter (m)}} = \frac{\text{w x h}}{2(\text{w} + \text{h})}$$

Where:

w = Width of Stope or Openings (m)
h = Height or Length of Stope (m)

The method proposed by the stability graph is based on Canadian case histories. It provides a preliminary guideline for open stope dimensioning. Modifications of its parameters are required to reflect the site conditions at Mactung and as detailed geotechnical information is available. Back analysis on the stope performance is essential to alter the parameters to suit the underground conditions.

Stability Graph Analysis

Table 18.14 shows stope dimensioning for the Mactung project. The modified stability number factors are presented in Table 18.14. The average dip of the orebody for long-hole mining is reported to be at 17° with the average stope panel dimension of 60 m in length and 20 to 40 m in height.

An estimated "A" factor of 1.0 is assumed if the strength of the intact rock is greater that the induced strength and where the hanging wall is at relaxation, in this case moderately dipping.

The "B" factor accounts for the joint orientation with respect to the stope surface. Estimation is made for the stope back and the walls based on the major critical joint set 1.

The gravity adjustment factor C is 2 for gravity fall and slabbing for the vertical hanging wall with an inclination of critical joint set 1 at 74° is 3.5 and for the stope back.

The rock mass tunnelling index for Mactung is good, with ground conditions for Q' at zones 2B and 2BL determined to be 82 to 98, respectively. The A, B, and C factors for variable stope dimension and length are presented in Table 18.14.

A stope with a strike length of 60 m, a height of 20-30 m, and a width of 17 m yields a hanging wall shape factor or hydraulic radius of 7.5 to 10 and 6.6 for the back; this equates to a stable stope (Table 18.14).

Table 18.14 Stability Numbers for Mactung

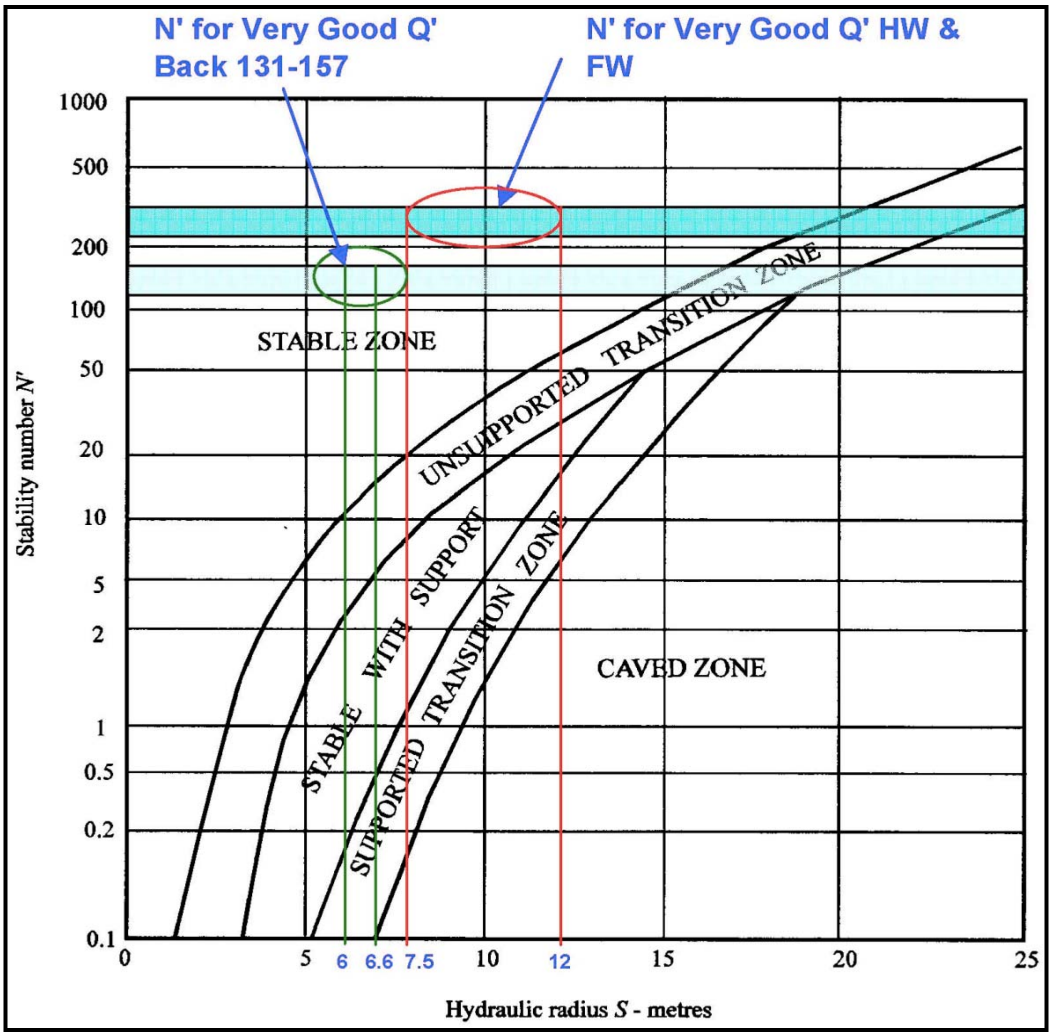
	Dimension (m)				Q'		A	B	C	N' (2B)	N' (2BL)	Stope
	Strike	Width	Height	S	2B	2BL						
HW and FW	60	-	20	7.5	82	98	1.0	0.9	3.5	258.3	308.7	**Stable**
	60	-	30	10.0								
	60	-	40	12.0								
	90	-	20	8.2								
	90	-	30	11.3								
Back	60	12	-	5.0	82	98	1.0	0.8	2	131.2	156.8	**Stable**
	60	15	-	6.0								
	60	17	-	6.6								

The report from Amax (May-October, 1973) confirmed that the XC4S test stope at the contact of B and C unit with excavation size of 9.1 m x 15.2 m (30 ft x 50 ft) with an average height of 2.8 m (9.1 ft) was very competent and required no rock bolting during development with no loosening of material from the stope back. One-third of the stope back was in hornfels (hanging wall) and the other two-thirds in skarn (ore). The contact was bolted afterward to prevent loosening of material.

Plotting these example values onto the stability graph provides a general guideline to the stability of the stope with respect to rock mass conditions and stope size.

Figure 18.12 shows a stope with hydraulic radius of 6-12. The stability of the stope can range from stable with support to stable, mainly depending on the quality of the ground conditions, stress factor, joint orientations, and orientation of the stope.

Figure 18.12 Stability Graph Analysis for Mactung

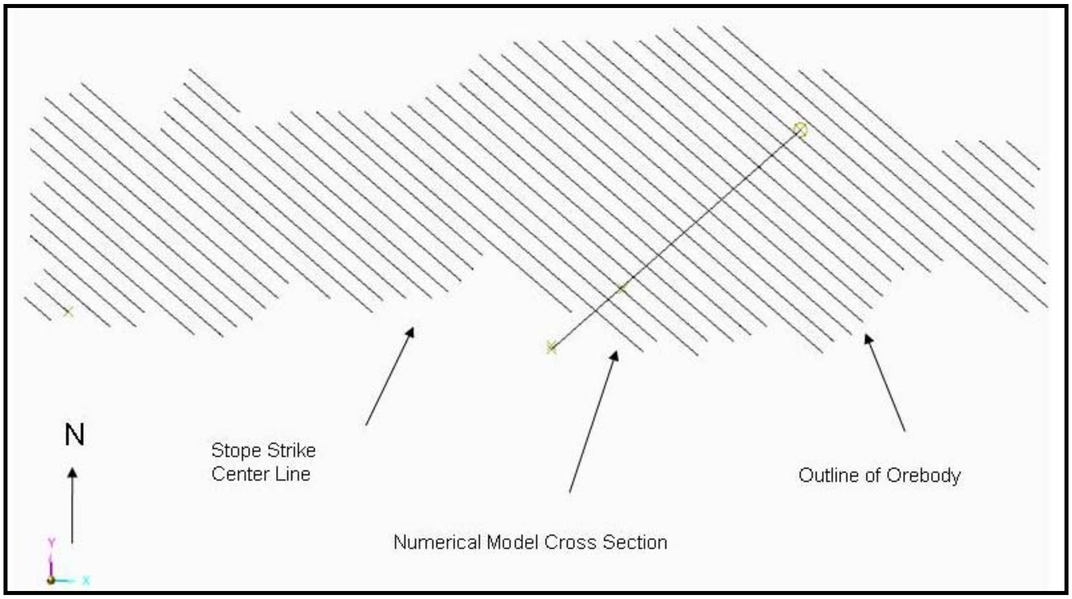


18.6.6 NUMERICAL MODELLING

A two-dimensional numerical modelling with Phase 2 was performed to analyze stope to pillar dimensioning and stress conditions around excavations for Mactung.

Mactung numerical models were built based on the mine layout plans prepared from Surpac and AutoCAD (Figure 18.13). The pillar-to-stope width ratio for this analysis is 0.235 or 4 m W pillar to 17 m W stope. This arrangement provides higher recovery compared to the initial design.

Figure 18.13 Numerical Model Cross Section



DEFORMATION AND STRENGTH PARAMETERS

The deformation and strength parameters (Table 18.15) for numerical modelling were based on rock mechanics test results presented in the report entitled “Room and Pillar Analysis for the Mactung Project” (D. Steward, 1980). The ore properties were assumed to represent hanging walls and footwalls since most of the mining stopes are located in the ore (2B zone). The assumed unconsolidated backfill parameters are shown in Table 18.16.

The backfill at Mactung is unconsolidated where initial loading occurs as body force or self weight and remains distressed. Walls on the open stopes are estimated to be relaxed before backfilling.

Hoek Brown failure criterion was used for modelling and the estimated intact parameters are presented in Table 18.17.

Table 18.15 Deformation and Strength Parameters for Ore Bearing Rock (D.Steward 1980)

Unit	Weighted UCS (MPa)	Weighted Young's Modulus (GPa)	Weighted Poisson's Ratio	Weighted Tensile Strength (MPa)	Weighted Cohesion (MPa)	Weighted Friction Angle (°)	Weighted SG
Combined Skarn Unit (Ore)	247	82	0.2	14.8	43.2	53.2	3.14

Table 18.16 Deformation and Strength Parameters for Backfill

	Young's Modulus (GPa)	Poisson's Ratio	Tensile Strength (MPa)	Cohesion (MPa)	Friction Angle (°)	Density (kg/m^3)
Backfill	0.5	0.2	0.1	0.1	30	2000

Table 18.17 Hoek-Brown Criterion Intact Rock Input Parameters for Modelling

GSI	UCS of Intact Rock (MPa)	m_i*	D Disturbance Factor	E_i * (GPa)
80	247	19	0.8	82

*Note: m_i – material
E_i – intact modulus

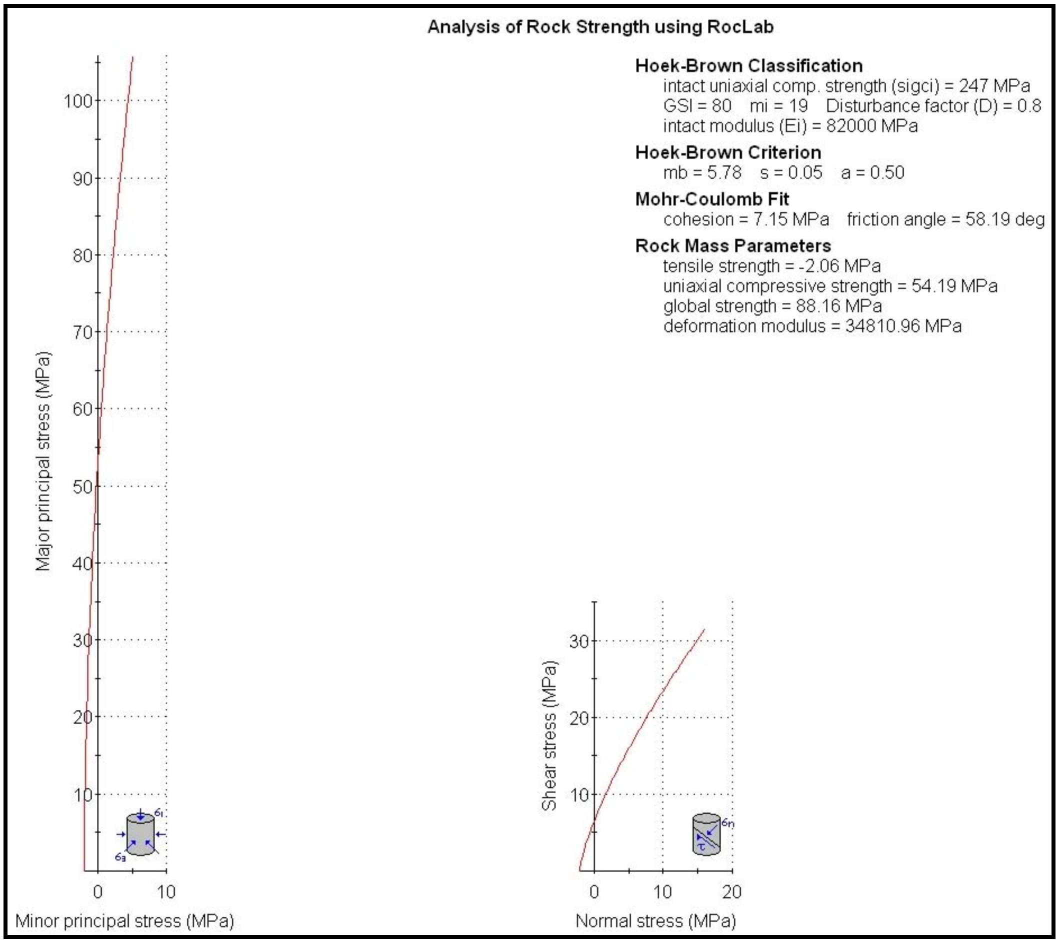
Table 18.18 Hoek-Brown Criterion and Rockmass Parameters

	m_b (peak)	m_b (residual)	s (peak)	s (residual)	Dilation
Ore	5.78	1	0.048	0.001	0

*Note: mb – value of constant "m" for the rock mass
s – constant which depend on the characteristics of the rock mass.

Figure 18.14 shows the analysis for rock strength made with Rocscience Roclab software to determine the rockmass, "mb" and "s" parameters. These values are presented in Table 18.18.

Figure 18.14 Hoek-Brown Criterion and Rockmass Parameters



IN-SITU STRESSES

Cantung in-situ stress measurement results (AMEC, 2000) from overcoring are utilized for Mactung numerical modelling. The Cantung site is about 160 km south of Mactung project, so it is assumed that the in-situ stress measurements are similar at the current level of study. Detail stress measurement is recommended for Mactung during the advance geotechnical analysis.

The principal stress values at 2900 ft (884 m) depth are presented in Table 18.19.

Table 18.19 The Principal Stress Values for (884 m) 2900 ft Depth (Cantung Mine)

Principal Stress	Azimuth	Plunge	Magnitude (MPa)	Gradient
Major	S87W	-48E	64	5.9*overburden stress
Intermediate	N01E	+2N	45	4.2*overburden stress
Minor	S89E	+37E	26	2.4*overburden stress

The major, intermediate, and minor principal stresses at overburden depth of 300 m are 55.5 MPa, 39.5 MPa, and 22.5 MPa, respectively, based on stress gradients listed in Table 18.19.

For the purpose of 2D numerical modelling, the major principal stress is along ore zone for stopes looking from the southeast where the horizontal stress is assumed to be 55.5 MPa, with overburden (minor principal stress) at 22.5 MPa, and 39.5 MPa out-of-plane stress.

NUMERICAL ANALYSIS

Figure 18.15 and Figure 18.16 outline the Mactung model geometry indicating stope and pillar array and mining sequences.

Figure 18.15 Base Model Geometry

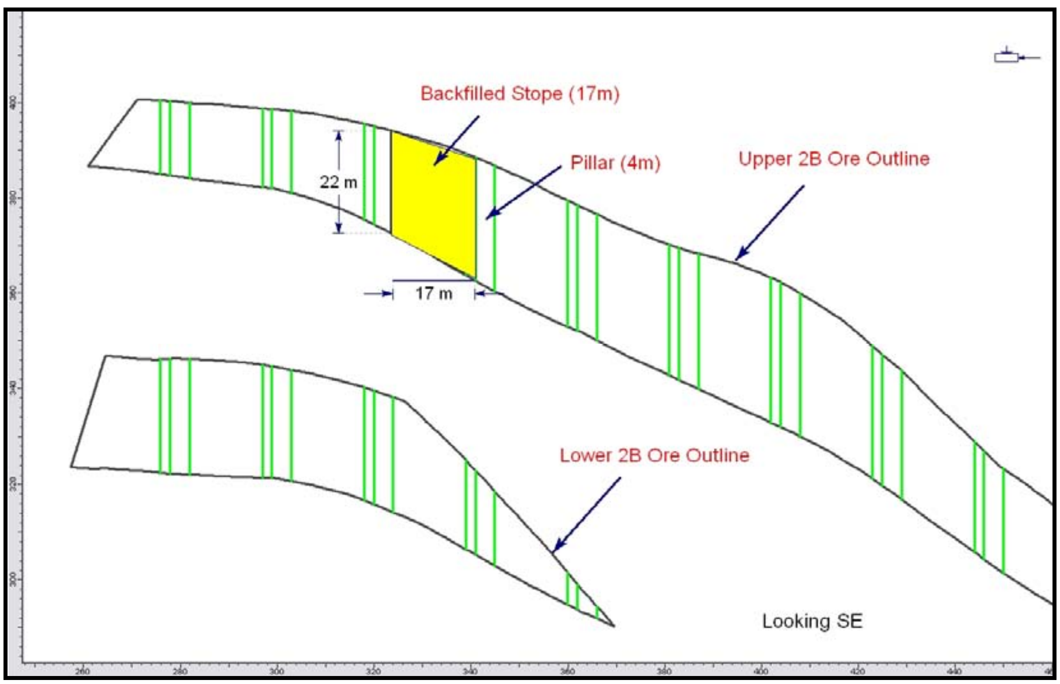
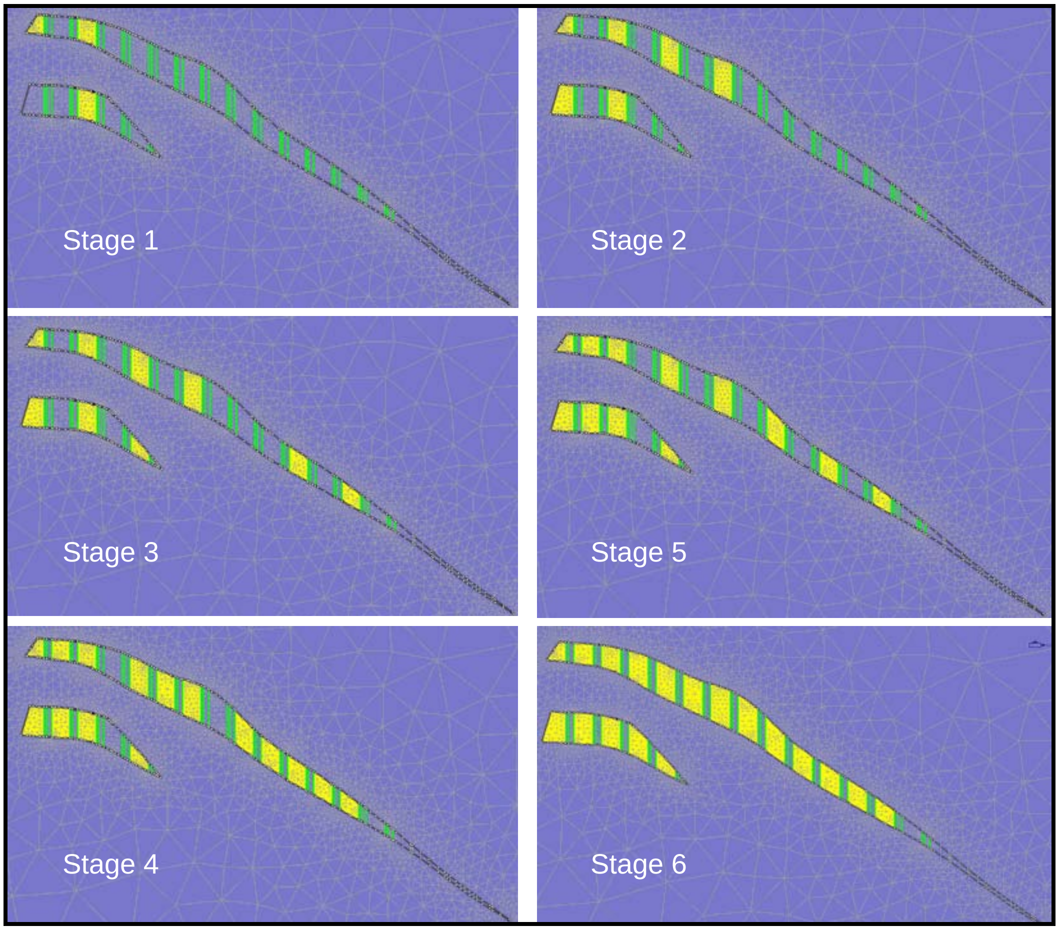


Figure 18.16 Mining Sequence Schematic



Mining Sequence

The 2D numerical analysis models the stope as being fully excavated along 200 m of ore strike length. At the current stage, the backfill material is considered to be unconsolidated backfill consisting of classified and filtered mill tailings and development waste rock. During actual mining, the 200 m strike length panel will not be fully extracted in a stage (approximately a year) and will be partially mined and filled.

The initial three stages simulate primary stope extraction and, at the fourth stage, two secondary and one primary stopes are mined. This sequence represents a typical cross section for the mine sequence.

This model provides initial assessment and understanding of mining induced stresses and the analysis is limited for complex mining sequences with array of ribs and regional pillars. To better analyze the mining sequence during detailed design, 3D numerical modelling is required.

ANALYSIS RESULTS

Tensile or relaxation zones and shear failures on the stopes' back and side walls are typical of open stoping and expected during mining stages 1 to 5 (Figure 18.17). Dilution is expected on stope walls having contact with waste. The supplementary support or sill support with cable bolts on full sill span will address issue pertaining to tensile or back relaxation.

The 4 m W pillar acts as a "skin pillar" or barrier preventing unconsolidated backfill from diluting into adjacent mining stope. Part of the 4 m pillar yields in tension and the rest of the pillar will be relaxed and confined by the backfill based on the analysis.

The backfill does not carry any load because of unconsolidated characteristics where backfill placement is not "tight" to stope back and the stope walls are in relaxation during deposition.

Secondary stopes can be observed to be under mid to low stress conditions in stages 2 and 3 on the Upper and Lower 2B zones. During stages 4 and 5, corners of the secondary stopes will experience higher stresses on the southern limb. This may cause strain bursting at the development and borehole squeezing may be expected.

Figure 18.18 outlines the extent of the yielding zone (13.3 m) where wedge formation or unravelling of the back may occur. Supplementary sill support, with cable bolts at 8.0 m length by 1.8 m by 1.8 m (Figure 18.11) based on an unwedge analysis, yields an FOS of 1.2, which is stable.

The model indicates that stoping with 17 m W stopes and with a 4 m W pillar is feasible with a pillar safety factor of 1.0 (Figure 18.19).

Figure 18.17 Mining Sequence

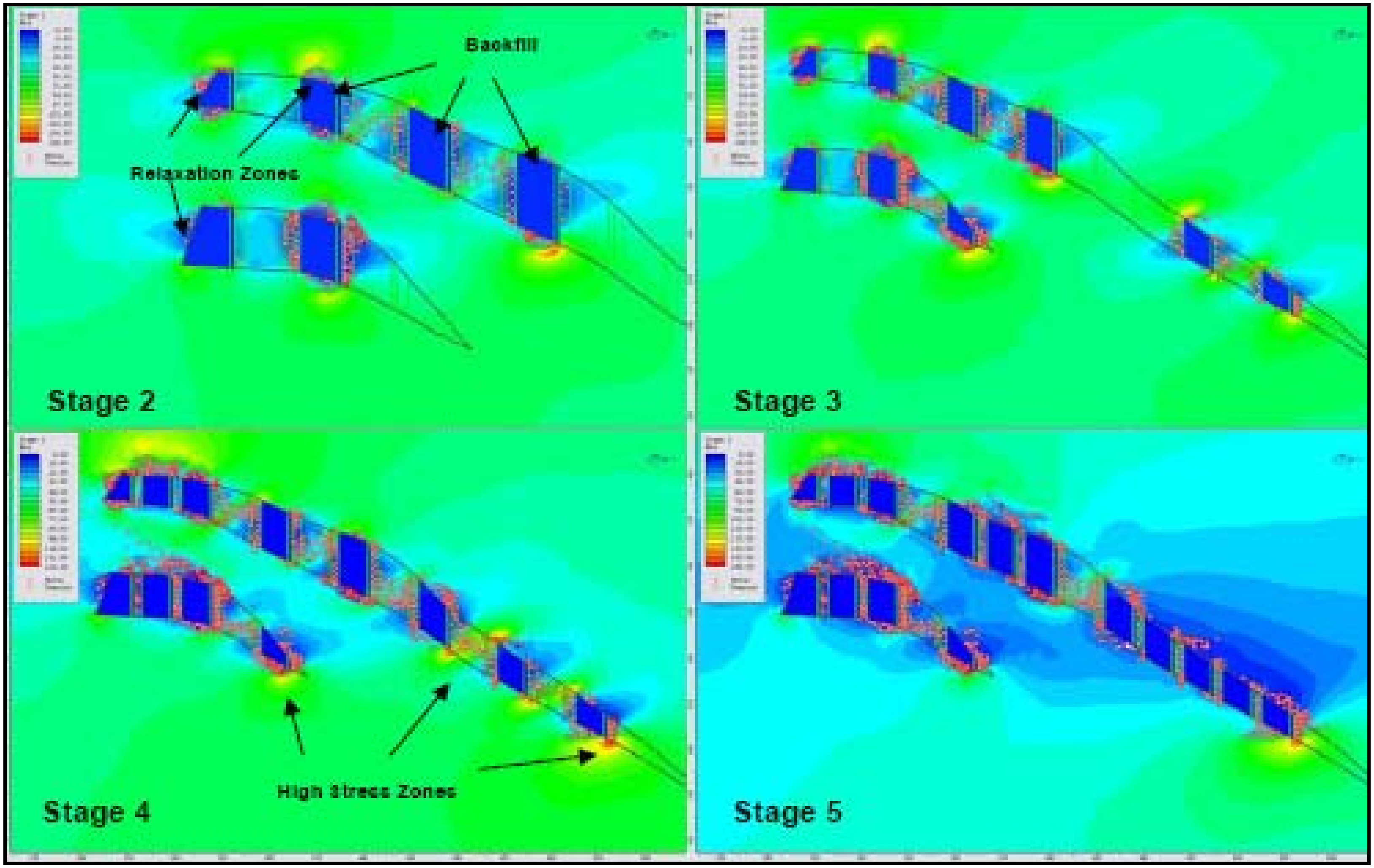


Figure 18.18 Tension Zones

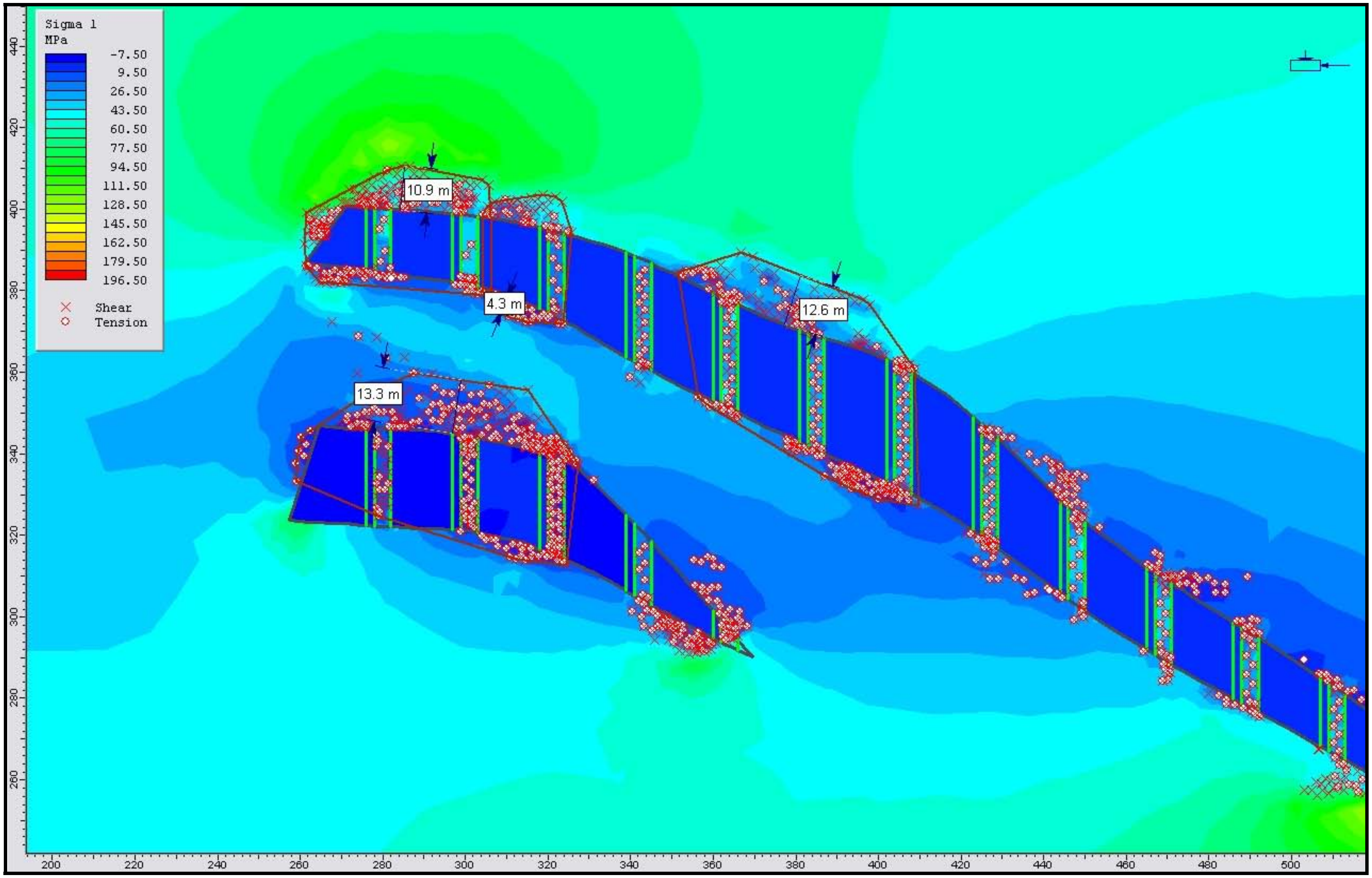
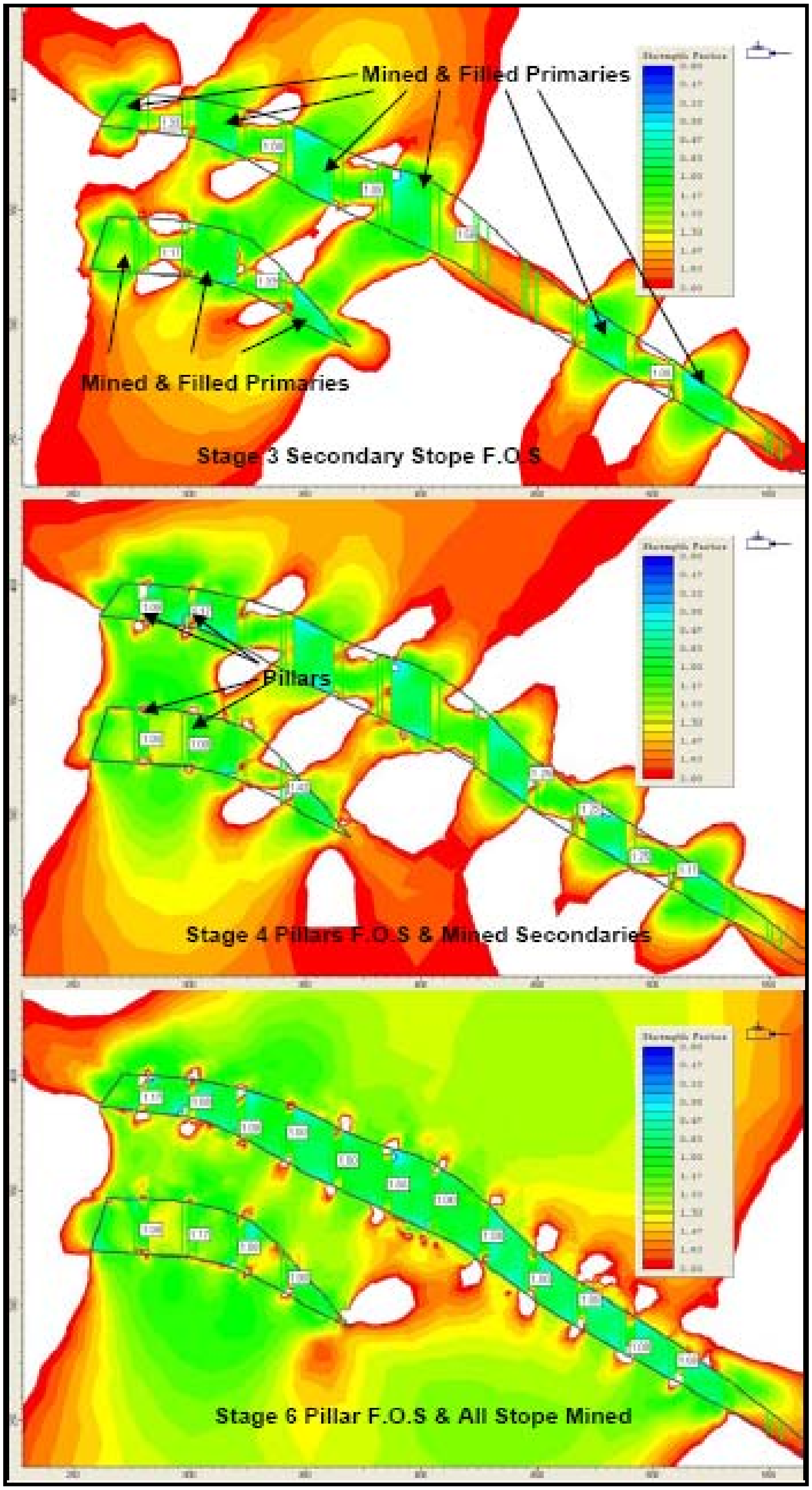


Figure 18.19 Factors of Safety



18.6.7 BACKFILL

Mactung's backfill material consists of classified, filtered, and unconsolidated mine tailings, and underground development waste rock. Wardrop recommends backfill of similar particle size distribution to hydraulic or slurry backfill.

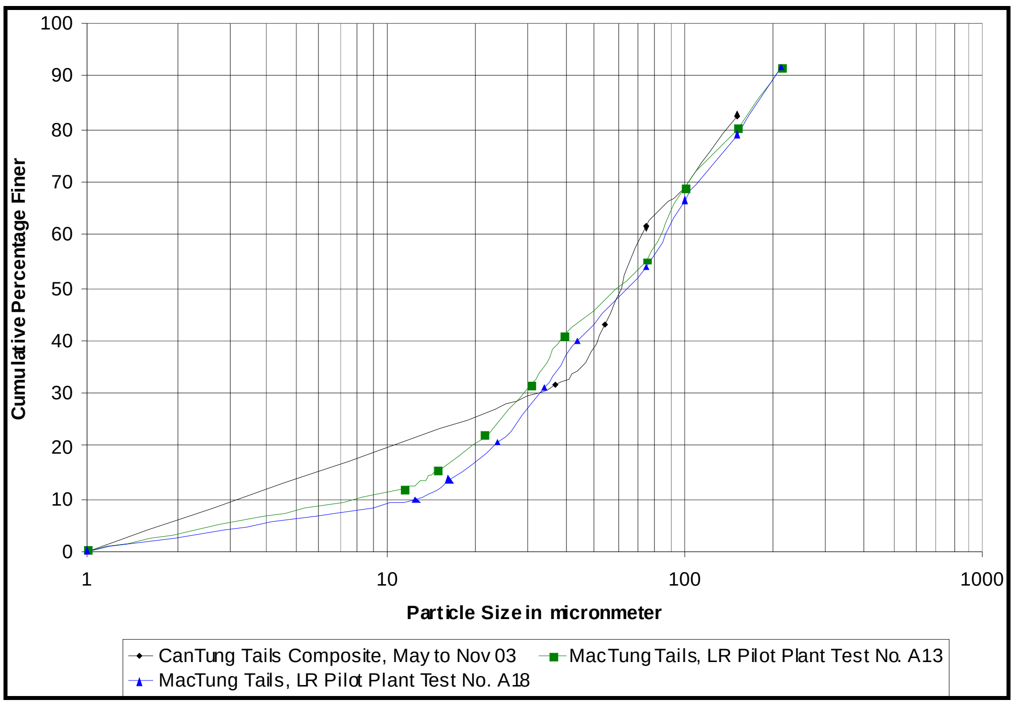
Hydraulic backfill specifications suggested by Hassani & Archibald (1998) are:

- particle size distribution containing less than 20% (by weight) of minus 10 µm
- percolation rate above 10 cm/h.

The presence of significant fines (sized less than or equal to 10 µm) reduces the effectiveness of stope drainage and may result in liquefaction. Tailings are typically de-slimed to reduce fines to less than 20% by weight for this application.

The particle size distribution for Cantung tailings, reported by Falcon test work, and Mactung tailings size analyses, by Lakefield Research (1985), indicates that less than 20% and 15% of the size distribution, respectively, is minus10 µm (Figure 18.20) based on extrapolation from the available data.

Figure 18.20 Particle Size Analyses for Cantung and Mactung Tailings Composite



The particle size distribution for Mactung's backfill is recommended to have at least similar specifications to the hydraulic fill even though the mine is considered to be in marginal permafrost where no water saturation is anticipated. This is to ensure that if there is a presence of water underground by external sources, it allows the unconsolidated backfill to drain. This reduces possibilities of liquefaction induced by dynamic forces such as blast vibration, etc.

The tailings will be filtered to approximately 10-15% moisture content and transported underground as backfill material. Although the tailings are filtered, containment barriers or bulkheads are still recommended at draw points to seal off backfill material, to prevent it from entering into crosscuts and drifts and to provide drainage to the system.

The particle size distribution of Mactung tailings will be suitable as backfill material according to the available data.

Wardrop recommends tests to confirm the percolation rate of the mill tailings to ensure that the backfill drains at a percolation rate above 10 cm/h (Hassani & Archibald, 1998).

MACTUNG BACKFILL REQUIREMENTS

The backfill material for Mactung is treated mill tailings and waste development rock. The estimated requirements are presented in Table 18.20.

Table 18.20 Development Waste and Classified Tailings as Backfill Material

Year	Development (t)	Stoping (t)	Development Waste (m^3)	Tailings Backfill (m^3)
1	66,493	604,850	28,278	115,629
2	66,299	650,828	22,151	177,297
3	38,458	690,875	17,029	194,691
4	15,021	717,578	11,567	208,336
5	38,394	715,358	18,257	200,965
6	24,409	681,803	8,373	200,560
7	59,177	670,408	16,878	188,570
8	63,709	707,931	21,017	195,930
9	60,666	649,830	18,862	180,279
10	13,791	697,044	9,431	204,179
11	0	726,452	0	222,650
Total	**446,415**	**7,513,047**	**172,292**	**2,130,093**

PERMAFROST AND FROZEN BACKFILL CONSIDERATIONS

Rock temperature measurements were performed at Mactung in September 1979; the measurements were revised in April 1982 by H. Heinicke and September 1982 by D. Stewart.

Rock Mass Temperature Measurement 1979-1980 by H. Heinicke

In the summer of 1979, nine Resistance Temperature Determination (RTD) devices and two thermometers were installed in crosscut No.1 north to measure rock mass heat response because it represents typical "weaker" rock. The 100 ft long (30.5 m) crosscut was curtained off and a Koehring K-Master Model B100D with a 90,000 BTU capacity heater was inserted under the curtain to heat the room.

RTDs were installed in drill holes at depths between 22 to 26" (558.8 to 660.4 mm) and held in place by wedges. One thermometer was hung beside a RTD; the other was taped to an RTD and inserted into a drill hole (H. Heinicke, September 1979, revised April 1982).

Heinicke experienced difficulty keeping the heater operating continuously and had no means of measuring the BTU generated by the heaters during the rock mass heat response measurement. He discovered that operating the heater for five hours raised the temperature above freezing; he also discovered that heat conduction through the rock is good (H. Heinicke, September 1979, revised April 1982).

In August and September of 1980, a 40 ft (12.2 m) remuck room was curtained off at the intersection of the main drift and heated by a space heater for 5 days while monitored by 18 RTDs. The remuck is located 500 ft (152.4 m) from the portal at 14 ft W x 10 ft H (4.3 m W x 3.0 m H) with muck pile at the face. The heater was a Koehring K-Master B1500 rated at 150,000 BTU.

Heinicke commented that the heat applied during the trial was excessive compared to diesel equipment and mine ventilation systems. However, the rapid heat transfer indicated that "it may be difficult to prevent a rind of thawed rock to develop around any mine openings exposed to diesel equipment" (H. Heinicke, September 1979, revised April 1982).

Thawing of frozen faults and fall of material were reported during this experiment.

Rock Mass Temperature Measurement 1982 by D. Stewart

D. Stewart performed rock temperature measurements in 62 previously-drilled holes from underground and surface in 1982. Temperatures in boreholes were measured by thermistor-type probes at 25-ft (7.6 m) intervals. Underground borehole measured temperatures ranged from -1.182°C to -2.086°C from vertical depth from surface at 163 to 448 ft (49.7 to 136.6 m).

Underground borehole measurements (47 holes) were utilized to derive linear and multiple regression expression. The ice point in the vicinity of temperature measurement locations is predicted to occur sub-parallel to the ground surface at depth of approximately 900 ft (274.4 m).

Stewart indicated that the Mactung property is located in a marginal permafrost environment.

FROZEN BACKFILL

Mactung backfill is composed of filtered mill tailings and underground development waste rock. At this stage of design the filtered mill tailings is assumed to be unconsolidated by freezing because of the marginal permafrost conditions. The use of filtered tailings as frozen backfill material is currently under investigation by mines in the Arctic.

Mactung backfill is therefore considered to be passive. “Skin pillars” are allocated on mined stopes to prevent unconsolidated fill from contaminating adjacent mining stopes. Drilling deviations are to be closely monitored for blast control to avoid pillar degradation during mining.

An example of the usage of frozen backfill is at the Polaris Mine, where broken shale with 10-13% moisture was left to freeze for two years before the adjacent pillar was mined, exposing the 80 m H x 14 m W frozen fill. Polaris permafrost conditions extend up to 500 m below surface with rock temperatures varying from -14°C near the surface to -3°C at the bottom of the mine. Freezing conditions exist throughout the mine workings where the mine is dry and the rock remains very competent when frozen (K. Dewing et al., 2006). Refrigerated intake air was delivered underground to counterbalance input heat from mining equipment, fresh water in the backfill, and electrical power for two months in the summer. Cemented rockfill was also utilized in areas with increased lithostatic loads induced by over strata settling.

Additional research is required to determine the mix design for treated mill tailings and development waste rock to obtain frozen fill. This design is also to be tested on site during operation.

18.6.8 GEOTECHNICAL RECOMMENDATIONS

The rock mass quality at Mactung was reported to be very good. The geotechnical data collection made in this investigation is based on RQD information provided by Mactung mainly on MS series drill holes; rock mass parameters were determined based on visual interpretation core photographs.

Initial ground condition estimates indicate good ground conditions, but detailed geotechnical drill information and data collection is required on the main infrastructure to make a more complete evaluation. Additional geotechnical drill holes are recommended to better define the rock mass quality surrounding major infrastructure and to determine the conditions of major faulting.

Although ground conditions are reported to be competent, ground support is required to ensure the safety of underground workers and equipment. Wardrop recommends installing of ground support to all permanent openings as the support guarantees safety of workers, equipment, and major infrastructure and to avoid future ground condition rehabilitation. The support requirement provided is based on the current understanding of Mactung ground conditions. The ground support systems design does not take into consideration the effect of permafrost. This is because the permafrost condition of the mine is reported to be marginal.

Ground movement monitoring is recommended so that the support system and its requirements can be optimized using the experience obtained during operations.

An open stope with dimensions of 20 to 40 m H with an opening of 17 m W x 60 m L is stable based on the stability graph analysis. These are non-man entry stopes. It is recommended to support the back (roof) of the 17 m stopes with cable bolts allowing the support wedge to be formed in the back. This enables production drilling and blasting activities, stoping, and backfill to be performed in a safe manner with reduced dilution from the back

The mining of the Upper and Lower 2B Zone is recommended to be performed from the bottom up (from lower to upper) when favourable higher grade is encountered in the lower panel. Middling in between the Upper and Lower 2B Zone is recommended to be greater than 15 m for mining from top to bottom. Wardrop suggests mining from the Lower to Upper Zone.

Preliminary numerical modelling indicated that 4 m skin pillar yields a factor safety greater than 1.0 and relaxation of the backs. Relaxation of the backs is anticipated thus cable bolting is required. Additional numerical modelling in 3D is recommended to define the extraction sequence and pillar dimension in detail. During the geotechnical drill investigation, it is recommended to test samples of cores for engineering properties to increase the accuracy of the numerical modelling.

The permafrost conditions at Mactung are reported to be marginal (D. Stewart, September 1982) and heat conduction through rock is good (H. Heinicke, September 1979, revised April 1982). Thus consolidated backfill induced by freezing is currently not considered. The heat generated by diesel power diamond drilling at the end of XC1N had been reported to start thawing the gouge zone (Amax, May-October 1973). Amax (1984) recommended that additional studies and test work be conducted to determine the extent of permafrost. Sprayed-on liners or foams are recommended to be applied on frozen faults to provide an insulation blanket and as a support component. On site investigations on the types of liners or foam are recommended to be performed on site.

A detailed investigation of the site's permafrost conditions will provide information essential to reinforce the mine design parameters. This will allow confirmation of dewatering requirements and provide additional fill strength if conditions are favourable to freezing.

The backfill is currently considered to be passive. The mill tailings particle size at Mactung confirms to the requirements as hydraulic fill material based on available data. Wardrop recommends tests to confirm the percolation rate of the mill tailings to ensure that the backfill drains at the percolation rate above 10 cm/h.

Underground testing of frozen, treated mill tailings are recommended once the mine is in operation to determine moisture content, the time required to freeze treated mill tailings, and frozen fill strength. The technology of frozen fill research is currently being performed, which will assist Mactung in defining the mix design, moisture content, and conditions. Refrigerated intake air may be required during the summer months to counterbalance heat generated from the equipment and surface air to maintain freezing conditions underground.

Detailed investigation on the tailings backfill will be conducted to have a better understanding of permafrost at Mactung. To be included in the investigation as earlier noted will be the percolation tests for drainage to meet or exceed the 10 cm/h rate.

18.7 Mine Ventilation

The proposed ventilation system for the Mactung underground mine will require an intake air volume capacity of 140 m^3/s. This intake volume requirement is based on the projected diesel equipment list for a 2,000 t/d operation. Table 18.22 provides the required air intake for the production mine.

Fresh air will be introduced to the mine via a 5.2 m wide x 4.6 m high adit, located at an elevation of 1895 m at the southeast end of the orebody. The adit will deliver fresh air through the stopes, MCF, and crusher workings.

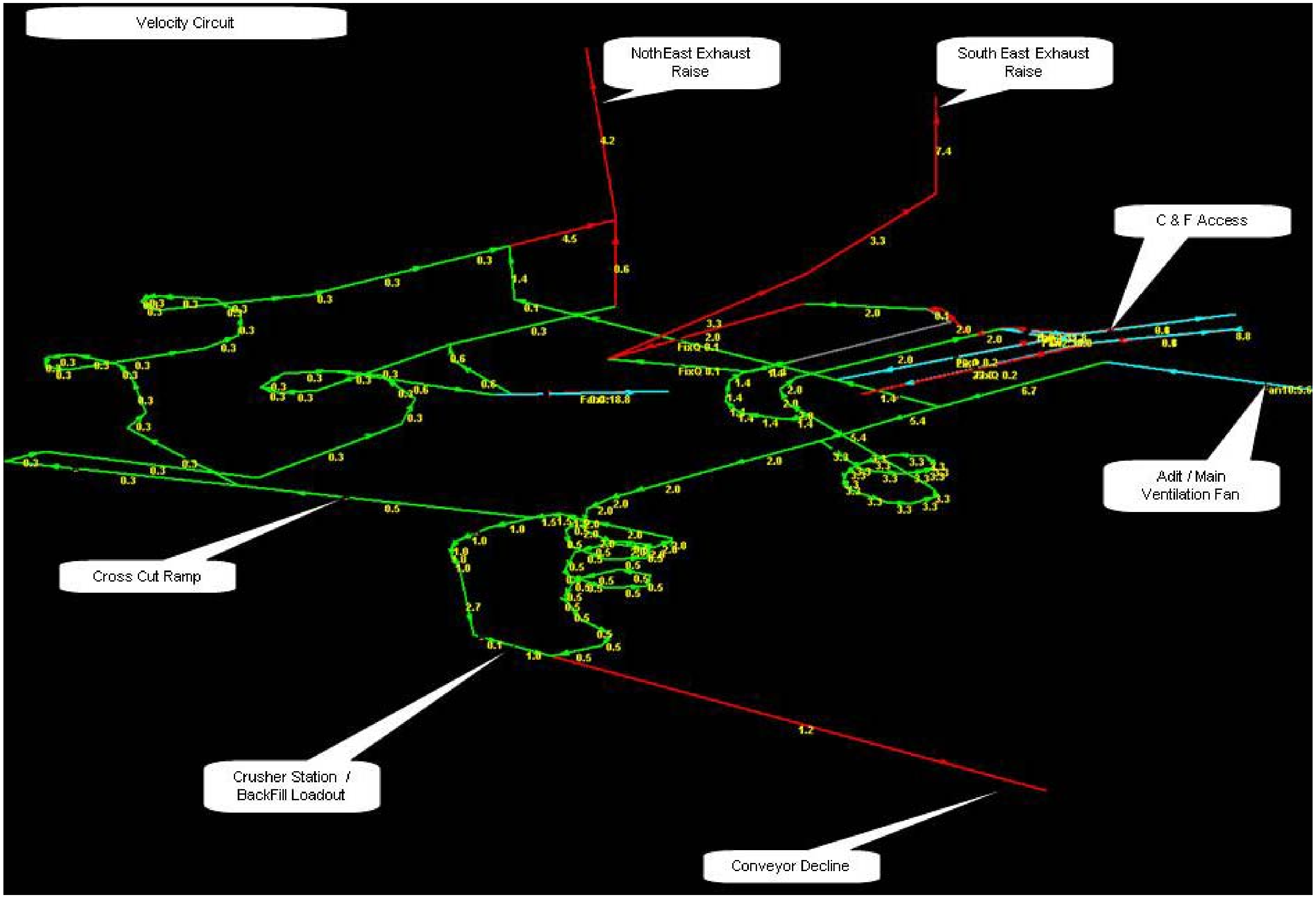
Exhaust air will exit the mine by one 3 m x 3 m and one 2 m x 2 m exhaust raise installed at the northeast end of the orebody and connected to surface. Approximately 30 m^3/s of air will be exhausted through the 5 m wide x 5 m high conveyor decline.

The mine ventilation system will operate during production as shown in Figure 18.21.

18.7.1 Primary Ventilation

Primary ventilation will be by one stand-alone mine axial fan installed on the adit level. The fan's adjustable blade pitch will provide the ability to alter air pressure and flow as mine development activities require. As the mine develops and deepens, operating pressures and air volumes to ventilate the mine will increase. Fan performance will be adjusted to meet the increased demand by changing fan speed and blade pitch. A second identical fan motor and blade will be available for backup.

Figure 18.21 Mine Ventilation Circuit for Production Years 1-5



DESIGN PARAMETERS

The ventilation system designed for the Mactung underground mine meets the Yukon Mine Safety Regulations for ventilation (section 15.61) and follows general practices employed throughout Canadian underground mines. The following is a summary of the ventilation limits.

Table 18.21 Design Ventilation Standards

Description	Unit	Value
Ventilation air requirements per kW	m^3/s	0.06
Ventilation air requirements per kW	cfm	127.13
Ventilation air requirements per hp	m^3/s	0.05
Ventilation air requirements per hp	cfm	100.00
Minimum air velocity (haulage)	m/s	0.25
Maximum air velocity (haulage)	m/s	6.00
Maximum air velocity (ventilation shaft)	m/s	12.00
Maximum air velocity (hoisting shaft)	m/s	8.00

The design basis for the ventilation system at Mactung is the air required to dilute and remove exhaust gases produced by underground diesel equipment. Equipment utilization factors were used to represent the diesel equipment in use at any time.

The ventilation system was modelled using Ventsim 3.9.2.f 2008. The input parameters include friction factor (k-factor), length, area, air quantity, and ventilation controls. The k-factors are standard for various types of drifts, raises, and openings. Ventilation controls such as bulkheads, regulators, and ventilation doors will be used to control air direction in the underground workings.

18.7.2 VENTILATION OF HEADINGS DURING DEVELOPMENT

Auxiliary fans will maintain a 22.4 m^3/s air flow in the development headings. This airflow rate is required to dilute and remove exhaust from the LHD and haul truck fleet. Using Atkinson’s equation for ventilation headings, a 75-hp fan with 0.9 m flexible ducting will be required for distances up to 100 m. For distances bewteen100 m and 600 m, twin ducting will be required.

Figure 18.22 and Figure 18.23 illustrate single duct tunnel location drawing and twin ducts application respectively.

Figure 18.22 Cross Section of Development Heading

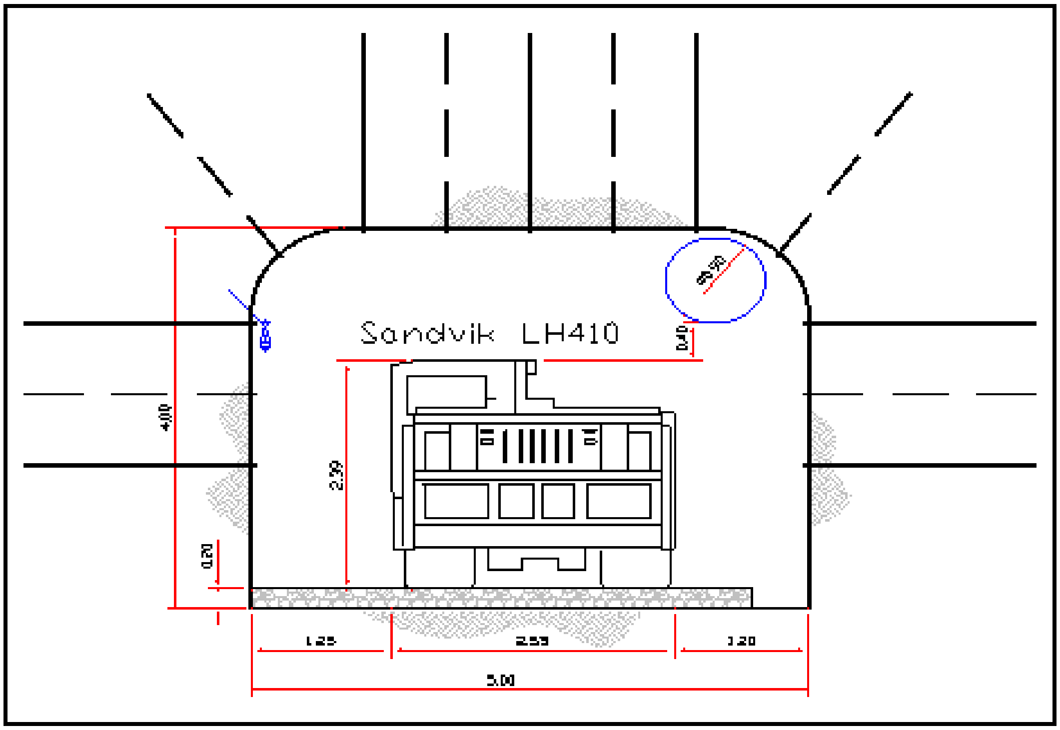
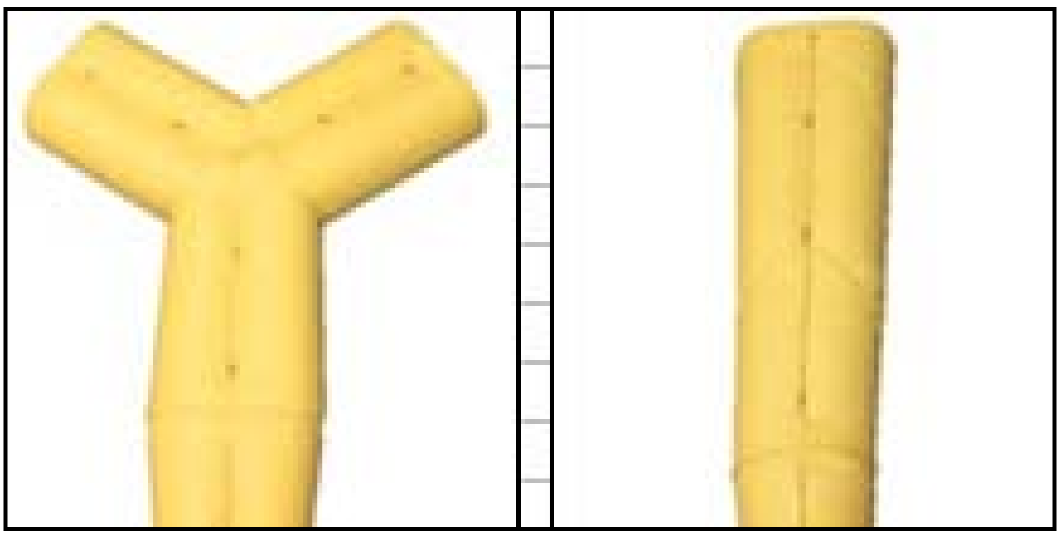


Figure 18.23 Twin Ducting



Production Ventilation Years 1-5

In the Upper 2B Zone long-hole operation, fresh air will enter through the south ramp and crosscuts. The airflow will leave the stopes through the crosscut that connect to the upper north ramp. This ramp connects to a ventilation raise that exhausts to surface.

In the Lower 2B Zone LH stoping operation, the fresh air will be drawn from the lower north ramp and pushed by a 75-hp fan through the ducting located in the mucking crosscut to the stope. Exhaust air will exit through the top crosscut to the lower north ramp, then out to surface through the exhaust raise system.

Development and production in the MCF section of the mine is designed so that the fresh air will be pushed using an auxiliary fan and flexible ducting to each of the cut and fill headings. The fresh air will exit through the drift and crosscuts back to the ramp and out to surface through the southeast ventilation raise.

Table 18.22 Mine Ventilation Requirements for Production Phase

Item	Equipment Detail	Units	Type	Qty	hp	Utilization	Total hp
1	Development Jumbo (2 boom)	ea.	Tamrock DD420-40C	2	149	10%	29.8
2	Long-hole DTH Drill	ea.	Tamrock DL310-7	2	99	10%	19.8
3	Secondary Breaking System	ea.	Maclean SB-6 Blockholer	1	150	10%	15
4	Rockbolter	ea.	Tamrock DS 310	2	200	20%	80
5	Exploration Drill	ea.	Diamec 252/1600U4PHC	1	61	0%	0
6	Development Load-Haul-Dump (5.0)	ea.	TORO LH410	1	295	60%	177
7	Production Load-Haul-Dump (5.0)	ea.	TORO LH410	2	295	75%	442.5
8	Haulage Truck (30 t)	ea.	EJC 30 SX	4	315	75%	945
9	Grader/D7 Dozer	ea.	GR 12 H	1	200	40%	80
10	ANFO Loader	ea.	Toyota HZJ79	1	125	50%	62.5
11	Mechanics Truck	ea.	Toyota HZJ79	1	125	50%	62.5
12	Supervisor Vehicle	ea.	Toyota HZJ79	2	125	20%	50
13	Electrician Vehicle – Scissor Lift	ea.	Toyota HZJ79	1	125	50%	62.5
14	Survey Vehicle	ea.	Toyota HZJ79	1	125	50%	62.5
15	Mine Engineering Vehicle	ea.	Toyota HZJ79	1	125	30%	37.5
16	Scissor Lift	ea.	Maclean SL-3	2	149	50%	149
17	Cassette Carrier	ea.	Maclean CS-3 Carrier	2	200	50%	200
18	Rockbreaker	ea.	Mobile (Caterpillar)	1	150	0%	0
Total hp							**2,475.6**
Total Utilization							**100%**
Ventilation Requirements 2.83 m^3/min/HP		m^3/min					7,005.948
20% Losses							1,401.1896
Total Ventilation Requirements		**m^3/min**					**8,407.1376**
Total Ventilation Requirements		**m^3/s**					**140.1**
Conversion Faction From m^3/ft^3		m^3/ft^3					0.0283
		cfm					29,7072

18.7.3 Fan Selection

The selection of the main intake fan is based on the maximum operating duty for specific conditions. For the main fan, Wardrop assumed an average fan efficiency of 75% and 1.36 kg/m^3 air density.

An axial flow fan with variable pitch control will be used for the permanent ventilation system. The simultaneous blade adjustment provides a wide range of fan settings within single speed; fan volume flow rate can also be adjusted to accommodate conditions.

Dust Control and Permafrost Conditions

Air ventilating the mine will be unheated to preserve the ground conditions provided by the marginal permafrost characteristics of the mine and its surrounding rock. All mine equipment will have heated cabins. Drilling dust will be controlled with air/water mist using water from hot water tank on the equipment. Dust collector systems will be installed in all drilling units.

18.8 Mine Production Schedule

The underground mine will operate on two ten-hour shifts per day, 365 days per year. Mine crews will rotate on a fly-in, three-week-on/three-week-off schedule. When weather conditions delay the arrival of incoming crews, the on-site crews will remain until relief crews arrive.

The mine will produce ore at the rate of 2,000 t/d from the Upper and Lower 2B ore zones. At this rate and at current market prices, sales contract, volume of reserves, and operating costs, the life of the underground mine will extend to 11 years.

The production schedule was developed using each stope line's tonnages and average grades, which were based on the reserve estimate calculation, the extraction sequence recommended by Wardrop's Geotechnical Study, and the preference to mine higher-grade ore during the initial years to reduce the payback period. There is an opportunity to enhance the annual average ore grade by selectively scheduling 60-m long individual stopes after calculating the ore grade for each one. There is also an opportunity to further enhance grade control from delineation drilling. Table 18.23 summarizes the production schedule for underground mining in the Upper and Lower 2B zones.

Table 18.23 Underground Mining Upper and Lower 2B Zones

	Year -1		Year 1		Year 2		Year 3		Year 4		Year 5	
	t	WO_3 (%)	t	WO_3 (%)	t	WO_3 (%)	t	WO_3 (%)	t	WO_3 (%)	t	WO_3 (%)
Plant Feed			730,000	1.447	730,000	1.314	730,000	1.222	730,000	1.175	730,000	1.109
Stockpiled Ore	74,247	1.464										
Development Ore			66,493	1.288	66,299	1.172	38,458	1.128	15,021	1.047	38,394	1.095
Stock ore/or Broken in Stopes	74,247	1.464	74,247	1.464	15,590	1.447	2,717	1.314	2,050	1.222	4,649	1.175
Ore from Stopes			604,850	1.463	650,828	1.326	690,875	1.227	717,578	1.177	715,358	1.109
Balance broken in stopes			15,590	1.447	2,717	1.314	2,050	1.222	4,649	1.175	28,401	1.109
Total Ore Mined			745,590	1.447	732,717	1.314	732,050	1.222	734,649	1.175	758,401	1.109

	Year 6		Year 7		Year 8		Year 9		Year 10		Year 11		Year 12	
	t	WO_3 (%)	T	WO_3 (%)	t	WO_3 (%)	t	WO_3 (%)	t	WO_3 (%)	t	WO_3 (%)	t	WO_3 (%)
Plant Feed	730,000	1.101	730,000	1.064	730,000	0.977	730,000	0.916	730,000	0.873	730,000	0.783	29,141	0.786
Stockpiled Ore														
Development Ore	24,409	1.052	59,177	0.945	63,709	0.902	60,666	0.827	13,791	0.814				
Stock ore/or Broken in Stopes	28,401	1.109	4,612	1.101	4,197	1.064	45,837	0.977	26,332	0.916	7,167	0.873		
Ore from Stopes	681,803	1.102	670,408	1.075	707,931	0.983	649,830	0.920	697,044	0.872	726,542	0.782	29,141	0.786
Balance broken in stopes	4,612	1.101	4,197	1.064	45,837	0.977	26,332	0.916	7,167	0.873	3,709	0.783		
Total Ore Mined	734,612	1.101	734,197	1.064	775,837	0.977	756,332	0.916	737,167	0.873	733,709	0.783		

table continues…

Table 18.25 (con't) Underground Mining Upper and Lower 2B Zones

	Total	
	t	WO_3 (%)
Plant Feed	**8,059,141**	**1.088**
Stockpiled Ore		
Development Ore	**446,415**	**1.042**
Stock ore/or Broken in Stopes	**74,247**	**1.464**
Ore from Stopes	**7,538,479**	**1.085**
Balance broken in stopes	**116,123**	**0.783**
Total Ore Mined	**8,175,264**	**1.089**

Notes:

- SG used for ore is 3.146 t/m^3 and 2.9 t/m^3 for waste.

- Scott Wilson RPA has based the SG value on densities established by the underground bulk sample taken from 2B zone in 1973.

18.9 WASTE MATERIAL MANAGEMENT

During the pre-production period, all waste material excavated from mine development will be stored in two rock dumps located on surface: one close to the adit portal (elevation 1895 m), and one close to the conveyor decline portal.

During the production phase, waste material excavated from ongoing mine development will be used to backfill mined out stopes, with priority given to mechanized MCF stopes used to circulate equipment in the backfilled areas.

Wardrop estimates that 95% of the mine-out stopes volume will be backfilled. Two materials types – mine waste and mill tailings – will be used in various ratios as fill. Wardrop estimates that 194,000 m^3 of fill material will be required each year. The geotechnical analysis has not considered the option of freezing backfill. Remote-controlled bulldozers will spread, level, and push the backfill material against the pillar walls and the back of the LH stopes. The mill tailings will be transported by two conveyor belts down to the underground backfill loading station. One conveyor will be installed beside the crushed ore conveyor in the 5 m x 5 m conveyor decline. The other conveyor will be transversal to the decline dumping on pile on the floor of the loading station (Figure 18.3).

An LHD unit will load the backfill tailings into a returning ore truck and transported a distance of approximately of 1 km up to the backfilling stope. The backfill material will be dumped into a dumping bay in the top crosscut of the stope, transported by another LHD unit up to the sill of the backfilling area, and pushed inside the stope with a remote-controlled bulldozer. When the backfill volume reaches the sill level, the bulldozer will push and level the backfill inside the backfilling stope, finally perch the material against the back of the stope. Due to the dip of the hanging wall contact, the backfilling stope can be reached from the drill sill of the next stope, pushing material through 4 m x 4 m windows in the rib pillars (Figure 18.4).

18.9.1 BACKFILL REQUIREMENTS

Mactung's backfill material will consist of filtered and unconsolidated mill tailings, and underground development waste rock. The backfill material requirement is presented in Table 18.20.

The tailings will be filtered to approximately 10-15% moisture content and transported underground as backfill material. Although the tailings are filtered, containment barriers or bulkheads at draw points are nonetheless recommended to provide drainage to the system and to prevent backfill material from entering into crosscuts and drifts.

18.10 WASTE STORAGE AREAS

Waste rock extracted during the pre-production phase of mine development will be stored in waste rock dumps on surface. To minimize trucking cycle time, the dumps will be located approximately 500 m from the portals. Wardrop estimates that a total volume of 124,000 m^3 of waste rock will be extracted and delivered to surface during this phase.

All soil and vegetation in designated rock dump areas will be recovered and a drainage system will be installed to collect run-off water into settling sumps and discharge into the natural drainage system.

The rock dumps are designed to dump the lower-level base bench first, and then benches on top of it. To control the long-range stability of the dump, benches will be up to 10 m with a 45° maximum effective slope angle. A portion of all the waste rock can be used for road construction, rock fill areas, and backfill material inside the mine.

Wardrop estimates that 92,000 m^3 will be stored in the rock dump close to the ventilation adit portal, and 32,000 m^3 in the rock dump close to the conveyor decline portal. Interference with the crusher and conveyor belt installations in the conveyor decline will not permit extraction of waste after the first year of pre-production development.

18.11 MINE EQUIPMENT FLEET

The main criteria for mining equipment selection are availability, dimensions, and capacity.

18.11.1 *EQUIPMENT AVAILABILITY*

An increase in mining activity has led to a worldwide mine equipment and components shortage. Many equipment items require purchase orders several months in advance; therefore, Wardrop recommends that NATC develop and operate the mine using the company's own personnel and equipment. Due to the remote location of the project and the higher productivity required in the development and operation of the mine, only new mine equipment was considered.

18.11.2 *DIMENSIONS AND CAPACITY*

The 30-t capacity rear dump truck was determined to be the best suited unit for mine works dimensions and 2,000 t daily production capacity based on design dimensions and configurations of the mine, ramps, drifts, drilling sills, and the access through the conveyor decline. The compatible loading units for these trucks are 10.0 t LHDs.

Truck loading and dumping operations require a loading/dumping station with back slashing to approximately 6.5 m high.

The two-boom drilling jumbos selected increase the productivity and offer good coverage of the drifting sections.

Rock bolting and cable bolting rigs have the capacity and dimensions appropriate for the vertical section of the working areas and the length of rock and cable bolts.

Mucking in LH stopes requires the use of remote control LHD units to operate in non-man entry areas.

The long-hole drill unit selected has a drilling capability of up to 36 m down and over 25 m up, with drilling towers/booms fitting in the 4 m drilling sills.

A list of major mine equipment operating parameters and productivities are shown in Table 18.3.

18.11.3 LOADING EQUIPMENT

Table 18.24 shows the loading productivities and calculated requirements for the Sandvik LH410 LHD unit.

Table 18.24 LHD Loading Productivities – Sandvik LH410

LHD Specifications	Unit	Ore Quantity	Development Waste Quantity	Backfill Quantity
Rated Payload	t	10.0	10.0	10.0
Selected Bucket Size	m^3	5.4	5.4	5.4
LHD Actual Payload	t/bucket	9.0	8.6	7.8
LHD cycles Required for Daily Production	trip	222	23	109
Truck Loading Production Estimate				
Mucking Productivity	t/h	80.4	148.5	135.1
LHD Requirements				
Average Trips per Shift	trips/shift	74	143	143
Average Mucking Production per Shift	t/shift	667	1,227	1,116
Average LHD Mucking Production per Day	t/d	1,333	2,453	2,232
LHD Availability	%	80	80	80
Number of LHD Required per Day	**ea.**	**2**	**1**	**1**
LHD Utilization	%	70	11	53

Note: Two LHD units are required to work on ore production and one LHD unit is required to work on development and backfill loading.

18.11.4 HAULAGE EQUIPMENT

Table 18.25 shows the production and productivities for the Tamrock EJC 30 SX 30-tonne truck.

Table 18.25 Truck Productivities and Requirements

Rear Dump Truck Specifications	Unit	Quantity Ore	Development Waste Quantity	Backfill Quantity
Truck Box Size Selection				
LHD Buckets per Truck	ea.	3	3	3
Selected Box Capacity	m^3	14.0	14.0	14.0
Tamrock EJC 30 SX (30-tonne)				
Truck Rated Payload	t	30.0	30.0	30.0
Cycle Time Estimate				
Truck Actual Capacity	t	27.0	25.7	23.4
Truck trips Req'd for Daily Production	trip	74	8	36
TRUCKING PRODUCTION ESTIMATE				
Truck on Ore Production				
Truck Trips per Shift	trip	13		
Ore Trucking Productivity (on ore production only)	t/h	41.2		
Truck on Ore Production and Backfill				
Truck Trips per Shift	trip	8		
Backfill Trucking Productivity (on ore production and Backfill)	t/h	22.0		
Ore Trucking Productivity (on ore production and Backfill)	t/h	25.4		
Truck on Development Waste				
Truck Trips per Shift	trip	20		
Trucking Productivity	t/h	62.0		
Truck Requirements				
Truck Availability	%	80		
Number of Trucks Req'd per Day	**ea.**	**5**		
Truck Utilization	%	80		

18.11.5 MINE DEVELOPMENT EQUIPMENT

Table 18.26 lists the mine equipment required for pre-production development.

Table 18.26 Pre-production Development Mine Equipment

Equipment	Model	No. Req'd
Drilling Equipment		
Development Jumbo (2 boom) Unit	Tamrock DD420-40C	2
Components - 16 ft feed	Tamrock	2
Cable Bolter Unit	Misc	1
Rock Bolter Unit	Tamrock DS 310	2
Jackleg Unit	PHQ250JHML	4
Stoper Unit	PHQ250SMCSR	4
Exploration Drill Unit	Diamec 252/1600U4PHC	1
Loading & Hauling Equipment		
Development Load-Haul-Dump Unit 10.0 t	TORO LH410	1
Production Load-Haul-Dump Unit, 10.0 t	TORO LH410	2
Haulage Truck Unit, 30 t	EJC 30 SX	4
Service Vehicles		
Grader	GR 12 H	1
ANFO Loader	Toyota HZJ79	1
Mechanics Truck	Toyota HZJ79	1
Supervisor Vehicle	Toyota HZJ79	2
Electrician Vehicle - Scissor Lift	Toyota HZJ79	1
Survey Vehicle	Toyota HZJ79	1
Mine Engineering Vehicle	Toyota HZJ79	1
Scissor Lift	Maclean SL-3	1
Cassette Carrier	Maclean CS-3 Carrier	2
- Flat Deck Cassette	Maclean CS-3 Flat Deck	1
- Hiab 095 Boom/Deck Cassette	Maclean CS-3	1
- Fuel/Lube Cassette	Maclean CS-3	1
- Personnel Cassette	Maclean CS-3	2

18.11.6 MINE PRODUCTION EQUIPMENT

Table 18.27 shows the production equipment, which includes all pre-production equipment as well as additional loading and trucking units, and equipment used for ground support and LH drilling.

Table 18.27 Equipment List for Full Production

Equipment	Model	No. Req'd
Drilling Equipment		
Development Jumbo (2 boom) Unit	Tamrock DD420-40C	2
Cable Bolter Unit	Misc	1
Rock Bolter Unit	Tamrock DS 310	2
Jackleg Unit	PHQ250JHML	4
Stoper Unit	PHQ250SMCSR	4
Rockbolter	Tamrock DS 310	2
Exploration Drill Unit	Diamec 252/1600U4PHC	1
Longhole DTH Drill	Tamrock DL310-7	2
Secondary Breaking System	Maclean SB-6 Blockholer	1
Loading & Hauling Equipment		
Development Load-Haul-Dump Unit, 10.0 t	TORO LH410	1
Production Load-Haul-Dump Unit, 10.0 t	TORO LH410	3
Haulage Truck Unit, 30 t	EJC 30 SX	5
Service Vehicles		
Grader	GR 12 H	1
ANFO Loader	Toyota HZJ79	1
Mechanics Truck	Toyota HZJ79	1
Supervisor Vehicle	Toyota HZJ79	2
Electrician Vehicle - Scissor Lift	Toyota HZJ79	1
Survey Vehicle	Toyota HZJ79	1
Mine Engineering Vehicle	Toyota HZJ79	1
Scissor Lift	Maclean SL-3	1
Cassette Carrier	Maclean CS-3 Carrier	2
- Flat Deck Cassette	Maclean CS-3 Flat Deck	1
- Hiab 095 Boom/Deck Cassette	Maclean CS-3	1
- Fuel/Lube Cassette	Maclean CS-3	1
- Personnel Cassette	Maclean CS-3	2

18.12 NET PRODUCTIVE OPERATING TIMES

Wardrop applied an 83.3% hourly efficiency (50 minutes per working hour) plus 1.75 hours of non-productive time per shift to arrive at an effective work time per shift of 6.90 hours per 10-hour shift.

18.13 MINE CAPITAL COSTS

Wardrop estimates pre-production capital cost to be $39.6 million. Mine equipment accounts for $20.9 million of this pre-production capital cost, and mine development accounts for $18.7 million. Pre-production capital cost details are provided in Table 18.28. Underground crusher and conveyor belt systems are not included. All figures are expressed in CDN$ in Q3 2008.

Table 18.28 Underground Mining Pre-production Capital Cost Summary

Underground Mining	Cost (CDN$)
Mobile Equipment	15,722,172
Underground Stationary Equipment	5,199,089
Underground Pre-production Development	18,667,660
Total Underground Mining Pre-production Capital Cost	**39,588,922**

Assumptions

Exchange Rate (US$/CDN$) = 0.88

All Costs are in CDN$

No Inflation/Escalation

No Freight Included

Underground excavations not included

18.14 Mine Operating Costs

The operating costs for underground mining at Mactung were estimated on the basis of a 2,000 t/d rate of production, 365 days per year, and 20 hours of operating time per day, split into 2 shifts.

The following input parameters were used for the operating cost estimate.

Table 18.29 Input Parameters

Operating Factors	Unit	Quantity
Underground production		
Mine days	days/month	30
Mine days	days/year	365
Nominal mining rate	t/day	2,000
Average mining rate	t/yr	730,000
Rock characteristics		
In situ density ore	t/m3	3.14
In situ density waste	t/m3	2.99
Swell factor	%	60%
Loose density ore	t/m3	1.96
Loose density waste	t/m3	1.87
Ore average thickness	M	26.00
Dip	deg.	18
Shift data		

Working days a week	ea.	7
Shifts per day	ea.	2
Shift length	Hrs	10
Shift change	Hrs	0.75
Lunch / break / delays	Hrs	0.75
Equip inspection	Hrs	0.25
Subtotal non productive	Hrs	1.75
Usable time/shift	Hrs	8.25
Shift efficiency	%	82.5%
Usable minutes/hr	Min	50
Hour efficiency (50 min in hr)	%	83.3%
Effective work time per shift	Hrs	6.9

All costs are expressed in Canadian dollars, with no allowance for escalation or interest during construction. These costs were estimated from first principles for each cost category such as development, production, haulage, maintenance, mine services, and labour. Labour rates are assigned according to the Cantung Mine wage rates for 2008.

The mining employees at the Mactung underground operation are divided into the following categories: salaried personnel, operating and maintenance labour.

Table 18.30 Mine Management and Supervision Salaried Personnel

Staff Mine Operation		Base Salary $/yr	Overhead 28%	Total $/year
Mine Superintendent	1	113,000	31,640	145,000
Mine General Foreman	1	92,000	25,760	118,000
Mine Shiftboss	4	84,650	23,702	433,000
Chief Mining Engineer	1	105,000	29,400	134,000
Sr. Mine Engineer	1	90,000	25,200	115,000
Sr. Geotechnical Engineer	1	90,000	25,200	115,000
Mine Engineer/Planner	2	75,000	21,000	192,000
Mine Technician	2	66,950	18,746	171,000
Chief Geologist	1	100,000	28,000	128,000
Senior Geologist	1	83,500	23,380	107,000
Geologist Technician	2	66,950	18,746	171,000
Surveyor	2	66,950	18,746	171,000
Surveyor Helper	2	66,950	18,746	171,000
Total Mine Operating Staff	21			2,171,000
Staff Mine Maintenance				
Mine Maintenance Foreman	2	90,000	25,200	230,000
Mine Electrical Foreman	2	90,000	25,200	230,000
Total Mine Maintenance Staff	4			460,000

Total Underground Mine Staff	25			$2,631,000
Salaried Personnel Cost per tonne ($/t)				3.60

Table 18.31 Mine Operating and Maintenance Hourly Labour

		Basic $/hr	OH 28%	Bonus 68%	Total $/man yr	Total $/yr
Mine Operating Labour						
Production Drillers	8	32.03	82,320	26.00	133,960	1,072,000
Jumbo Operators	8	32.03	82,320	26.00	133,960	1,072,000
Rockbolter Operators	8	32.03	82,320	24.00	129,980	1,040,000
Ground Support	8	32.03	82,320	25.00	131,970	1,056,000
Scoop-Loader Operators	12	28.88	74,230	22.00	117,920	1,415,000
Truck Drivers	16	25.73	66,110	12.00	89,940	1,439,000
Dozer/Grader Operators	4	26.78	68,830	17.00	102,590	410,000
Service Crew	4	25.50	65,540	12.00	89,370	357,000
Mine Nippers	2	22.50	57,830	10.00	77,690	155,000
Mine General Labourer	4	25.50	65,540	12.00	89,370	357,000
Diamond Driller	1	26.50	68,110	20.00	107,830	108,000
Diamond Drill Helper	1	24.50	62,960	10.00	82,820	83,000
Blaster	2	29.50	75,810	40.00	155,250	311,000
Blaster Helper	2	24.50	62,960	15.00	92,750	186,000
Sub-Total Operating	80					**9,061,000**
Operating Labour Cost per tonne ($/t)						**12.41**
Mine Maintenance Labour						
HD Mechanic	10	32.03	82,300	15.00	112,090	1,121,000
Electrician	4	32.03	82,300	15.00	112,090	448,000
Welder	4	25.00	64,260	15.00	94,050	376,000
Mine Maintenance Helper	2	23.00	59,110	15.00	88,900	178,000
HD Apprentice	2	23.00	59,110	15.00	88,900	178,000
Elec. Apprentice	2	23.00	59,110	15.00	88,900	178,000
Millwright	2	30.50	78,390	15.00	108,180	216,000
Total Mine Maintenance	26					2,695,000
Maintenance Labour Cost per tonne ($/t)						**3.69**

Productivities, equipment operating hours, labour, and supply requirements were estimated for each type of underground operation, namely: drilling, blasting, mucking, ground support, haulage, and services. Total hourly labour requirements were estimated to achieve the daily mining production rate based on 2 shifts at 10 h/d with 4 crews: two on-site and two off.

The supply costs were based on 2008 Canadian supplier's prices for drill bits and steel, explosives, ground support, and services supplies, and were included in development and stoping costs. Prices for maintenance and operating supplies were obtained from supplier quotations freight on board Cantung Mine as of June 2, 2008.

A freight component was factored in to convert the estimated costs to the projected operating conditions at Mactung.

The fuel price used in calculations was $1.31/L as delivered to the site.

The cost of production development per tonne of ore was estimated based on the amount of underground development required per one production stope and stope tonnage.

Summary of development cycle time estimation and development costs is shown in the Table 18.32.

Table 18.32 Summary of Development Cycle Time and Cost

Equipment Detail	Units	Adit Slash	Ramp	Conveyor Decline	Cross Cut	Ore Drift	Ore Drift Slash	Vent Raise	Slot Raise
Width	m	5.2	5	5	5	5	17	3	2
Height	m	4.6	4	5	4	4	4	3	2
Gradient	%	0.3	0.3	15	2	0.2	0.2	90	90
Summary cycle times									
Drilling	hrs	3.15	5.77	6.13	5.77	5.77	6.24	8.22	21.40
Blasting	hrs	1.55	2.10	2.23	2.10	2.10	2.60	2.71	22.80
Re-entry	hrs	0.50	0.50	0.50	0.50	0.50	0.50	0.50	
Mucking	hrs	2.07	2.43	2.83	2.43	2.43	5.43	1.48	4.67
Support	hrs	5.59	5.66	6.63	5.95	5.72	42.78	2.93	
Services	hrs	0.59	0.52	0.52	0.52	0.40	0.27	1.20	
Secondary mucking	hrs	2.19	4.14	3.86	4.08	5.03	13.89		
Trucking	hrs	2.24	4.39	4.01	4.32	5.39	14.22	1.54	4.73
Critical path cycle time	hrs	13.45	16.97	18.83	17.27	16.92	57.82	17.04	48.86
Advance per shift	m	2.80	1.94	1.75	1.91	1.95	0.65	1.20	3.04
Advance per day	m	5.59	3.88	3.50	3.81	3.89	1.30	2.39	6.08
Labour	$/m	$404.44	$639.16	$683.43	$646.54	$667.79	$1,934.88	$1,061.03	$378.33
Equipment cost	$/m	$496.17	$792.75	$841.61	$712.35	$760.74	$1,942.82	$754.70	$179.13
Drill steel and bits	$/m	$91.33	$162.00	$176.91	$173.17	$171.41	$357.82	$57.58	$128.44
Explosives	$/m	$140.29	$239.62	$260.24	$239.62	$239.62	$275.91	$112.64	$99.92
Ground support	$/m	$344.98	$309.11	$366.05	$198.36	$188.50	$768.82	$121.95	
Piping	$/m	$158.46	$158.46	$158.46	$163.71	$46.97			
Electrical	$/m	$30.06	$30.06	$30.06	$30.06	$30.06	$30.06	$220.00	
Ventilation	$/m	$13.17	$13.17	$13.17	$13.17	$13.17	$0.00	$0.00	
Miscellaneous	$/m	$3.74	$3.74	$3.74	$92.00	$90.14	$267.72	$3.74	
Total Cost	$/m	$1,683	$2,348	$2,534	$2,269	$2,208	$5,578	$2,332	$786

The stope production cost was estimated based on average stope size as well as estimated cycle times for each operation, labour, supplies and equipment requirements. Over the 11-year mine life, 89% of the orebody is extracted by LH stoping and the remaining 11% by MCF. During the first four years of operation, all production is done by LH. In year five, a small volume of production is to be obtained by MCF from the southeast mineralized area in the Upper 2B Zone.

The operating cost for the LH mining method was based on average 17 m wide, 26 m high, and 60 m long stope size. Stope crosscut development was considered as stope waste development. Development of the drilling level on the top of the stope, which is the initial 5m wide x 4m high ore drift and slash to the full 17m stope width, development of the 5m wide x 4m high mucking drift on the bottom of the stope and development of slot raise were included in stope ore development cost.

The LH production cost was based on the estimated cycle time for parallel drilling, blasting and ore mucking. Operating costs did not include ore haulage, as it was estimated as a separate cost item and included in total mining operating costs (see haulage estimation below).

Table 18.33 Summary of LH Operating Cost

Stope Production	Cost $/tonne
Labour	$2.16
Equipment cost	$2.23
Drill steel and bits	$0.32
Explosives	$0.68
Total LH Production Cost	$5.40
Ore Drift	$3.29
Drift Slash	$3.67
Slot Raise	$0.15
Total Ore Development Cost	$7.11
Equipment cost	$2.23
Drill steel and bits	$0.32
Explosives	$0.68
Total LH Production Cost	$5.40

The operating cost for the MCF method was based on an average 17 m wide, 12 m high, and 140 m long stope size. The stope mining will be done in three 4m high cuts. It was assumed that the first cut, which is the development of initial 5m wide and 4m high ore drift and slash to the full stope width of 17m, was considered as stope ore development and a crosscut was considered as stope waste development. The next two stope cuts were considered as stope production. Ore haulage was included.

Table 18.34 Summary of MCF Operating Cost

	Ore Development		Ore Production		Total P&D
	Ore Drift 5m x 4m $/tonne	Drift Slash To 17mW $/tonne	Second Cut 17m x 4m $/tonne	Third Cut 17m x 4m $/tonne	Total MCF 17m x 12m $/tonne
Labour	$0.98	$3.43	$2.24	$2.24	$8.90
Equipment cost	$1.12	$2.98	$2.81	$2.81	$9.72
Drill steel and bits	$0.25	$0.73	$0.47	$0.47	$1.92
Explosives	$0.35	$0.41	$0.53	$0.53	$1.81
Ground support	$0.28	$1.51	$0.74	$0.74	$3.26
Piping					
Electrical	$0.04	$0.04	$0.04	$0.04	$0.18
Ventilation	$0.02	$0.00	$0.00	$0.00	$0.02
Miscellaneous	$0.13	$0.53	$0.29	$0.29	$1.24
Total MCF Cost	**$3.26**	**$9.66**	**$7.13**	**$7.13**	**$27.04**

The haulage cost was estimated based on average haulage distances as follows: ore from production stopes to underground crusher station, waste from access development to the mined out stopes as backfill during production period, and backfill haulage from backfill stockpile to the mined out stopes. Then haulage truck cycle times and trucking productivities were estimated based on 30 t trucks. It was assumed that some trucks will be loaded with backfill material after dumping the ore into the crusher station. Trucks will deliver backfill material to the mined out stopes on the return haul instead of coming empty. The waste haulage from access development to the surface during pre-production period was included in the initial capital costs.

Table 18.35 Summary of Truck Haulage Cost

Mucking/Trucking Costs		Truck
Labour Rates per Hour	$/hr	$45.29
Utilized Labour Cost per Hour	$/hr	$69.00
Equipment Operating Cost	$/hr	$112.20
Equipment Productivity per Hour	t/hr	33
Mucking Cost	$/t	$5.42

The mine exploration cost was estimated based on the assumed amount of exploration and delineation drilling required and drilling costs.

The mine services cost was estimated based on equipment working time and materials supply required for ventilation, compressed air, transportation of personnel and materials, ore handling, mine and road maintenance, mine dewatering, and underground construction.

Table 18.36 Summary of MCF Operating Cost

Cost Description	Total Cost $/day
Ventilation	
Main Ventilation Fan	$755
Vent. Doors/Regulators/Bulkheads	$290
Compressed Air	
Compressor	$95
Transporting Men and Materials	
U/G Personnel Carrier	$453
Fuel-Lube Truck	$422
Service Vehicle	$410
Road Maintenance	
Grader	$659
Underground Construction	$660
Miscellaneous, 10%	$374
Total Services Cost per Day	$4,118
Daily Production, t/day	2,000
Services Cost per Tonne, $/t	$2.06

The mine maintenance cost was estimated based on required maintenance labour, equipment, tools, and supplies.

Table 18.37 Summary of Maintenance Cost

Cost Description	Unit	Cost
Maintenance Labour Cost	$/t	$3.69
Maintenance Parts & Supplies	$/t	$3.71
Tires	$/t	$0.44
Oil and Lubricants	$/t	$0.45
Total Maintenance Cost	**$/t**	**$8.29**

Mine safety cost was estimated based on the number of underground mine personnel and required personal protective equipment, first-aid and safety supplies, mine rescue, and safety training.

Table 18.38 Mine Safety, Training, Mine Rescue Cost

Cost Description	Unit	Cost
Safety Worker Labour Rate	$/hr	$52.23
Working Hours	hrs/month	300
Labour Cost per Month	$/month	$15,669
Mine Rescue Equipment	$/month	$7,167
First Aid Equipment	$/month	$833
Cap Lamps	$/month	$990
Cap Lamp Charger	$/month	$408
Personal Protective Equipment	$/month	$5,942
Equipment Operating Cost	$/month	$15,341
First Aid Supplies	$	$1,000
Safety Equipment Supplies	$	$6,550
Supplies Cost per Month	$/month	$7,550
Small Tools/Accessories/Misc	$/month	$3,856
Total Safety Cost per Month	$/month	$42,416
Monthly Production Rate	t/month	60,000
Mine Safety Cost per Tonne	$/t	$0.71

The miscellaneous cost was assumed to be 5% of the total operating costs in this study.

The operating costs are summarized in Table 18.39.

Table 18.39 Operating Cost Summary

Cost Distribution By Item	Unit	LH	MCF	Average
Total Production by Stope	%	89	11	
Labour Operating	$/t	12.06	15.21	12.41
Labour Maintenance	$/t	3.59	4.52	3.69
Management & Supervision	$/t	3.60	3.60	3.60
Operating Supplies	$/t	6.63	10.79	7.09
Maintenance Supplies	$/t	4.47	5.64	4.60
Freight	$/t	0.98	1.23	1.01
Power	$/t	2.71	2.71	2.71
Fuel	$/t	3.03	3.03	3.03
Total Mining Operating Cost per Tonne	**$/t**	**37.07**	**46.74**	**38.14**
Cost Distribution by Operation	**Unit**	**LH**	**MCF**	**Average**
Total Production by Stope	**%**	**89**	**11**	
Stope Production	$/t	5.40	8.81	5.78
Stope Development (Ore)	$/t	7.11	12.92	7.76
Stope Access Development (Waste)	$/t	1.18	1.43	1.21

Stope Backfill	$/t	0.78	0.36	0.73
Truck Haulage	$/t	5.42	5.42	5.42
Exploration	$/t	0.29	0.29	0.29
Services	$/t	2.06	2.06	2.06
Maintenance	$/t	8.29	8.29	8.29
Mine Safety, Training, Mine Rescue	$/t	0.71	0.71	0.71
Miscellaneous	$/t	1.25	1.61	1.29
Freight of Materials and Parts	$/t	0.98	1.23	1.01
Salaried Personnel	$/t	3.60	3.60	3.60
Total Mining Operating Cost per Tonne	**$/t**	**37.07**	**46.74**	**38.14**

18.15 MINE DEWATERING

Dewatering equipment is included in the equipment list. In the event permafrost conditions are not sustained during the summer, pumps will be installed to dewater the underground workings.

18.16 PERSONNEL

18.16.1 ROSTER

Underground shifts will be 10 hours and underground personnel will work a three-week on/three-week off schedule. This exceeds the hours allowed underground by regulation and a variance will be sought. Given the nature and location of the mine, and referencing other northern mining operations where similar variances have been given, it is expected that this variance will be granted.

18.16.2 LABOUR SCHEDULE

Due to the remote location of the project, current labour shortages, and a short construction period, all mine pre-production development will be carried out with NATC's personnel and equipment, without external contractors. Table 18.40 and Table 18.41 respectively show the underground labour requirements during pre-production and production phases.

Table 18.40 Pre-production Phase Labour

	Number of Personnel		
Labour	**On Days**	**On Shifts**	**Total**
Mine Operation Staff	9	10	19
Staff Maintenance	5	5	10
Hourly Mine Crews	38	38	76

Labour	Number of Personnel		
	On Days	On Shifts	Total
Hourly Mine Maintenance	8	8	16
Total	**60**	**61**	**121**

Table 18.41 Production Phase Labour

Labour	Number of Personnel		
	On Days	On Shifts	Total
Mine Operation Staff	21	21	42
Staff Maintenance	4	4	8
Hourly Mine Crews	80	80	160
Hourly Mine Maintenance	26	26	52
Total	**131**	**131**	**262**

18.16.3 SALARY AND HOURLY WAGE RATES

The salary and wage rates are based on Cantung Mine data (2008).

18.17 RECOMMENDATIONS

Based on the findings in this report, Wardrop recommends the following:

- Investigate the potential for an open pit to mine shallow higher grade ore to replace lower grade ore in the underground mine, to further enhance project economics.
- Investigate the potential for an ore/waste pass from the open pit to the underground crusher.
- Conduct geotechnical drilling for the potential open pit in order to collect data to incorporate it with the underground mine during detailed engineering.
- Closely monitor the temperature and permafrost conditions inside the mine.
- Conduct detailed ground control monitoring oriented to minimize the length of ground supported with steel mesh in ramps and the number of long cable bolts in LH and MCF.
- Conduct detailed monitoring of freezing progress of backfilled areas.

19.0 TAILINGS DISPOSAL

Three tailings disposition systems — conventional, thickened and dry-stacked — were considered for use for the Mactung Project. Of the three deposition systems dry-stacked tailings was selected for use.

19.1 DRY-STACKED TAILINGS FACILITY

The mill tailings will be thickened, filtered, and then either transported to the underground workings as backfill or to the dry-stacked tailings facility (DSTF). Both conventional and thickened tailings disposal systems were evaluated for the project; however, the steep sloping valley sides make the required containment areas very large and the required tailings dam very high for both conventional and thickened tailings deposition systems. These requirements made the use of conventional or thickened tailings deposition systems appear difficult technically and potentially very expensive.

19.1.1 DSTF INVESTIGATIONS

In the summer of 2007, EBA excavated seven test pits and drilled four closed-ended Becker Hammer boreholes. The foundation conditions can generally be described as gravel and sand colluvium, containing some silt directly overlying bedrock. The foundation soils are fairly permeable ($K=4x10^{-2}$ cm/s) and compact ($N=10$ and $\phi' = 35°$) (EBA, 2007).

19.1.2 TAILINGS PLACEMENT

Milling 2,000 t/d produces approximately 717,600 t/a of tailings. The estimated tailings storage requirements are summarized in Table 19.1.

Table 19.1 Estimated Tailings Storage Breakdown

Year	Total Generated (m^3)	Backfilled Underground (m^3)	Surface Stacked (m^3)
1	387,328	156,629	230,699
2	387,328	177,297	210,031
3	387,328	194,691	192,637
4	387,328	208,336	178,992
5	387,328	200,965	186,363
6	387,328	200,567	186,761

table continues…

Year	Total Generated (m^3)	Backfilled Underground (m^3)	Surface Stacked (m^3)
7	387,328	188,570	198,758
8	387,328	195,930	191,398
9	387,328	180,279	207,049
10	387,328	204,179	183,149
11	387,328	222,650	164,678
Total	**4,260,608**	**2,130,093**	**2,130,515**

A conveyor will discharge the tailings to a pile outside the mill, where they will be loaded into a truck and hauled approximately 3 km to the DSTF. The truck will then dump the tailings, and a bulldozer will spread them in lifts not thicker than 600 mm. After spreading, the tailings will then be compacted with a 15-t vibratory compactor to 95% of the maximum dry density as determined by ASTM D698. The assumed compacted dry density of the tailings will be 1,800 kg/m^3. The in situ moisture content of the compacted tailings will be approximately 20%.

The foundation of the DSTF will consist of the natural deposit of sand and gravel, which will act as a drain to dissipate seepage water from the tailings pile. The seepage rate of the tailings is anticipated to be approximately 74,000 m^3/a (2.3 x 10^{-3} m^3/s). The DSTF will be constructed as a sidehill fill dump with a 4H:1V slope. The tailings pile location and typical cross-section are shown in Figure 19.1 and Figure 19.2, respectively.

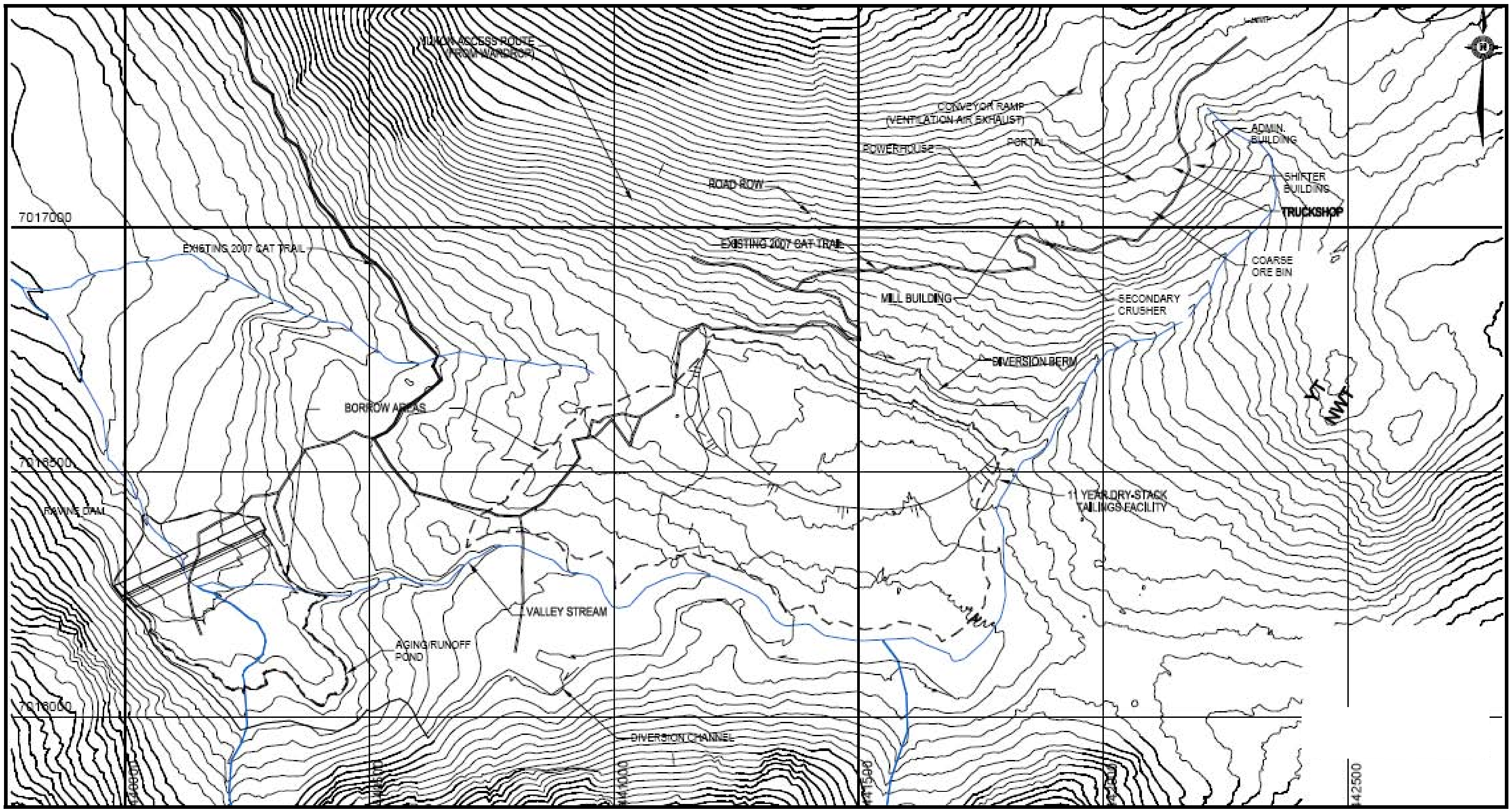
19.1.3 DSTF STABILITY DESIGN CRITERIA

There are currently no guidelines for mine waste pile construction in Yukon, so design guidelines from the British Columbia Mine Waste Rock Pile Research Committee were used. The design seismic event is 1:500 (10% chance of being exceeded in 50 years) corresponding to a ground acceleration of 0.137 G.

19.1.4 DSTF STABILITY

For stability analyses, the shear strength (ϕ') of foundation soil and tailings were 35° and 26°, respectively. The shear strength of the tailings was estimated from the particle size distribution, as shown in Figure 18.20 (Section 18.0 Mining), as well as EBA's experience with DSTFs at other Canadian mine sites. The FOS associated with several slope stability scenarios are summarized in Table 19.2.

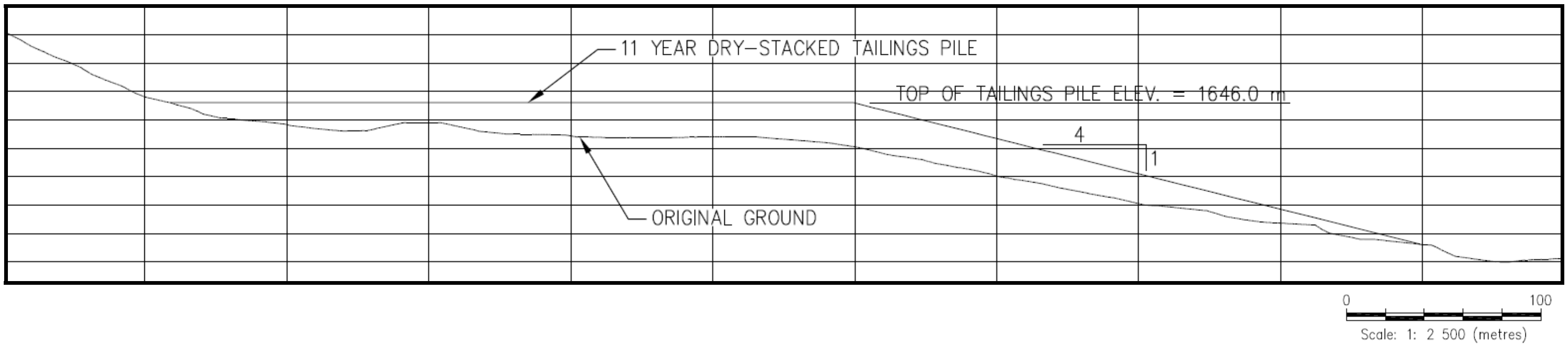
Figure 19.1 General Site Plan



Note: Contours shown on 10 m intervals

0 250
Scale: 1: 7 500 (metres)

Figure 19.2 11-Year DSTF Typical Section



19.1.5 Runoff Water Diversion

Runoff diversions structures will be required in two locations. A 760-m runoff diversion berm will be required uphill of the DSTF to divert water which would otherwise flow over the tailings pile.

The diversion berm will require foundation preparation and 30,000 m^3 of engineered fill. On-going maintenance will be required and erosion protection may be required, depending on the final gradation of the engineered fill.

Table 19.2 DSTF Slope Stability Factors of Safety

Stability Condition	Event	Recommended FOS (BC Guidelines)	Calculated FOS
Stability of Dump Surface			
Short Term (During Operation)	Static	1.1	1.9
	Pseudo-static (1:500)	1.1	1.2
Long Term (Reclamation and Abandonment)	Static	1.1	1.9
	Pseudo-static (1:500)	1.1	1.2
Deep Seated Stability			
Short Term (During Operation)	Static	1.3	2.0
	Pseudo-static (1:500)	1.0	1.2
Long Term (Reclamation and Abandonment)	Static	1.3	2.0
	Pseudo-static (1:500)	1.0	1.2

19.2 Ravine Dam

A dam is required to store the process water from the tailings thickener overflow for at least 30 days before it can be reused in the process plant, as described in Section 16.0. The aging pond reservoir will also act as a runoff storage pond to collect runoff from the site infrastructure and DSTF.

19.2.1 Ravine Dam Investigations

In the summer of 2007, EBA excavated six test pits and drilled two open-ended and two closed-ended Becker Hammer boreholes. The foundation conditions can generally be described as between 2 to 5 m of gravel and sand containing some silt, underlain by shale bedrock. The foundation soils are fairly permeable ($K=4x10^{-2}$ cm/s) and compact (N=10 and ϕ' 35^o). The shale bedrock is generally of poor quality across the bottom of the valley, good quality in the northeast abutment and unknown quality (no boreholes into rock in 2007) in the southwest abutment (EBA, 2007).

19.2.2 Ravine Dam Design

The ravine dam is designed to retain 114,643 m^3 of process water (to allow it to age for 30 days before it is re-used in the mill), and a maximum of 520,000 m^3 of runoff water. The dam is designed as a geomembrane lined structure, to account for the lack of natural low permeability core material on or near the site. The dam also incorporates the use of geosynthetic products in place of high quality aggregates, which are also not available on or near the site. The location plan and a typical cross-section of the dam are presented in Figure 19.1 and Figure 19.3, respectively.

The dam is approximately 315 m long and up to 35 m high. The dam has a crest width of 25 m to allow two-way traffic of haul trucks that will be used for construction of the dam and diversion channel. The downstream slope will be 2.5H:1V and the upstream slope will be 3H:1V with a 4H:1V toe buttress.

Engineered Fill

Engineered Fill will make up the bulk of the dam structure, and will be sourced from a borrow pit upstream of the ravine dam, increasing the size of the reservoir. Engineered Fill processing should be limited to removal of boulders at time of truck loading and avoiding areas of high silt content in the pit. The Engineered Fill will be a well-graded gravel and sand with some silt.

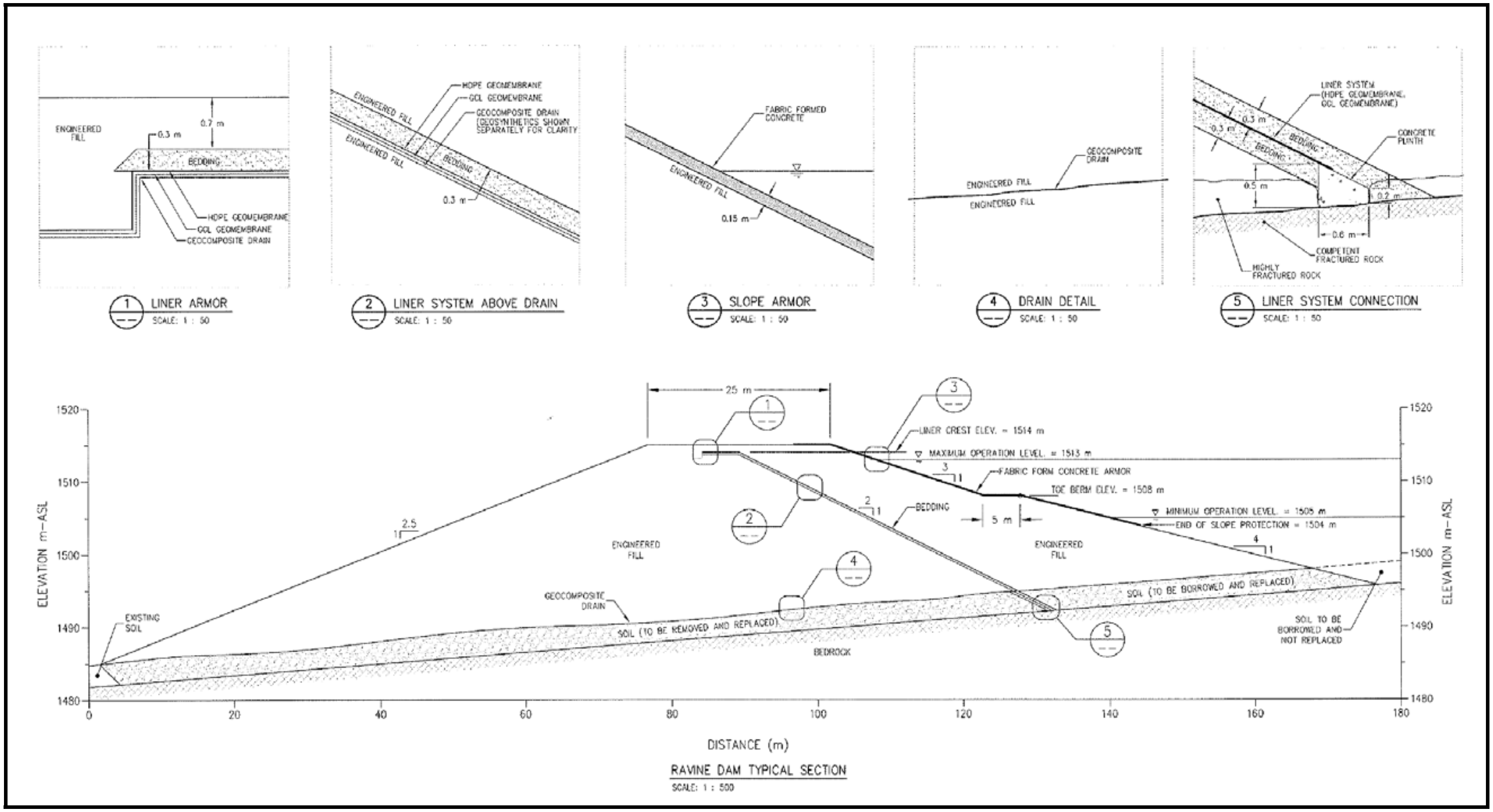
Geocomposite Drain

The Geocomposite Drain is a single piece of Geo-Net (a plastic net used for soil reinforcement) bonded between two pieces of non-woven geotextiles. It will be used to prevent porewater pressure build-up in the Engineered Fill in the event of liner leakage; it will also be used as downstream bedding for the liner system. The Geocomposite Drain will be a Geo-Comp 5, supplied by Layfield Plastics, or approved equivalent.

Liner Bedding

Liner Bedding will be used to protect the liner system from the Engineered Fill, and will be sourced from a borrow pit upstream of the ravine dam. Processing of the Liner Bedding will require screening. The Liner Bedding will be well-graded sand with trace silt.

Figure 19.3 Ravine Dam with Geosynthetics Typical Section



Notes:

1. All dimensions in metres unless otherwise noted.
2. Not for construction.

GROUT-FILLED FABRIC FORMED ARMOUR

Grout-Filled Fabric Formed Armour (Fabric Form) will be used to prevent erosion of the dam crest when the reservoir is operating. Fabric Form is an enclosed sewn sock of non-woven geotextile. The sock is sewn together in the factory to customized dimensions. Each sock is then pumped full of grout (20 – 30 MPa) in sections creating a series of grout panels. The grout panels will be 150 mm (6") thick after hydration. Fabric Forms are available from Layfield Plastics.

OVERBURDEN EXCAVATION

The overburden excavation quantity reflects the removal of the unconsolidated soils beneath the foundation of the dam. Review of existing geotechnical information shows that the material should meet qualifications for use as borrow material for Engineered Fill, with the removal of boulders and avoidance of areas of high silt content.

ROCK EXCAVATION

Rock excavation occurs immediately beneath the concrete plinth to avoid a direct short-circuit route at that mechanical connection. Rock excavation will be limited to the highly-weathered and fractured bedrock surface. Rock excavation will be conducted mechanically (by a ripper tooth attached to a bulldozer, an excavator, or hand tools). The use of drilling and blasting is not recommended as it may further fracture the underlying rock. Since the bedrock excavation is limited to highly fractured and weathered rock, it cannot be re-used as construction material, except for access roads.

CONCRETE

The Liner System must be bolted to the sloped face of a concrete plinth to facilitate the connection of the competent bedrock surface to the Liner System. The concrete plinth will be a 30 MPa reinforced concrete beam. Concrete aggregate will be sourced from a quarry, which will likely be in the area of the proposed open pit. Processing of the concrete aggregate will involve controlled blasting in the quarry, crushing with both a jaw and hydraulic cone crushing plant, screening, and washing.

LINER SYSTEM

The primary containment barrier will be a 40 mil double-sided textured high-density polyethylene (HDPE). The secondary containment barrier will be a geosynthetic clay liner, which will be 4.5 kg/m^2 of bentonite needle-punched between two 300 g/m^2 non-woven geotextiles. These two geosynthetics together comprise the Liner System.

19.2.3 RAVINE DAM DESIGN CRITERIA

The ravine dam has been classified as a “Significant” consequence dam, according to the 2007 Canadian Dam Association Dam Safety Guidelines (CDA Guidelines).

The “Significant” rating was selected for the following reasons:

- There is only a temporary population at risk downstream.
- There is no anticipated loss of life.
- There is no significant loss or deterioration of fish or wildlife habitat.
- Restoration or compensation in-kind is highly possible.
- Potential infrastructure and economic losses will be limited to public recreational facilities, seasonal workplaces, infrequently used transportation routes, and owner’s infrastructure and mill production (CDA, 2007).

The CDA Guidelines stipulate the design seismic and flood events for the various dam classifications. For the “Significant” classification, the design seismic event is 1:1000 year return period and the inflow design flood can be between 1:100 and 1:1000 year flood. For the purpose of the design, the peak ground acceleration associated with a 1:1000 year return period of 0.179 G was used and reservoir was sized to the 1:100 year flood event (peak storage of 520,000 m^3).

19.2.4 RAVINE DAM STABILITY

For stability analysis the shear strength (ϕ’) of foundation soil and engineered fill were both 33° and the shear strength of the liner system (including bedding) was 14°. The shear strength of the liner system interface was determined from literature (PGI, 1997). The FOS associated with several slope stability scenarios are summarized in Table 19.3.

Table 19.3 Ravine Dam Slope Stability Factors of Safety

Loading Condition	Minimum FOS (CDA, 2007)	Slope	Calculated FOS
Static	1.5	Upstream	2.0
		Downstream	1.7
Pseudo-static	1.0	Upstream	1.0
		Downstream	1.1

19.2.5 STORAGE ASSUMPTIONS

The reservoir was sized to the 1:100 year flood event corresponding to a peak runoff storage of 520,000 m^3 and a peak total storage of 640,000 m^3, including the process water. A frequency analysis for the reservoir inflow is shown in Table 19.4. The stage-storage chart for the ravine dam is presented in Table 19.5.

Table 19.4 Frequency Analysis for Ravine Dam Reservoir

Return Period	Probability of Exceedence in any Given Year	Peak Flow (m^3/s))
2	50%	1.5
5	20%	2.0
10	10%	2.2
20	5%	2.5
50	2%	2.8
100	1%	3.0
200	0.5%	3.2
500	0.2%	3.4

Table 19.5 Stage Storage for Ravine Dam

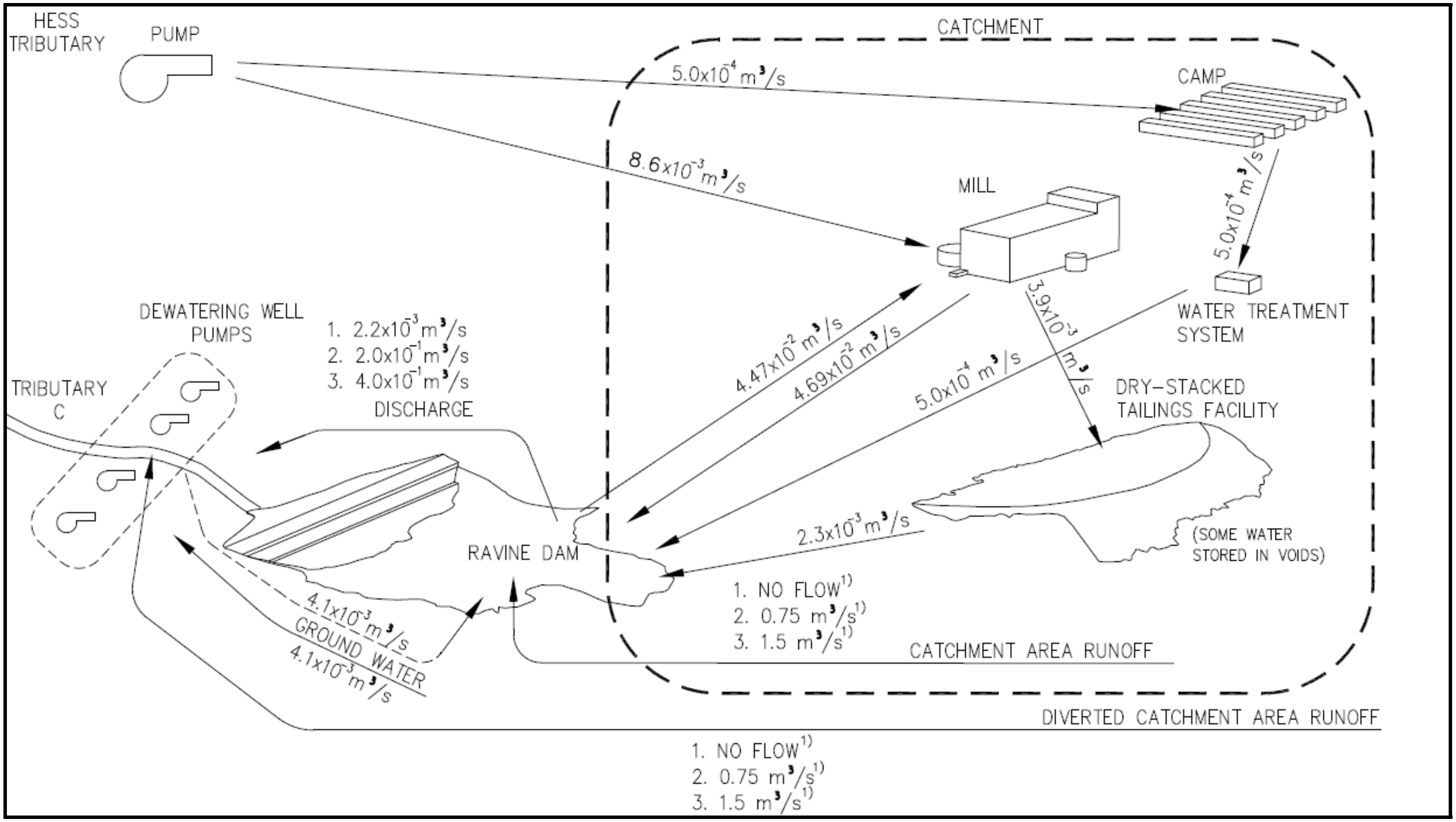
Water Elevation (m)	Reservoir Capacity (m^3)
1504	98,000
1505	130,000
1506	170,000
1507	220,000
1508	280,000
1509	340,000
1510	410,000
1511	480,000
1512	560,000
1513	640,000

A reservoir water balance for the average (1:2 return period) and maximum (1:100) year return period is shown in Figure 19.4. To keep the dam from overtopping, controlled discharge is required. The discharge rate will vary with the precipitation inflow. The discharge during the winter months will be limited to the difference in the process water discharge rate and the process water reclaim rate.

The maximum discharge rates associated with the 1:100 year and 1:2 year events are 0.4 m^3/s and 0.2 m^3/s, respectively.

The anticipated groundwater flow under the dam is 4×10^{-3} m^3/s. Four wells will be required approximately 100 m downstream of the dam toe to pump the groundwater back into the reservoir, should the seepage water not meet discharge criteria.

Figure 19.4 Water Balance



Notes:

1. OVER-WINTER CONDITIONS
2. 1:2 YEAR (AVERAGE) CONDITIONS
3. 1:100 YEAR (MAXIMUM) CONDITIONS

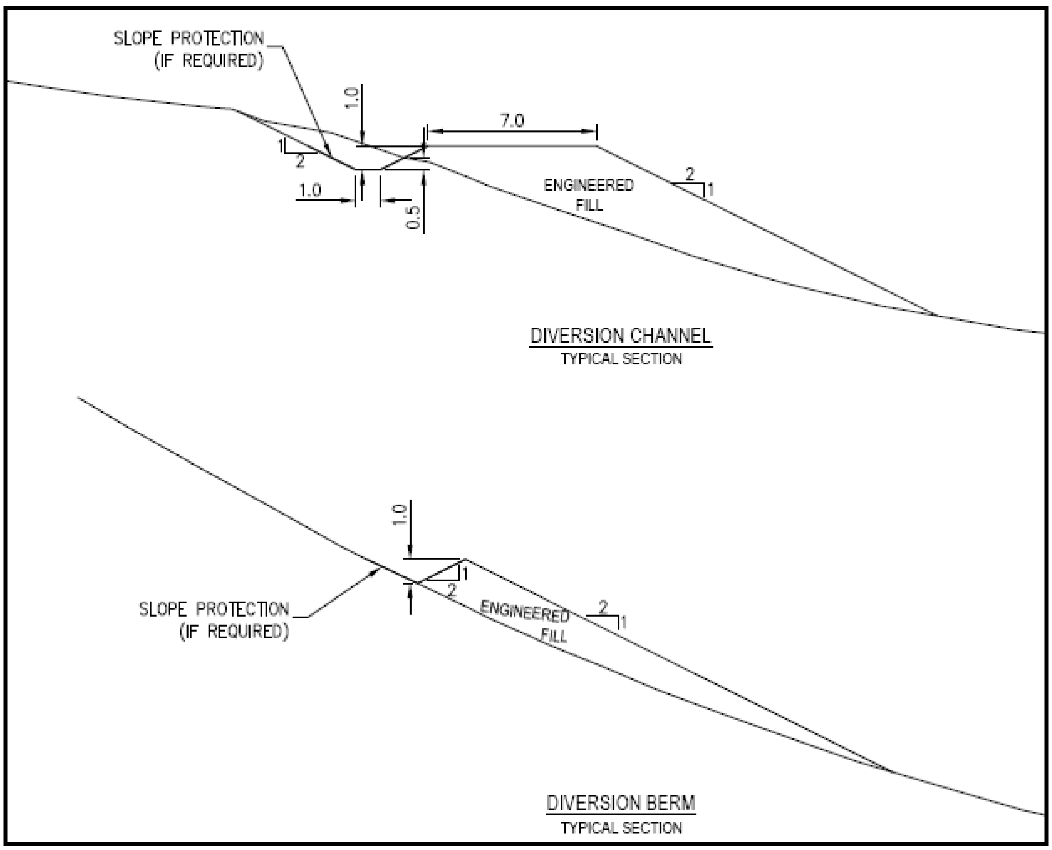
1) PEAK FLOW VALUES, FLOWS VARY
2) ALL OTHER FLOWS CONSTANT YEAR ROUND

19.2.6 *RUNOFF WATER DIVERSION*

A 2,160 m runoff diversion channel will be required on the north side of the valley to divert runoff water around the dam reservoir. Typical runoff diversion structure sections are presented in Figure 19.5.

The diversion channel will require an excavation of 9,000 m^3 and 85,000 m^3 of engineered fill. On-going maintenance, including spring snow removal, will be required and erosion protection may be required, depending on the final gradation of the engineered fill.

Figure 19.5 Diversion Structures – Typical Sections



NOTES:
1) ALL DIMENSIONS IN METRES UNLESS OTHERWISE SHOWN
2) NOT FOR CONSTRUCTION

Scale: 1: 250 (metres)

20.0 INFRASTRUCTURE

The infrastructure planned at Mactung to support the mining and processing operations includes:

- site and access roads
- an airstrip upgrade
- water supply and distribution system
- waste disposal system
- ancillary facilities
- power supply and distribution system
- communication system.

20.1 SITE LAYOUT

The layout of the site consists of the process mill building, ancillary buildings, an underground primary crushing facility, and the dry stack tailings area. The site layout was designed to minimize the amount of disturbed area.

The location of the main project facilities is shown in Figure 20.1.

20.2 ROADS

20.2.1 ACCESS ROADS

Approximately 48 km of upgraded/new roads will provide access from the existing Upper Canol Road to the mill site. A road to the pump house at the Hess River Tributary C will branch from the main access road. All access roads will be designed for the high snowfall and long winters typical of this area.

Wardrop evaluated two possible access road routes. The first option is an approximately 25-km route that follows through a narrow valley with steep slopes. The second option is an approximately 35-km route through a wider and flatter valley. The second option was selected for its flatter terrain. The access road layout is shown in Figure 20.2.

Figure 20.1 Site Layout – Main Project Facilities

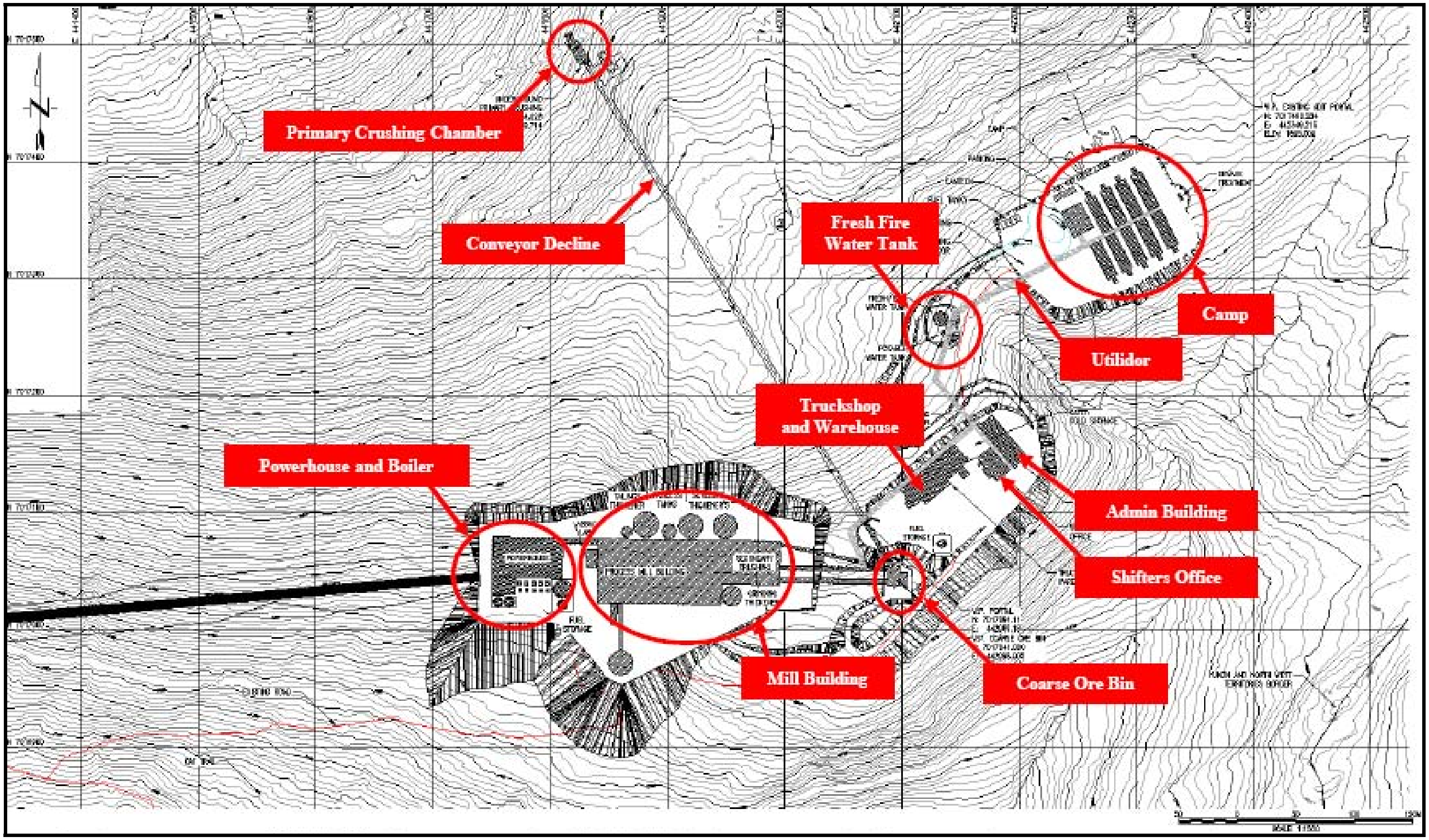
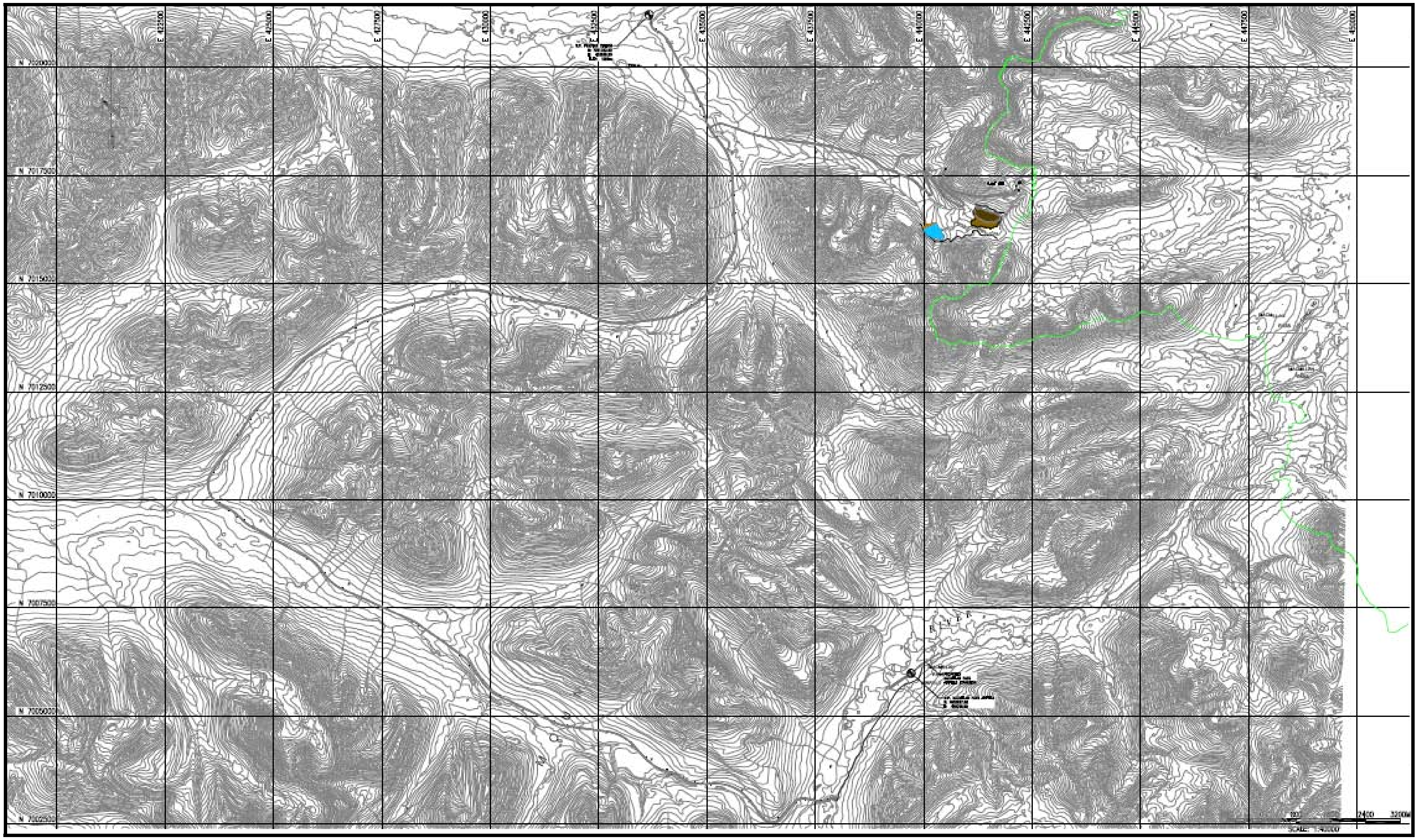


Figure 20.2 Access Road Layout



The design of the access roads assumed the following:

- an average vehicle speed of 30 to 50 km/h
- an overall 8% maximum grade, increasing to 12% grade in some sections
- a 0.2 m surface of 25 mm crushed gravel/rock
- an 8 m wide main access road
- a 5 m wide pumphouse road
- 1.0 m deep ditch lines
- 2H:1V cut slopes
- 3H:1V fill slopes.

20.2.2 SITE ROADS

The site roads connect the process plant, coarse ore storage area, truck shop, and accommodation camp. The design of the access roads assumed the following:

- an average vehicle speed of 30 to 50 km/h
- an overall 8% maximum grade, increasing to 12% grade in some sections
- a 0.2 m surface of 25 mm crushed gravel/rock
- a width of 6 m
- 0.5H:1V cut slopes
- 1.5H:1V fill slopes.

20.3 AIRSTRIP UPGRADE

The MacMillan Pass airstrip is located south of the Mactung site adjacent to the North Canol Road. The airstrip is reported to be 460 m long by 15 m wide (1,500' x 50'). The airstrip is owned and maintained by the Government of Yukon. NATC will upgrade and maintain the airstrip during the life of the project.

The airstrip will be upgraded to 1,375 m long by 30 m wide, to accommodate a 19-person Beechcraft 1900 or similar aircraft. A total of 25.4 ha will be cleared around the strip to provide the necessary space for the airstrip, apron, and obstruction clearances.

An apron area measuring 60 m by 90 m will be provided to simultaneously park two Beechcraft 1900 aircraft or similar.

Construction will be a compacted granular sub-base with a crushed rock cover. Longitudinal and transverse slopes will be limited to 2%. The finished elevation of the airstrip will be above the flood level of the MacMillan River and, if required, the airstrip will be provided with an armoured berm to protect against erosion during river flooding.

The airstrip will be designed for visual flight rules (VFR); no runway lights will be provided, similar to the operating runway at Cantung Mine.

20.4 WATER SUPPLY AND DISTRIBUTION

Fresh/fire water will be pumped from the Hess River Tributary C to the Mill Site by a pipeline approximately 10 km. The pipeline will be insulated and heat-traced to prevent freezing.

20.4.1 FRESH AND FIRE WATER SUPPLY

Fresh water and fire water will be stored in one 10 m high by 11 m diameter fresh/fire water tank. The tank will be made of carbon steel and insulated. Fresh water will use 128 m^3 of the 810 m^3 capacity tank, and the remaining 682 m^3 will be used for fire water.

20.4.2 SITE WATER DISTRIBUTION

Fresh/fire water will be gravity-fed to the process mill building and ancillary buildings, and pumped to the permanent camp. Pipelines will run through a series of utilidors to prevent freezing.

20.4.3 POTABLE WATER

Potable water will be stored in a 4.2 m high by 4.2 m diameter carbon steel, double-walled and insulated potable water distribution tank. A hypochlorinator will disinfect the water prior to use.

A potable water pump house will be located next to the tank and will house two 7.5-kW distribution pumps.

20.5 WASTE DISPOSAL

Waste will be disposed of on-site to minimize or eliminate the requirement for off-site waste removal services.

20.5.1 WATER WASTE DISPOSAL

Grey water will be transported to the sewage treatment plant, located west of the camp. The grey water, along with the thickener water, will then be transported to the aging/runoff pond, where it will be retained for 30 days, then pumped to the ravine dam.

20.5.2 Solid Waste Disposal

Large waste items will be transported to the dry stack tailings area and disposed of along with the tailings filter cake. All garbage and small waste items will be incinerated on-site.

20.6 Plant Ancillary Facilities

The following is a general description of the ancillary facilities recommended for the Mactung site.

20.6.1 Truck shop/Warehouse

The warehouse/truck shop will be a single story, 60 m long by 16 m wide by 13 m high stick-built building, designed for truck maintenance and repair, and for storage.

The warehouse/truck shop includes indoor truck bays, a waste oil system, an exhaust system, lube-oil systems, water systems, coolant systems, a machine shop and equipment, a welding bay and tire-change area. The building will include offices for maintenance and warehouse personnel.

20.6.2 Cold Storage

The cold storage warehouse will be an unheated spring structure used to store large equipment.

20.6.3 Administration Building/Mine Dry

The administration building will be a two-storey, 44 m long by 14.5 m wide by 6 m high pre-engineered building.

The administration building will include working space and offices for engineering, technical, surveying and administration personnel. The offices of the general manager, mine manager, mill superintendent, mine operations superintendant, maintenance superintendent and mine foremen will be located in this building. A first aid safety area, control station, kitchen, lunch room facilities will also be located in the building.

The mine dry area will include washing areas and men's/women's locker rooms.

20.6.4 BUNKHOUSE COMPLEX AND CANTEEN

The bunkhouse complex and canteen will accommodate 150 personnel. After construction, the bunkhouse and canteen will be refurbished to make it suitable for operations personnel.

20.6.5 FUEL STORAGE

Fuel for the mining equipment and process and ancillary facilities will be stored in fuel storage tanks located adjacent to the south side of the power plant.

20.6.6 EXPLOSIVE STORAGE

Explosives storage facilities will be both above ground and below ground. Two powder magazine containers and one cap magazine container will be located above ground.

An underground concrete explosives storage facility will include concrete and gates for security and an ANFO kettle mobile.

20.6.7 ASSAY LABORATORY

The assay laboratory will be a single story, 15 m long by 8 m wide by 3 m high pre-engineered building located adjacent to the process mill building. It will be equipped to perform analyses of mine and process samples.

20.7 POWER SUPPLY AND DISTRIBUTION

Multiple on-site diesel generator units will generate electrical power. A power plant structure, including five diesel generators with heat recovery modules, will be located adjacent to the process plant to minimize power distribution losses and connection costs.

The number of generator sets will be N + 1, where N is the number of units required to meet the maximum power demand. N is determined to be four 2.58 MW generator units (de-rated 10% for 1,900 m elevation). One stand-by unit will be used as a backup unit. Each generator is capable of 10% extra power generation to cover momentary surges in power demand.

Each generator unit will recover waste heat from engine coolant, engine-lubricating oil, and the exhaust gases. When in full production and with four generator sets operating under load, there will be sufficient recovered heat to heat surface facilities in the winter. The camp will be heated via a combination of electrical heat and propane.

Portable propane heaters will provide heat for surface facilities when waste heat is not available from the generator units, typically when the mill is shut down for maintenance or repair.

The generator system will provide electrical power at 4.16 kV. Power will be distributed throughout the mine site at this voltage for large electrical loads, such as rod mills. A number of centrally-located electrical rooms will transform 4.16 kV power to 600 V. These electrical rooms will include motor control, lighting panels, and other electrical equipment necessary for facility operation. Electrical power will be stepped up to 13.8 kV for distribution by overhead power line to locations such as the tailings and booster pump station.

Underground electrical power distribution equipment will power the underground ventilation, primary crusher, and miscellaneous loads via two underground power stations. Power will be delivered to underground operations via a single 4.16 kV underground tunnel power cable.

20.8 COMMUNICATION

The telecommunications system will have adequate data, voice, and other communications channels available. The telecommunications system will be supplied as a design-build package. The base system will be installed during the construction period then expanded to encompass the operating plant.

The system will include:

- a VoIP telephone system
- satellite communications for voice and data
- ethernet cabling
- wireless Internet access
- 2-way radio communications at site
- satellite TV.

A main telecommunications Central Equipment Office will consist of a pre-manufactured trailer in which the main communications contractor will install and test all the main sub-systems for the facilities, prior to shipment. The trailer will form the first block in a system that must support the construction needs of the project first, and the operating needs of the project following construction.

Spare parts for critical and main components will be provided to ensure maximum reliability, and minimum down time. A variety of communications media (copper and wireless during the construction phase and fibre optic during the operating phase) will be incorporated in the overall design.

Communications requirements, particularly satellite bandwidth, are expected to peak during the plant construction phase, and then taper off slightly as initial construction crews yield to plant operations.

Technologies and services will include the following:

- construction phase:
  - local VoIP wireless network
  - satellite link for voice, data and video services
  - personal computer local/wide area network (PC LAN/WAN)
  - trunked mobile radio system
  - internet service
  - private telephone system for voice and fax.
  - video conferencing
  - ground-to-air communications system (VHF Radio)
  - independent satellite television system.
- operation phase (includes selected services above):
  - process monitoring and control for efficient operation and maintenance
  - fibre optic cabling for plant wide communications
  - security access control
  - closed circuit television (CCTV) for process, security and safety.

The underground mining operation communication system will use leaky feeder technology.

20.9 PROCESS PLANT CONTROL SYSTEM

20.9.1 OVERVIEW

The process plant control system will consist of a Distributed Control System (DCS) with PC-based Operator Interface Stations (OIS) located in two separate control rooms. The DCS, in conjunction with the OIS will perform all equipment and process interlocking, control, alarming, trending, event logging, and report generation. DCS input/output (I/O) cabinets will be located in electrical rooms throughout the plant, and interconnected via a plant-wide fibre optic network.

Field instruments will be microprocessor-based “smart” type devices. Instruments will be grouped by process area, and wired to each respective area local field instrument junction boxes. Signal trunk cables will connect the field instrument junction boxes to DCS I/O cabinets.

Intelligent-type Motor Control Centres (MCC) will be located in electrical rooms throughout the plant. A serial interface to the DCS will facilitate the MCC's remote operation and monitoring.

The plant control room operators will be in and out of the control rooms; therefore, the rooms will be equipped with alarms to alert the operator of potential problems.

20.9.2 PRIMARY CRUSHING CONTROL SYSTEM

A single OIS will be installed in the primary crushing underground electrical room to monitor all crushing and conveying operations onto the coarse ore stockpile. Control and monitoring functions will include:

- plugged chute detection at all transfer points
- zero speed switches, side travel switches, emergency pull cords, and belt rip detection of all conveyors
- weightometers on selected conveyors to monitor feed rates and quantities
- equipment bearing temperatures and lubrication system status
- vendor instrumentation packages.

20.9.3 MILL BUILDING

To control and monitor all mill building processes, two OIS's will be installed in the mill building's central control room. These OIS's will control and monitor:

- coarse ore reclaim, crushing and grinding conveyors (zero speed switches, side travel switches, emergency pull cords and plugged chute detection)
- secondary and tertiary crushers (speed, bearing temperatures, lubrication systems, motors, and feed rates)
- grinding mills (mill speed, bearing temperatures, lubrication systems, clutches, motors, and feed rates)
- pump box, tank, and bin levels
- variable speed pumps
- thickeners (drives, slurry interface levels, underflow density, and flocculant addition)
- flotation cells (level controls, reagent addition, and airflow rates)
- samplers (for flotation optimization)
- concentrate pressure filter and load out
- reagent handling and distribution systems
- tailings disposal

- water storage, reclamation, and distribution including tank level automatic control
- air compressors
- fuel storage
- vendor instrumentation packages.

20.9.4 Remote Monitoring

CCTV cameras will be installed at various locations throughout the plant, including the primary crushing facility, the stockpile conveyor discharge point, the stockpile reclaim tunnel, the secondary crushing area, the grinding area, and the concentrate handling area. The cameras will be monitored from the two plant control rooms.

21.0 YUKON ASSESSMENT AND PERMITTING REGIME

21.1 INTRODUCTION

The Mactung property is located in the Selwyn Mountain Range and covers the area around Mt. Allan along the Yukon/NWT border. It is a remote site with seasonal access by gravel road, and is a six hour drive from the community of Ross River.

The Selwyn Range, characterized by sharp peaks, steep sideslopes, and narrow rounded valleys, forms the landscape of the project area that varies from gentle to flat terrain on the valley floor to steep relief on upper bedrock slopes. Mountain elevations range from 745 masl to 2970 masl.

Most of the proposed Mactung Project area is located in un-forested high alpine terrain, where forest stands occur only at elevations below 1400 masl. Upper bedrock and colluvium slopes are mostly bare and lower valley slopes and valley floors are typically vegetated with grasses, mosses, lichens and alpine willow and birch.

The area is used by nesting migratory birds in the summer. Mammal species which frequent the area include caribou, grizzly and black bear, wolf, lynx, wolverine, fox, marten, hare, and other small mammals.

21.2 LICENSING AND PERMITTING

Project development activities proposed to occur in the Yukon are subject to the *Yukon Environmental and Socio-economic Assessment Act* (YESAA). This assessment regime has been formed from the Umbrella Final Agreement and is tailored to fit the governing structures of Yukon. Yukon-settled First Nation Governments govern their respective settlement lands and hold traditional treaty rights for areas designated traditional territories. Of the fourteen First Nations Governments in Yukon, eleven have settled land claims and developed Final Agreements with the Government of Yukon and Government of Canada.

The Mactung project is proposed to occur in an overlap area of the traditional territories of three Yukon First Nations, including: the Ross River Dena Council (RRDC), the Liard First Nation (LFN) (both non-settled First Nations), and the First Nation of Na-Cho Nyak Dun (NND) (a settled First Nation). The project site, as currently designed, does not cross settlement land or land set aside by the non-settled First Nations.

Beyond meeting the requirements of the assessment process, proponents of development projects are required to meet the regulatory obligations of both the Government of Yukon and Canada.

21.2.1 *YUKON ENVIRONMENTAL AND SOCIO-ECONOMIC ASSESSMENT ACT*

YESAA is the legal basis for the assessment of projects in Yukon. It provides a single assessment process for projects proposed on all Yukon lands, including federal, territorial, and First Nation lands.

The assessment process is governed by federal legislation and regulations as well as rules created by the Yukon Environmental and Socio-economic Board (YESAB). YESAB is an arms-length independent board which makes recommendations to decision bodies, comprised of federal departments, Government of Yukon, and First Nations Governments.

21.2.2 *REQUIRED YUKON PRODUCTION PERMITS*

This section contains an overview of the permitting required in Yukon for the production phase of a tungsten mine, as well as any perceived risks as identified through the consideration of the Yukon permitting regime.

In 2003, the responsibility for land, water, mineral, and other natural resources were transferred from the federal to the Government of Yukon. The majority of the permits required for a producing mine have since been issued by the Yukon government, or by arms-length boards. The main permits required for a producing tungsten mine include a Type A Water Licence and a Quartz Mining License. Depending on the land tenure, land use permits or land leases may also be required. In addition, a number of other ancillary permits will be required by Yukon government and federal agencies. A complete list cannot be provided at this time, as permitting requirements will be determined upon completion of project design and planning.

QUARTZ MINING LICENSE

Within Yukon Territory, the Government of Yukon, Department of Energy, Mines, and Resources form the major regulator for mine exploration and production which occurs on Crown land. Regulatory requirements stem from the *Quartz Mining Act* (Yukon).

The process has been designed to meet the realistic nature of mining projects which have received a positive determination of the assessment process. As a result, the department will issue the Quartz Mining Licensing in two phases:

1. The first phase will include authorization to proceed with construction activities which do not require licencing through the *Waters Act* (Yukon).
2. The second phase will form the final component of the major licence requirements for the proponent and will provide authority to finalize construction activities and proceed with the proposed mining program.

TYPE A WATER LICENCE

In Yukon, water licencing is conducted by the Yukon Water Board under the authority of the *Waters Act* (Yukon). The responsibility of the Water Board is specific to the provision of Water Licences, which may only be issued once a project has been assessed under YESAA. Terms and conditions issued by the Water Board may not contravene a decision document issued under the authority of YESAA. Licences issued by the Water Board provide the Government of Yukon with an enforcement mechanism, based on the technical parameters pertaining to quantity of water used and quality of effluent released.

The Mactung project will require the issuance of a Type A Water Licence for its production activities. As such, Section 19 of the *Waters Act* establishes the requirement for a public hearing in the review of the Water Licence application prior to permitting.

ANCILLARY PERMITTING REQUIREMENTS

Other permits which may be required by the Yukon government include:

- Type B Water Licence
- Land Use Permit/Lease
- Work in the Right-of-Way of a Highway
- Formation of a Memorandum of Understanding (for the expansion of an aerodrome)
- Environmental Health Permits
- Building, Development, and Electrical Permits
- Storage Tank System Permit
- Special Waste Generator Permit
- Air Emissions Permit
- Solid Waste Commercial Dump Permit

- Permit to Install a Sewage Disposal System
- Blasting Permit
- Explosive Magazine Permit.

Permits which may be required from the Government of Canada include:

- Explosives Factory Licence
- Explosives Magazine Licence
- Blasting Explosives Purchase and Possession Permit
- Non-Mechanical Ammonium Nitrate and Fuel Oil (AN/FO) Certificate
- Permit to Transport Explosives
- Authorization for Works or Undertakings Affecting Fish Habitats
- Metal Mining Effluent Regulation
- Approval under *Navigable Water Protection Act*.

This list is not meant to provide a comprehensive list of potential permits, but rather to identify authorizations which may be required as a result of the proposed project.

21.3 ENVIRONMENTAL SETTING

A number of studies have been completed since 1968 to study and document baseline environmental conditions in the area. A current comprehensive baseline environmental assessment was initiated in 2006 (EBA, 2007).

21.3.1 SURFICIAL GEOLOGY/SOILS

The general site geology of the Mactung Project area has been extensively described and documented in previous reports. Detailed terrain mapping was completed in the project area in 2007, as it was known at this time.

Soils in the project area originate from glacial, colluvial, and minor fluvial processes. The higher areas of the mountains have little or no surficial soils while the lower slopes and valley bottoms are covered with thin deposits of residual soils, colluvium and glacial tills. Till (moraine) is the most common surficial material mapped in the project area and occurs on middle to lower valley hillslopes

Due to the generally weak and fissile nature of the parent rocks in the area, which are generally highly schistose metamorphic rocks, most granular deposits are of poor quality. No granular deposits suitable for road surfacing or concrete aggregates have been identified and initial indications are that these materials may have to be obtained by crushing local competent bedrock.

Steep terrain and climate factors such as rain and snowmelt contribute to potentially high surface soil erosion hazard on the lower and middle slopes from May to September, particularly during spring freshet.

Information required for the completion of the surficial geology/soils portion of the YESAA application and regulatory documentation was available for the project area, however, it was decided that more information would be required on the surficial geology and soil for the mine access road from the North Canol Road to the mine site. This information was collected and analyzed during the summer 2008 field program and will be included in the YESAA Project Proposal.

21.3.2 *Acid Rock Drainage Potential*

Several historical papers and studies completed for this site between 1976 and 1982 have been reviewed with regards to geochemical studies. The reviewed reports contained all available relevant geochemical data and it appeared that no prior acid-base accounting (ABA) analysis was conducted at the mine site.

There has been considerable geological characterization of the rock units at the Mactung site. Four of the nine mappable units at the site are mineralized. The primary ore mineral is scheelite and occurs in varying quantities within the mineralized units. Sulphide mineralization occurs as pyrhotite, pyrite, and chalcopyrite within the ore zone to varying degrees. There has also been considerable surface mapping, diamond drill coring, and petrographic analysis including thin section and X-ray diffraction (XRD) studies. Sample descriptions and petrographic analysis show that mineralogical content within a single lithologic unit can be highly variable.

Geochemical investigations were completed in 1978 and 1979 by the Colorado School of Mines (CSM) bulk sample collected in 1973 (CSMRI, 1976). These reports documented an 8-week and 44-week testing program, in which a tailings sample was rinsed with aerated water. The leaching program was designed to include several variants such as temperature, inoculation with bacteria and circulation vs. free flow.

The CSM study was related to tailings produced from the pilot scheelite flotation plant of AMAX Inc. (the site owner at that time). The sample used for the test work consisted of Mactung pilot plant tailings material of known composition (8.07% calcium, 13.80% carbonate, 5.83% total sulphur, 5.81% sulphide sulphur, and 0.02% sulphate sulphur).

Rock samples were submitted by NATC for ABA and shake flask analyses to provide information on acid rock drainage potential and metals leaching. A total of 27 samples were submitted for the underground workings and surface borrow sites to provide geochemical characterization information on the different rock types.

This information should satisfy YESAA information requirements for characterization with respect to acid rock drainage; however, the age of the core may be of concern.

The only core from the site that is less than two years in age is from geotechnical investigations conducted in 2007. It was recommended that core collected from the proposed 2008 exploration program be submitted for ABA analyses to confirm the results of the older core. Sample collection was not conducted as part of the 2008 exploration program for ongoing static and kinetic testwork.

Based on Wardrop and EBA data analysis, NATC has assumed that the tailings for the project are similar in composition to the tailings produced at the nearby Cantung Mine located in the NWT. A comparative study of Mactung and Cantung undertaken by NATC and Wardrop has confirmed this assumption.

Samples of tailings are planned to be collected during subsequent metallurgical test programs and on a continual basis during mine operation to confirm potential impacts to the environment. To ensure that impact to the environment is minimal during and after mine operation, potentially acid-generating (PAG) and metal-leaching tailings will be stored using the most conservative method allowable. If subsequent testing shows that the tailings are more benign, then less conservative storage methods may be employed.

The results of ABA analyses indicate that all ore grade samples are PAG. Low grade ore was overall net neutralizing but some PAG samples were identified. Unit 1 waste rock sample analyses identified 67% of the samples as PAG. Unit 3 waste rock sample analyses identified 83% of the samples as PAG.

The underground nature of the development means that there will be no above ground ore stockpiles and wastes identified as having acid rock drainage or metal leaching potential may be able to be backfilled underground as part of the mining process.

Based on the PAG classification of the ore samples, it is anticipated that the tailings produced at Mactung will be PAG. Due to the lack of a metallurgical program, it has been assumed that the tailings will reflect the ore and are therefore considered PAG. If tailings are assumed to be PAG, there may be a need to encapsulate the tailings at closure to minimize risk.

21.3.3 TERRAIN HAZARDS

Active geomorphological processes in the study area include rockfall, debris slides, debris flows, avalanches, gully erosion, and periglacial processes (e.g. rock glaciers).

Rapidly-drained bedrock slopes of the upper valley walls and cirques are typically steep (greater than 70%) and moderately steep (50 to 70%). Rockfall is the primary erosional process on these slopes and the moderately steep to steep slopes are likely subject to avalanching.

A possible relic (inactive) debris slide about 10 ha in area was identified on a north valley hillslope about 6 km west of the Yukon/NWT border. Small, periodic debris flows are probably an ongoing process within valley sideslope stream channels and contribute to colluvial fan deposits mapped in the study area.

Snow accumulations in the project area are heavy and the moderately steep to steep slopes have potential for avalanching. Avalanches are an annual occurrence in this region and probably play a part in some downslope transport of colluvial material in the study area.

Gullies on the valley sideslopes are mostly active, with most erosion probably taking place during spring run-off. A number of rock glaciers are mapped in the study area, mostly on north-facing slopes but are not located within areas impacted by proposed development and there is a low probability that the project will be impacted by large rock glaciers.

The project area is located in an area of moderate seismic hazard, rated 3 on a scale of 5 (GSC, Natural Resources of Canada, 2005). The maximum ground accelerations associated with 1:475 and 1:1000 year return earthquakes are 0.137 G and 0.179 G, respectively.

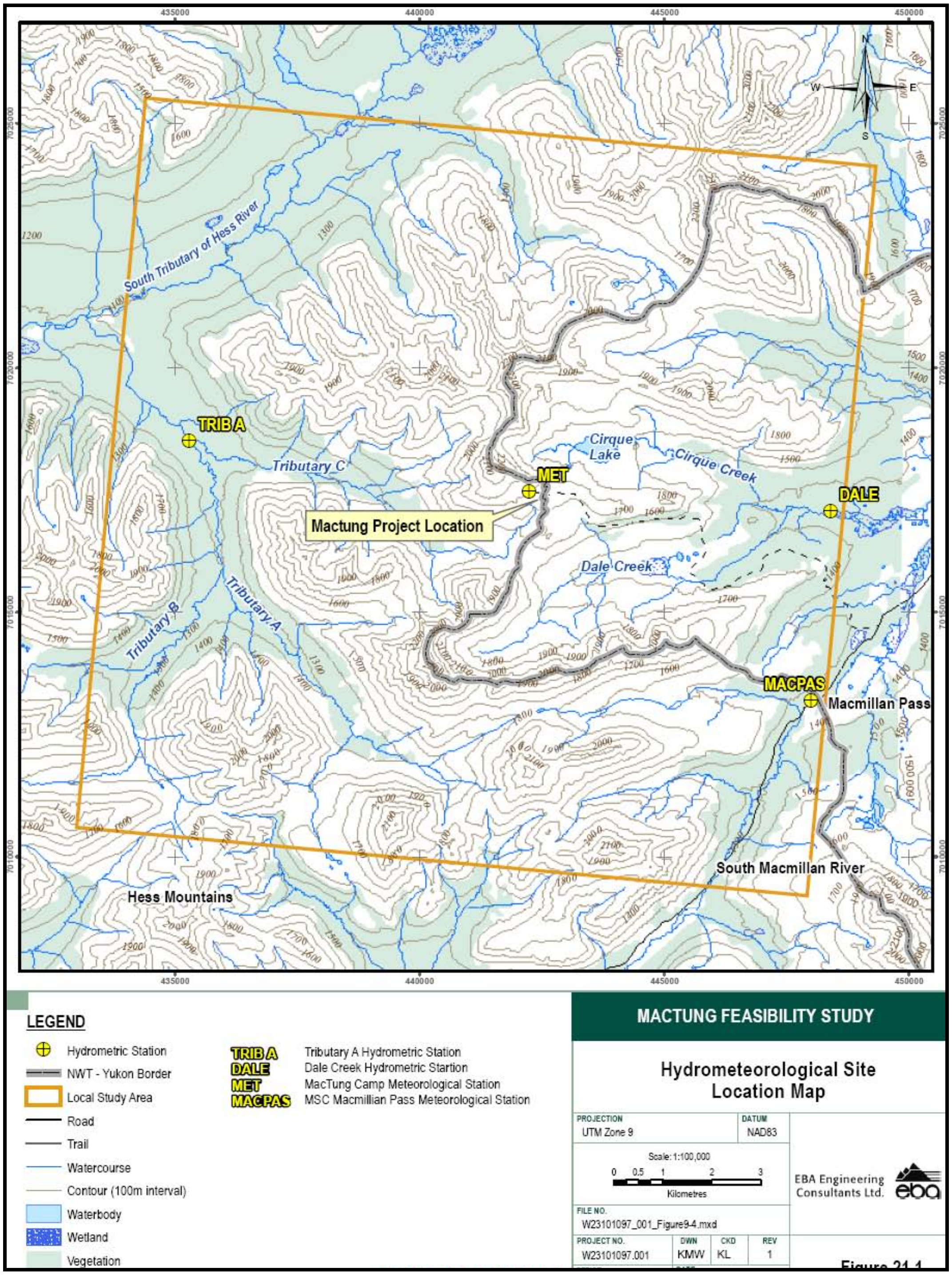
Information required for the completion of the terrain hazards section of the YESAA application was considered to be available at the time of feasibility analysis. The terrain hazards mapping being completed for the mine access road from the North Canol Road to the mine site will be incorporated into the YESAA application.

21.3.4 HYDROLOGY

During the summer of 2006, hydrometric stations were installed on two Yukon creeks draining the Mactung Project site, and Tributary A. The locations of these stations are shown in Figure 21.1. The ongoing study involves the continuous open water season measurement of creek stages and temperatures as well as numerous discrete discharge measurements and subsequent determination of a time history of discharge from the stage.

Tributary C originates in Yukon close to the Yukon/NWT border within the Mactung property, and has a catchment of 24.2 km^2. Tributary C flows into Tributary A, which flows northwest into the south tributary of the Hess River, and eventually into the Stewart River in Yukon. Time histories of creek discharge were determined over the periods from July 9, 2006 to September 20, 2006 and July 10, 2007 to September 5, 2007.

Figure 21.1 Hydrometeorological Site Location Map



Due to the lack of a suitable site for the installation of a hydrometric station on Tributary C, it was necessary to estimate creek flows for this basin based on the larger Tributary A discharges collected just downstream of the confluence of Tributary C with Tributary A. Discharges were measured manually for both creeks at similar times throughout the course of the study to determine a discharge relationship for the two creeks. The ratio of discrete discharge measurements between Tributary A and Tributary C is 0.308. The ratio of the catchment area is 0.302, providing confirmation of this method. The similarity of these two ratios validates the estimation of Tributary C discharges from Tributary A flows. This methodology enabled discharges from two separate basins, Tributary A basin and Tributary C basin, to be determined.

The average discharge for Tributary A over the two year period of record was 3.4 m^3/s. Both years of record had very similar average discharges. Short period increases in flow on the order of 10 m^3/s may occur during the summer, in response to large precipitation events. The maximum recorded summer flow on Tributary A was 10.5 m^3/s, recorded on July 25, 2007. Water temperatures recorded at the hydrometric station indicate that the average water temperature for the open water period of record was 5.6°C. Over the two years of data collected for Tributary A, creek temperatures varied over the range from 1 to 10°C.

The average flow on Tributary C over the period of record was 0.92 m^3/s. Both years on record had very similar average discharges. Short period increases in flow on the order of 1.5 m^3/s can occur during the summer, as a response to large precipitation events. The maximum recorded summer flow on Tributary C was 3.1 m^3/s which occurred on July 25, 2007.

Monitoring during May 2008 measured flow in both systems and identified a thermal spring in the Tributary C drainage, which suggests year round flows in this stream. A winter hydrology program was conducted during February and March of 2009 and the flows were observed in both tributaries at this time.

Total basin runoff over the period of record was calculated for the two creeks draining the Mactung property. Runoff during the 2006 and 2007 study period, as well as the mean discharge for Tributary C is summarized in Table 21.1.

Table 21.1 Total Basin Runoff for Tributary C in 2006 and 2007

Parameter	Tributary C	
Year	2006	2007
Mean Discharge (m^3/s)	0.90	0.94
Runoff (mm)	193	237

The construction of the Mactung mine will decrease the flows in Tributary A only during the period of filling of the Ravine Dam. At other times during the production phase of the project the flows in this tributary will be generally unaffected with the exception of the winter months.

The addition of a reservoir in the upper drainage basin is comparable to a lake headed drainage system. It is expected that there will not be significant changes in water temperature that could be observed as a result of the Ravine Dam; however, these temperature effects should be minimal based on the elevation and size of the reservoir.

The proposed Mactung mine is designed to operate on a year-round basis. The process water balance indicates a water surplus of 8 m^3/h reporting to the polishing pond. This will mean winter discharges to the Ravine Dam with overflow from this structure into Tributary C. No information is available to estimate the relative impact to surface water flows as a result of these discharges.

Low-flow analysis of the southern tributary of the Hess River, just downstream of the confluence of Tributary A was conducted using data recorded by Water Survey of Canada. Hydrometric data for the Tsichu and Hess rivers has shown that water withdrawal requirements for mining operations may exceed creek discharge, posing potential effects to downstream aquatic resources and wildlife. The percent occurrence of this low flow is projected to be less than 1%. Mitigation of potential effects from water withdrawal in this tributary will be managed through the setting of maximum allowable water withdrawals as a percentage of total streamflows. Limited data exists on the drainage of the proposed pumphouse; however, low flow monitoring from winter 2008 measured flows that were within the modelled range for this tributary.

A detailed hydrological program has been designed to conduct year-round monitoring on the southern tributary of the Hess River. The monitoring program will target the winter low flow period to determine the accuracy of the low flow modelling for the system. The detailed assessment of aquatic resources in the stream sections where water withdrawals are proposed to allow for evaluation of potential habitat effects resulting from proposed water withdrawal. The winter hydrology program of 2008/2009 will also provide information to determine the relative contribution of groundwater to streamflows, which is identified as an information requirement under YESAA.

Hydrological analysis of the Mactung site is complicated by the absence of active Water Survey of Canada hydrometric stations within 60 km of the site. The three stations closest to the property were operated from the mid-1970s to the mid-1990s; therefore, there is no direct comparison of the short-term onsite hydrometric data reported to the longer term record which would enable more accurate determination of hydrological parameters such as low- and high-flow frequency analysis and period of return for various flows.

The initial assumption of no winter flows for Tributary A and Tributary C was re-evaluated to reflect physical measurements. Determination of flood frequencies becomes difficult without long-term hydrology stations for comparative purposes. Modelling of drainage basin run-off response is being conducted to allow for the flood frequency determinations required to meet YESAA information requirements.

Indications from existing information show that water quality from the tributaries draining the property is of poorer quality than that of the southern tributary of the Hess River. Winter monitoring of flows and water quality in both drainage systems was recommended and undertaken to evaluate and quantify potential effects from water withdrawals during the low flow period.

Hydrological data to determine sizing of drainage structures for the proposed 40 km access road has been recommended for collection upon determination of stream crossing locations. This data may be of interest to assessment and regulatory agencies.

21.3.5 SURFACE WATER QUALITY

In June, July, August, and September of 2006 and 2007, EBA conducted water quality sampling at watercourses within the Mactung site including the Hess River and Tributaries A and C of the Hess River in Yukon, as well as Dale Creek, the Tsichu River, and Cirque Creek in NWT. During these sampling events, surface water samples were collected eight times from sampling stations WQ1, WQ2, WQ3, and WQ4 and twice from sampling station WQ9.

All of the stations were located on the Yukon side of study area; WQ1 at Tributary C of the Hess River, WQ2 at Tributary A of the Hess River, WQ3 on the Hess River upstream of its confluence with Tributary A, and WQ4 on the Hess River downstream of its confluence with Tributary A. WQ1 through WQ4 are on the western side of MacMillan Pass on watercourses which drain northwest into the Stewart River in Yukon. Station WQ9 was located on the South MacMillan River which flows into the Pelly River (Figure 21.2).

The results from these events combined with previous water quality studies completed for the Mactung site (CSMRI, 1976) and AMAX Northwest Mining Company Limited (1983) were interpreted for this water quality summary.

The project area contains large mineral deposits below and at ground surface. There are underground springs seeping in at various locations. These conditions influence the area's water quality; however, most parameter concentrations were found to be below laboratory detection limits. Exceptions included aluminum, cadmium, copper, extractable iron, iron, nickel, selenium, and zinc which were consistently found to be above the Canadian Council of Ministers of the Environment (CCME) guidelines for the Protection of Aquatic Life.

In general, higher trace metal concentrations were seen at stations WQ1, WQ2, WQ4, and WQ9 while Station WQ3 on the Hess River upstream of the confluence of Tributary A (and Tributary C) had lower metal concentrations. The elevated metal concentrations appear to be natural occurrences with no anthropogenic influences suspected.

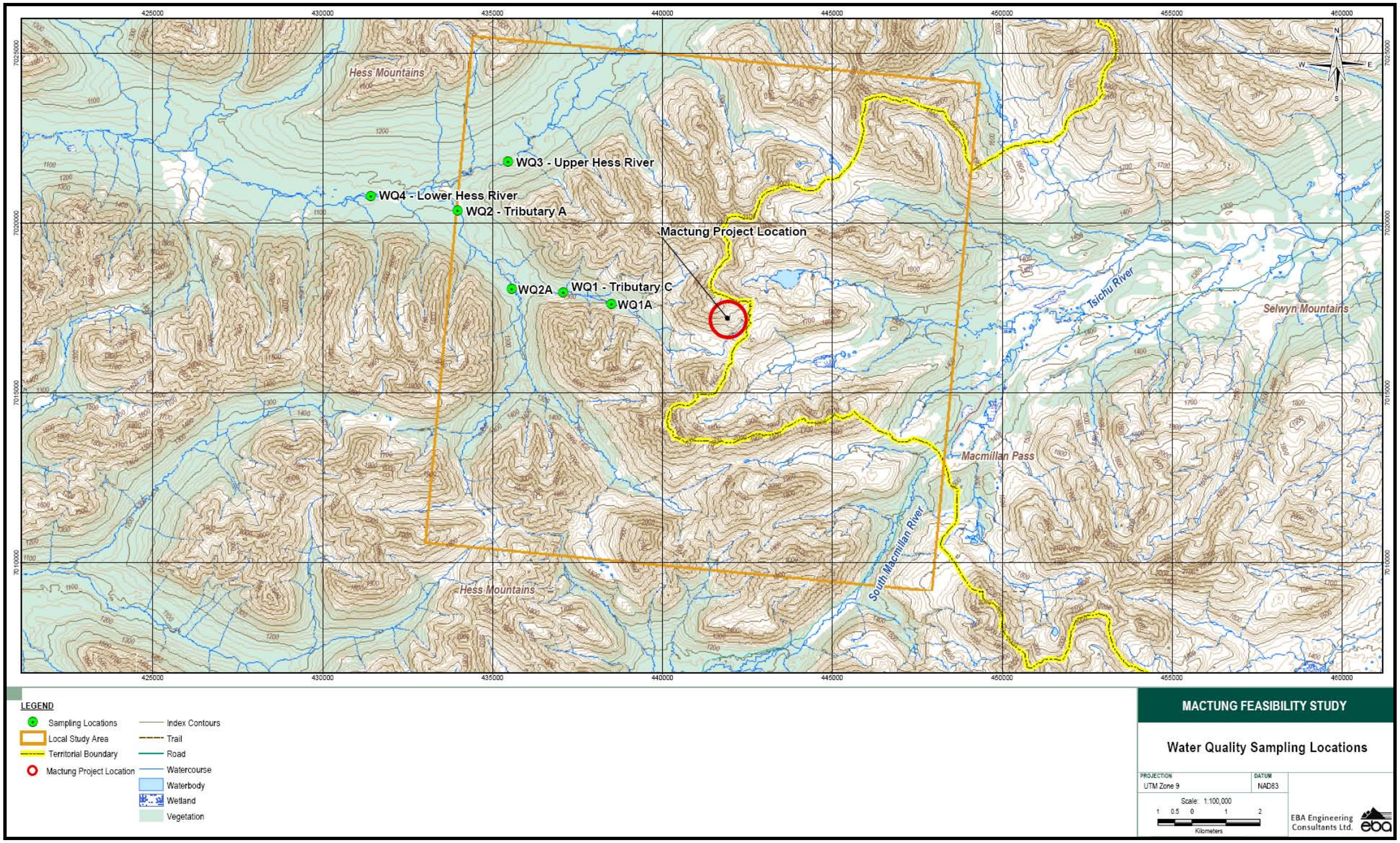
Tailings from the project will be deposited below the mine mill site. Thus, potential effects on downstream water quality, if any, would be greatest at Station WQ1 on Tributary C, due to its proximity to the project area and the direction of water flow from the proposed tailings location. If water quality in Tributary C were to be measurably affected, then Tributary A at Station WQ2 and the Lower Hess River Station WQ4 might also potentially be affected. Therefore, the collection of baseline data has begun for Tributary A prior to its confluence with Tributary C. The monitoring of the current water sampling stations will continue. Sampling of Tributary A upstream of the confluence with Tributary C was conducted during May 2008 and the results indicate that Tributary A has poor water quality.

Metals leaching from the tailings storage facility at the mine are deemed to have the highest potential for effects on receiving water chemistry in the project area. CSMRI (1976) evaluated effluent quality for a tailings sample from the site and this study showed that effluent chemistry is not anticipated to have negative effects on the surface water receiving environment; however, water quality guidelines have changed since this study was conducted. Relative changes to surface water parameters as a result of discharges from the Mactung site can be calculated once more detailed hydrology and water chemistry data for the winter months is available.

There is a potential for changes to surface water flow and water chemistry in the southern tributary of the Hess River downstream of the confluence with Tributary A as a result of water withdrawals. Winter flow and water quality monitoring are required to be able to evaluate the potential changes and determine whether mitigation measures may be required.

Water quality data for the Mactung project currently only extends over the approximate 4 month open water period for each of the two years of record. These summer months include periods of large dilution from snowmelt and precipitation events. At the time of preparation of the Feasibility Study, data was being collected to better understand low flow periods, as this information will be required for a project proposal submission to YESAA. Continued water quality from 2008/2009 monitoring program will be used for the development and implementation of mitigation strategies where these are required.

Figure 21.2 Water Quality Sampling Locations



21.3.6 HYDROGEOLOGY

A Detailed Hydrogeological Assessment (DHA) was conducted by EBA for NATC to assess the groundwater conditions at the Mactung property. The purpose of the DHA was to provide enough background information for the description of the existing (baseline) hydrogeological conditions and to present a hydrogeological effects assessment for the YESAA process.

The project area is bordered by surface water divides to the north, east, and south. The regional groundwater flow divide is assumed by EBA to coincide with surface water divides (i.e. groundwater from the project area discharges to the valley of Tributary C south of Mt. Allan).

The groundwater flow direction generally mimics the slope of surface topography. Deep groundwater flows from the highest areas of Mt. Allan southwards, turning southwesterly or westerly in the valley of Tributary C. Shallow groundwater flow within the overburden is characterized by local, small-scale flow cells, with its flow direction closely following local topography. Groundwater recharge typically occurs at higher elevations with groundwater ultimately discharging to surface water bodies at lower elevations in valleys. The presence of permafrost in the upland areas tends to reduce and/or inhibit infiltration to groundwater. At lower elevations where permafrost is discontinuous or absent, groundwater recharge to the deep aquifer occurs where a hydraulic connection exists between the shallow overburden and underlying bedrock aquifers.

The horizontal hydraulic gradient in the bedrock aquifer ranges from about 0.02 (i.e. 2 m vertical per 100 m horizontal) in the upland areas of Mt. Allan to about 0.5 in locations south of the proposed mill site. The hydraulic gradient in the valley south of Mt. Allan appears to be fairly constant at 0.1 to 0.15. Based on the hydraulic conductivities obtained from hydraulic tests, the hydraulic gradients inferred from groundwater level measurements, and assuming a bedrock porosity of about 0.05 to 0.15, the mean flow velocity of the deep groundwater is estimated to range from about a few metres to several tens of metres per year.

Natural background concentrations for aluminum, cadmium, iron, selenium, and zinc slightly exceeded the Canadian Council of Ministers of Environment guideline values for the protection of aquatic life. However, all concentrations were below the Yukon Contaminated Sites Regulation aquatic life standards.

UNDERGROUND WORKINGS

Measurements of static piezometric levels in observation wells installed during the detailed hydrogeologic assessment indicate that a portion of the underground workings will occur below the existing static piezometric water level. Hydraulic testing results suggest that the rock mass where underground workings are planned to be advanced has a low to moderate hydraulic conductivity. It is expected therefore that there will be some drainage into the mine from the surrounding rock mass, and a resulting requirement for dewatering during later mining. Further, because a portion of the mine will be wet, the interaction of sulphide minerals in the open drifts and adits of the mine may create the potential for acidic mine drainage and metals leaching.

Groundwater seepage is anticipated to occur in the underground workings below an elevation of about 1775 masl. Mining at these depths will occur only on the west end of the deposit and will take place only from Year 5 to Year 11 of mine operation. A consequence is that water flowing into the mine may create waters with elevated metals concentrations that must be handled during mining operations.

Because the reservoir will largely be infilled with oxygenated, near neutral pH surface water diverted from the catchment areas upslope, dissolved metals will precipitate as/or adsorb onto mineral surfaces (e.g., iron oxyhydroxides), and further metal transport through surface water movement from the reservoir will be mitigated. With this change in redox conditions and pH, precipitation of some metals would likely occur within the timeframe that the water is retained within the reservoir. As part of the mine closure, once the dam is decommissioned, sediments that had settled out from the base of the reservoir and containing high metals concentrations will be relocated and disposed of on the DSTF.

During mine dewatering, the base flow to some creeks with headwaters on Mt. Allan will also be diverted towards the underground workings. Comparison of these volumes to the estimated flow into the mine suggests that the mine drainage water will amount to less than 1% of the volume in the reservoir (annually), and less than 2% in low flow (winter months).

Mined-out stopes and drifts will be backfilled with tailings. After dewatering has ceased, the piezometric level will start to recover to its natural condition. As a result, the lowest part of the underground workings below an elevation of about 1775 masl will flood and groundwater will move through the abandoned underground workings following the natural hydraulic gradient to the south towards Tributary C. Stope and drift walls as well as backfill materials will have been exposed to atmospheric oxygen during the operation phase and sulphide minerals may have started to oxidize. This provides the potential for the generation of acid rock drainage at least during the initial phase of flooding when oxygen will be available.

However, after flooding, oxygen availability will be very limited due to the fact that all underground workings will be sealed off with bulkheads preventing any air circulation from atmosphere. Furthermore, the groundwater flow through the mine will contain no significant amounts of oxygen and chemical conditions will be reducing. Therefore, potential acid generation and associated metals leaching will widely be restricted to the initial phase of flooding of the underground workings. If acid generation and metals leaching does occur, there may be effects to groundwater that will flow southwards and ultimately discharge into Tributary C upstream of the ravine dam.

The overall significance has been identified as being low because acid generation and metals leaching will be very limited due to the low oxygen content in water within the flooded underground workings; and, the potentially affected groundwater represents only a small percentage (<1% to 3%) of the total discharge of Tributary C at the point of surface water discharge.

DRY STACKED TAILINGS FACILITY

Water drainage through the active layer of the tailings during the warmer months may affect shallow groundwater quality during tailings placement. The primary interface between leachate (process water, and/or precipitation that has infiltrated the dry-stacked tailings) and groundwater at the DSTF occurs primarily at the south slope of the DSTF where water leaching from the active layer of the tailings has the potential to mix with groundwater. The southern part of the DSTF area is a groundwater discharge area with an upward groundwater gradient expected. The upward gradient is expected to limit percolation of leachate from the tailings to the shallow groundwater flow system. The potentially acidic and metals-rich water generated within the tailings will flow preferentially to the south end of the DSTF through the active layer at the upper surface of the stack. Any seepage or leachate discharging from the DSTF will flow on surface, or in shallow groundwater towards Tributary C, and during operation, will ultimately discharge to the reservoir. This seepage is expected to have chemistry similar to the process water.

An impermeable geomembrane will be placed over the DSTF at the end of operation. This will greatly reduce or eliminate the risk of infiltrating water transporting acidic drainage or metals leachate post closure (after Year 11). The water quality in Tributary C will not be affected significantly and the effect is unlikely to represent any threat for fish habitat further downstream in Tributary C where dilution will be much greater. As well, the effects on groundwater are reversible due to the natural flushing of groundwater that will occur through the area of the DSFT after closure. The overall significance of the effect of water seepage from the DSTF on the water quality of Tributary C after mine closure is low.

WASTE ROCK STORAGE

Groundwater recharge through the PAG waste rock stored at surface near the mine portal will only occur seasonally. Since these materials will only be stored on surface for less than five years, acidic recharge is not expected. If acidification of the waste rock occurs more rapidly than expected, accelerated placement as mine backfill is the most prudent mitigative option to limit the exposure of the PAG source to oxygen and moisture. With this mitigative strategy, the potential effect on shallow groundwater quality will be minimal.

Reservoir Dam Underflow

The reservoir will collect groundwater discharge, mine drainage water, mill process water, and water directed to the reservoir from selected surface water catchments. The location of the proposed reservoir is currently a groundwater discharge area, and will remain a groundwater discharge area during the time that the reservoir exists. The reservoir dam will be constructed upon fractured bedrock which will allow some groundwater seepage under the dam. This underflow will mix with the underlying groundwater and potentially change the quality of the shallow groundwater.

The rate of dam "underflow" has been predicted by EBA using a scoping 2D flow model to be in the order of 500 to 750 m^3/d. It is assumed that groundwater underflow will be very similar in chemistry to the surface water quality contained within the reservoir. In the event that the reservoir water quality does not meet discharge criteria, mitigation to prevent the uncontrolled discharge of seepage water has been formulated and is included in the YESAA project proposal.

21.3.7 Climate

Data from two existing meteorological stations in the Mactung project area were used:

- A meteorological station was installed during the summer of 2005, approximately 50 m south of the existing Mactung camp site (Figure 21.1). Weather data recorded by this meteorological station have been analyzed for a period of record of over two years from July 15, 2005 to August 31, 2007.
- The MacMillan Pass meteorological station, which is operated by Meteorological Services of Canada from 2003-2005.

The summer period at the Mactung site runs from June to late August and the remainder of the year may be classified as winter, with transitions occurring between April and May and between late August and September.

Mean summer air temperatures are typically between 5°C and 10°C, with daily maximums around 15°C and minimums around 5°C. The maximum air temperature recorded by the station over the study period was 20°C

Winter temperatures have more day-to-day variance, but typically range between -10°C to -20°C. The minimum air temperature recorded at the site over the period of record was -36.6°C. During the winter months, air temperatures rarely rise above freezing.

Winds at the site come predominantly from the west/southwest and the northeast, with average wind speeds less than 6 m/s occurring approximately 90% of the year. Wind speeds greater than 6 m/s tend to occur more often during the winter months.

Average daily maximum wind gusts are typically 7 m/s; however, the maximum wind gust recorded by the station for the period of record was 23 m/s. Relative humidity is typically near 90% throughout the year, but frequently drops as low as 30% for periods of up to a day.

The site receives the highest amount of incident solar radiation between April and August. During this period, maximum incident solar radiation is in the order of 900 W/m^2. In the months of December and January, daily maximums are approximately 50 W/m^2. There are about 20 hours of daylight on June 21 (summer solstice), while daylight is restricted to approximately 4 hours on December 21 (winter solstice).

Precipitation data recorded at the MacMillan Pass meteorological station, provided by the Meteorological Service of Canada, have been used to estimate yearly precipitation at the Mactung property. The MacMillan Pass station is located on the lee-side of the mountains (481 m lower in elevation), approximately 1.6 km southeast of the Mactung property. An analysis of the data showed that 663 mm of water-equivalent precipitation per year fell on the area over the period 2003 to 2005. Months with greater than 50 mm of precipitation commonly occur throughout the year, with January seeing the least, typically under 30 mm. Based on temperature data recorded at the Mactung property, this precipitation could be expected to fall as snow from October to April. Precipitation occurring in September and May has the potential to fall as snow, freezing rain, rain or as mixed precipitation.

A complete record of monthly and daily precipitation at MacMillan Pass exists between August 2002 and June 2006. The precipitation record outside of this period is sparser. In addition, due to its location on the lee-slope of the divide, recorded precipitation totals may not be completely indicative of conditions at the Mactung site. As a result, it is difficult to accurately assess the study area basin runoff or potential for flood. An onsite precipitation gauge is planned to be installed for the collection of relevant precipitation data.

The MacMillan Pass meteorological station collects data on snow depth; however, no data with respect to snow depth has been collected for the site. Field observations at the site during March 2008 indicated that the influence of wind on snowpack distribution makes snowpack highly variable. Heavy drifting was noted in the area of the camp while other areas near to the proposed Mill building showed little to no snow cover.

The majority of the potential project related risks described in this section can be addressed during the detailed engineering of the project infrastructure.

Wind speeds up to 23 m/s have been recorded on site; therefore, there is a potential for minor wind damage to structures. There is also a possibility of the transport of contaminated sediment fines from the mine or tailings storage facility during dry periods to areas outside of the property. Aeolian transport from the tailings facility can be mitigated through the use of sprinklers to maintain surface moisture.

Precipitation could pose a potential risk with respect to increased potential for erosion and sediment transport in ditches and diversion channels at the site, including the ditchlines along the proposed access road to the site. This impact can be mitigated through proper ditch and diversion design relating to anticipated flow volumes and rip-rap requirements. Revegetation, where moisture conditions allow, can also be conducted to mitigate impacts from erosion and provide sediment control.

Cold winter temperatures pose a risk with respect to freezing water lines used for plant feed and discharge lines, potable water and septic systems. Ground temperatures readings from the site will be used to assist in service designs. The use of adequately insulated or heat-traced water lines will need to be installed as appropriate to mitigate this concern. Health and safety issues related to cold temperatures for mine operation personnel will be mitigated through development of operational procedures for the production phase.

Currently there is no on-site information for precipitation or snowpack distribution for the Mactung site. The highly variable distribution of snow at the site makes determination of site values for this parameter difficult. The application of an orographic modifier to the available data from the MacMillan Pass Atmospheric Environment Service (AES) station can be conducted to reasonably estimate site values for both precipitation and snowpack.

Climate change effects were considered for the submission of the YESAA project proposal, as per the information requirements.

21.3.8 *Air Quality*

The proposed Mactung mining area is located in a remote area of Yukon that has no other sources which would affect the air quality. As a result, it is assumed that the air quality in the area would be in the same range as background levels for the Canadian north.

There is currently no air quality monitoring equipment in the area of the mine site or any other adjacent areas. The only ongoing air quality monitoring in Yukon is conducted in Whitehorse. The Whitehorse station is part of the National Air Quality Surveillance (NAPS) Network. The air pollutants monitored in Whitehorse include carbon monoxide, nitrogen dioxide, nitric oxide, ground level ozone, and fine particulate matter (PM_{25}). The monitored ambient air pollutants in Whitehorse are compared with the National Air Quality Objectives (NAQOs), and summary reports for 1998, 2000, 2001, and 2004 were reviewed for this station. Overall, the reports conclude that the ambient air pollutant levels monitored at the Whitehorse NAPS station are good and rarely exceed the levels specified in the NAQOs.

A direct comparison cannot be made between Whitehorse ambient air pollutant levels and those at the Mactung mine site. Some air emissions will be created from fuel-fired generator sets and traffic at the mine site. However, due to the location of the mine site, it is assumed that the ambient air quality at the site will have lower concentrations of air emissions than in Whitehorse.

Based on the remoteness of the mine site location, it is not believed that air quality will pose a risk to the feasibility of the proposed Mactung project.

21.3.9 NOISE

The proposed mine site is situated in a remote area where the background noise consists of the natural setting. Some noise is present during some intermittent mining exploration and mine planning activities. Noise levels at the mine site have not been monitored but are commensurate with regular exploration activities. Ambient noise is expected to be mainly of a temporary nature during the construction and operation phase and will only impact the immediate vicinity of the above-mentioned activities. The daily and long-term averages for ambient noise in the area are anticipated to be low.

Based on the remoteness of the mine site location, it is not believed that noise will form a risk to the feasibility of the proposed Mactung project.

21.3.10 ARCHAEOLOGY

In 2006, Points West Heritage Consulting Ltd. (Points West) conducted a preliminary archaeological assessment of the Mactung site. The primary objective was to determine if there was sufficient archaeological potential to require more detailed investigation once finalized development plans are available.

The results of this study identified the areas that may have sufficient archaeological potential to justify further archaeological investigation. Areas that would not require further work were also identified. The report predicted that any archaeological sites located in this project area would be small. Further, it is most likely that they would be characterized by sparse to moderate quantities of artifacts. However, because little is known of the archaeology of the region, any information that can be collected would be of value.

Points West continued its work in August 2007, with the primary objective to follow-up on the recommendations of a 2006 preliminary assessment. The proposed mine facilities will be located in a high west facing alpine valley on the Yukon side of the border; and as a result of this study, is considered to have very low archaeological potential.

No new archaeological sites were encountered during the 2007 study. Little to no archaeological potential was found in the majority of the area proposed for mine facilities in the upper alpine valley area within Yukon. No archaeological sites were discovered and there are no previously recorded sites in this area.

The new access road approach within Yukon from the vicinity of the MacMillan Pass airstrip, and this airstrip, were not included in the 2007 study. These two areas were instead subject to an additional archaeological assessment in July 2008. The results of this study were not available at the time of finalizing this report.

Subject to the results of the 2008 study, no conflicts between the Mactung development and archaeological sites have been found, and there appears to be no archaeological issues that will impact the feasibility of this project.

21.3.11 HUMAN AND ENVIRONMENTAL HEALTH

There are no full-time residents in the area, and there is limited use of the area for recreational purposes. Traditional activities are expected to occur in the area; however, the timing and extent to which these will occur is unknown.

The predominant activities known to have occurred for the past two years have been centered on exploration activities and scientific research as a result of the exploration activities and plans. Once construction activities begin there will be a camp located at the site which will be used to provide temporary accommodations for workers involved with project development. Camp facilities are expected to be provided from construction through to operation and eventually for closure and reclamation.

Yukon does provide a regulatory regime which will monitor site conditions and inspect monitoring reports from the site. As well, Yukon government's Environmental Health Services will be responsible for the issuance of permits and monitoring of human and environmental health.

Human and environmental health is important matters which will need to be considered throughout all stages of the project and will be regulated by Yukon government. The project's impact on human or environmental health conditions appear to be minimal.

21.3.12 VEGETATION

The types of vegetation present and vegetation cover are highly variable in the Mactung Vegetation Local Study Area (VLSA) due to elevation, aspect, microtopography, and soil conditions. Valley bottoms tend to be vegetated by willow and scrub-birch thickets, wetlands, and sedge-forb meadows. Black and white spruce communities are rare in the Hess Tributary. Subalpine fir dominates the wooded taiga from 1200 to 1550 masl at higher elevations the canopy becomes sparse and is typically replaced by Krummholz and dwarf shrub communities in the alpine. At elevations above 1800 masl, vascular plants become rare and bare rock and epilithic lichen communities dominate.

Two studies of special interest provided a detailed description of the vegetation of the area. The Amax (1983) and Kershaw and Kershaw (1983) reports reviewed all physical, biological, and cultural aspects of their identified study area (500 km^2). The vegetation portion of these reports summarized both the existing vegetation and plant communities and potential environmental impacts associated with the construction of the proposed mine at that time; and further identified 17 plant communities within their study area. The reports also stated that upon review of rare species lists from Yukon, four species were to be considered rare with a potential to occur in the area. None of the four plant species were identified or recorded in areas planned for proposed construction at that time.

More recently, new biophysical assessments were conducted by EBA in 2006 and 2007 entitled "Vegetation Environmental Baseline Studies" and "Vegetation and Ecosystem Land Classification" respectively (EBA 2007a; 2007b).

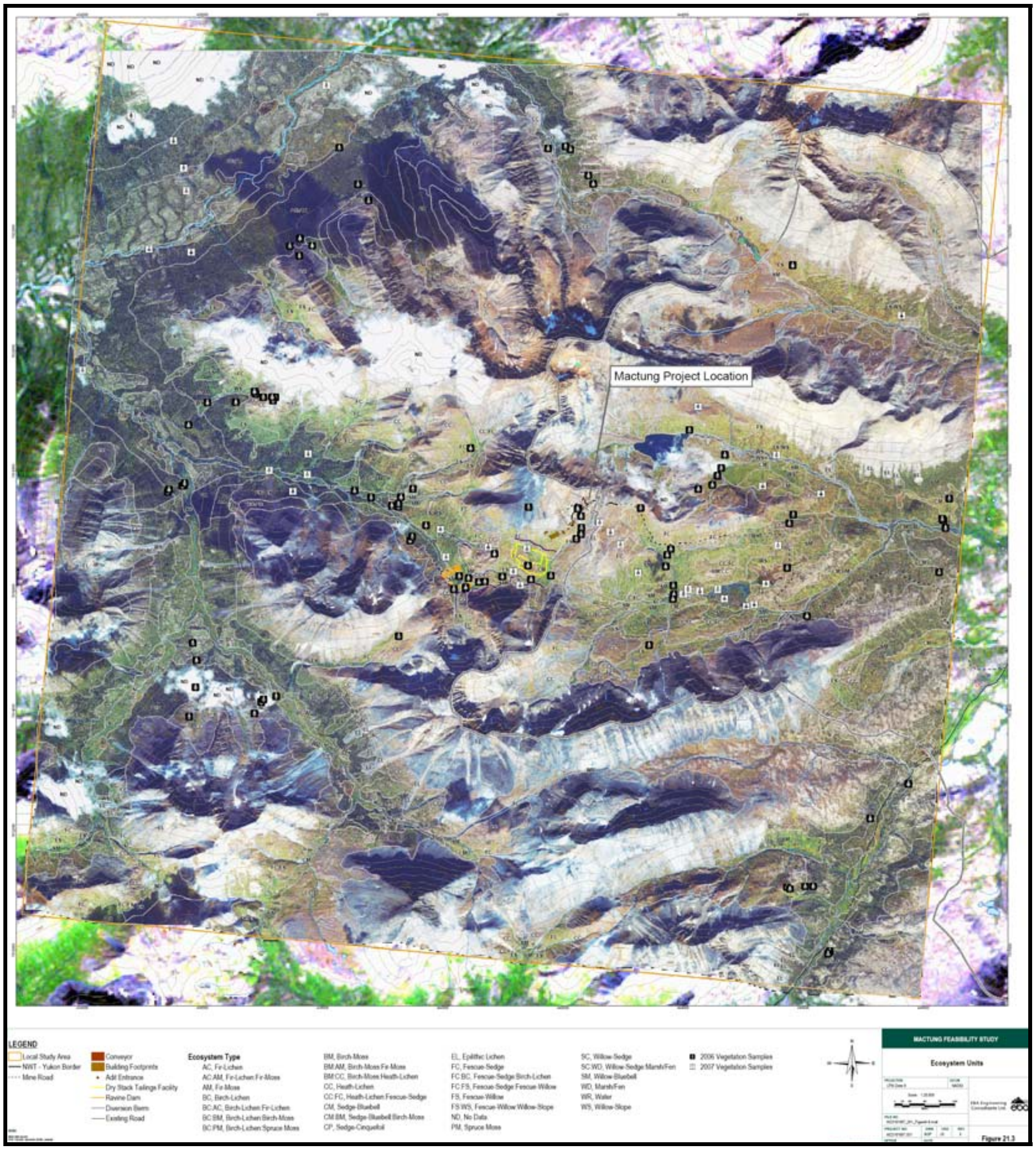
Fifteen distinct ecosystem units/map units and 12 complex polygon associations were mapped based on the sampling performed by EBA in 2006 and 2007 (Figure 21.3).

A total of 145 plant species were observed within the Mactung VLSA. One hundred twenty-three different plants were recorded in 2006, with an additional 22 observed in 2007. Twenty-nine of these recorded plants were identified only to genus and are assumed to be different than those identified to species. A list of plant species expected from historical studies is presented in the Vegetation Baseline Study produced by EBA.

A rare plant survey was completed over a two year period as part of on-going environmental baseline studies. A three day rare plant survey was conducted from August 13 to August 15, 2007 in the proposed disturbance footprint for the Mactung Project, with two days of effort in Yukon and one day of effort in the NWT. A rare plant reconnaissance (functionally different from a rare plant survey) was conducted and 14 vegetation plots were sampled within the area of the proposed footprint during the 2006 season. Data collected in July 2006 were used to supplement the 2007 rare plant survey.

In 2007, no rare species were observed in areas proposed for development. A baseline assessment of trace element concentrations in plant tissue was performed within the LSA. A total of 14 vegetation samples were collected with two samples collected at each of seven sample locations. Data were collected to establish baseline values for trace element concentrations in plants in and around the project area. The results indicated no exceedance of any existing Canadian standards and values are only elevated with regards to within treatment values.

Figure 21.3 Ecosystem Units



In review of the existing vegetation baseline requirements of the YESAA proposal guidelines, as well as the existing data there appears to be low risks to the project for development proposed to occur in the mine footprint. Site clearing and construction will impact small amounts of alpine plant communities in specified areas, however, given their common occurrence throughout the Mactung region, these plant communities are expected to return following reclamation.

Additionally, no forested areas occur within the proposed development area; therefore, no risks related to the loss of merchantable timber exist. The information collected in their 2006 and 2007 baseline studies will support future regulatory submissions leading to project approvals and have not identified any outstanding risks or impacts that may affect the feasibility of the mine footprint development.

The project will include the development of a 48 km all season access road through lower elevation valleys to the southwest of the project area. The road may traverse heavily vegetated habitat in the boreal high and subalpine ecozones, approximately two thirds of which would lie outside of the area covered by the 2006 and 2007 baseline studies. A 2008 field program has recently been completed which catalogued and assessed vegetation resources along this route and an inventory report was used for the YESAA application.

21.3.13 WILDLIFE

EBA conducted an extensive baseline study program at the Mactung project area with the objective of documenting and characterizing wildlife within the study area. Both ground and aerial surveys were carried out in October 2005 and continued during the summers of 2006 and 2007 (June, July, August, and September survey events) (Figure 21.4 and Figure 21.5). These surveys may also form the basis for future monitoring programs associated with project implementation and operation. Baseline surveys were carried out in a 720 km^2 wildlife study area that was centered on the existing Mactung camp.

Figure 21.4 Wildlife Observations in the Study Area

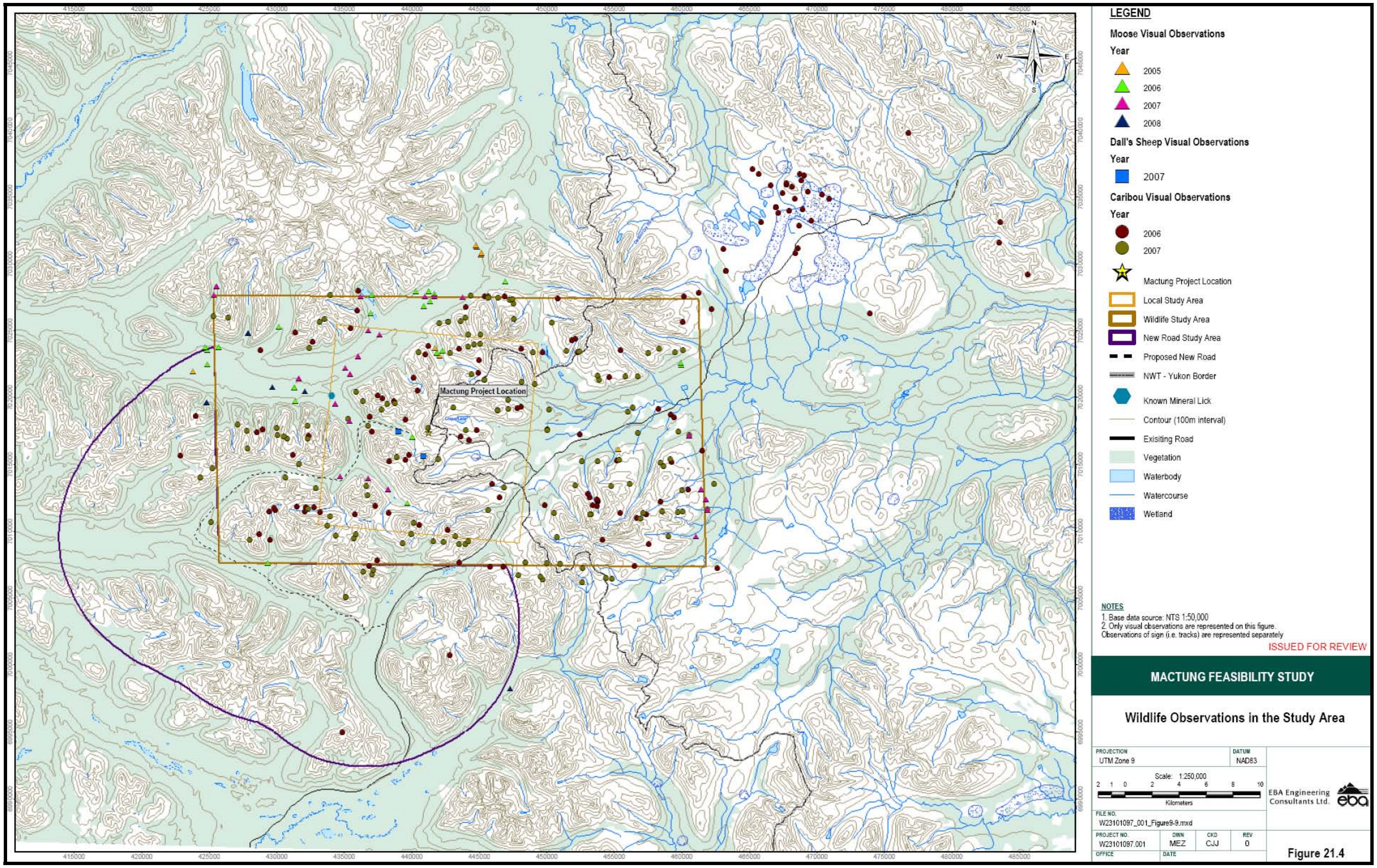
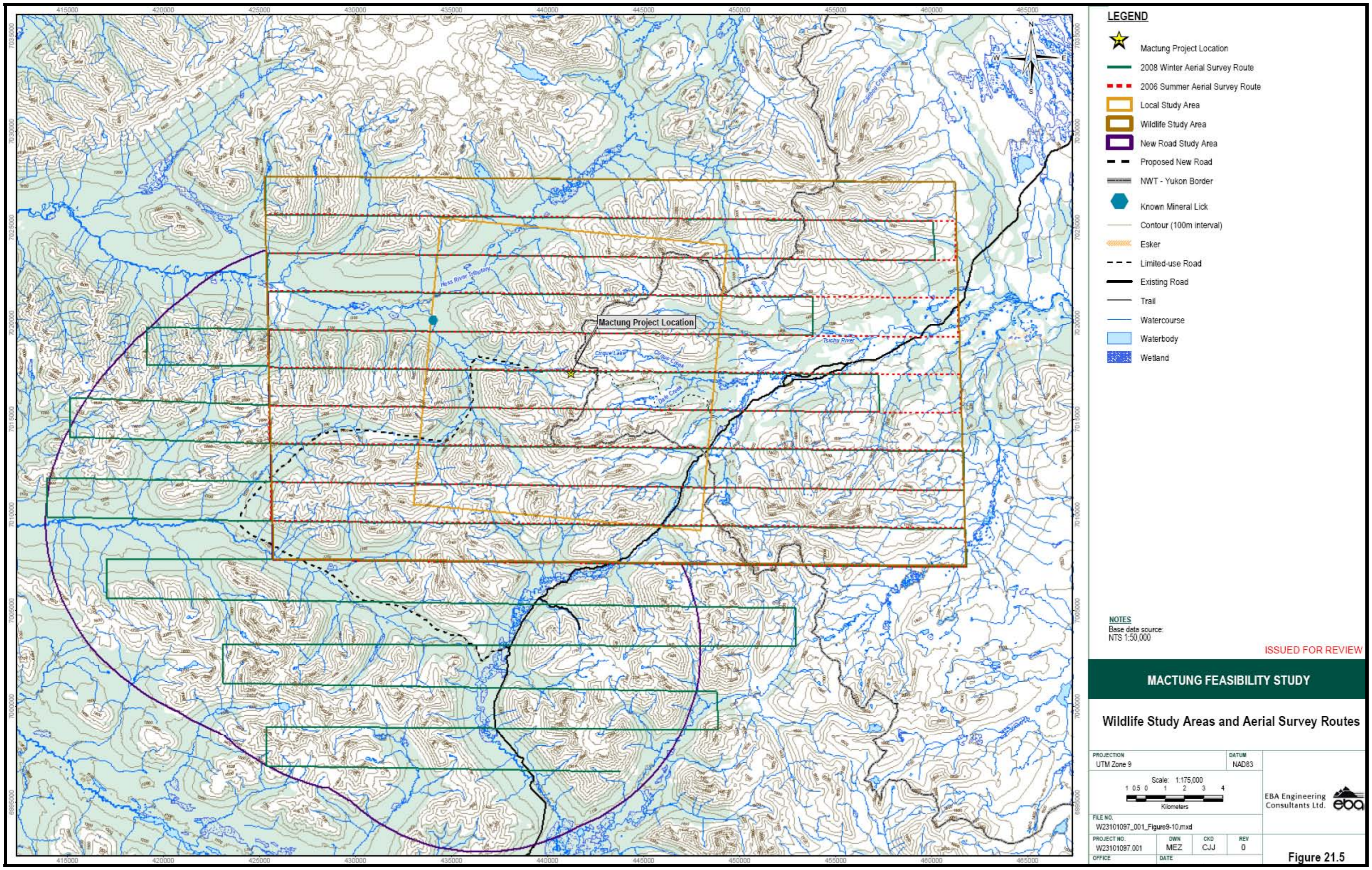


Figure 21.5 Wildlife Study Areas and Aerial Survey Routes



Overall, wildlife use of the primary footprint area was found to be consistent with other areas in the region. Consequently, disturbances to wildlife are expected to range from low to moderate in scale, and highly localized in spatial scope. Potential disturbances may include limited effects on an active wildlife corridor between valleys in Yukon and the NWT that connect across the pass situated in the project footprint In addition, potential habitat losses for other species local to the development area may arise. Such potential effects may be of concern to regulators and other stakeholders. However, with the application of appropriate mitigation measures any potential effects are expected to be manageable.

Woodland Caribou are likely to be the species of greatest interest during the YESAA assessment process, as many herds of caribou across the North, have been slowly declining due largely to the effects of encroaching human development. Although the Redstone herd is currently believed to be stable, assessors and managers will be interested in predicting effects of future developments. Mitigation measures have been included within the YESAA application. Monitoring programs to minimize potential effects may be required as part of the permitting process.

Moose are often considered to be a 'bread and butter' species in Yukon, and wildlife managers tend to be concerned about the indirect effects of development on this species. Mitigation measures have been included within the YESAA application.

Both Dall's sheep and grizzly bears are also species that are considered of high value. While the study area does provide good habitat for grizzly bears, this species is secure in the region, and should not pose a project risk. The relative abundance of sheep in the project area remains poorly understood, mostly due to the limited winter abundance data available for this species.

While several other species were identified as occurring within the study area, no other keystone species or species at risk were identified. EBA believes that there is sufficient baseline data from recent studies to properly assess the risks to these species, as well as to develop effective mitigation strategies.

To address the limited winter abundance data, additional baseline work was conducted during March 2008, as well as further work in the summer of 2008, to gain a better understanding of abundance and distribution of this key species. This work will provide a better understanding of consequences related to the project.

Similar to all other disciplines, the risks associated with access to the Mactung site during all phases of development and operation is anticipated to be addressed in regulatory applications. Access considerations for the Mactung project were not originally included in the baseline studies, as they had not been identified at the time these studies took place. As a result further baseline studies have been undertaken in 2008 to provide for the information requirements held under YESAA. Based on these studies, general wildlife usage and concerns were found to be similar to those in the area of the mine site. Also considered within the submission were effects associated with increased hunting pressure, direct and indirect habitat avoidance in adjacent areas, and wildlife mortality, as well as potential mitigations.

21.3.14 FISHERIES AND AQUATIC RESOURCES

FISHERIES RESOURCES

Fisheries baseline studies were collected during 2006 and 2007 in the Mactung study area. These studies focused on assessing the fisheries and aquatic resources in both the primary tributaries leading from the project area, as well as those higher order watercourses further downstream. Generally, these studies were conducted within the project LSA with the intent of characterizing the presence and characteristics of local fish populations, documenting habitat quality, identifying barriers to upstream fish passage, and identifying the fish bearing status of watercourses in direct influence of the project.

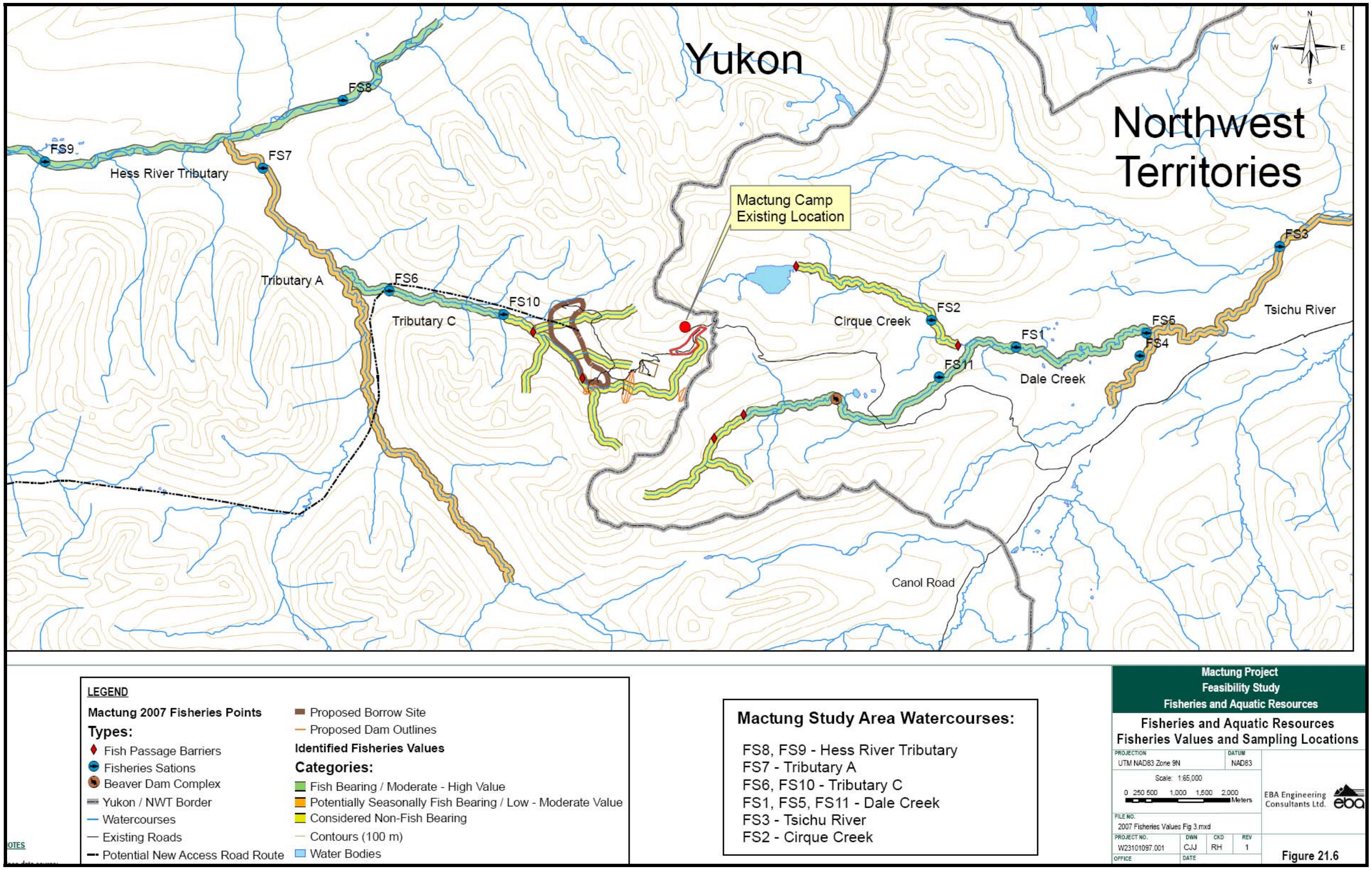
Prior to the current baseline studies by EBA, no previous information had been collected for the local tributaries on the Yukon side of the Mactung property. The current baseline study program was conducted by EBA and it focused efforts primarily within the greater project footprint and downstream impact areas on the Yukon side of the border, namely Tributary C (the primary tributary leading from the footprint area), Tributary A (a lower elevation tributary into which Tributary C flows), and the Hess River Tributary (into which Tributary A empties).

The baseline studies determined the major tributaries downstream of the project area to be fish bearing. Tributary C, the primary tributary flowing out of the footprint area, was found to support a resident population of dolly varden[1] at lower elevations and gradients. In these areas, Tributary C was found to provide good fish habitat conditions and is believed to support populations throughout the year. Favourable habitat conditions deteriorate closer to the Mactung Project area at higher elevations, and a sampling effort approximately 700 m downstream of the proposed project footprint determined poor conditions and an absence of fish in this area. It was also determined that areas upstream of this point were non fish bearing due to the numerous barriers to fish passage and the prevailing steep gradients.

Further downstream from the project area, Tributary A was found to not support fish at the time of baseline surveys, presumably due to poor water quality. This tributary, however, is likely to offer seasonal migration or other habitat potential due to the moderating effects of runoff water in the system. Finally, the Hess River tributary was found to provide excellent characteristics for year round fish survival, and supports slimy sculpin, arctic grayling, and dolly varden. Figure 21.6 presents the fisheries values and sampling locations.

[1] The dolly varden (*Salvelinus malma*) is a member of the charr family that is a common resident of fluvial systems in the Yukon. This fish occupies watercourses from large to small, and does revert to an alternate “dwarf” life history form that occupies high elevation streams.

Figure 21.6 Fisheries Values and Sampling Locations



Feasibility Analysis

Overall, risks to fisheries resources resulting from the mine development are expected to be moderately low and manageable through the implementation of appropriate and effective mitigation measures for the release of deleterious substances and maintenance of site hydrology. Generally, no direct effects (loss of habitat) are expected to result from the development or operation of the Mactung mine, as the footprint is at a high elevation and above steep gradient areas that naturally exclude fish. As it has been determined that resident fish populations exist directly downstream of the project area, the maintenance of downstream habitat and water quality will be a requirement, through appropriate mitigative measures, to ensure that fish habitat quality and fish populations are maintained.

It is anticipated that streams directly affected by project developments (within the development footprint) will be properly re-aligned to maintain downstream hydrological characteristics.

The development, upgrading, or maintenance of roads can result in potentially negative effects on fish and fish habitat, in particular at watercourse crossings. As expected, the development of suitable new mine access road is an important aspect of the mine feasibility. Thus, the proposed development of the new road will need to consider lower elevation tributaries in the area outlined for development.

Baseline information regarding the extent, quality, and characteristics of fisheries resources along potential access road route has been recently collected and preliminary results suggest that some tributaries along the route are fish-bearing. Consequently where the installation of a crossing structure may result in the loss of fish habitat, an Authorization pursuant to the Federal Fisheries Act will be required. For the permitting and authorization stage, a field habitat inventory and habitat compensation plan prepared by a professional biologist may be required.

AQUATIC HABITAT

During the aquatics baseline studies at the Mactung site in 2006 and 2007, a survey of benthic invertebrate resources was conducted in conjunction with fisheries sampling. Overall, the diversity and abundance of benthic resources was found to be closely tied to water quality characteristics (and consequently fish presence/abundance). Sampling in both years focussed on Yukon streams, and provided a suitable baseline of information for use in both regulatory submissions and future biomonitoring programs.

Based on a review of YESAB's "Proponent's Guide to Information Requirements for Executive Committee Project Proposal Submissions", that all listed information requirements can be met with the current baseline information. Some temporary or permanent direct loss of benthic resources can be expected during development and operation through the realignment of several primary tributaries within the lower footprint area or through potential water quality impacts. These streams have not been included in EBA's baseline studies to date as NATC was still working to define the project specific components. However, while these tributaries are expected to support some local primary production and to supply downstream areas with drift invertebrates used by fish, their relative contribution to the system in whole is predicted to be minor.

21.4 SOCIO-ECONOMIC BASELINE CONSIDERATIONS AND RISKS

This section of the report for the proposed Mactung Project considers the socio-economic conditions for the development. As the project will be located solely in Yukon, the socio-economic effects of the project were considered under YESAA. The "Guide to Socio-economic Effects Assessments" produced by YESAB identifies the following six steps to a socio-economic effects assessment:

1. Determine project scope.
2. Determine assessment scope.
3. Compile baseline data.
4. Characterize potential effects.
5. Evaluate mitigation and enhancement strategies.
6. Determine the significance of potential effects.

Each of the above steps relates to late identification of "Valued Socio-Economic Components" (VSECs). The 2006 Census provided much of the baseline data required.

21.4.1 SOCIAL AND ECONOMIC CONDITIONS

In the Yukon, three First Nations (RRDC, LFN, and NND) and the communities of Ross River, Faro, Watson Lake, and Mayo are most likely to be economically and socially affected by the Mactung mine. The sections below summarize some of the key economic and social conditions of these communities and First Nations.

LAND CLAIMS IN THE YUKON

In 1992, the Council for Yukon Indians (today the Council of Yukon First Nations) and the federal and territorial governments signed the Umbrella Final Agreement to serve as the template for individual land claims agreements. The land claims agreements are accompanied by self-government agreements that allow First Nation governments to pass legislation in a number of areas. The Mactung site is located within the Traditional Territories of the LFN, the RRDC, and the NND. The NND in Mayo was among the first four land claims signatories. However, significantly for this project, the RRDC and the LFN have not signed the agreements and remain Indian Act bands.

YUKON

The Yukon is large in area (483,450 km^2) but has a small population (32,714 inhabitants as of December 2007). The territory's economy is heavily reliant on government and, to a lesser extent, on natural resources and tourism; a pattern that is typical of relatively remote jurisdictions across the circumpolar north. Although Yukon has a variety of economic and industrial sectors, government is by far the largest and is largely financed through federal transfers.

Until the 1990s, mining was the mainstay of the Yukon's private sector industry. The dependence on mining gave rise to boom-and-bust cycles typical of economies dependent on natural resources. Until this century, the size of the non-aboriginal population and of the economy fluctuated with the fortunes of the mining industry. The current importance of government, and of mining in the past, has resulted in Yukon having high average incomes, typically among the highest in Canada.

The Yukon's mining industry entered into a prolonged slump following the last closure of the Faro lead-zinc mine in January of 1998. By 2002 there were no operating hard-rock mines in Yukon, mineral exploration spending had declined steeply, and even placer gold production had fallen to a 23-year low. Beginning in 2004 expenditures on mineral exploration and development began to rise sharply, a rise that has continued through 2007.

Whitehorse is the Yukon's capital, has close to three-quarters of the territorial population, is the service centre for the territory, and dominates the territorial economy. Rural communities, where the First Nation population is concentrated, are generally not benefiting from the high incomes and economic prosperity of the capital.

The communities most affected by the mine are likely to be Ross River, Faro, and Watson Lake. The First Nation of NND, centered in the Village of Mayo, will also be affected due to the location of the mine in their traditional territory. Ross River, Faro, and Mayo all have populations of approximately 400, while Watson Lake is larger with about 1,500 inhabitants.

LABOUR FORCE

According to the 2006 Census, about one quarter of the Yukon's population is of Aboriginal origin. That proportion is much greater in rural communities with the sole exception of Faro, which was originally built to service a mine.

For the proposed Mactung mine, NATC considered the potential benefits provided through employment as well as the potential effects associated with labour force depletion for inclusion within the YESAA application. Further, the assessment application outlined potential mitigation measures to minimize negative effects and potential means of enhancing positive effects.

ROSS RIVER

Ross River is the community closest to the Mactung mine. Located on the south bank of the Pelly River near the confluence of the Ross and Pelly Rivers where the North Canol Road crosses the Pelly, Ross River is a mainly Kaska First Nations community (86%) with a population just under 400 (349 in December 2007). The South Canol Road runs through the community with a seasonal ferry providing access to the North Canol Road during the summer. The community is approximately 10 km from the Campbell Highway.

Mining exploration increased in the region around Ross River through the 1950s and an exploration and mining boom occurred in the 1960s and 1970s with the discovery and development of the Faro mine. Although Ross River Dena people did work in mining exploration the mining boom did little to benefit the First Nation. The more recent increase in mineral exploration and development activity in the region have benefited Ross River residents as employment levels have increased to the point that in 2007 mining companies have had some difficulty recruiting local residents for work. It is likely that the Mactung mine could get at least a portion of its labour force from Ross River, especially if it is willing to provide the necessary training and work environment to attract and keep a local work force.

Despite the exploration boom in the region, unemployment rates were still high in Ross River in 2006 with an unemployment rate of 21% compared to 9% for Yukon as a whole.

The RRDC is the First Nation Government in Ross River. The Ross River Dena has developed a mining strategy to guide their relationship with mining companies.

FARO

Faro, located above the north bank of the Pelly River approximately 70 km from Ross River, is Yukon's newest community. It was built in 1969 to house the Faro mine workers and their families. The community's relatively short history has been marked by wild swings in population as the mine has opened and closed.

Cyprus Anvil made the decision to go into production in 1967 and construction of the town began in the fall of 1968. The 1970s were boom times for Faro. The mine was profitable, wages were high, there were the benefits of heavily subsidized housing, and the town grew. In 1981, it was the highest income community in Canada and reached a peak population of over 2,000 people.

In the early 1980s metal prices dropped, and Cyprus Anvil began losing money and building up debt. A temporary shutdown was announced in June of 1982 and the shutdown continued as metal prices remained very low. In May 1985, the company announced it was mothballing the mine.

In November 1985, Curragh Resources Inc. took over the mine and in the spring of 1986 the Faro mine and mill were back in operation; however, by 1992, the company was mired in difficulties and in April of 1993 the Faro mine was shut down for the second time. Again, the town's population dropped precipitously,

Anvil Range Mining Corporation (Anvil Range) bought the property in 1994 and production began again in August of 1995. In November of 1996, Anvil Range suddenly announced a temporary closure of the mine by the end of the year. The mill would continue to operate until March 1997 using stockpiles of ore. Lower metal prices and a higher Canadian dollar were given as the reasons for the shutdown. Anvil Range declared bankruptcy in April 1998. Since 2000, the town's population has remained stable at just under 400 residents.

The 2006 census showed that Faro had a labour force of about 75 people, of whom 17% were employed in natural resource industries (i.e. mining as there is very little agriculture and forestry in Yukon). The occupational distribution provided by the Census showed that 50 worked as tradespeople, transport and equipment operators, or in occupations unique to primary industry. It is therefore likely that the Mactung mine could get a portion of its labour force from Faro and that a number of other workers might be interested in moving there to spend less time commuting between their shifts.

WATSON LAKE

Watson Lake is situated in the south-eastern corner of Yukon at the junction of the Alaska Highway and the Campbell Highway. The Stewart Cassiar Highway whose northern end is just north of Watson Lake provides a link to tidewater in Stewart, British Columbia. The community is considered to include not only the Town of Watson Lake, but also the adjoining Kaska First Nation Settlements of Upper Liard along the Alaska Highway, and Two and One-Half Mile Village and Two Mile Village on the Campbell Highway. The settlement of Lower Post just across the border in British Columbia is also often considered part of Watson Lake.

The economy of the Watson Lake area has depended on being a transportation hub and a supply centre for mines in the area, notably Cassiar for asbestos, the Sa Dena Hes for lead zinc, and Cantung for tungsten. There has also been logging and sawmilling in the area as the southeast is home to Yukon's largest trees associated with its relatively wet climate. Yukon's one producing natural gas field, Kotaneelee, is located in the territory's south-eastern tip.

As with the Cantung mine, Watson Lake is likely to provide a number of services to the new Mactung mine as well as a portion of its work force.

MAYO

Mayo is quite far from the Mactung mine, about 716 km by road, but both the Mactung mine and the community are in the Stewart River watershed, hence the interest in the mine by First Nation of NND. The First Nation may be concerned about potential downstream environmental effects and may wish to discuss positive economic spin-offs from the mine for its citizens.

Mayo is another Yukon community that began as a mining centre. Mayo served as the transhipment point for the silver ore from Keno and Elsa until the United Keno Hill mine closure in 1989. There continues to be active placer mining for gold in the Mayo area as well as exploration activity, and for uranium in the Wind River area further to the north.

Community Considerations

The affected communities have had considerable experience with the mining industry in the past; as a result, some residents have worked or are currently working in the mining industry. Given this experience and knowledge of the industry, the affected communities will have a fairly sophisticated approach to new mining development. They can be expected to be generally supportive of new mining development and will wish to maximize the potential economic benefits to their communities; however, it is also expected that these communities will be highly aware of and concerned about long-term environmental effects.

One major socio-economic consideration that may have a material impact on the feasibility of the Mactung mine is the relationship that the company develops and maintains with affected First Nations. In practical terms, this relationship may be shaped by the negotiation and signing of agreements, for instance “Impact-Benefits Agreements” (IBA) or “Socio-economic Participation Agreements” (SEPAs). These agreements typically speak to a variety of issues including: jobs for community members, training, business opportunities, relations with the community, etc.

Many of these provisions can also be advantageous to the company, as a local labour force is likely to be more stable and ultimately less expensive. While it is not a legal or regulatory requirement, a SEPA/IBA may provide a framework for a positive relationship during the assessment and regulatory process as well as to provide a forum for ongoing communication while the mine is in operation.

21.4.2 NON-TRADITIONAL LAND USE

The area proposed for the Mactung mine site is located in a remote region of Yukon. Current use of the site has been limited due to this remoteness and limited access. At present, the site can be accessed through summer use of the North Canol Road, or through air access. Due to the access limitations for the site, non-traditional land use has been limited to mineral exploration, and recreational use such as trapping, hunting and hiking.

Based on the limited existing non-traditional land use as well as the relatively short temporal scope associated with the proposed project, conflicts between non-traditional land use and this project are expected to be minimal.

21.4.3 TRADITIONAL KNOWLEDGE

In preparation of the YESAA application, NATC has been seeking to obtain Traditional Knowledge from Yukon First Nations who have territories in which the Mactung Project is located. Consultation and Traditional Knowledge gathering are separate but parallel processes, and are both required under YESAA.

Guidelines provided by YESAB entitled the "Proponent's Guide to Information Requirements for Executive Committee Project Proposals" (Proponent's Guide) provides distinct guidance which requires proponents to seek Traditional Knowledge in addition to scientific knowledge. The guidelines specifically state "The proponent is encouraged to develop a sound understanding of First Nations issues and expectations with respect to the incorporation of Traditional Knowledge" (YESAB, 2005). This Proponent's Guide describes the methods for the identification of valued components. The guidelines produced by YESAB state, "Focus on the components identified as being most important according to the issues and concerns raised by government, stakeholders, First Nations, and the public, and include a consideration of: perceived intrinsic value, economic importance, traditional use, recreational value, rarity, legal, scientific value, and sensitivity" (YESAB, 2005).

The Proponent's Guide also identifies that the project proposal should contain information describing the past and present land use by First Nations for traditional, commercial, and recreational purposes. Information regarding social systems and economics is expected to be presented, where available, in both the description of existing conditions as well as the effects assessment. The importance of recording best efforts to obtain Traditional Knowledge is identified in the consultation requirements of the Proponent's Guide.

The Mactung project is proposed to occur within the traditional territories of LFN, RRDC, and NND. Information collected during project planning, identification of methodologies, and through consultation activities was incorporated into the environmental and the socio-economic existing conditions section, as well as the environmental and socio-economic effects assessment section of the YESAA project proposal.

At the time of the Feasibility Study, RRDC had developed a proposal to gather Traditional Knowledge for the company's consideration and NND had provided their Government's protocol for the gathering of Traditional Knowledge. NATC has also sent letters to NND and LFN, specifically requesting that Traditional Knowledge be collected where available.

21.5 CONSULTATION

Consultation requirements for the proponent are stated under YESAA, specifically Section 50 (3) states:

> *"Before submitting a proposal to the executive committee, the proponent of a project shall consult any first nation in whose territory, or the residents of any community in which, the project will be located or might have significant environmental or socio-economic effects."*

Based on this requirement, NATC will be required to consult with the RRDC, LFN, and the First Nation of NND as well as the communities of Ross River and Faro.

NATC has been actively engaged in consultation activities which included meetings in Ross River and Faro as well as with NND. The company has also engaged with LFN Chief and Council, although, to date no formal meetings have taken place with LFN membership. NATC is working through the consultation process to fulfill its requirements under Section 3 of YESAA. The information utilized and gathered from this process was part of the YESAA application for review by the Board.

The consultation process ensures that community members are provided with an opportunity to understand the project and participate in the project component planning, effects identification, and the potential formation of mitigations. Through carrying out consultation activities prior to submission of the project proposal, the proposal included the values presented and, as a result, provide for a more efficient and effective assessment.

21.6 WASTE AND WATER MANAGEMENT PLAN

Waste will be generated during all phases of the proposed project. NATC is committed to managing its waste streams to minimize potential impacts to the environment and local wildlife. Within this section, a number of items have been outlined to demonstrate the range and types of mitigation measures which can be applied.

Domestic waste generated at the site will be incinerated to minimize the potential for attracting wildlife to the project area. Bear-proof garbage containers will be used at the site to minimize potential attraction of bears; ash from incineration will be disposed of in an approved disposal location.

Sewage generated from the operations will be treated and disposed of on-site. A suitably sized modular state-of-the-art sewage treatment system such as a rotating biological contactor (RBC) will be utilized. The sewage treatment and disposal system will require permitting and the design requirements of the system will reflect appropriate regulatory requirements.

Industrial wastes will be handled, stored, and disposed of using accepted and appropriate methods to be determined by project planning and regulatory requirements. Waste petroleum products such as oil will be collected and, if suitable, will be recycled or used to generate heat.

Production and volumes of hazardous materials from mine processing has yet to be determined, but any such wastes will be safely stored and disposed of in accordance with appropriate regulatory requirements.

During the de-commissioning and abandonment phase of the project there will be substantial wastes generated. Any materials and equipment that are potentially salvageable will be sold or removed from site. Remaining waste materials will be handled and disposed of in accordance with accepted practices and regulatory conditions.

21.6.1 TAILINGS MANAGEMENT

Tailings produced at the proposed Mactung mine will be permanently stored using the dry stack process. Dry stack technology will produce a sandy tailings product that will have a moisture content of less than 20%. Approximately 50% of the tailings produced at the Mactung site are required for use as backfill in the underground workings. The surface tailings material will be trucked from the mill site to the tailings storage facility (TSF) for disposal. Water draining from the TSF will report to the Ravine Dam polishing pond where it will become part of the pumpback system used to supply the mill with process water.

The use of dry stack technology eliminates much of the logistical issues associated with the disposal of wet tailings under winter conditions. The tailings produced from the underground mining program are expected to be primarily classified as potentially acid generating based on accepted ABA practices. The time to acidity for the tailings is expected to be greater than the duration of the underground mining for the reasons discussed below.

There are some higher sulphide ores (>15%) that will be milled during the production phase which appear to be more reactive than the remainder of the deposit. The higher sulphide material is not concentrated in the deposit and does not represent a significant portion of the deposit. As a result it is expected that the higher sulphide material will become blended with the less reactive tailings and should not result in the onset of acid rock drainage during the production phase. The bulk of the ore is expected to have greater than 30 years to potential onset of acid rock drainage based on an examination of ore samples from the 1970s. To date no kinetic testwork is available to support the assumptions with respect to the possible time to onset of acid rock generation.

A suitable engineered cover will be installed at the end of the underground production phase to cap and isolate the tailings from water and oxygen.

21.6.2 Water Management

The proposed Mactung project requires freshwater for potable water supply, pump glands, and mixing of reagents. It is estimated that the peak demand for fresh water will be approximately 34 m^3/h. Fresh water will be supplied to the project from a primary pumping station on the southern tributary of the Hess River, and possibly a seasonal pumping station located closer to the mine site. This water demand is required year-round during periods of active milling.

The fresh water utilized by the project will be collected with the process water during tailings dewatering activities and report to the polishing pond. Process water for milling will be sourced from a reclaim barge located at the Ravine Dam. The overall process water demands for the project are estimated to be 161 m^3/h.

Diversion ditches and berms will be installed to divert water away from the TSF. Waters routed away from the TSF end up in the Ravine Dam which will serve as a polishing pond during periods of active discharge to the receiving environment. It is anticipated that the Ravine Dam will discharge periodically during the year and possibly during extended periods of suspended operations when process water is not required for milling. Pumping will be used to discharge excess water from the Ravine Dam; the pipe discharge will be directed into an engineered energy dissipation structure prior to re-entering the natural drainage channel.

The need for treatment of the discharges from the Ravine Dam remains to be determined and will be based on the Metal Mining Effluent Regulations (MMER) and water quality objectives for the project. Should the Water Licence for the project be based on the MMER, then the treatment requirements would not be as stringent as opposed to water quality objectives under the CCME.

21.7 Environmental Management and Planning

NATC is planning to minimize environmental risk through the development of an Environmental Management System (EMS) for the project. The EMS will contain detailed information on spill and disaster response, waste materials handling and emergency contact information for the project. Operating plans for mining and milling will be reviewed periodically and developed in accordance with the EMS for the site.

A list of locations and quantities of reagents and other hazardous materials used at the site will be maintained throughout the production phase. This information will allow emergency and environmental personnel to be fully aware of hazards that may be encountered.

The underground mining methods to be used at the Mactung mine are favourable to the effective management of waste materials to reduce potential long-term risks at the site. The use of backfill in the mine development process will create the opportunity for underground disposal of dry stack tailings with low carbonate mineralization in addition to waste rock materials identified as having either acid-rock drainage or metals leaching concerns. The mining and primary crushing processes will also occur underground, minimizing the potential for releases to the receiving environment based on accidents or spills from these processes.

A production-phase geochemical characterization program will be developed to track materials being stored. This production phase program will include field and laboratory static and kinetic testwork to confirm the assumptions made during the application and permitting phases and also to provide information for verification of the post closure site environmental model.

21.7.1 *Biological Monitoring*

Under the Yukon's regulatory and assessment process for mine development of major mine development projects, NATC may be required to undertake a number of wildlife, aquatic, and vegetation monitoring components. It is not anticipated that any monitoring programs will exceed those normally expected for similar projects. Consequently, monitoring concerns are not anticipated to limit project feasibility. The potential scope and subject of potential monitoring requirements are outlined below.

Wildlife

Based on the sensitivity of wildlife features in the Mactung project area, it is expected that the scope of recommended or required monitoring will fall within generally accepted monitoring practices. Those anticipated and generally accepted practices include:

- General Monitoring Program: Wildlife sightings within the project area will be logged, including time, location, activity, group sizes, etc. This information would aid in monitoring migration routes, and presence of wildlife near project infrastructure.
- Wildlife Incident Log: All wildlife incidents and interactions with people will be logged (e.g. road kills or injuries, bear attacks, bears in waste). Based on the information gathered, NATC will cooperate and work with governments and other interested stakeholders to improve wildlife management programs at the project area.
- Hunting Policies: NATC will incorporate a no hunting/firearms policy for employees while they are at the site.

Fishing

Required fisheries monitoring activities will conform to generally accepted industry Best Management Practices (BMPs) or government requirements and would form the conditions of fisheries permitting, including the following components:

- Construction Monitoring: An environmental professional would monitor construction activities around watercourses to provide assurance that permit conditions are met. This may also include components such as fish salvage, or completion of erosion/sediment control plans.
- Post-construction Monitoring: An environmental professional would monitor the effectiveness of fisheries compensation measures, long term watercourse stabilization measures, as well as the effectiveness of re-vegetation in riparian areas.

AQUATIC ENVIRONMENT

Comprehensive aquatic environment baseline studies have been conducted at the Mactung site, which includes the surveying of benthic fauna, stream sediment metals and grain size, as well as periphyton and primary productivity. Based on these studies, no aspects of the receiving environment or project plan have been identified that are predicted to require unconventional ongoing monitoring programs.

It is anticipated that NATC will be required to undertake standard aquatic monitoring under the terms of a 'Type A' Water Licence for the Mactung site and the Federal MMER Environmental Effects Monitoring (EEM) program. NATC plans to initiate EEM monitoring as the project progresses as the implementation of such a program will provide an ongoing, structured regime of monitoring that will identify changes in aquatic health throughout the life of the project, and will protect NATC from future liabilities at the site.

VEGETATION

Based on vegetation baseline study program conducted at the Mactung site, a low overall risk to vegetation communities identified based on chemical, community level, and species level surveys. Consequently, it is anticipated that those monitoring programs required of NATC under the terms of development permitting will be conventional studies related to the ongoing effects of operations and site reclamation. Examples may include trace element concentrations monitoring and invasive plant monitoring.

22.0 RECLAMATION, DECOMMISSIONING, AND CLOSURE

This section outlines the Closure Plan requirements as part of the permitting process for the proposed Mactung project. EBA reviewed available information for the major mining components to determine if it is feasible to meet these requirements. Further, each project component has been identified in reference to the overall reclamation goals and objectives as well as the final land use objectives for the site.

Based on the review, EBA determined that the project, as proposed, would provide suitable closure techniques and that final abandonment would be feasible for the site.

22.1 RECLAMATION GOALS AND OBJECTIVES

NATC is committed to using environmental management techniques during all phases of the operation that will minimize potential environmental impacts from the Mactung project. Progressive reclamation activities to reduce site liability will be conducted when possible; however, much of the site infrastructure cannot be reclaimed until the end of the Production Phase.

The main surface infrastructure for the project will include:

- a new, approximately 40-km long access road in Yukon
- an extension and upgrade of an existing airstrip in MacMillan Pass, Yukon
- a freshwater pumphouse located on the southern tributary of the Hess River, with associated access road, powerline and piping
- power generation facilities (including potential wind turbines) with associated site power distribution
- a process water polishing pond with associated diversion ditching
- a dry stack tailings facility with associated diversion berm
- a mill building
- an equipment maintenance shop and office complex
- access roads between site infrastructure
- a 150-person camp.

The proposed resource access road to the Mactung site will be constructed entirely within Yukon. Following end-of-site decommissioning, EBA anticipates that responsibility for this road may be returned to the Government of Yukon, if the road is deemed to be a government asset. The Government of Yukon may request financial security for the resource access road during the initial construction and production phase of the project.

The extension and improvements to the surface of the MacMillan Pass airstrip will result in an overall improvement of this facility. It is unlikely that Government of Yukon will require reclamation of the airstrip, since this infrastructure provides for a public aerodrome.

Approximately 50% of the tailings produced during the underground mining phase of the project will be used to progressively backfill the underground workings. Using tailings as mine backfill will reduce the long-term environmental concerns for surface storage. The use of dry-stacked tailings also eliminates the necessity for a tailings impound dam. The 50% portion of tailings that will be dry-stacked and surface stored must be capped prior to abandonment. The anticipated construction procedure of the dry-stacked tailings facility does not allow for progressive capping. The process water aging and runoff storage pond created by the ravine dam will be recontoured. The dam will be breached in a v-notch fashion and an erosion preventing streambed will be constructed through the dam structure. Drainage patterns will be re-established in the area of the previous reservoir. Once complete, the stream will once again be continuous from its natural headway to the confluence of Tributary A. All water diversion structures constructed will be removed. The diversion channel on the north side of the valley will be filled in and runoff water will be able to flow into the stream at the base of the valley.

During decommissioning activities, all building, salvageable equipment and inventories will be removed from site. Hazardous materials will be removed from site and disposed of in an approved disposal facility. Any non-salvageable waste materials and structures will be broken down to ground level and preferably disposed in the underground workings.

Periodic inspection and monitoring of the site after closure will be required. The MMER regulations will continue to apply to the Mactung site for a period of three years following closure; it is also expected that Government of Yukon will require some form of more frequent inspection and monitoring for a period of up to 5 years post closure.

22.2 LAND USE OBJECTIVES

Land use objectives for decommissioning and closure are normally developed for a project area to guide the overall reclamation strategy for the project. These objectives are often developed in accordance with objectives laid out in higher-level management plans, as well as through consultation with First Nations, community members, and other stakeholders.

Current land use in the project area is primarily wildlife habitat, with most wildlife utilizing a portion of the site during the year. The limited road access into the project area has limited the level of general use of the area. There has been some past exploration activity in the area of the proposed access road to the Mactung site. There is also a network of trails near the start of the proposed access route to the mine site.

During the construction and production phases of the project, the lands will be considered industrial usage, and access to the site will be controlled to ensure safe operations. The underground nature of the project ensures that the disturbance footprint of the mine site is minimal and that effects of changes in the land use are only temporary. Following site decommissioning and abandonment, reclamation activities will return the site to as close to the pre-mining state as practical.

It is expected that the post-closure wildlife usage of the site will return to pre-mining levels. There may be an increased usage during the initial post-closure phase because of that initial high availability of forage from reclamation activities. The lack of a tailings pond and the proposed encapsulation of the tailings wastes will ensure that these facilities are capable of meeting the final land use objectives for the Mactung project.

23.0 PROJECT EXECUTION

23.1 INTRODUCTION

This section of the report outlines the proposed philosophy and approach required to execute engineering and construction activities needed to put the Mactung project into production.

This Project Execution Plan (PEP) defines the relationship between engineering, procurement, and construction activities.

23.2 IMPLEMENTATION STRATEGY

NATC will design and direct pre-production mine development activities, and appoint a small project team to oversee project execution. The initial steps of the implementation strategy will include:

- the selection of an EPCM company to undertake the design of the material handling, process plant, and infrastructure facilities and to manage the full project construction activity
- the selection of an engineering group for the engineering and procurement (EP)
- the ordering of long-lead equipment items.

Project risks will be managed using the EPCM scheduling, cost reporting and control, and purchasing and contracting strategy. The construction contracting strategy may be based on either:

- the award of multiple contracts commensurate with when the various engineering components will be delivered
- a general contractor that will supply the majority of labour and equipment and make use of specialty sub-contractors as needed.

23.3 PROJECT MANAGEMENT STRUCTURE

23.3.1 ORGANIZATIONAL OVERVIEW

Working under the administration of NATC's project team, the EPCM contractor will be lead by a Project Manager, who will be responsible for the execution of the project's established schedule, capital cost, safety, environmental, and quality targets. The organizational chart is shown in Figure 23.1.

23.3.2 OWNER'S RESPONSIBILITIES

NATC will retain responsibility for the following activities:

- mine planning and underground mine development
- communication with local communities and the media
- accounting and invoice payment
- all permits or licenses required for construction, operation, and environmental compliance
- operation staffing, operator training, commissioning, and start-up.

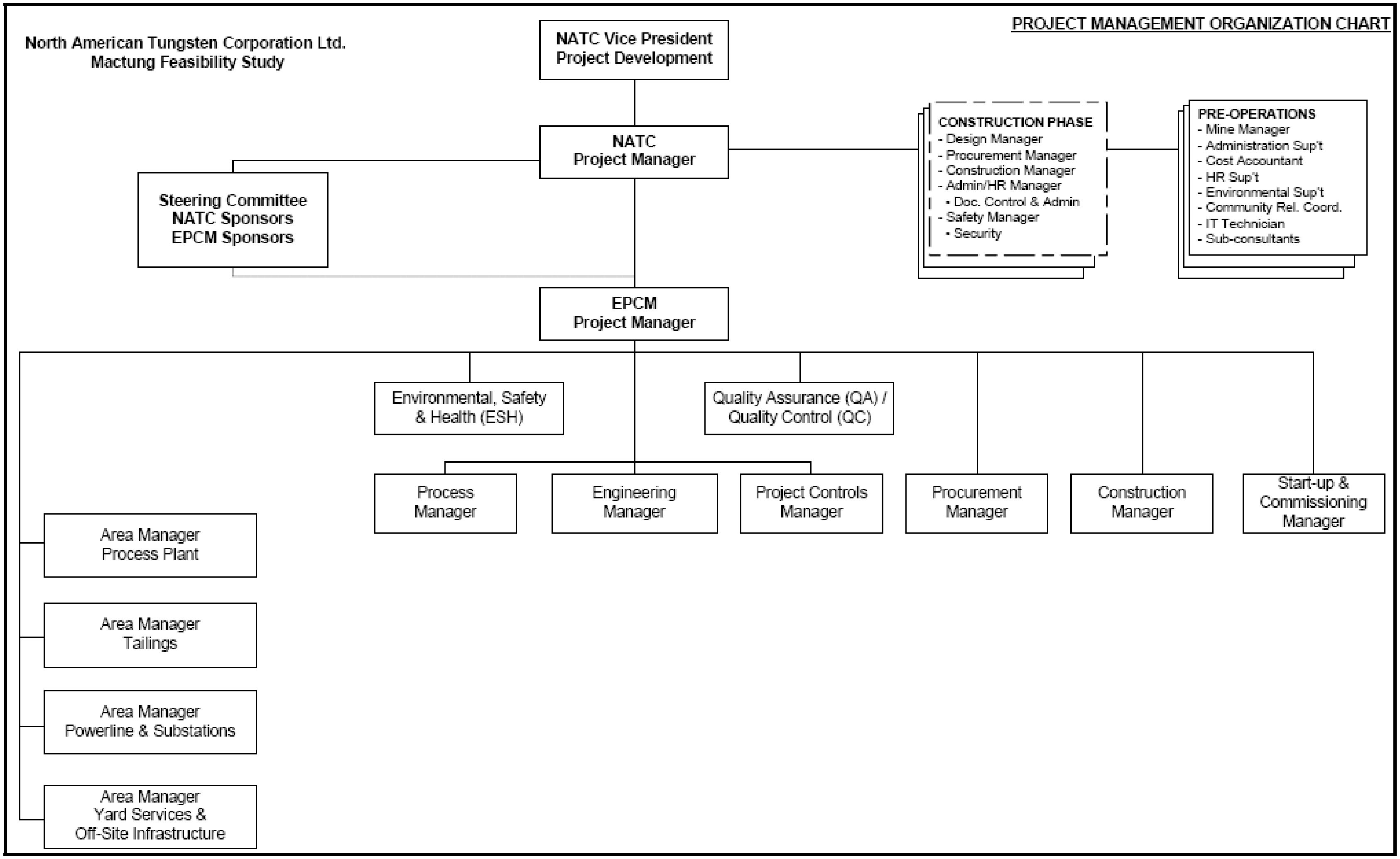
23.3.3 EPCM CONTRACTOR'S RESPONSIBILITIES

The EPCM team will establish a project management system that will contribute to the overall cost reporting and scheduling for the project, as well as establish a comprehensive baseline plan to monitor project progress and productivities. This plan will help project teams quickly identify and correct early departures from the plan.

23.3.4 PROJECT PROCEDURES MANUAL

A comprehensive Project Procedures Manual will outline the procedures and requirements for the execution of the project administrative activities, as well as Owner and EPCM contractor rights, authorities and obligations. The overall Project Procedures Manual will be prepared by the EPCM contractor, and then issued to NATC for review and comment prior to formal issue.

Figure 23.1 Project Management Organization Chart



23.4 SET-UP AND BASELINING PHASE

The period prior to project financing approval will be used to establish the core project team, freeze project concepts, set-up, and baseline the project and advance the design and procurement of long-lead items.

The intention is to commence this phase as soon as the set-up and baselining contracts are in place. This phase is expected to take six months to complete and will proceed in parallel with basic engineering and project financing. It will also overlap with the commencement of the detailed engineering and construction phases to a certain extent.

Once the project has received full funding, the engineering and construction phase will then proceed to detailed engineering design, procurement, and construction activities.

23.5 ENGINEERING PHASE

23.5.1 ENGINEERING STRATEGY

Primary out-sourced engineering work will include the following categories:

- water, waste and tailings design
- process facilities, site development and infrastructure, including the main access road and the transmission line.

23.5.2 BASIC ENGINEERING

Basic engineering tasks will include the following:

- Updating of the PFDs, Equipment List, P&IDs, General Arrangement drawings, Electrical SLDs, and Load Flow reports.
- The engineering contractor will provide licencing and permitting assistance to NATC as required.
- Specifications will be developed to obtain competitive and firm price bids for major equipment that requires a long lead for delivery. The EPCM contractor will evaluate the bids and make recommendations to NATC for purchasing; the purchase orders will be issued by the EPCM contractor on behalf of NATC.
- Once the capital cost estimate from the Feasibility Study has been approved, it will be “converted” to a control budget and updated with recent cost information including the values of the released purchase orders.

- The project schedule will be updated.
- Purchasing procedures and standard documentation will be developed.
- A construction contract boilerplate will be developed for NATC's approval.

23.5.3 DETAILED ENGINEERING

Calculations, specifications, drawings, material requisitions, and other items related to detailed engineering will be completed in accordance with the engineering procedures. In addition, the detailed design engineering team will:

- Complete the engineering calculations, detailed drawings, and specifications associated with the construction of new facilities.
- Produce design and construction drawings based on the results of site investigations, including surveys and inspections, in accordance with the details set out in the Feasibility Study as much as possible.
- Develop equipment specifications for the remainder of the non-long lead equipment items, together with the commercial and technical analysis and purchase recommendations. All manufacturing hold points will be included in the specifications.
- Provide material requisitions for the bulk materials purchases based on material take-offs for items such as electrical cables, cable trays, hi-bay lighting, and piping.
- Provide the construction technical specifications to be included in the construction contract bid packages as well as pertinent drawings, vendor information when available, and quantities for the Form of Tender. Also provide all technical reports that will be referred to by the contractors when bidding the construction work for items such as the geotechnical analysis, studies, and special information from vendors for installation of equipment and special requirements for handling.

23.6 PROCUREMENT AND CONTRACTS

23.6.1 PROCUREMENT AND EXPEDITING

The EPCM Contractor's Purchasing Group will provide capital equipment procurement, vendor drawing expediting and, when required, equipment inspection. The procurement department will package the technical and commercial documentation and manage the bidding cycle for equipment and materials to be supplied by NATC to the contractors.

The Construction Management (CM) group will organize bulk materials purchases, assemble contract tendering documents, establish qualified bid lists, issue tenders, analyze and recommend suitably qualified contractors to NATC, and prepare executed contracts for issue.

A field procurement manager will support ongoing construction needs for miscellaneous materials and services to be provided by NATC as well as provide expediting services. They will also be responsible for the receipt, storage, and disbursement of purchased materials and equipment at the job site.

A plan for expediting equipment purchase orders based on the project schedule and equipment list will be prepared. Expediting will be coordinated by the construction group.

Expediting reports will be entered into the material control reporting system after each contact with suppliers.

23.6.2 LOGISTICS

The EPCM team will direct logistics and freight for incoming equipment and materials, to be transported by rail and truck through Prince Rupert and/or Vancouver when possible. A single-point freight forwarding company will coordinate with manufacturing facilities, establish shipping points and dates, forward the shipments to the most convenient ports, and complete trans-shipments to the project site.

23.7 CONSTRUCTION PHASE

The construction phase is subdivided into three main phases: Construction Management, Field Engineering, and Construction Contracting.

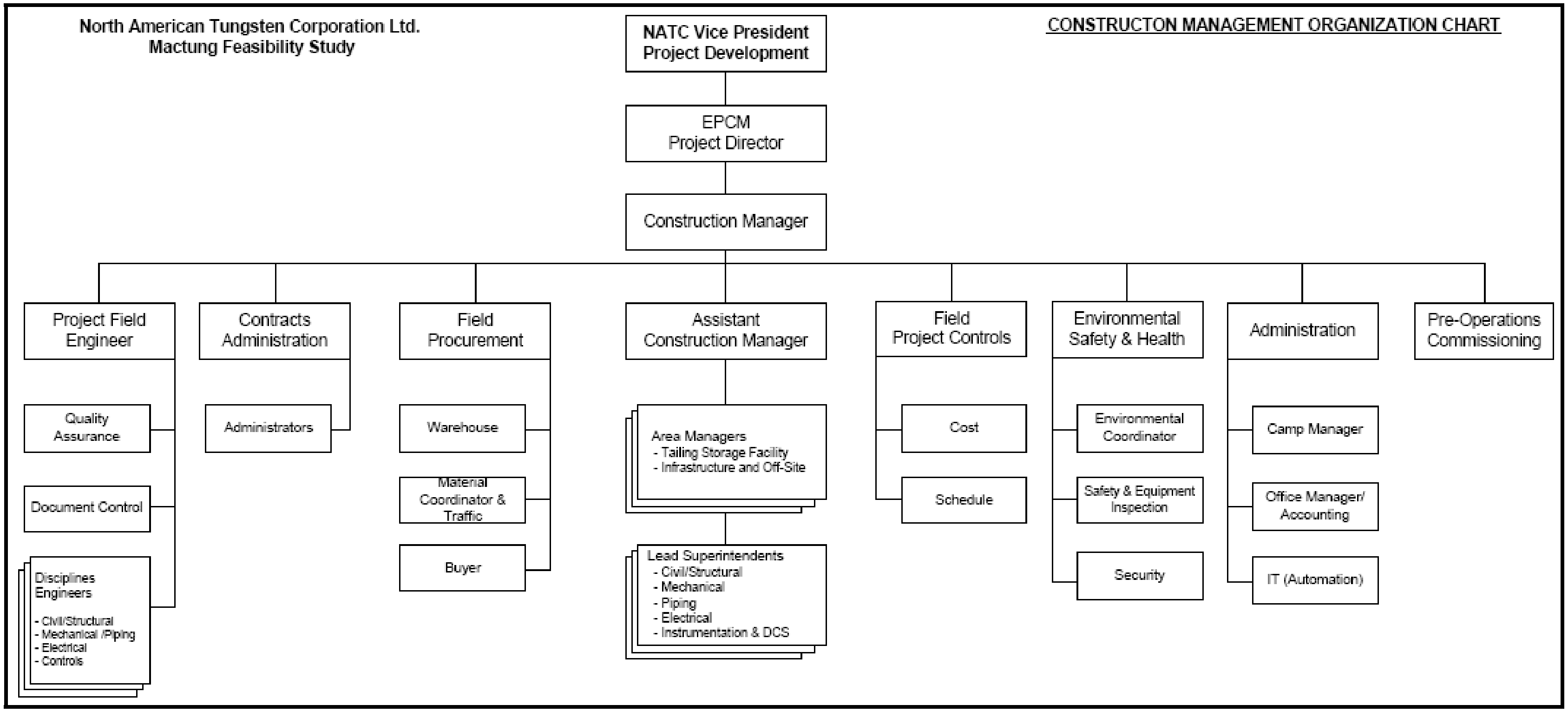
23.7.1 CONSTRUCTION MANAGEMENT

The CM group will be responsible for the management of all field operations. Reporting to NATC, the Construction Manager will plan, organize, and manage construction quality, safety, budget, and schedule objectives.

Construction will be performed by contractors under the direction of the CM team, reporting to NATC's representative.

The CM organization chart (Figure 23.2) details the CM team organization plan for the site.

Figure 23.2 Construction Management Organization Chart



23.7.2 FIELD ENGINEERING

SURVEYING

The CM survey crew will verify the accuracy of the existing control system before construction begins. The Construction Manager will verify surveys prior to construction. Contractors will supervise day-to-day field surveying, and the CM team will provide spot checks.

QUALITY CONTROL/QUALITY ASSURANCE

Contractors will establish and observe their own Quality Control program in accordance with the construction technical specifications and the applicable codes and standards. The CM Field Engineering Team will employ independent CSA-qualified Quality Assurance specialists to ensure quality control.

23.7.3 CONSTRUCTION CONTRACTING

The contracting strategy will be designed to maximize the local labour force, create a responsible and sustainable relationship with the nearby communities, and provide a mix of senior management and specialists to support the safety, quality, schedule, and cost objectives of the project. In addition, contracts will be designed to combine timing, scope, battery limits, and contract value into manageable packages.

Approved contract pro formas will be utilized for construction and service contracts.

23.7.4 MATERIALS MANAGEMENT

WAREHOUSING

The Site Materials Management group will receive, inspect, and log all incoming materials, assign storage locations, and maintain a database of all materials received and dispensed to the contractors.

SITE HOUSEKEEPING AND HAZARDOUS WASTE MANAGEMENT

Procedures for waste management and spill response will be implemented for the construction period. These procedures will be established in the Project Procedures Manual and will outline compliance, auditing, and reporting requirements.

CONSTRUCTION EQUIPMENT

Individual contractors will be responsible for the equipment required to meet their contract obligations. All equipment must comply with Mine Safety Branch requirements; NATC's CM team will perform regular spot checks. Large cranes may be supplied by a single company, managed by the CM team.

Construction Camps

Development of a full-service camp for construction contractors will begin in June 2010, immediately following the receipt of construction permits.

The camp – a modular design with propane heat – will accommodate up to 250 workers. Accommodations types will include single occupancy rooms. A recreation complex will house games and gym facilities.

Transportation, potable water, waste management and other support services will be scaled to support the various development stages. The CM team will ensure that catering contractor meets all facilities, staffing, hygiene, food handling, storage, and meal expectations.

Only prescription drugs will be permitted on site. No firearms or alcohol will be allowed on site.

Temporary Facilities and Construction Site Infrastructure

Communication

NATC's systems manager will determine the appropriate telecommunications technologies for the project. Requirements include voice and data link technologies adequate to support growth construction and plant operation growth.

The communications framework for management offices will be installed early in the construction period.

Construction Power

Approximately 1 MW of operating power capacity and one spare for camp utilities (excluding heat) and limited construction facilities will be required during the 2010 to 2012 construction period. By the second quarter of 2012, permanent power will supply all mine equipment and peak construction power loads for the remaining duration of the construction phase.

First-aid and Site Security

NATC will provide a fully-equipped first-aid facility and ambulance for project-wide use. The facility will normally be staffed 12 h/d, with on-call services ensuring continuous coverage. The first-aid staff will live at the camp. Contractors will be expected to provide basic first-aid stations at the site.

NATC will supply a 24-hour staffed site security program during the initial field mobilization in 2010. Access to the site will be controlled at the principal road entrance and will be limited to personnel who have attended induction training, as well as approved visitors.

Early Construction Activities

Receipt of the construction permits in May 2010 will launch initial construction activities from June to September.

The first trucks on the Canol Road will transport the construction camp trailers to the project site. Heavy civil equipment including bulldozers, excavators, gravel trucks, loaders, bobcats, graders, water trucks, fuel trucks, and pick-up trucks will be transported. Trucks will also transport generators, fuel, electrical supplies, plumbing supplies, aggregate plant, carpentry supplies, and food.

The camp will include approximately 13 modules, comprised of:

- six modules for accommodations
- three modules for kitchens
- one module for the sewage treatment plant (STP)
- one module each for water supply, recreation, and change room.

The second 49-person camp module will be transported and set up at the MacMillan Pass Airstrip. Generators, STP, and water storage will be set up. More heavy civil equipment will be transported and off-loaded at the airstrip.

The permanent 150-person camp will be transported to the site and operational in September.

Work will commence to upgrade the airstrip. The access roads will be constructed from both ends and will be completed in September.

Infrastructure Construction Activities

The activities during this period will involve major construction of non-temporary infrastructure, including the STP, the fuel tanks, and the process plant. This construction phase is scheduled to start in the last week of September and will last for 14 weeks, after which the camp will have a capacity of 250 people.

Major Construction and Mining Activities

The final phase of construction will involve the commencement and completion of all remaining site facilities and services. The bulk of the work will focus on completing the process plant, mine development and underground facility and services installation, and generator installation and start up. Additional work will be the construction of auxiliary buildings and the completion of all site services including the development of a fresh/fire water system.

The DSTF will be constructed from dewatered tailings hauled to the site by a combined conveyor belt and haul trucks. The tailings material is spread out into a 600 mm thick lift using a dozer and then compacted using a vibratory compactor. Construction will continue during the operating years.

A dam will be constructed downstream from the plant to collect surface run-off from the tailings facility, and to age process water for re-use in the mill.

The activities will commence in Q1 2011 and be completed 93 weeks later in Q3 2012. The project will be in full production in Q1 2013, after three to four months of plant commissioning.

Health, Safety, and Environmental

A fully-integrated workplace safety and environmental responsibility program will be implemented to help achieve a "zero-harm" goal. The development of effective Health, Safety, and Environmental (HS&E) practices will require a high level of communication, motivation, and involvement, including alignment with site contractors on topics such as safety training, occupational health and hygiene, hazard and risk awareness, safe systems of work, and job safety analysis. Tools will be implemented for performance tracking and accountability, including procedures for incident management.

All design and engineering stages incorporate criteria for responsible methods of process flows, effluent, and waste products, in order to meet established capture and containment guidelines. The design also incorporates clean plant design standards, including operational safety and maintenance access requirements.

A Hazard and Operability Analysis (HAZOP) will be conducted by the project design team during each plant area's detailed design stage. Environmental protection will be incorporated in both the design of the main plant processes, as well transportation, storage, and material disposal.

23.8 Pre-production Mine Development

Underground mining equipment delivery is expected in Q4 2010. The pre-production mine development will commence in early Q1 2011 with the installation of generators and air compressors. A temporary mine dry will be installed.

Two ramps to the underground mine will be developed to provide permanent access to future underground mine workings. Concurrent with the existing adit rehabilitation will be the construction of the 512 m by 5 m by 4 m conveyor decline at a –13%

gradient, with projected completion in Q4 2011. At about this time, the drift, ore pass, and Crusher Station will be excavated at 1778 m level elevation. The Crusher Station will be completed in Q1 2012.

A drift will be driven from the conveyor decline to commence the excavation of the Tailings Backfill Load-out Station for completion in Q1 2012.

The completion of the conveyor decline is subsequently followed by the excavation of the North Ramp to access the Upper and Lower 2B ore zones in Q4 2011.

The pre-production stope developments will commence in Q4 2011 all the way to Q2 2012 in order to prepare the underground mine for production.

The power plant will be completed and commissioned in Q2 2012. The electrical underground substation will be installed in Q2 2012 followed by the completion of the electrical feeders and distribution systems in Q3 2012.

The commissioning of the primary crusher and the ore conveyor in the decline will be completed in Q2 2012 and followed by the tailings backfill conveyor in Q4 2012.

The mine will be ready for full production in Q1 2013 with development ore supplied to the mill during the commissioning phase in the Q4 2012.

23.9 PROJECT CONTROLS

23.9.1 COST

The project Work Breakdown Structure (WBS) defines the elements of project scope. The first step in the project implementation process will be to confirm the project WBS and distribute the control estimate.

The definitive estimate produced at the conclusion of basic engineering will become the control estimate for project execution. Budgets will then be cast for the approved scope, which will become the baseline control document against which the project will be measured.

Cost management allows the EPCM team to quickly evaluate and minimize any potential negative effects of capital control budget deviations. A project cost management system will provide comprehensive cost reporting, forecasting, and trending.

The project team will implement a coordinated program to review the project Estimate at Completion (EAC), together with engineering, procurement, and the construction management team.

A cash flow will be established once the contracting strategy and construction schedules are finalized and the estimate is recast into procurement packages. Deviations from the scope of the control estimate will be evaluated with respect to cost and schedule impact against the funds approved for the project.

23.9.2 Planning

The project master schedule highlights project milestones and critical sequences. Executive level reports provide an ongoing overview of project status. Detailed schedules track the planned and actual progress throughout the duration of the project. Progress reporting is based on earned or weighted value progress measurement for work in place, trend analysis, and other techniques as required to ensure precise reporting of project status.

23.9.3 Document Control

A collaborative document control system will be implemented that provides status and version control for all issued documents. Documents will be linked to the equipment database to ease the delivery of an accessible control system to the operations group at the completion of the design and construction phases.

23.9.4 Risk Management Strategy

A formal risk management program will commence during the Feasibility Study phase and will continue through commissioning. The project team will periodically review risks and opportunities, and take appropriate action to minimize the impact on overall costs and scheduling.

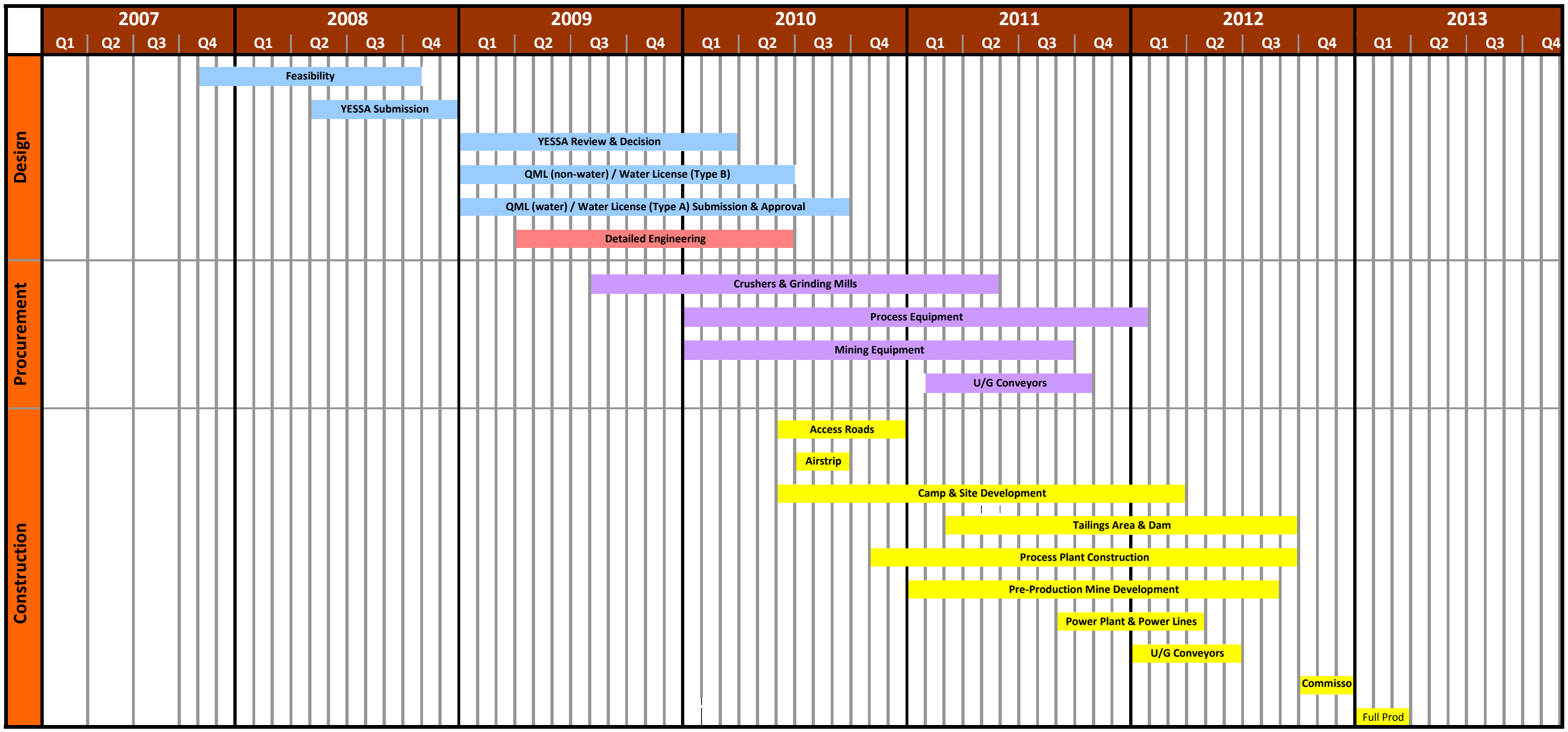
23.10 Project Schedule

23.10.1 General

The first scheduled project implementation tasks are long delivery equipment (grinding mills, crushers, filters, thickeners, dryers, power generators, conveyors, underground mining equipment) ordering, process facilities basic engineering commencement, and facilities construction. The overall project execution duration, from the ordering of long lead time equipment to mechanical completion, is expected to be about 37 months. The duration from commencement of field construction (Phase 1 camp construction) to mechanical completion is about 28 months. The overall duration assumes that required financing will be in place to allow all phases of the project to proceed at their projected start times, and that all permits will be in place for the work to proceed as planned and without stoppages.

The project development schedule is summarized in Figure 23.3.

Figure 23.3 Project Development Schedule Summary



23.10.2 CONSTRUCTION LABOUR

The schedule is based on a 70-hour work week. Crew rotations are assumed to be three weeks on-site and one week off-site.

It is anticipated that there are about 1.0 million direct and indirect construction man-hours, excluding mine pre-development and engineering. A peak of about 250 persons on-site is also expected during the construction phase of the project. The distribution of labour is shown in Figure 23.4.

23.10.3 MILESTONES AND SCHEDULE STRATEGY

The schedule reflects a traditional approach to project execution. Field construction will start only after engineering tasks are adequately advanced, in order to accommodate major equipment long lead delivery times. Engineering will continue early in the project, and many of the construction work packages can be issued on a fixed-price basis, thus reducing cost risk.

The project schedule requires interim project financing shortly after the completion of the Feasibility Study. This interim financing will accommodate long delivery items and ensure camp and power transformer preparedness.

DSTF construction activities are highly weather-dependent. The phased completion of the DSTF containment areas and the ravine dam have accordingly been scheduled to collect surface runoff from the tailings facility and to age process water capture for re-use in the processing plant.

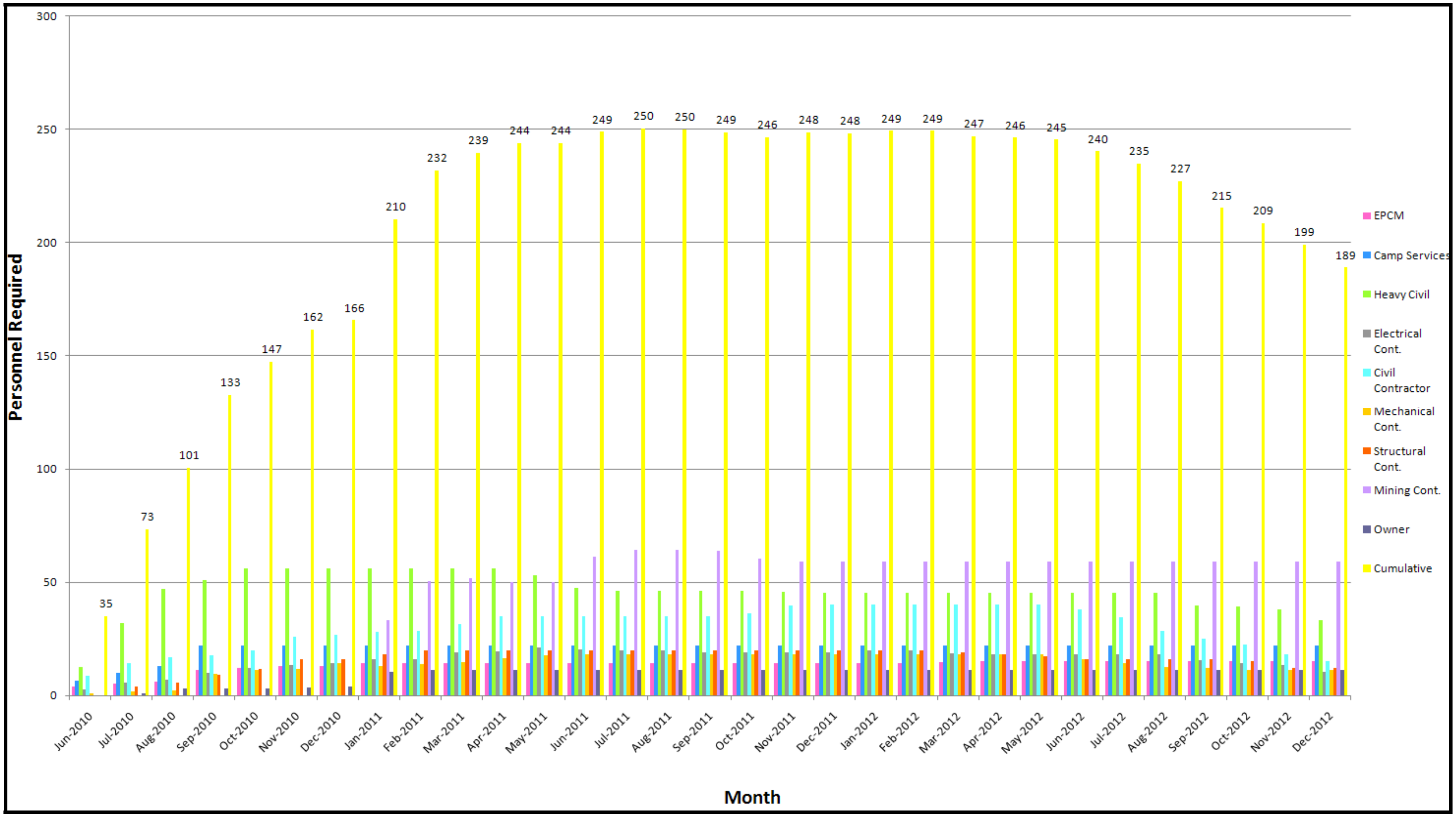
23.10.4 CRITICAL ACTIVITIES

The availability of early interim financing is critical to achieve sustained operations in Q1 2013. The following tasks should be completed as soon as possible after the Feasibility Study is issued:

- the execution of the EPCM contract will ensure that basic engineering commences, allowing a level of design adequate to order long delivery items
- transmission line design
- DSTF design
- long delivery capital equipment ordering.

Permits must be in place to allow initial construction to proceed in June 2010. Operators must be trained and ready to take over the operation of the facility upon project turnover.

Figure 23.4 Distribution of Labour



23.11 MECHANICAL COMPLETION AND COMMISSIONING

Mechanical Completion indicates that systems are ready for safe dry-testing. It also indicates that final work inspections can be completed, contract deficiency lists generated and resolved, and documentation for contract closeouts can be processed.

The project will phase the mechanical completion sequence to level the resource requirements and stage the transition from construction to operation. Critical site infrastructure facilities will be completed first, followed by critical process utilities commissioning, and then the main process equipment.

23.11.1 MECHANICAL COMPLETION AND COMMISSIONING INTERFACE

With the exception of mining activities, the EPCM contractor will have responsibility over all facilities until Mechanical Completion. Then, system-by-system, the plant will be transferred to NATC's operations team for commissioning.

After Mechanical Completion, the EPCM contractor and the construction contractors will provide assistance during the commissioning and start up phase as requested.

23.11.2 PLANT OPERATION AND PERFORMANCE TESTING

The plant operations group will be responsible for plant operation and performance testing, including facility operation at various design sizes and capacities. NATC's operational personnel will conduct plant performance tests, with assistance from suppliers. Selected members of the project commissioning team may be asked to assist as required.

24.0 CAPITAL COST ESTIMATE

The capital cost estimate (CAPEX) consists of four main parts: direct costs, indirect costs, contingency, and owner's costs. Owner's costs were developed by NATC. Quantities for tailings disposal were developed by EBA. All other areas were developed by Wardrop engineers, procurement specialists, and cost estimators. As of August 11, 2008 the CAPEX for the Mactung project is CDN$402,055,378. The estimate is subject to qualifications, assumptions, and exclusions all of which are detailed in this report. The CAPEX summary is shown in Table 24.1.

Table 24.1 Capital Cost Summary

Area	Cost (CDN$)
Direct Costs	
Overall Site	32,355,025
Mining	39,588,922
Crushing	9,132,813
Crushed Ore Stockpile and Reclaim	7,305,481
Grinding and Flotation	74,588,118
Tailings	23,132,203
Site Services and Utilities	15,399,567
Ancillary Buildings	33,940,283
Plant Mobile Equipment	1,232,050
Temporary Services	16,136,159
Off Site Infrastructure Facilities	19,665,217
Airstrip	1,753,888
Total Direct Cost	**274,229,726**
Indirect Costs	
Project Indirects	
Construction Indirects	15,773,536
Spares, Initial Fills	5,407,867
Freight & Logistics	12,102,000
Commissioning & Pre-operational Startup	1,575,000
Engineering and Procurement	17,053,688
Construction Management	9,225,000
Total Project Indirects	**61,137,091**
Owner's Costs	21,094,560
Total Indirects	**82,231,651**
Subtotal	**356,461,377**
Contingency	45,594,000
Total Project	**402,055,377**

24.1.1 PRICING

None of the pricing for commodities or the design/supply of equipment is based on binding quotations. Budgetary quotations have been obtained from vendors and contractors for major equipment and unit rates.

Equipment items valued under $100,000 may be priced from in-house data and previous similar projects if pricing was recently updated, unless the equipment was of a specialized nature.

24.1.2 TAXES

All taxes, such as GST, are excluded from the CAPEX.

24.1.3 PROJECT CURRENCY, ESTIMATE BASE DATE, AND FOREIGN EXCHANGE

All project capital costs are expressed in Canadian Dollars. Costs are based on April to June 2008 market conditions with no provision for cost escalation beyond this date.

Costs submitted in other currencies have been converted to Canadian Dollars. Foreign currency exchange rates applied to the CAPEX relative to Canadian Dollars are set out in Table 24.2.

Table 24.2 Foreign Exchange Rates

Base Currency	Foreign Currency
CDN$1	US$0.88

24.1.4 ACCURACY

The CAPEX, including contingency, for the mine, process plant, tailings disposal, and infrastructure has been prepared to a level of ±15%.

24.1.5 PROJECT IMPLEMENTATION

The CAPEX is based on the assumption that NATC will follow the project execution plan described in Section 23.0. Any deviation from this plan may have an impact on both project schedule and costs.

24.2 BASIS OF ESTIMATE

The direct costs have been based on the following information:

- Process flow diagrams, site layout and general arrangement drawings, equipment lists, electrical single line diagrams and, piping and instrumentation diagrams.
- Budget submissions for the design/supply of new major and secondary equipment provided by vendors in accordance with specifications and/or datasheets developed by the engineering groups involved.
- Prices for permanent materials based on supplier quotations, in-house data, and current market conditions.
- Quantity take-offs for materials, provided by Wardrop and EBA.
- Labour rates provided by local and regional construction contractors.
- Productivities for installing equipment and materials were obtained from northern construction contractors who are familiar with the project location and local conditions. Expected productivities have been discussed for this location and compared to actual results from other projects as well as with in-house data.
- Supply and installation prices from experienced vendors of pre-engineered and modular buildings as well as in-house data.

24.2.1 DIRECT COSTS

Direct costs include all new equipment and materials, as well as installation for all permanent facilities associated with:

- crushing, material handling and processing
- process equipment
- infrastructure roads and site preparation
- power supply and distribution
- pre-production mining
- tailing storage
- assay, metallurgical and environmental laboratories
- warehousing
- administration
- truck shop
- yard services and other utilities

- control and communications systems
- plant mobile equipment
- fuel storage
- cold storage
- explosives storage.

24.2.2 INDIRECT COSTS

Indirect costs include the following:

- temporary construction facilities, including worker's camp, secure lay-down areas, warehouses, etc.
- temporary construction services including some construction equipment
- freight
- vendor representatives
- first fills and capital spares
- EPCM services (including travel expenses and CM costs by Merit)
- third-party engineering
- pre-operational testing services and associated materials
- quality assurance
- surveying
- vendor representatives
- Owner's costs
- start-up and commissioning allowance.

24.2.3 QUALIFICATIONS AND ASSUMPTIONS

The CAPEX assumes that:

- Construction work will be based on unit and fixed-price contracts, not cost plus or time and materials arrangements.
- Budget quotes from vendors for equipment and materials will be valid to within ±15% of the purchase price.
- Concrete aggregate will be locally available and suitable for use.
- Competent backfill material will be suitable for use and locally available.
- Soil conditions will be adequate for foundation-bearing pressures.

- Construction activities will be continuous.
- Labour productivities have been adjusted for northern locations and established with input from experienced northern contractors and data from current projects.
- Bulk materials such as cement, rebar, structural steel and plate, cable, cable tray, and piping will be available when they are required.

24.2.4 EXCLUSIONS

The CAPEX does not include allowances for:

- cost escalation during construction
- scope changes
- interest costs during construction
- schedule delays and associated costs, such as those caused by:
  - scope changes
  - unexpected ground conditions
  - extraordinary climatic events
  - labour disputes
  - permit applications
  - receipt of information beyond the control of EPCM contractors
  - schedule recovery or acceleration
- financing costs
- sunk costs
- research and exploration drilling costs
- permitting costs
- the cost of the Feasibility Study and Technical Report.

The following items are excluded from the CAPEX, but included in the financial model:

- property taxes
- corporate and mining taxes
- sustaining capital
- working capital
- closure costs
- salvage values.

24.3 PROJECT DIRECT COSTS

24.3.1 *QUANTITY DEVELOPMENT AND PRICING*

All quantities developed are based on general arrangement drawings, process design criteria, process flow diagrams, and preliminary P&IDs. Design allowances are applied to bulk materials based on discussions between the respective discipline and the estimator. The respective discipline quantities are as detailed in the following sections.

BULK EARTHWORKS

Bulk earthworks quantities are based on rough grading designs done using Autodesk Land Development Desktop Civil Package. Excavation of overburden and allowances for rock excavation are based on geotechnical information available at the time of the study. Structural fill is costed based on aggregates being produced at site utilizing a portable crushing and screening plant. The price of the aggregate plant is included in the processed material prices contained in the capital cost estimate.

Earthwork quantities do not include an allowance for bulking or compaction of materials; these allowances are included in the unit prices. Percentages of total quantities allocated to rock (ripped or drill/blast) waste material, and cut/fill is assigned based on available soils information.

PLANT SITE SERVICES

Plant site services quantities for fire water, fresh water, and sewage disposal are based on engineering designs, sketches, and P&IDs which identify pipe sizes and routing.

CONCRETE

Concrete quantities are based on “neat” line quantities from preliminary drawings and sketches. For estimating purposes, designers provided neat quantities in the following breakdown:

- lean mix levelling concrete
- footings and foundations
- retaining walls
- grade beams and pedestals
- perimeter walls
- slab on grade

- elevated slabs
- pads curbs and sumps
- embedded metals and anchor bolts.

Quantities are provided by area as defined by the project WBS. Unit rates for each type include formwork, reinforcing steel, additives, placing, and finishing of concrete. Costing is based on ready-mixed concrete delivered from an onsite batching plant.

STRUCTURAL STEEL

Steel quantities are based on quantities developed from preliminary drawings and sketches. Allowances are included for cutoffs, bolts, and connections. For estimating purposes, the designers have provided steel quantities as per the following breakdown. The designers provided neat quantities and the allowance is added by the estimator.

Categories of steel are as follows:

- light weight steel sections: 0 to 30 kg/m (tonnes)
- medium weight steel sections: 31 to 60 kg/m (tonnes)
- heavy steel sections: 61 to 90 kg/m (tonnes)
- extra heavy steel sections: >90 kg/m (tonnes)
- platforms (tonnes)
- stairways (metres)
- checker plate (m^2)
- grating (m^2)
- handrail with kickplate (metres)
- handrail without kickplate (metres)
- ladders (metres).

Supply unit rates are based on the current world pricing for steel. The unit price for new steel purchase includes detailing and fabrication

Stick build quantities are obtained from layout sketches. “S-Frame” structural modelling and analysis software was utilized for major building structures. Cranes are included for all categories at a rate of $150/t.

PLATEWORK AND LINERS

Quantities for all platework and metal liners for tanks, launders, pumpboxes, and chutes are calculated based on detailed quantity take-offs developed from preliminary drawings and sketches and are provided in kilograms of steel. Rubber lining for pumpboxes is provided on a square metre basis. Wear plate liners and rubber liners are included as appropriate.

HVAC

Quantities for HVAC systems have been designed and calculated by Wardrop's building services engineer or obtained from in-house data. Prices have been obtained from suppliers on budgetary quotes.

Dust Collection/Suppression

All dust collection and suppression pricing has been sourced by Wardrop. Dust suppression system and dust ducting sizes/capacities are designed by normal methods.

The amount of ducting is calculated on anticipated duct runs at the project site.

The installed price of dust ducting will be based on rates supplied by Wardrop's building engineer based on previous project information. Installation costs for the equipment and ductwork are based on previous information from similar projects.

Piping

Piping quantities are based on detailed quantity take-offs for pipe over 3" (75 mm) diameter and fittings. The quantity take-offs are developed from pipe routing 'red line' drawings based on the detailed general arrangement drawings and the P&IDs, which identify pipe size. Piping is provided as separate line items and sorted by WBS area and pipe specification. Piping under 3" diameter (including fittings) is included as a total number of lines per area at an average length of 15 m. Special piping such as stainless steel is listed separately; flanges and bolt-ups are included. Allowances are included for specialty items such as flexible hoses, etc.

Painting and tagging are included as appropriate.

Pipe insulation is quantified and quoted by an insulation supplier.

Specifications for electrical heat tracing for the major pipelines are prepared and issued to suppliers for budgetary quotations. Electrical heat tracing for all other main service lines is priced by Wardrop's electrical engineer, based on Wardrop's in-house data from previous projects.

Two types (categories) for piping supports are included as follows:

- Type 1 = >300 mm pipe
- Type 2 = 50 mm – 300 mm pipe

Some small bore piping is supported on cable tray.

VALVES

All valves are listed as separate line items in the estimate and priced from budgetary quotes.

ELECTRICAL

The electrical estimate is based on budgetary quotations for electrical equipment and associated cabling.

The equipment list is used to estimate loading and generation requirements. The equipment list and site plan are used to locate electrical buildings to minimize cable runs.

A single line diagram is developed to indicate major electrical equipment including generators, transformers, feeders, power distribution centres, motor control centres, and requirements for variable frequency drives as appropriate.

Electrical buildings are designed and quoted to be fabricated off-site and delivered as complete modular units. The on-site work consists of connecting the incoming transformer feed and motor feeders. The electrical buildings are self contained with all necessary auxiliary equipment.

INSTRUMENTATION

The instrumentation estimate is based on the preliminary instrumentation index. Each piece of equipment, including cables, is based on project drawings and sketches. Instrument types and quantities are included based on developed P&IDs.

Quantities will be estimated based on electrical room locations, instrument quantities, and area layouts.

PRE-ENGINEERED BUILDINGS

Pre-engineered building costs are based on Wardrop's in-house data from previous projects.

24.3.2 Labour Cost Development

Labour Rates

Labour rates include the following:

- base rates include basic rate (based on combined union and non-union)
- payroll burdens (employment insurance, vacation pay)
- overtime shift premium rates are included; an allowance for incidental overtime will be included in the indirects section
- any applicable contractor's monthly fee will be covered in the hourly rates.

An average labour rate of CDN$81 will be utilized for the estimate.

The following activities are included in the rates as a percentage of the base rate:

- contractors field supervision including managers, general foremen, secretarial, and office personnel (overhead)
- office supplies, running costs, and vehicle costs (overhead)
- temporary facilities and utilities such as tool sheds, cribs, small equipment, maintenance facilities, power, water, and sewer (overhead)
- small tools and consumables
- miscellaneous indirect costs such as power and water distribution, clean-up, safety supervision and protection, safety training, welder's certification, etc. (overhead)
- contractors profit and overhead included as a percentage including home office overhead costs and profit
- sub-contractors mark-ups (10%) on materials.

The following labour rated items are calculated separately and included in the indirects section:

- all additional construction equipment, cranes, and incidental equipment rentals; contractors and construction costs are included in the individual line items of estimate
- mobilization and demobilization costs
- freight costs relating to contractors materials are included in the mobilization and freight sections
- turnaround costs for contractors and construction management/owner personnel
- travel time for personnel if appropriate

- construction camp catering, housekeeping, and maintenance costs (daily rate).

Labour Premiums

The labour rate is based on a construction schedule of three week in and one week out, 70 hour work week. Scheduled site hours are: 7 days x 10 hours = 70 hours (40 hours x 1 and 30 hours x 1.5).

Productivity

The labour hours productivity factor for the capital cost is estimated to be 1.15 to 1, or 87% efficiency. This productivity factor was agreed to by Merit.

24.4 Project Indirect Costs

24.4.1 *Temporary Construction Facilities and Services*

The following contractor costs are included in the indirect cost estimate:

- contractors' mobilization and demobilization
- construction equipment
- construction field offices, furnishings and equipment
- contractor travel and accommodations
- temporary power supply
- temporary water supply
- temporary heating and hoarding
- warehouse and laydown costs
- temporary toilets
- temporary communications
- on-going and final clean-up
- yard maintenance
- janitorial services
- site safety personnel and training.

Indirect costs associated with construction equipment are absorbed by contractors' unit prices, and are therefore not included in major earthmoving costs.

24.4.2 CONSTRUCTION CAMP AND CATERING

A construction camp will be built to accommodate up to 250 construction workers, construction management staff, and visitors.

Average catering costs are estimated to be $65 per camp man-day. This estimate accounts for varying construction camp occupancy levels throughout the life of the camp.

24.4.3 SPARE PARTS AND FIRST FILLS

An allowance for spare parts is included as follows:

- commissioning spares – 1.5 % of process equipment
- capital spares – 3% of process equipment
- capital mining spares – 5% of mine rolling stock.

Industry-standard allowances have been included for first fills for items such as start-up grinding media, reagents, and fuel.

24.4.4 START-UP AND COMMISSIONING

In some cases, vendor representatives, contractors' crews, and management staff will be required on-site to supervise equipment installation perform pre-start-up inspections, in order to satisfy equipment performance warranty requirements. Costs associated with this requirement have been included in the estimate.

24.4.5 FREIGHT

Freight costs for large and/or heavy equipment items are based on quotes for shipping directly from manufacturers' fabrication plant. Some equipment purchase estimates include shipping. In cases where freight costs are not available, an allowance has been included in the estimate. All estimates include packing and crating.

24.4.6 ENGINEERING, PROCUREMENT, AND CONSTRUCTION MANAGEMENT

Engineering and procurement costs were provided by Wardrop based on the required specifications and estimated number of drawings. Construction management costs were provided by Merit.

24.4.7 TAXES AND DUTIES

The CAPEX does not include PST and GST.

24.5 OWNERS COSTS

The estimate for Owner's costs, developed by NATC, includes items such as:

- NATC project management
- environmental permitting, testing, and monitoring
- socio-economic management
- insurance, taxes, and fees
- geotechnical and resource drilling
- recruitment and relocation.

24.6 CONTINGENCY

A contingency of 13.5% of the total direct and indirect costs has been included to meet unanticipated costs within the scope of the estimate.

The contingency was estimated on a discipline-by-discipline basis, taking into account items that have been quoted, estimated, or factored. The contingency is a subjective allowance based on the degree of confidence that study contributors feel should be applied to their work

25.0 OPERATING COST ESTIMATE

On site operating costs are estimated to be CDN$103.65/t of ore mined including mining, processing, general and administrative, and surface operations. The unit costs, summarized in Table 25.1, are based on annual ore production of 730,000 t/a and 365 days of operation.

Table 25.1 Operating Cost Summary

Area	Unit Cost ($/t milled)
Processing	36.39
Mining	38.14
General and Administrative	13.65
Surface Services	15.48
Total Operating Cost	**103.65**

The operating costs were developed based on the organizational chart presented in Figure 25.1.

25.1 BASIS OF ESTIMATE

The operating cost estimate has an accuracy of ±15% and is categorized as labour, power, and consumables.

25.1.1 LABOUR

Sources used for establishing labour rates included Western Mining Engineering's published figures and NATC's operating Cantung Mine. Loaded salary for hourly wages is calculated based on 1,986 hours per year, 1.1% overtime rate, and 28% overhead. Loaded salary for salary based wages is calculated based on 28% overhead. Overhead costs account for benefits such as CPP, WCB, EI, life insurance, and long-term disabilities, pension plan, and statutory holidays. Labour wage rates reflect different crew schedules and hours of work. The proposed operating schedule is shown in Table 25.2.

Figure 25.1 Operating Cost Organizational Chart

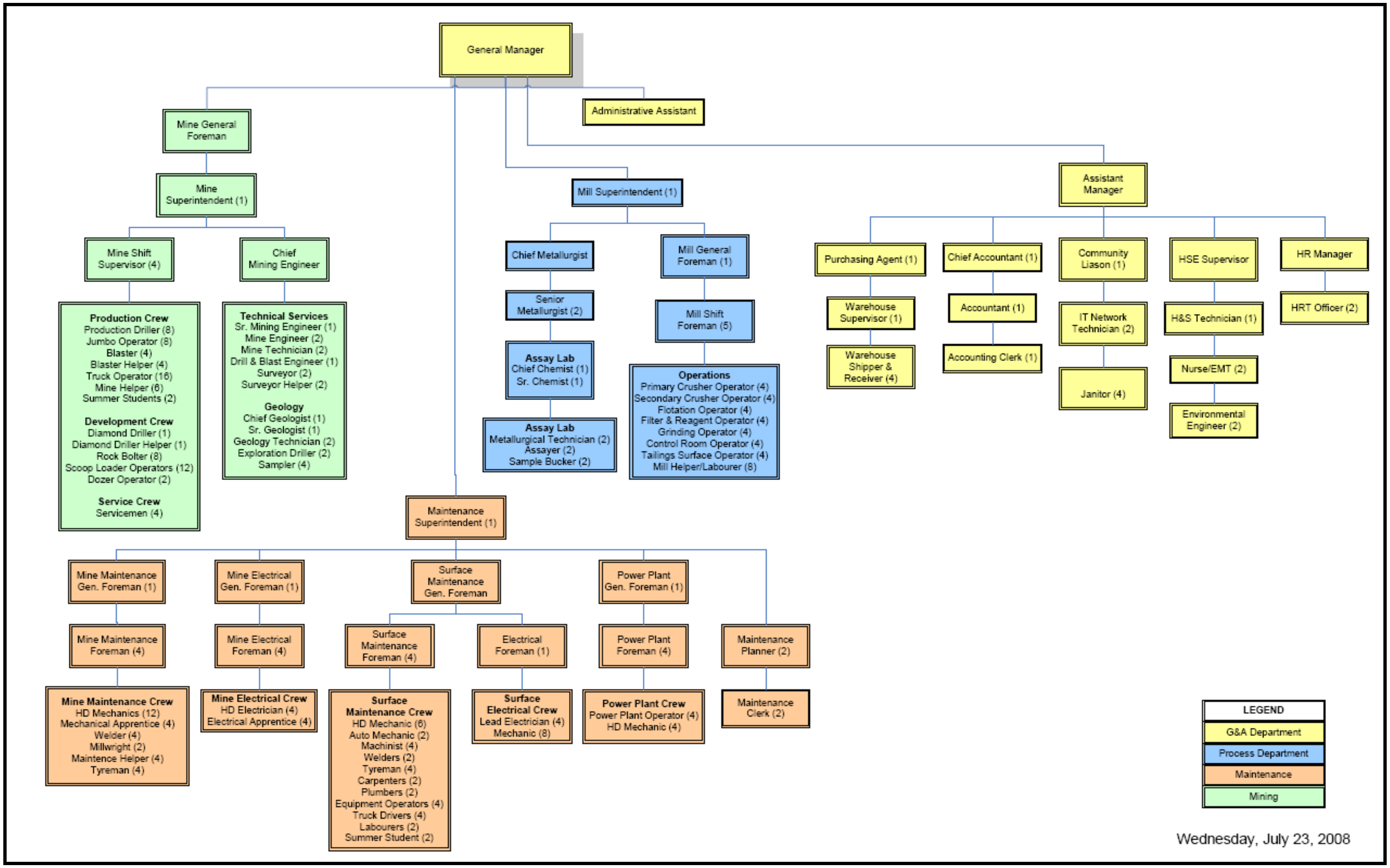


Table 25.2 Operating Schedule

Personnel	Shift Length (h/d)	Rotation	Shifts per Day
Senior Management	12	3 weeks on/3 weeks off	1
Mining (Underground)	10	3 weeks on/3 weeks off	2
Process Plant	12	3 weeks on/3 weeks off	2
Surface Services	12	3 weeks on/3 weeks off	2

25.1.2 POWER

Pricing for power is based on the use of a diesel power generating facility with a fuel consumption rate of 230 g/kWh and a fuel cost of CDN$1.31/L. Based on these factors, the electrical energy cost is $0.31/kWh. Fuel cost of CDN$1.31/L is based on current Cantung delivered price and calculated to an equivalent landed price at Mactung.

25.1.3 CONSUMABLES

Consumables include operating maintenance supplies as well as fuel. Operating supply costs are based on industry standards and budgetary prices from vendors of consumables and reagents.

25.2 MINING

Unit mining costs have been developed for the annual tonnages mined in the Production Schedule as shown in Table 18.23. The distribution of costs has been identified as direct mining (drilling, blasting, loading, and hauling) and general mine expense (supervision, engineering, geology, and mine electrical and mechanical services).

The years 2013 to 2024 life-of-mine average cost of CDN$37.07/t ore mined by LHB and CDN$46.74/t ore mined by MCF, thus giving an average operating cost of CDN$38.14/t ore mined. These costs are summarized in Table 25.3 and Table 25.4. Costs incurred during pre-production and totalling CDN$39.6 million, reported in Table 18.28, have been excluded from these unit costs. Further details regarding the mining operating costs are provided in Section 18.14.

Table 25.3 Direct Mining and General Mine Expense

	Unit Cost by LHB (CDN$/t Ore)	Unit Cost by MCF (CDN$/t Ore)
Direct Mining	28.18	37.23
General Mine Expense	8.89	9.51
Total	**$37.07**	**$46.74**
Portion of Production	89%	11%
Average Mining Cost	**38.14**	

Table 25.4 Mine Operating Cost Summary

Description	Personnel	Annual Cost (CDN$)	Unit Cost (CDN$/t ore)
Labour			
Mine Staff	21	2,171,000	2.97
Mine Labour	80	9,061,000	12.41
Mine Maintenance Staff	4	460,000	0.63
Mine Maintenance Labour	26	2,695,000	3.69
Sub-total Staff	**131**	**14,387,000**	**19.71**
Utility			
Power		1,975,374	2.71
Sub-total Utility		**1,975,374**	**2.71**
Consumables			
Operating Supplies		5,616,831	7.69
Maintenance Supplies		3,648,314	5.00
Fuel		2,211,613	3.03
Sub-total Supplies		**11,476,757**	**15.72**
Total Mine Operating Cost		**27,839,131**	**38.14**

25.3 PROCESS

The process operating cost per tonne of ore is calculated at CDN$36.39. The mill is sized to process 2,000 t/d feed with an availability of 94%.

Table 25.5 shows a summary of the process operating costs.

Table 25.5 Process Operating Cost Summary

Description	Personnel	Annual Cost (CDN$)	Unit Cost (CDN$/t ore)
Labour			
Mill Staff	10	1,036,000	1.42
Mill Labour	34	2,401,440	3.29
Mill Maintenance Staff	2	218,000	0.30
Maintenance Labour	18	1,392,000	1.91
Sub-total Staff	**64**	**5,047,440**	**6.91**
Utility			
Power		13,401,000	18.36
Sub-total Utility		**14,156,000**	**18.36**
Consumables			
Operating Supplies		6,207,000	8.50
Maintenance Supplies		1,908,000	2.61
Sub-total Supplies		**8,115,000**	**11.12**
Total Process Operating Cost		**26,563,440**	**36.39**

The process operating supplies include reagents consumption, comminution wear, assay, and dewatering supplies. Reagent consumption rates are based on plant metallurgical balance and the reagent costs are from vendors. The comminution wear estimate is based on industry experience and vendors for the key consumables.

The process maintenance supply costs are factored from equipment costs.

25.4 GENERAL AND ADMINISTRATIVE, AND SURFACE SERVICES

G&A and surface services are based on current operations at Cantung. The operating costs include:

- salaries for administrative personnel
- expenses and services related to human resources, safety, and environment
- legal, insurance, property tax
- expenses for personnel travel
- general maintenance
- consultant fees
- freight charges.

A summary of the G&A and plant services are provided in Table 25.6 and Table 25.7, respectively.

Table 25.6 G&A Operating Cost

Description	Personnel	Annual Cost (CDN$)	Unit Cost (CDN$/t ore)
Labour			
G&A Staff	6	747,000	1.02
Purchasing/Warehouse	7	654,000	0.90
Human Resources	2	230,000	0.32
Health and Safety	4	374,000	0.51
Environment	3	302,000	0.41
Sub-total Staff	**22**	**2,307,000**	**3.16**
Expenses and Services			
Safety Supplies		769,693	1.05
Administration		301,776	0.41
Human Resources		98,928	0.14
Legal, insurance and taxes		1,258,409	1.72
Head office, travel, and consultants		5,065,677	6.94
Maintenance Supplies		29,181	0.04
Freight		132,300	0.18
Sub-total Supplies		**7,655,963**	**10.49**
Total G&A Operating Cost		**9,962,963**	**13.65**

Table 25.7 Surface Services Operating Cost

Description	Personnel	Annual Cost (CDN$)	Unit Cost (CDN$/t ore)
Labour			
Surface Maintenance Staff	8	883,000	1.21
Surface Maintenance Labour	28	1,856,000	2.54
Sub-total Staff	**36**	**2,739,000**	**3.75**
Utility			
Power		4,858,000	6.65
Propane		326,000	0.45
Sub-total Utility		**5,184,000**	**7.10**
Consumables			
Operating Supplies		581,000	0.80
Maintenance Supplies		202,000	0.28
Fuel		2,593,000	3.55
Sub-total Supplies		**3,376,000**	**4.62**
Total Surface Operating Cost		**11,299,000**	**15.48**

26.0 PRELIMINARY MARKET REVIEW

The supply and demand of most industrial metals and minerals is cyclic and is influenced by many factors. These factors include changes in demand patterns and the substitution of alternative products, the discovery and availability of new mines, and the political and economic structures of the key producing regions. There is almost no exception to these rules.

GBRM has prepared this review for the purpose of providing an overview of the global tungsten industry and to examine both the activity of existing producers and current and forward consumption patterns. In preparing this review, GBRM has reviewed a large amount of data and while in some instances the validation of such data cannot be guaranteed, GBRM knows of no reason why such data should not be relied upon for the preparation of this review. This review is intended to form part of the feasibility study and while no guarantee can exist with respect to future metal prices, the most likely scenario for tungsten metal prices based on all of the known factors is reasonable based on the author’s professional skill, experience and judgement as well as the reasonable expectation that the factors outlined which can influence tungsten prices have been considered in determining such prices.

26.1 PRODUCTION AND SUPPLY

In the late 1980s and early 1990s, the supply structure of many of the industrial minerals and metals traditionally produced and consumed in Western markets was disrupted by two major changes in the market. These were increases in Chinese production, and the change of the Commonwealth of Independent States (CIS) (former Soviet Bloc) from importer to exporter.

Tungsten was one of the minerals quite dramatically affected by these changes. By the early 1990s, China with significant reserves and low cost of production, had secured over 90% of world markets for the production and sale of base concentrates. The Chinese processors also flooded the market with APT at prices marginally above the concentrate price. As a direct result and despite previously strong financial histories, the great majority of Western tungsten miners were driven from the market. Additionally, most APT capacity outside China was eventually forced to close.

Tungsten prices (both concentrate and APT) remained extremely low during the 1990s and in fact well below the true cost of production. Many Chinese mines were ‘high graded’ and APT plants were artificially supported by government export incentives. As a direct result, exploration and mine development programs outside China were almost totally abandoned.

However, commencing in the early part of this decade, the global market structure for tungsten again began to change, particularly driven by a rapid increase in domestic demand of tungsten products in China. This signalled to the global market, two key issues:

- The rapid increase in demand in China would tighten raw material availability to other markets, particularly given the Chinese Government policy of curtailing mining programs to maintain reserves for future domestic requirements
- The Chinese program of developing downstream processing would place increased pressure on processing companies outside China. In order to remain competitive and secure sufficient base concentrates, these companies urgently needed to develop alternative supplies outside China.

Recognition of these circumstances has since led to a significant increase in exploration and mine development activities outside China and particularly in Vietnam, Australia, and in North and South America. However, despite this increased activity, and apart from the restart of the Canadian Cantung mine in 2005, only small projects (i.e. Los Santos project in Spain, Malaga Project in Peru, and the Wolfram Camp project in Australia) have commenced operation. No new major production has actually been realized and this is unlikely to occur until 2010 at the earliest.

Rapid increases in mine development and operating costs, and also the very limited availability of high grade deposits, is increasingly indicating that a further advance in price structures is necessary before major new mining programs can actually be achieved.

The lack of new production has been recognized by Chinese processors which have recently entered into a number of agreements with companies such as King Island Scheelite and NATC in order to assist development through input of capital and off-take agreements.

26.2 Growth Analysis and Other Factors That Will Influence Future Demand

26.2.1 Expansion in Current Markets

The consumption of tungsten is reasonably broad based, both in industrial and geographical terms. Major applications are metal-cutting tools, drill bits, light bulb filaments, high temperature alloys, x-ray shielding, military use, and chemical applications. Regionally, the largest consuming area is China, followed by Europe and North America. China also has by far the fastest growth; however, India is also showing signs of rapid growth in demand albeit from a low base.

Most applications for tungsten are relatively mature and it is difficult to identify major new uses. Minor exceptions include the replacement of lead bullets and lead fishing weights by tungsten, primarily for environmental reasons. However, this would still represent a relatively small percentage of the market, which is unlikely to greatly impact on current supply and demand structures.

More conservative projections are that China's demand will be approximately 10% per annum. Other major markets such as Europe, Japan and the USA are mature and are expected to show a long-term growth of 2%. Developing markets such as Russia, South America, Korea, India etc, while not at buoyant as China have been projected to experience a medium term (5 year) growth rate of 5%.

Scrap is projected to increase over the period and the overall percentage of scrap in world-wide tungsten consumption has been increased from 22% to 25%.

Over the next five years to the end of 2012, global consumption of tungsten is expected to increase from its current level of approximately 85,000 t W (64,500 t W virgin tungsten) to 110,000 t W (82,000t W virgin tungsten) per annum (See Table 26.1). Mature markets such as Europe and North America are only expected to grow by an average of 2% per annum. Chinese domestic consumption is forecast to continue growing in excess of 10% per annum, largely driven by the increase in requirements for cutting and drilling tools.

However, it should also be noted that China's growth over the past five years has averaged 15% and if these levels were to be maintained world tungsten demand will exceed 122,000 t W (91,500 t W virgin demand) by 2012. In addition, China's continual move to end products may influence the direct consumption of tungsten in USA and Europe as imported finished goods replace domestic production.

Table 26.1 Tungsten World Demand Projected 2006-2012 (t W)

	Growth Rate	Multiplier	2006	2007	2008	2009	2010	2011	2012
China	10	1.1	23500	25850	28435	31279	34406	37847	41632
Japan	2	1.02	12000	12240	12485	12734	12989	13249	13514
Europe	2	1.02	20000	20400	20808	21224	21649	22082	22523
USA	2	1.02	13000	13260	13525	13796	14072	14353	14640
Other	5	1.05	12700	13335	14002	14702	15437	16209	17019
			81200	85085	89255	93735	98553	103739	109328
Recycle %			22	23	24	25	25	25	25
Stockpile			3500	1000	0	0	0	0	0
Virgin demand			59836	64515	67834	70301	73915	77805	81996
China	15	1.15	23500	27025	31079	35741	41102	47267	54357
Japan	2	1.02	12000	12240	12485	12734	12989	13249	13514
Europe	2	1.02	20000	20400	20808	21224	21649	22082	22523

USA	2	1.02	13000	13260	13525	13796	14072	14353	14640
Other	5	1.05	12700	13335	14002	14702	15437	16209	17019
			81200	86260	91899	98197	105248	113159	122053
Recycle %			22	23	24	25	25	25	25
Stockpile			3500	1000	0	0	0	0	0
Virgin demand			59836	65420	69843	73648	78936	84869	91540

26.2.2 POTENTIAL NEW DEMAND

There are a few applications that provide potential for increased consumption.

- In principle, the US military has recently approved the use of tungsten as a replacement for lead core bullets. However, replacement is likely to be a gradual transition, rather than a total industry change in the immediate future.
- Expanded super alloy milling capacity has provided new product potential for tungsten and tungsten alloys.
- Other relatively new uses including weighting in golf clubs and golf balls, significant use in motor vehicles, replacement of lead fishing weights, self darkening window panes and corrosion resistance of optical glass.

26.3 MEDIUM TO LONG TERM TUNGSTEN SUPPLY ANALYSIS

26.3.1 MEDIUM TERM ANALYSIS 2006-2012

On the basis of 10% growth in China, global consumption of tungsten is expected to grow steadily from its current level of approximately 85,000 tonnes W per annum in 2007 (64,500t virgin tungsten) to 109,328 tonnes W (81,996t virgin tungsten) per annum by 2012. However, should China continue to grow at the 15% rate experienced between 1996-2006, then demand will be 122,053t W (91,540t of virgin tungsten) as shown in Table 26.2.

Consequently, increased supply required to meet growth in tungsten demand from 2006-2012 will be between 28 and 40kt W. In addition some current day operations are likely to be worked out in the next five to six years. Based on a 10% decline in current operations over the period 2006-2012 a further 5kt W of new production will be required taking overall requirements to between 33 and 45kt W. Areas, from which this increased supply could be serviced, are detailed in Table 26.2.

As the majority of these projects are still under feasibility it is considered unlikely that they will all progress to production. On this basis tungsten is projected to remain in tight demand until 2012 and probably beyond (Figure 26.1). Additional tungsten projects to those already announced will be required in order to meet supply growth over this period.

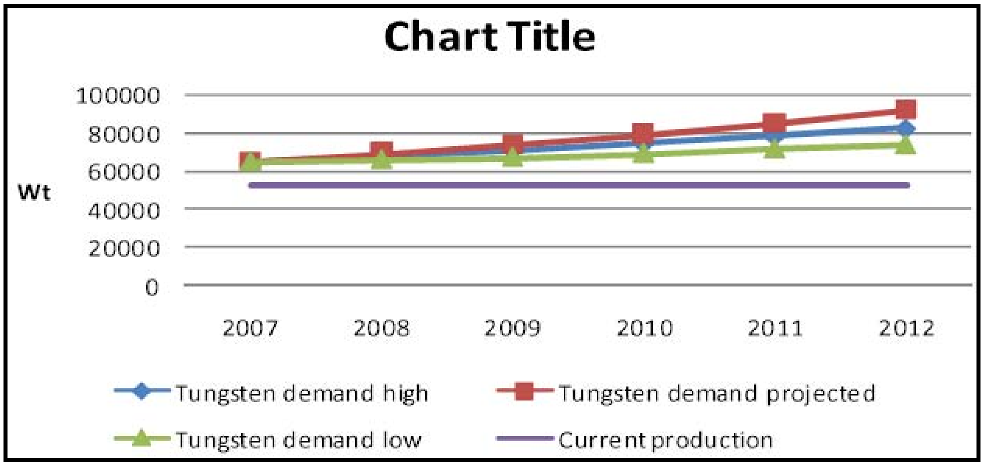
Even if China's production dropped to 5% per annum and Western world consumption was limited to 1.5% per annum, an additional 19ktpa W will be required by 2012 accommodating some 75% of propose projects.

Table 26.2 Potential Additional Tungsten Production (2006-2012)

Area		Tpa Tungsten	Year
Nui Phao		3,800	2009
Peru		1,400	2006/7
Brazil		1,000	2007/10
Russia		4,000	By 2010
Australia			
	Vital	3,100	2010
	KIS	2,250	2009-2010
	Thor	1,000	2009-2010
	QOL	330	2009
Canada		2,000	By 2012
Thailand		2,000	By 2012
China 10% net increase		4,000	
Africa		1,200	2009
Germany		780	2010
Spain		1,000	2008
Total		**27,860**	

The recent increase in US demand of 9.0% between 2006/2007 indicates that tungsten growth will potentially exceed projected demand and could currently be restricted by supply.

Figure 26.1 Projected Tungsten Supply and Demand 2007-2012

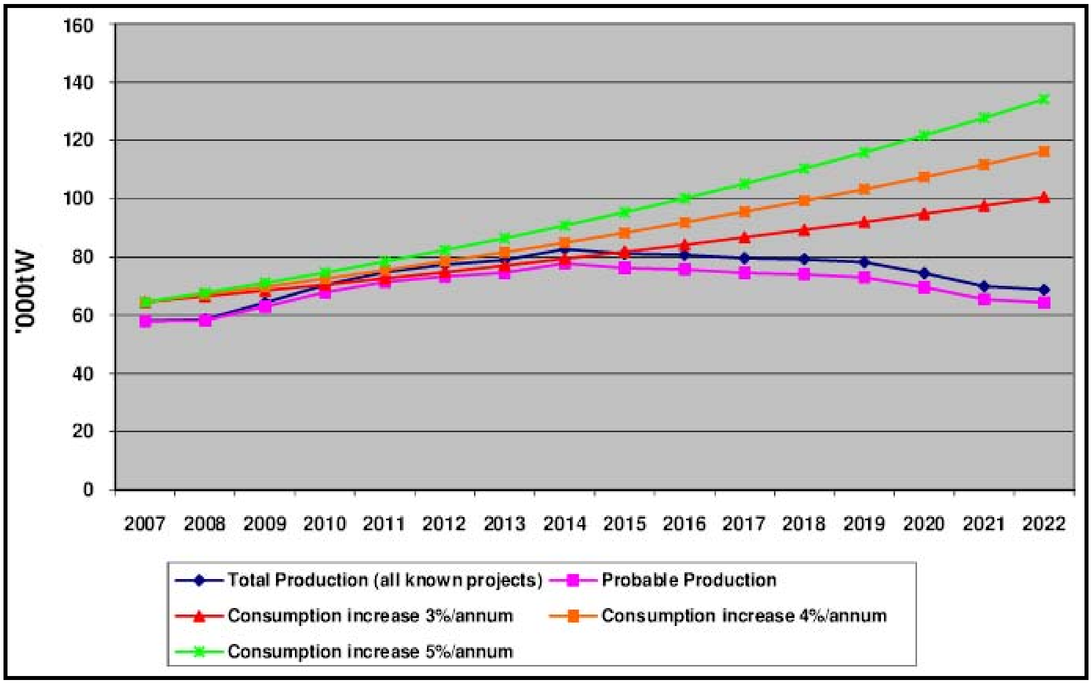


26.3.2 *LONG TERM (15 YEAR) ANALYSIS 2007-2022*

In preparing a 15 year assessment (Figure 26.2), the following assumptions have been made.

- Total production – all known projects. All announced projects, plus some general production increases for small projects in a range of countries.
- Probable production – large projects under feasibility have been given a 90% probability of coming on stream as per the timelines projected by the companies. Smaller projects have been given an 80% probability. These probabilities are considered to be extremely aggressive, and it is quite probable that actual production build-up will be lower than projected.
- In all cases it has been assumed projects will take three years to build-up to projected capacity. Mine life has been applied based on company projections.
- Long term consumption trend lines have been based on growth trends of 3, 4 and 5% per annum. This is considered reasonable based on long-term world growth projections of 3.6% GDP and the fact that industrial growth has historically exceeded GDP.

Figure 26.2 Supply Projection



It is clear from this projection that:

- There is likely to be continued shortage of supply from 2008-2012.
- Development of the large and medium projects Mactung, Vital, King Island Scheelite and Nui Phao are critical in maintaining the long-term supply demand balance.
- Should the projects not be brought on line as per the current projections by the companies, the projected short-term supply shortages will continue past 2010.
- The only dedicated tungsten project that currently offers significant production and long (>10 years) mine life is Mactung. The development of this project by 2013 will be critical to long-term market supply and stability.
- Additional significant deposits need to be located and developed in the timeframe 2014 onwards to negate the potential for further supply shortfalls, as smaller short-term mines are worked out.

In order to support the development of new projects, it is critical that tungsten prices are at levels that provide ongoing adequate returns to these projects.

26.4 PRICING STRUCTURE

26.4.1 MARKET PRICE CORRELATION FOR APT AND CONCENTRATES

The price of APT is related to the price of concentrate and with corresponding trends. However it is not always an absolute correlation and prices can vary depending upon processing costs and the prevailing attitude of individual APT producers.

In turn, as a general guideline and under current market circumstances it is reasonable to assume that the market price for APT is approximately US$30-$50/mtu higher than the corresponding price for concentrates. Alternatively put, APT is increasingly becoming the traded commodity in tungsten, and tungsten concentrate has been declining. Therefore it is generally safer to adopt the prevailing price for APT and then deduct US$30-$50/mtu as a guideline price for concentrate prices, however some variation in price will prevail depending upon quality and the point of delivery etc.

26.4.2 APT PRICE STRUCTURES 2001-2007

Over the past six years there has been significant variation in APT prices.

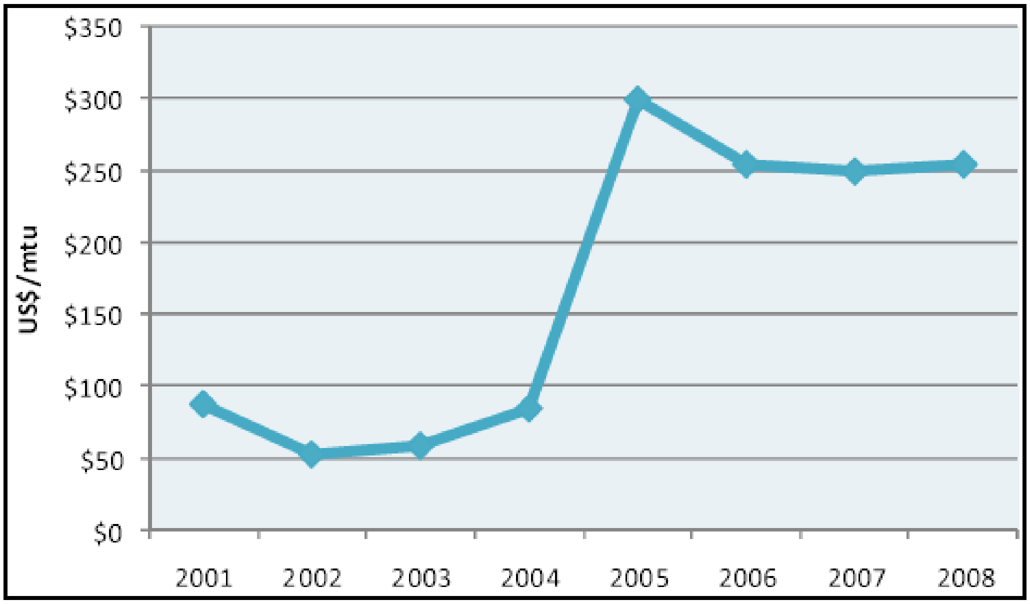
In 2001 market demand was strong as companies built up stocks, being particularly concerned about the potential impact of recently introduced Chinese export restrictions and diminished supplies from the CIS strategic stockpile. This provided an upward impact on prices.

However, the projected shortages did not eventuate due to Chinese production remaining high and Cantung Canada coming into production. Consequently consumers drew down their stocks and prices again declined.

In 2003 APT prices averaged approximately US$60/mtu and there did not appear to be much potential for a price increase with supply and demand in balance.

However commencing in mid 2004, and on the back of rapidly increasing demand, prices increased quickly, moving to a new price structure well in excess of US$200/mtu and for a short period reaching US$300/mtu. However, following this dramatic increase, prices have since stabilised and have remained relatively steady during the past two years. The current price level for APT is US$250-260mtu FOB China.

Figure 26.3 APT Price Structure 2001-2008



26.4.3 FUTURE PRICING 2007-2015

Whilst APT prices currently appear to have stabilised at between US$250 and US$260/mtu FOB China, tungsten processing companies both within China and more particularly outside China remain extremely concerned about future raw material supply. Furthermore in Japan and South Korea, there is even some discussion about Government agencies establishing strategic stockpiles

With no new mining projects actually confirmed and with demand projected to increase between 17-26kt W over the next five years, this concern seems well justified.

Challenges for the development of new tungsten mining projects outside China continue to increase, particularly in relation to economic grade availability, higher capital costs, and environmental constraints, with the net result that the time taken to actually establish new mining programs continues to extend.

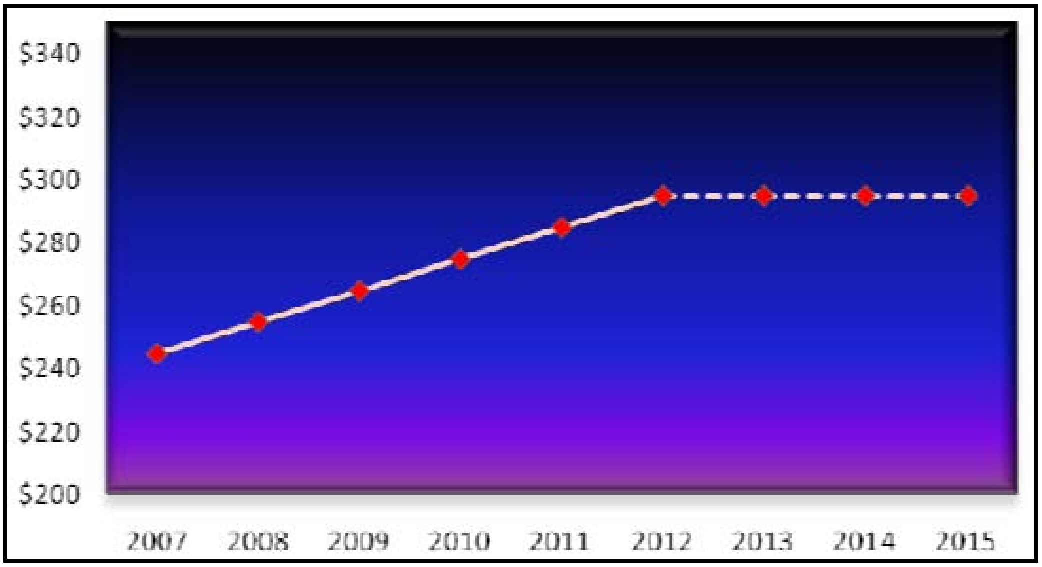
Increased mine development and operating costs is in fact the major issue, and investors in these projects will need to achieve a suitable return on investment before the new projects are confirmed and realised. With tungsten production based on low grade deposits (i.e. <1% WO3), the fuel costs associated with mining of significant tonnages/annum and power costs incurred in crushing and grinding these tonnages are, together with increasing labour costs, having a significant impact on operating costs.

These increased costs and the indicated timing issues in actually establishing new mining programs strongly suggests that supply pressure on the market will continue to increase. In turn, and to provide greater encouragement to mine developers, further price increases seem inevitable and during the next five years (if not sooner) it now seems certain that global market prices for APT will at least reach US$300/mtu (See Table 26.3).

Table 26.3 Forecast APT Prices 2007-2012

2007	2008	2009	2010	2011	2012
$240/250	$250/260	$260/270	$270/280	$280/290	$290/300

Figure 26.4 Forecast APT Prices 2007-2015



While it is difficult to accurately forecast pricing past 2012, this assessment clearly indicates that,based on projected supply and identified demand, APT prices will continue at, or above, US$300/mtu.

The only major development currently proposed for the period 2012-2015 is the Mactung Project which, based on supply and demand projections, the market clearly requires in order to develop a stable base around larger long-term producers.

Many of the tungsten projects which are at the early stage of assessment that could be brought into production in the longer term are based on much lower WO3 grades than Mactung, Vital and King Island Scheelite. Consequently these projects will require high tungsten prices in order to be viable.

The need to develop these types of lower grade projects, and the associated high operating costs, could be a further driver towards prices above the projected US$300/mtu from 2012 onwards (Figure XXX).

26.4.4 SUMMARY

As a result of the constant over-supply situation through the 1990s and early 2000s, prices fell to low levels of US$45t/mtu for concentrates and only marginally higher for APT. However, as indicated, a number of important events have occurred since then that have greatly influenced current and forward price structures.

China has not only curtailed domestic mining programs, but has now become a significant importer of tungsten concentrates and tungsten scrap. Also the Chinese Government has moved from a position of providing export incentives for tungsten exports to now introducing production and export quotas, and consistently increasing export tariffs.

The net result of these changes coupled with the continuing growth in global demand has resulted in strong price increases commencing in 2004, to a current level of approximately US$220/mtu for concentrates and US$250-$260/mtu for APT (quoted price by Asian Metal News May 2008).

With the Chinese Government strongly encouraging an even higher level of downstream processing (with a large percentage of these downstream products being consumed domestically, plus expected further increases in export tariffs for semi processed products) the availability of tungsten units to markets outside China in all forms will continue to tighten. Furthermore, China has now become a major importer of tungsten concentrates and scrap.

The net result of these changing circumstances (continued strong growth in China's consumption, the recognition of increasing costs to develop and operate mines outside China, plus the fact that tungsten has a 'high value in use' in most applications and substitution is unlikely) result in the conclusion that a continuing escalation in price structures is highly likely. During the next five years, it is forecast that global prices for APT will reach or even exceed US$300/mtu.

26.5 OPPORTUNITY FOR NEW PRODUCERS

The projected strong growth in tungsten demand over the period 2007-2012 will require the development of between 33-45 kt W of new production.

Analysis of current projects that are in the feasibility study stage indicates that, even if the majority come on stream, the market will continue in tight supply through to 2012 and onwards. However, a number of these projects are associated with junior mining companies that are likely to find it difficult to obtain support in today's tight financial markets.

Many of the tungsten projects that are in the early stage of assessment that could be brought into production from 2012 onwards, are based on much lower WO_3 grades than the projects currently under feasibility and will require stable tungsten prices at high levels to be feasible.

Consequently there is a definite requirement for the development of larger (+4,000 t/a W) mines with 15 to 20 year mine life in order to support long-term market demand.

These types of world-class deposits are rare and, of the projects under feasibility study, only NATC's Mactung deposit is capable of achieving these production rates over 10 years plus.

This project is currently scheduled to come on line in 2013, with production of 30,000 t W over its initial 5 years of operation. Projections are that the market will clearly require this additional production by 2013 and could in fact accommodate an earlier start-up.

In addition to Mactung, there are only four other projects under feasibility; Vital Metals Watershed Project (Australia), King Island Scheelite's King Island Project (Australia), the Nui Phao Deposit in Vietnam, and the Hemerdon deposit in the UK that are projecting production levels of between 2,000 to 4,000 t/a W.

The majority of other projects currently scheduled by the owners to come on line by 2012 will produce less than 2,000 t/a and are based on small relatively short-term deposits. On this basis, it is hard to identify sufficient supply to meet demand in 2012, and even harder in the longer term (2012 onwards).

With both capital and operating costs increasing significantly with processing of lower grade material, the tungsten price will need to remain high (+US$300/mtu APT) over the long term in order to attract the development of the lower grade deposits currently being assessed.

This strong price trend will conversely underwrite the profitability of the projects scheduled to come on line between 2010 and 2013, which are based on higher grades (i.e. Mactung, Vital Metals, and King Island Scheelite).

Barriers of entry for a new producer outside China with a good grade tungsten deposit are expected to be minimal. In the projected strong market, long term letters of intent to purchase either concentrates or APT should be readily achievable and these would underwrite project finance. However, these letters of intent would be firmly subject to the following conditions:

- Confirmation that the operation would be relatively competitive with existing production programs, and that this position could be sustained for the life of the supply contract. European, North American, and Japanese processors must still be able to access cost competitive concentrates in order to maintain a competitive position against finished products from China.

- Confirmation of the required quality, with particular attention to the nominated impurity levels, including radioactivity.

26.6 CONCLUSION

The global market for tungsten is forecast to maintain strong growth over the next five years and beyond. While China continues to dominate world mining and primary processing, the availability of tungsten units to non-Chinese markets will continue to decline. Equally important, China is now becoming a major importer of tungsten concentrates and scrap materials. Ongoing rapid growth in demand by China will ensure that competition for raw materials between Chinese and non-Chinese processors will continue to intensify with the result that there is now an urgent need for increased mining programs outside China.

A strong escalation in prices has already occurred over the past three years. However with producers struggling to meet demand, global mining costs continuing to increase, mining grades dropping, and the Chinese Government likely to impose tighter production quotas and higher export tariffs to maintain reserves, further global price escalation appears certain.

Barriers of entry for new producers are relatively low apart from actual development costs, and new mine projects will continue to receive strong encouragement from processors both in and outside China.

Despite this, only limited new tungsten production has been developed over the past 3 to 5 years. The only major new production has been the Cantung mine in Canada with smaller production from Malaga (Peru), Los Santos (Spain), and the recent commencement of mining at Wolfram Camp (Australia).

While there are a number of other projects at various stages of development, from exploration to Bankable Feasibility Study, the timing to bring these projects into production is proving to be considerably longer than originally projected.

Much of this delay has been due to the world-wide concerns impacting on the mining industry regarding equipment supply, labour, and increased complexity of government approvals. In addition, in the tungsten industry, many of the potential producers are junior miners that are finding it difficult to raise the necessary finance for the projects.

Consequently, for the long term stability of supply consumers are keen to form relationships that will assist in the development of the larger, longer term producers with good trade tungsten deposits.

27.0 FINANCIAL ANALYSIS

27.1 INTRODUCTION

Wardrop developed an economic evaluation of the Mactung project based on a pre-tax financial model. For the expected 11 year underground mine life, and 8 Mt reserve, the following base case pre-tax financial parameters were calculated:

- 23.5 % internal rate of return (IRR)
- 2.9 years payback on US$402.1 M initial capital
- CDN$276.8 M at an 8% discount rate.

The base case is calculated using an APT price of US$300/mtu. Refer to section 26.0 Market Review for more detailed commodity pricing information.

The post-tax model was provided by NATC and audited by W.H. Taylor Inc. calculating the following:

- 17.6 % IRR
- 3.2 year payback on US$402.1 M initial capital
- CDN$ 147.7 M NPV at an 8.0% discount rate.

A sensitivity analysis was carried out to evaluate the project economics against the base case and alternative pricing scenarios.

The pre-tax and post-tax discounted cash flow analysis for the base is shown at the end of this section.

27.2 PRE-TAX MODEL

27.2.1 FINANCIAL EVALUATIONS – NPV AND IRR

The pre-tax base case financial model was calculated based on the following parameters:

- US$255 per mtu, gravity concentrate
- US$300 per mtu, APT concentrate
- a 24-month pre-production period with a distribution of 45% and 55% for the initial capital costs of year 1 and year 2 respectively
- an 11.2 year mine life

- off-site charges including transport, marketing, and insurance
- direct on-site operating charges including mining, milling, maintenance and surface, and general & administrative
- a four-year average currency exchange rate of CDN$1.00 = US$0.88 as of July 18, 2008
- royalty of 1% of the net revenue starting in Year 1.

The mine production schedule shown in Table 18.25 was incorporated into the 100% equity pre-tax financial model to develop annual recovered commodity production from the relationships of tonnage milled, head grades, and recoveries. The market price for tungsten was adjusted to a realized price level by applying transport, marketing, insurance, and toll conversion costs to determine the net revenue.

Unit operating costs for mining, milling, maintenance, and general & administrative areas were applied to annual mined and milled tonnages to determine the overall mine site operating cost which has been deducted from the net revenue to derive annual cash flows.

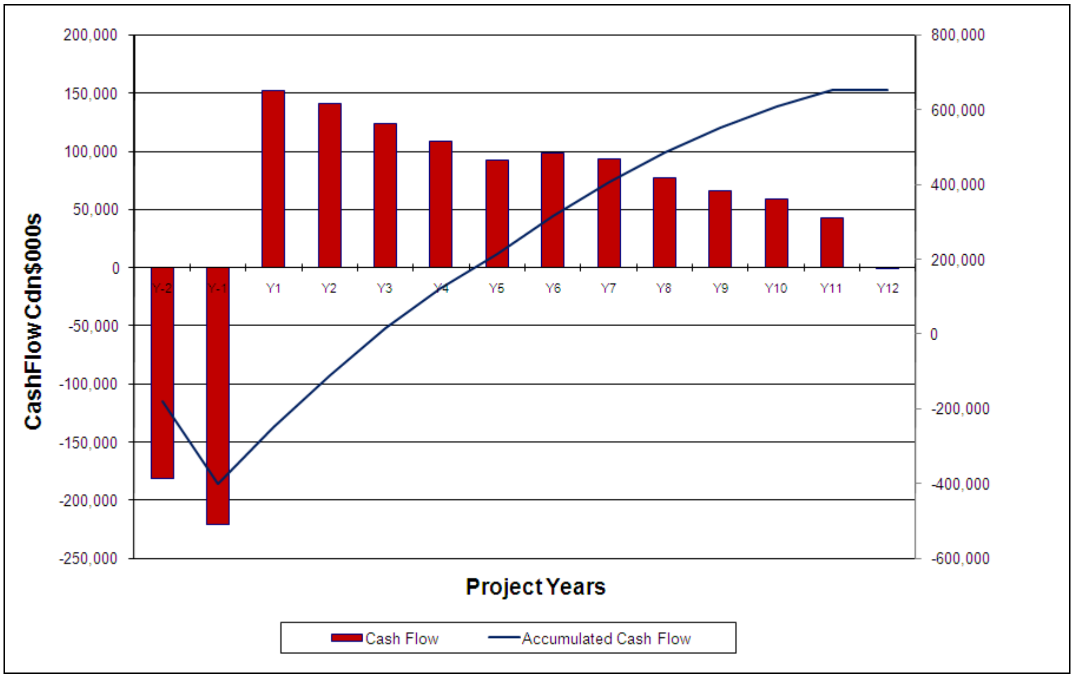
Initial and sustaining capital costs have been incorporated on a year-by-year basis over the mine life and were also deducted from net revenue to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated before concentrate is first produced. Sustaining capital includes expenditures for mining and milling additions, replacement of equipment, and tailing facility construction.

Working capital of CDN$12.6 M was determined on the basis of two months' mine site operating costs and applied to the first year of expenditures. Working capital is recovered at the end of the mine life and aggregated with the salvage value contribution before being applied towards reclamation during closure. This effectively sets the final year capital cost as zero.

The undiscounted annual cash flows are illustrated in Figure 27.1.

Net revenue is defined as gross revenue less off-site costs incurred for flotation toll conversion, concentrate transportation, insurance, and marketing charges. Operating cash flow is defined as the net revenue less mine site operating costs. Payback is calculated using the undiscounted cash flows.

Figure 27.1 Pre-Tax Cumulative Cash Flow



27.2.2 COMMODITY PRICE SCENARIOS

Commodity prices used for economic evaluation are:

- Base Case: GBRM APT forecast prices (US$300/mtu APT)
- Case 2: current Cantung APT price (US$254/mtu APT)
- Case 3: three-year average APT price (US$252/mtu APT).

BASE CASE

GBRM, an independent third party, was commissioned by NATC to provide a preliminary market review. Tungsten prices were adopted from this GBRM study.

CASE 2

The current selling prices are available from the Cantung Mine, and the current prices are available at http://www.metalprices.com for APT price as of July 18, 2008.

CASE 3

Three-year average APT prices were determined from http://www.metalprices.com. Historical prices were determined as of July 18, 2008.

SUMMARY

The GBRM Forecast, current and three-year average APT prices are summarized in Table 27.1.

Table 27.1 Summary of Pre-tax Commodity Price Scenarios

Scenario	APT (US$/mtu)	Gravity (US$/mtu)
GBRM Forecast (Base Case)	300.00	255.00
Current Prices	254.00	215.90
3-Year Average (FXR based on 3 year average)	252.00	214.20

Note: prices are as of July 18, 2008.

The pre-tax financial model was established on a 100% equity basis, thereby excluding debt financing, loan interest charges, and capital depreciation. The financial outcomes have been tabulated for NPV, IRR, and payback of capital. Discount rates of 8% and 6% were applied to all cases identified by commodity price scenario. The results are presented in Table 27.2.

Table 27.2 Summary of Pre-Tax NPV, IRR, and Payback

Scenario	NPV at Selected Discount Rates (million CDN$)		IRR (%)	Payback (Years)
	8%	6%		
GBRM Forecast (Base Case)	276.8	346.4	23.5	2.9
3-Year Average	84.8	128.5	13.3	4.3
Current Prices	92.8	137.6	13.8	4.2

27.2.3 SENSITIVITY ANALYSIS

Sensitivity analyses were carried out on the following parameters:

- tungsten prices (gravity price and APT price)
- exchange rate
- initial capital expenditure
- operating costs
- tungsten head grade.

The analyses are presented graphically as financial outcomes in terms of NPV and IRR. The project NPV (8% discount) is most sensitive to the foreign exchange rate and in decreasing order: head grade, gravity price, operating, and capital costs (Figure 27.2). Similarly, the project IRR is most sensitive to FXR, head grade, and initial capital costs (Figure 27.3).

Figure 27.2 Pre-tax NPV Sensitivity Analysis

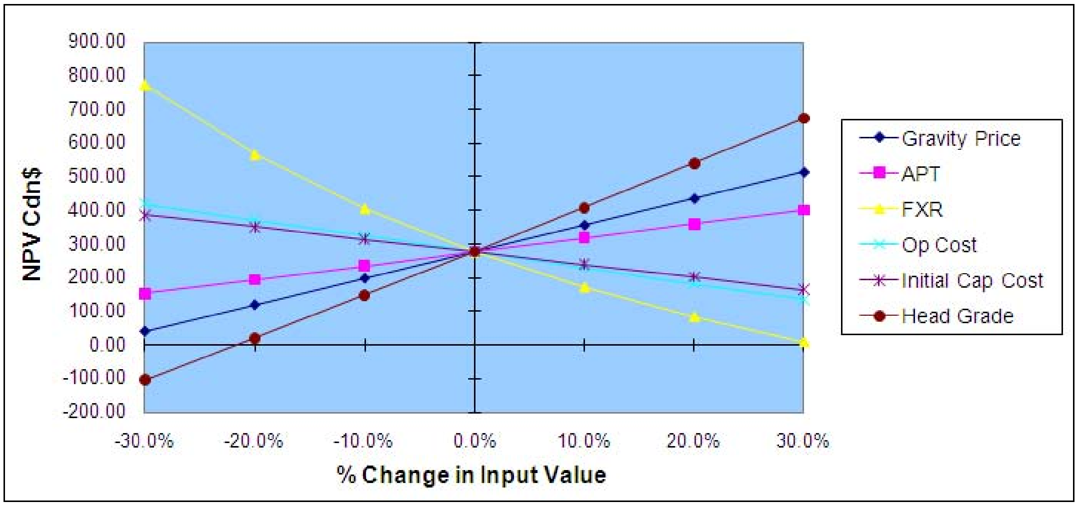
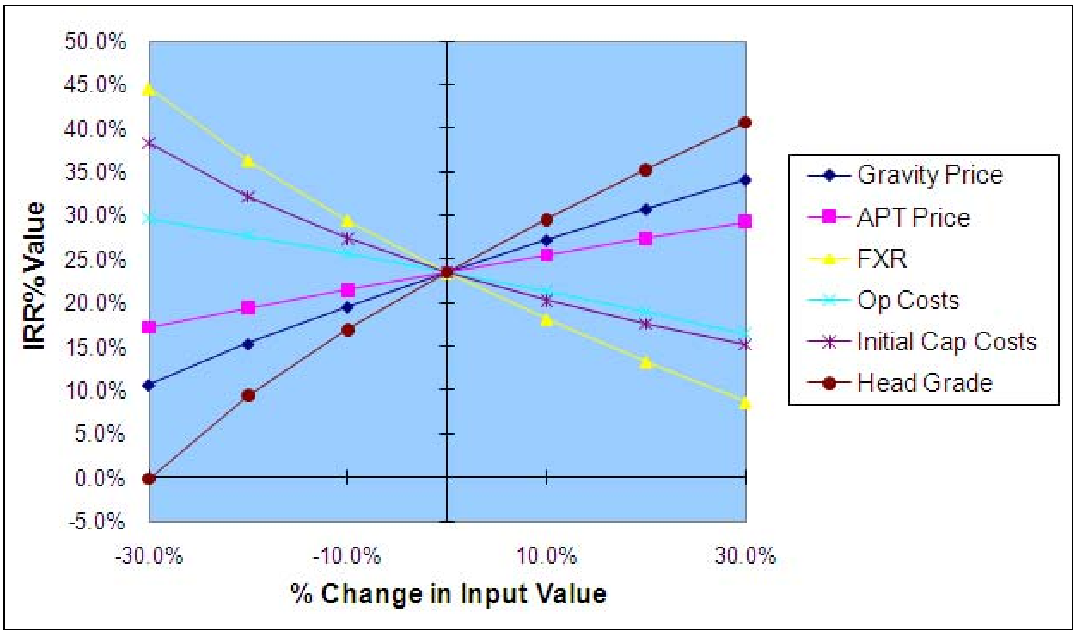


Figure 27.3 Pre-tax IRR Sensitivity Analysis



27.3 CONCENTRATE TRANSPORT LOGISTICS

Gravity concentrate is shipped to several costumers in China, US, Japan, or Korea. Flotation concentrate from the mine site will be truck transported to Vancouver and shipped to China to be converted to APT. The costs for shipping 163,000 t/a (wet basis) are shown below quoting current pricing:

- land transport: Mactung to Vancouver– CDN$216/t (wet basis)

- terminal charges Vancouver – CDN$30/t (wet basis)
- ocean transport – US$80/t (wet basis)
- moisture content – 0.5 %.

27.3.1 CONCENTRATE TRANSPORT INSURANCE

An insurance rate of CDN$0.10/mtu (wet basis) will be applied to the concentrate to cover land-based and ocean transport from the mine site to the smelter.

27.3.2 MARKETING

A marketing rate of CDN$0.30/mtu (wet basis) will be applied to the gravity concentrate for marketing the product. A marketing rate of US$1.27 per dry mtu of APT is applied to APT sales from flotation concentrate.

27.3.3 CONCENTRATE LOSSES

Concentrate losses are minimal due to shipping of products in concentrate bags.

27.4 POST-TAX MODEL

The post-tax model was provided by NATC and W.H. Taylor Inc. calculating the following:

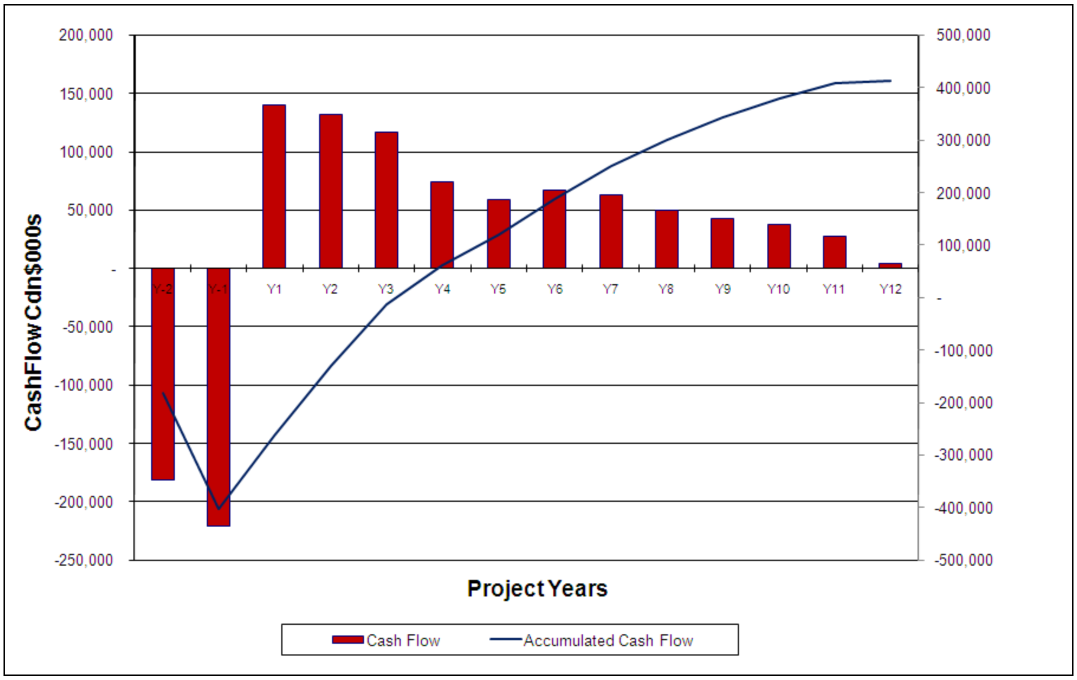
- 17.6 % IRR
- 3.2 year payback on US$402.1 M initial capital
- CDN$ 147.7 M NPV at an 8.0% discount rate.

A post-tax model was prepared to evaluate the impact of the base case with the following:

- Federal and Territorial Corporation Taxes
- Yukon Quartz Mining Royalty.

The post-tax base case financial model used the same inputs as the pre-tax economic evaluation.

Figure 27.4 Post-Tax Cumulative Cash Flow



27.4.1 CORPORATION TAXES – FEDERAL

A federal tax rate of 15% will be assessed on taxable income.

Accelerated provisions will apply in determining taxable income. These include deductions for:

- exploration and pre-production development expenditures at 100%
- Class 41(b) – ongoing capital expenditures at 25% declining balance
- Class 41(a) – initial capital expenditures at 100% and claimed up to income from mine operating profit
- CEE – initial mine pre-strip capital expenditures at 100% and claimed up to income from mine operating profit
- loss carry forward provision – 20 years
- Provincial resource taxes (see below).

27.4.2 CORPORATION TAXES – TERRITORIAL

A territorial tax rate of 15% will be assessed on taxable income.

The Yukon territorial corporate taxable income base is the same as the federal tax base. Similar write-off deductions are applied.

27.4.3 YUKON QUARTZ MINING ROYALTY

The following summary of the "Yukon *Quartz Mining Act*" is taken from Yukon Government Website:

Mining in the Yukon is administered under the Quartz Mining Act (QMA) enacted by the Yukon government in 2003. Section 102 of the QMA specifies an annual royalty payable to the Yukon Commissioner on mining profits. Under the QMA, royalty is a share of profits from a mine in the Yukon acquired under the QMA or the (predecessor) Yukon *Quartz Mining Act*, reserved for the Yukon government as owner of the mineral rights, for permitting extraction of mineral resources. It is paid by a mine owner or operator to the Yukon government (Table 27.3).

The royalty is payable annually, on an escalating rate basis to a maximum rate of 12%, for any profits from mining that exceed $10,000. The royalty-able profit is the amount by which the value of annual output from mining (revenues) exceeds eligible deductions (costs and a depreciation allowance for capital expenditures) for the year.

Table 27.3 QMA Royalty Rate Table

On annual profits of greater than:	And up to:	Royalty rate (applied to the increment)
$0	$10,000	0%
$10,000	$1 million	3%
$1 million	$5 million	5%
$5 million	$10 million	6%
$10 million	for each additional $5 million	A proportional increase of 1%

27.5 CASH FLOW ANALYSIS

Table 27.4 and Table 27.5 show the pre- and post-tax discounted cash flow analysis.

Table 27.4 Pre-tax Discounted Cash Flow

Period Ending	Source	Units	Y-2 2011	Y-1 2012	Y1 2013	Y2 2014	Y3 2015	Y4 2016	Y5 2017	Y6 2018	Y7 2019	Y8 2020	Y9 2021	Y10 2022	Y11 2023	Y12 2024	Total
Period Length			1.0	2.0	3.0	4.0	5.0	6.0	7.0	8.0	9.0	10.0	11.0	12.0	13.0	14.0	
Mill Feed		t			730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,000	29,141	8,059,141
Head Grade		g/t			1.45	1.31	1.22	1.17	1.11	1.10	1.06	0.98	0.92	0.87	0.78	0.78	1.09
Mill Recovery		%			83%	82%	82%	81%	80%	80%	80%	79%	78%	78%	76%	76%	80%
Contained WO3		million lb			23.3	21.2	19.7	18.9	17.8	17.7	17.1	15.7	14.7	14.0	12.6	0.5	193.3
Recovered WO3		million lb			19.4	17.4	16.0	15.3	14.3	14.2	13.7	12.4	11.5	10.9	9.6	0.4	155.3
Dry Concentrate Production																	
Gravity Concentrate		mtu			591,247	529,802	487,713	466,231	436,411	432,854	416,501	377,611	350,840	331,915	293,065	11,699	4,725,890
Flotation Concentrate		mtu			290,249	260,085	239,423	228,877	214,238	212,492	204,464	185,373	172,231	162,940	143,868	5,743	2,319,982
Concentrate Sale Price																	
Gravity Concentrate	NATC	US$/mtu			255.00	255.00	255.00	255.00	255.00	255.00	255.00	255.00	255.00	255.00	255.00	255.00	3,060
APT	NATC	US$/mtu			300.00	300.00	300.00	300.00	300.00	300.00	300.00	300.00	300.00	300.00	300.00	300.00	3,600
Concentrate Gross Revenue																	
Gravity Concentrate		000's US$			150,768	135,100	124,367	118,889	111,285	110,378	106,208	96,291	89,464	84,638	74,732	2,983	1,205,102
APT From Flotation Concentrate		000's US$			78,803	70,613	65,003	62,140	58,166	57,692	55,512	50,329	46,761	44,238	39,060	1,559	629,875
Gross Revenue (US$)	**W**	**000's US$**	**0**	**0**	**229,571**	**205,713**	**189,370**	**181,029**	**169,450**	**168,069**	**161,720**	**146,620**	**136,225**	**128,877**	**113,792**	**4,542**	**1,834,977**
Transportation Cost	NATC																
Gravity Concentrate		000's US$			2,631	2,557	2,507	2,481	2,445	2,441	2,421	2,375	2,343	2,320	2,273	1,936	28,731
Flotation Concentrate		000's US$			1,573	1,410	1,298	1,241	1,161	1,152	1,108	1,005	934	883	780	31	12,577
Total Transportation Costs		000's US$			4,205	3,967	3,805	3,722	3,607	3,593	3,530	3,380	3,276	3,203	3,053	1,967	41,308
Insurance and Marketing	NATC																
Gravity Concentrate		000's US$			208	187	172	164	154	153	147	133	124	117	103	4	1,665
Flotation Concentrate		000's US$			359	322	296	283	265	263	253	229	213	202	178	7	2,871
Total Insurance and Marketing		000's US$			567	508	468	447	419	415	400	362	337	319	281	11	4,536
Flotation Toll Conversion Cost		000's US$			6,567	5,884	5,417	5,178	4,847	4,808	4,626	4,194	3,897	3,687	3,255	130	52,490
Royalty	NATC	000's US$			2,182	1,954	1,797	1,717	1,606	1,593	1,532	1,387	1,287	1,217	1,072	24	17,366
Net Revenue (US$)		**000's US$**	**0**	**0**	**216,049**	**193,399**	**177,884**	**169,965**	**158,972**	**157,661**	**151,633**	**137,297**	**127,428**	**120,452**	**106,130**	**2,410**	**1,719,278**
Net Revenue (Cdn$)		**000's Cdn$**	**0**	**0**	**245,288**	**219,572**	**201,957**	**192,966**	**180,486**	**178,997**	**172,153**	**155,877**	**144,673**	**136,752**	**120,493**	**2,736**	**1,951,950**

table continues…

Table 27.4 cont'd

Period Ending	Source	Units	Y-2 2011	Y-1 2012	Y1 2013	Y2 2014	Y3 2015	Y4 2016	Y5 2017	Y6 2018	Y7 2019	Y8 2020	Y9 2021	Y10 2022	Y11 2023	Y12 2024	Total
On-Site Costs																	
Mining		000's Cdn$			27,842	27,842	27,842	27,842	27,842	27,842	27,842	27,842	27,842	27,842	27,842	1,111	307,376
Process		000's Cdn$			26,565	26,565	26,565	26,565	26,565	26,565	26,565	26,565	26,565	26,565	26,565	1,060	293,272
Maintenance and Surface		000's Cdn$			11,300	11,300	11,300	11,300	11,300	11,300	11,300	11,300	11,300	11,300	11,300	451	124,756
G&A		000's Cdn$			9,965	9,965	9,965	9,965	9,965	9,965	9,965	9,965	9,965	9,965	9,965	398	110,007
Vancouver Office		000's Cdn$			0	0	0	0	0	0	0	0	0	0	0	0	0
Exploration		000's Cdn$			0	0	0	0	0	0	0	0	0	0	0	0	
Direct Operating Cost (Cdn$)	**W**	**000's Cdn$**	**0**	**0**	**75,672**	**75,672**	**75,672**	**75,672**	**75,672**	**75,672**	**75,672**	**75,672**	**75,672**	**75,672**	**75,672**	**3,021**	**835,411**
HEAD OFFICE COSTS																	
Head Office Costs		000's Cdn$			2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	100	27,600
Total Head Office Costs					**2,500**	**2,500**	**2,500**	**2,500**	**2,500**	**2,500**	**2,500**	**2,500**	**2,500**	**2,500**	**2,500**	**100**	**27,600**
		000's Cdn$	**0**	**0**	**167,116**	**141,400**	**123,785**	**114,794**	**102,314**	**100,825**	**93,982**	**77,705**	**66,501**	**58,581**	**42,321**	**-384**	**1,088,940**
Preproduction Capital		000's Cdn$	180,925	221,130													402,055
Sustaining Capital		000's Cdn$			2,413	400	400	6,050	9,395	2,445	400	400	400	0	0	0	22,303
Working Capital		000's Cdn$			12,612												12,612
Salvage Value and WC Recovery		000's Cdn$														-21,107	-21,107
Reclamation Cost		000's Cdn$														21,107	21,107
Total Capital Costs	**W**	**000's Cdn$**	**180,925**	**221,130**	**15,025**	**400**	**400**	**6,050**	**9,395**	**2,445**	**400**	**400**	**400**	**0**	**0**	**0**	**436,971**
Net Revenue		000's Cdn$	0	0	245,288	219,572	201,957	192,966	180,486	178,997	172,153	155,877	144,673	136,752	120,493	2,736	1,951,950
Direct Operating Costs		000's Cdn$	0	0	75,672	75,672	75,672	75,672	75,672	75,672	75,672	75,672	75,672	75,672	75,672	3,021	835,411
Head Office Costs		000's Cdn$	0	0	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	100	27,600
Capital and Sustaining Capital Costs		000's Cdn$	180,925	221,130	15,025	400	400	6,050	9,395	2,445	400	400	400	0	0	0	436,971
Pre-Tax Cash Flow																	
Cash Flow		000's Cdn$	-180,925	-221,130	152,091	141,000	123,385	108,744	92,919	98,380	93,582	77,305	66,101	58,581	42,321	-384	651,969
Cash Flow		000's US$	-159,359	-194,772	133,961	124,193	108,677	95,782	81,843	86,653	82,427	68,091	58,222	51,598	37,277	-338	574,254
Accumulated Cash Flow		000's Cdn$	-180,925	-402,055	-249,965	-108,965	14,420	123,164	216,083	314,463	408,045	485,350	551,451	610,032	652,353	651,969	
Discounted Cash Flow at 8% Discount Rate		000's Cdn$	-167,523	-189,584	120,735	103,639	83,974	68,527	54,217	53,152	46,814	35,807	28,350	23,263	15,561	-131	276,802
Accumulated Discounted Cash Flow @ 8%		000's Cdn$	-167,523	-357,107	-236,372	-132,733	-48,759	19,768	73,985	127,137	173,951	209,758	238,108	261,371	276,933	276,802	

Discount Rate	%	8.0%
Pre-Income Tax Net Present Value (NPV)	million Cdn$	276.8
Pre-Income Tax Net Present Value (NPV)	million US$	243.8
Pre-Income Tax Internal Rate of Return (IRR)	%	23.5%
Initial Capital	million Cdn$	402.1
Total Operating Cost	Cdn$/t	103.66
Mine Life	Yrs	11.0
Payback Period	Yrs	2.9

Discount Rate	%	6.0%
Pre-Income Tax Net Present Value (NPV)	million Cdn$	346.4
Pre-Income Tax Net Present Value (NPV)	million US$	305.1
Pre-Income Tax Internal Rate of Return (IRR)	%	23.5%
Initial Capital	million Cdn$	402.1
Total Operating Cost	Cdn$/t	103.66
Mine Life	Yrs	11.0
Payback Period	Yrs	2.9

Legend
W - Wardrop
LME - London Metal Exchange
NATC - North American Tungsten

Table 27.5 Post-tax Discounted Cash Flow

		Y-2 2011	Y-1 2012	Y1 2013	Y2 2014	Y3 2015	Y4 2016	Y5 2017	Y6 2018	Y7 2019	Y8 2020	Y9 2021	Y10 2022	Y11 2023	Y12 2024	Total
After-Tax Cash Flow																
Cash Flow before tax	000's Cdn$	(180,925)	(221,130)	152,091	141,000	123,385	108,744	92,919	98,380	93,582	77,305	66,101	58,581	42,321	(384)	651,969
Income Tax	000's Cdn$	-	-	-	-	(218)	(28,795)	(29,838)	(28,125)	(25,710)	(20,408)	(17,600)	(15,624)	(11,408)	4,875	(172,850)
Quartz Mining Royalty	000's Cdn$			(11,923)	(8,798)	(6,656)	(5,457)	(3,775)	(3,551)	(4,912)	(7,330)	(5,972)	(5,053)	(3,211)	-	(66,639)
Cash Flow after tax	000's Cdn$	(180,925)	(221,130)	140,168	132,202	116,511	74,492	59,306	66,705	62,960	49,568	42,529	37,903	27,703	4,491	412,481
Cash Flow after tax	000's US$	(159,359)	(194,772)	123,460	116,443	102,623	65,612	52,237	58,753	55,455	43,659	37,460	33,385	24,401	3,956	363,313
Accumulated Cash Flow after tax	000's Cdn$	(180,925)	(402,055)	(261,887)	(129,686)	(13,175)	61,317	120,622	187,327	250,287	299,854	342,383	380,287	407,990	412,481	
Discounted Cash Flow at 8% Discount Rate	000's Cdn$	(167,523)	(189,584)	111,270	97,172	79,295	46,942	34,604	36,038	31,496	22,959	18,240	15,052	10,186	1,529	
Accumulated Discounted Cash Flow @ 8%	000's Cdn$	(167,523)	(357,107)	(245,837)	(148,665)	(69,370)	(22,427)	12,177	48,216	79,711	102,671	120,911	135,963	146,149	147,678	

Discount Rate	%	8%	6.0%
After Income Tax Net Present Value (NPV)	million Cdn$	147.68	196.9
After Income Tax Net Present Value (NPV)	million US$	130.07	173.5
After Income Tax Internal Rate of Return (IRR)	%	17.6%	17.6%
Payback Period	Yrs	3.2	3.2

28.0 RISK AND MITIGATION

28.1 INTRODUCTION

Risk identification allows mitigating strategies to be devised and resources to be allowed for their implementation, thus enhancing the project's security. While unforeseeable risks by their nature cannot be predicted, the effort to identify risks has been comprehensive.

Table 28.1 outlines the risks identified for the project.

Table 28.1 Risks and Mitigation

Risk	Mitigation
Project Execution	
Project Financing	Ensure appropriate resources and efforts are directed early to secure financing.
Delays in project schedule due to delivery delays, engineering delays	Execution plan includes early procurement of long-lead items. Selected portions of engineering have been advanced to near basic engineering level during feasibility phase.
Capital cost overruns	Costs to be monitored and trended through project execution phase.
Contractor Non-Performance	Close monitoring and managing of contractor critical items scheduling and costs to minimize unavoidable cost over-runs. Contingency allowed for in the study's capital costs estimate will cover some of the unforeseeable over-runs.
Non-availability of key personnel (management, engineering, supervisory, and tradesmen)	Staffing from Cantung Mine. Ensure early placement of contracts, prompt and effective recruiting at start of project, and the expanded use of contractors and consultants.
Transportation and Logistic	
Foul Weather	Backup and storage of essential supplies, medical, food, fuel, etc. for foul weather conditions.
Seasonal River Transport Transition	Increase material transportation rate before and after ice freezes over the Pelly River allowing additional essential material delivered to site and allowing concentrate to leave site. Allocate designated air evacuation plan for emergencies and essential material transport.
Pelly River Crossing	Expeditiously contact all territorial, provincial, and federal agencies to confirm river crossing permits. An emergency spill response plan is probably required for transporting materials across the river.

table continues…

Risk	Mitigation
Mining and Operations	
Initial Production	Productivity may be reduced and production targets not met because of: • Training of new personnel. • Development schedule delays, etc • Mining and milling equipment start up and trouble shooting. Monitor and advance detail engineering work to ensure timely pre-production mine development, assess mining extraction sequences, and milling operational delays during start-up.
Fuel Price Sensitivity	Project designed to minimize fuel consumption at estimated fuel prices. Investigation into technology of utilizing heat generated from the generator sets as heat exchange for camp and buildings. Alternate power generation sources such as wind energy will be pursued to supplement the diesel generators.
Metal Prices and Mining grade	Monitor exchange rate and metal price trends and implement forward selling strategies.
Mining Ore Grade	Monitor and enhance quality control during mining of ore and develop flexible mine plans with multiple ore sources.
Skilled Manpower Availability	By retaining services from mining contractors and consultant to cover temporary shortfall in certain key areas until appropriate staff are employed. Transfer of skilled and trained employees, technical staff and miners from the Cantung mine who are already accustomed to company operation strategy. Early recruitment and training during pre-production.
Mining Equipment Availability	Retain contractor services to cover shortfall, transfer of equipment from Cantung Mine, or acquiring contractor's equipment or purchasing of used or rebuilt equipment.
Backfilling	Establish safety manual and work procedures to address backfilling and equipment salvage during breakdowns in open stope. Additional remote control dozer is proposed to be available on site.
Skin Pillars	Skin pillars located in between stopes are to reduce possible dilution of unconsolidated backfill material into the adjacent mining stope. Backfill dilution into the adjacent mining stope will reduce the overall grade and effect stope stability. External factors such as drilling deviation and blasting practices, structure geological conditions and pillar strength, etc. will govern the stability of the pillars. Detail geotechnical data and advance numerical modelling will assist in predicting pillar behaviour.
Underground Material Handlings & Equipment	Adhere closely to mobile equipment, crusher and conveyors, etc in implementing preventive maintenance as per manufacturer's recommendation and company policy to reduce unexpected failures and downtime. Allow temporary ore stockpile at the surface.
Marginal Permafrost Condition	
Ground Water & Mine Dewatering	Perform permafrost and hydrogeological investigation to understand permafrost temperature sensitivity and rockmass permeability. Increase capacity in pump design & requirements.
Rockmass Quality	Mine design and geotechnical analysis estimation made based on non-permafrost conditions. Permafrost conditions may increase rockmass quality. Perform advance and detail permafrost investigation.

table continues…

Risk	Mitigation
Underground Rock Mechanics/Geotechnical	
Geotechnical Data Collection	Advance geotechnical data collection and analysis to confirm mine design, rockmass competency, and numerical analysis parameters made by the current study.
Advance Numerical Modelling	Advance numerical modeling to predict induced mining stress on excavation openings and pillar configuration for optimization.
Support Quality Control and Assurance	Appropriate application of support system to ground condition. Quality control on support installation, destructive testing, and material specification inspection.
Unconsolidated Backfill	
Mill Total Tailings	Permeability testing performed on tailings to verify suitability of total mill tailings as unconsolidated backfill material and to be free draining.
Unconsolidated Backfill	Investigation on frozen fill alternative if permafrost conditions allows for achievement in strength and modification to mining method by increase recovery.
Backfill Bulkhead	Investigation on marginal permafrost, hydrogeological conditions, and tailings characteristics to determine requirement for backfill bulkheads.
Mine Ventilation and Mine Refrigeration	
Drilling Dust	Dry drilling with dust collector on drilling equipment.
Fresh Air Requirement	Design based on current equipment selection and fresh air requirement based on the current study. Auxiliary ventilator may be required as demand and production increase in the future.
Refrigeration	Depends on permafrost, rock type and ground water condition. Refrigeration may be required.
Dry Stack Tailings Facility	
Deep Seated Slope Failure	Facility is designed to the standards and placed upstream of the runoff dam.
Surface Slope Failure	Facility is designed to the standards and placed upstream of the runoff dam.
Pile Erosion	Diversion berm included up hill to minimize surface runoff from facility.
Toe Liquefaction	Tailings are specified to be compacted with a steel drum vibratory compactor. N values of foundation soils will be in acceptable range.
Snow and Ice Buried During Pile Construction	Operational procedures will be developed to minimize this.
Ravine Dam	
Deep Seated Slope Failure	Dam is designed to the required safety standards.
Surface Slope Failure	Dam is designed to the required safety standards.
Liner Leakage	A two-liner system is used. Primary liner is HDPE. Secondary containment is GCL.
Dam Overtopping	Dam design will incorporate an emergency spillway. Dam designed to required safety standard.
Surface Erosion	Armour will be placed on the upstream slope of the dam across the normal reservoir operating elevation.
Toe Liquefaction	N values of foundation soils will be in acceptable range.
Seepage through Foundation	Keying the liner system into the bedrock will force any seepage though the bedrock, avoiding piping failures in the soils.

28.2 Permitting and Environmental Risks

The requirements of the Yukon's assessment and permitting regime have been utilized to review the environmental and socio-economic information available for the Mactung project. As a result of the review each section of the report has been divided into a summary of existing conditions as well as a description of the feasibility or potential risks. In conducting this review, EBA recommended that studies be conducted which investigate ways to minimize risk to the project as well as to identify where mitigations may be required to minimize effects.

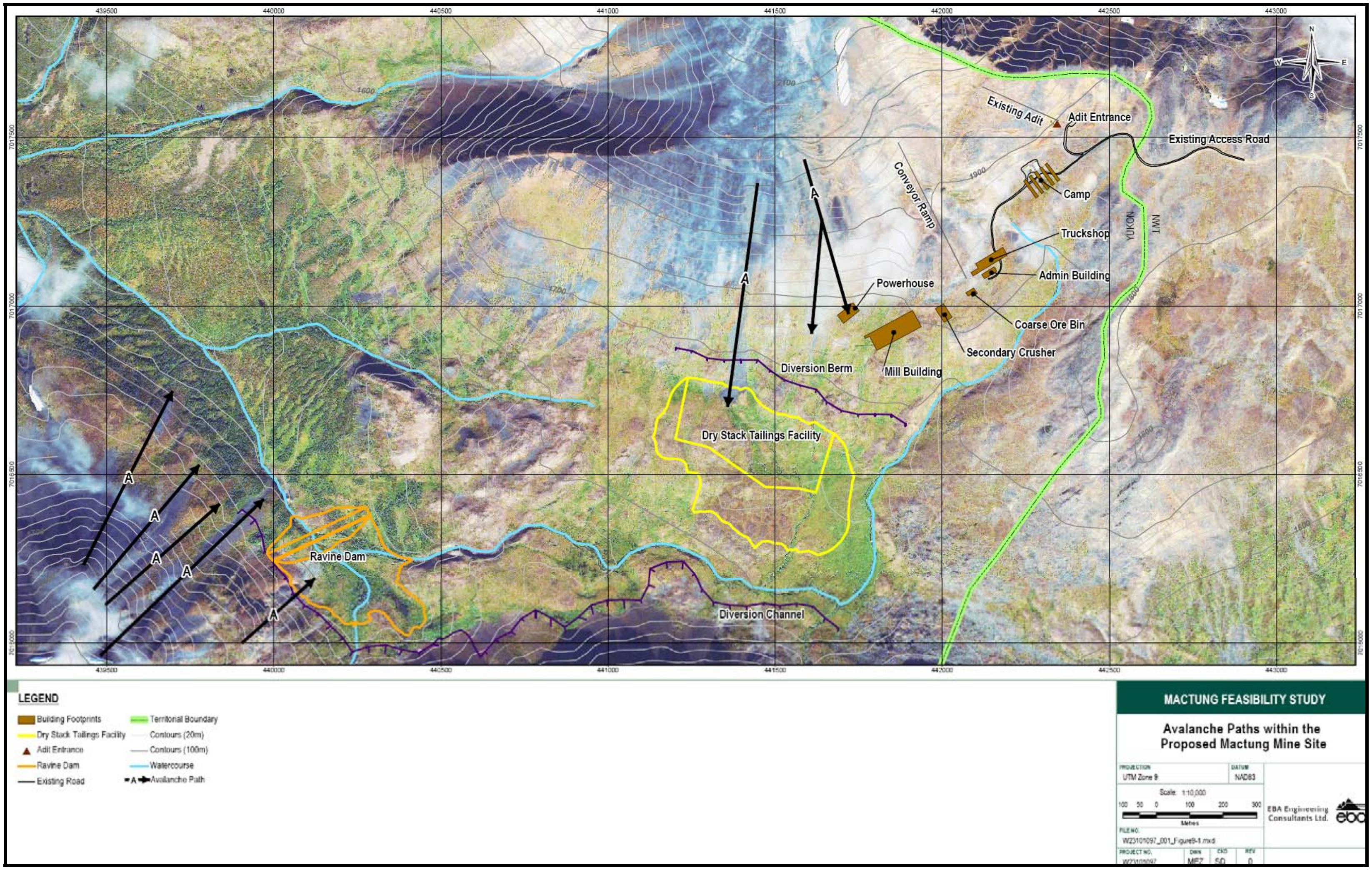
With the completion of identified studies, continued planning and design of the project, and further work on consultation and traditional knowledge activities, it is EBA's view that the Mactung project, as proposed, is feasible from an environmental and permitting perspective.

28.2.1 Avalanche Hazard

Potential avalanche hazards were identified in some areas of the proposed mine site during field work completed in the spring of 2008 (Figure 28.1). The consideration of avalanche hazard in the areas of the Ravine Dam and DSTF are necessary due to the presence of workers in these areas during the winter and spring months. The access road to the explosives magazine and mine site runs along the same slope as the DSTF and there will be year round use of this road by mine personnel and contractors.

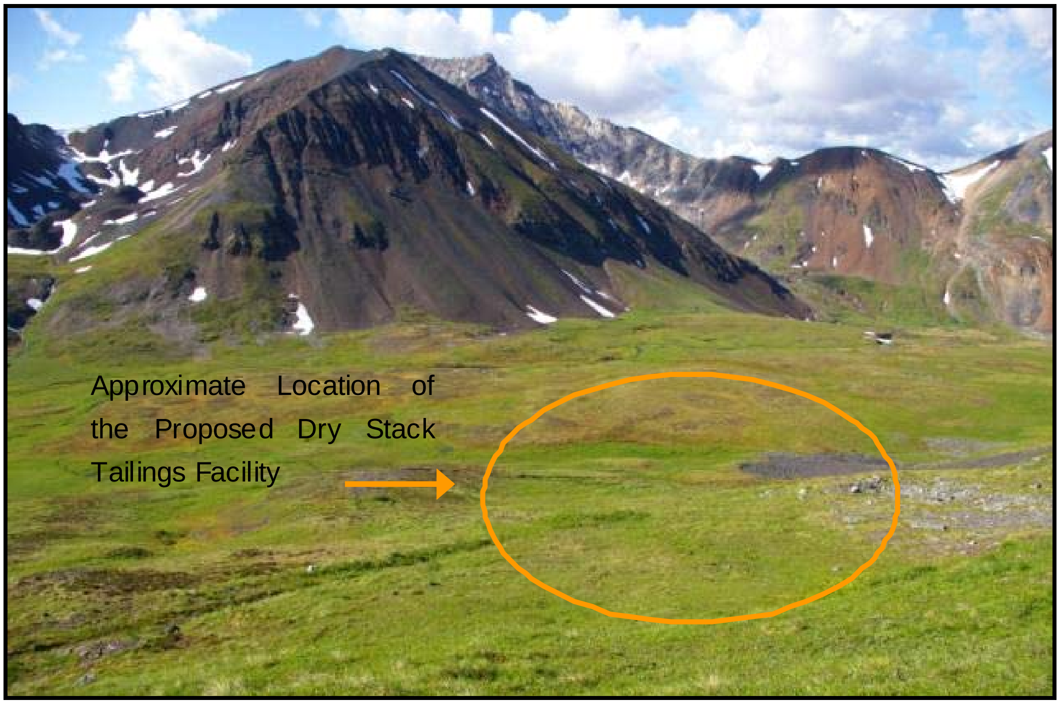
The southeastern valley slopes above the proposed Ravine Dam show evidence of avalanche runout that extend towards the valley bottom. The proposed diversion ditch on this side of the valley has a high likelihood of being impacted by avalanches. Cleaning of debris from the diversion ditching will be required seasonally to ensure that the ditching operates as per design. Spring cleaning of the ditching may require prior assessment or management of the avalanche hazard prior to allowing equipment and personnel to conduct this work.

Figure 28.1 Avalanche Paths within the Proposed Mactung Mine Site



There is also evidence of avalanche runout from upslope that extends into the area of the proposed DSTF (Photo 28.1). The proposed diversion berm, to divert surface waters away from the tailings facility, is located upslope of this infrastructure and has a high probability of avalanche impact. Consideration should be given during the detailed design of this berm to provide some protection from avalanche runout to the tailings facility. There are similar concerns with maintenance activities conducted on the diversion berm during winter and spring months.

Photo 28.1 Looking West-South to the Area of the Proposed DSTF

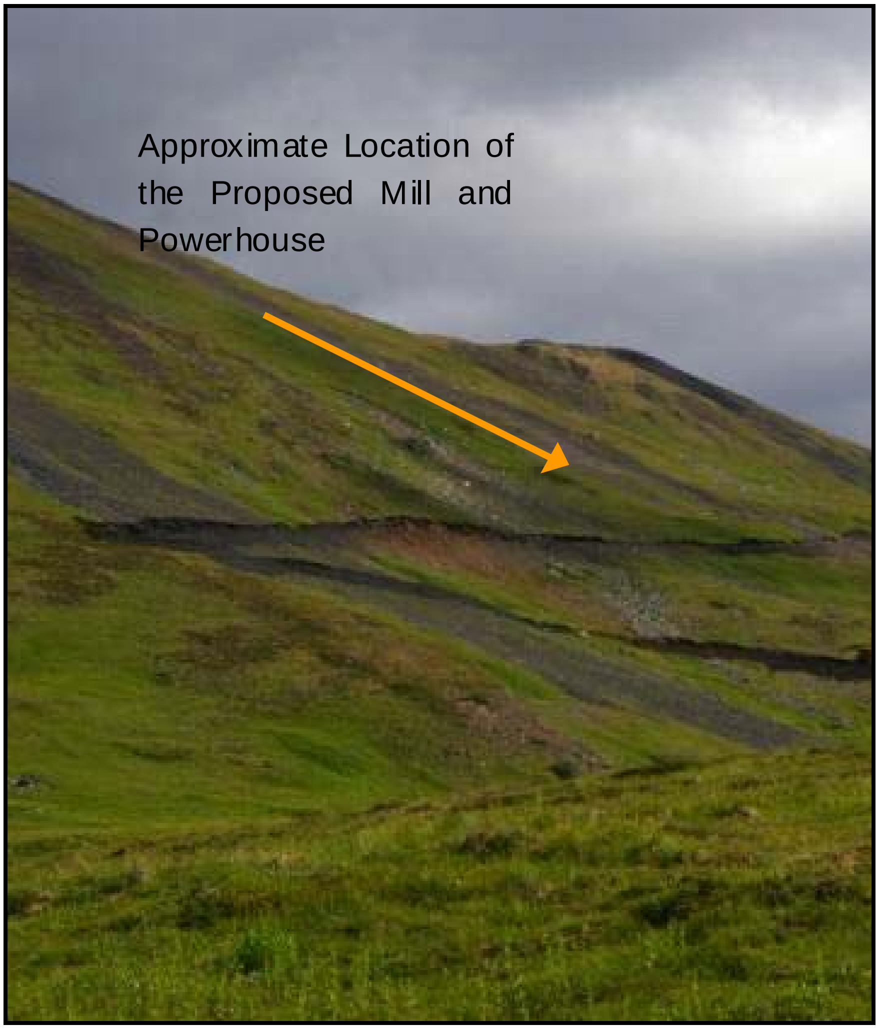


Note: Photo taken by Chris Dixon (2007).

The area of the proposed powerhouse and mill were inspected in the field, where terrain with high avalanche potential is located upslope of these components. The proposed powerhouse location is within an avalanche runout zone. Relocation of the powerhouse will be considered during detailed engineering to reduce the risk of avalanche impact.

The proposed mill building site appears to be in an area with a low to moderate probability of avalanche impact; however, further avalanche assessment is required to fully determine the avalanche impact potential in this area. One section of moderate slopes is located above the proposed camp (Photo 28.2). Further assessment is required to determine if this slope is steep enough for avalanche initiation.

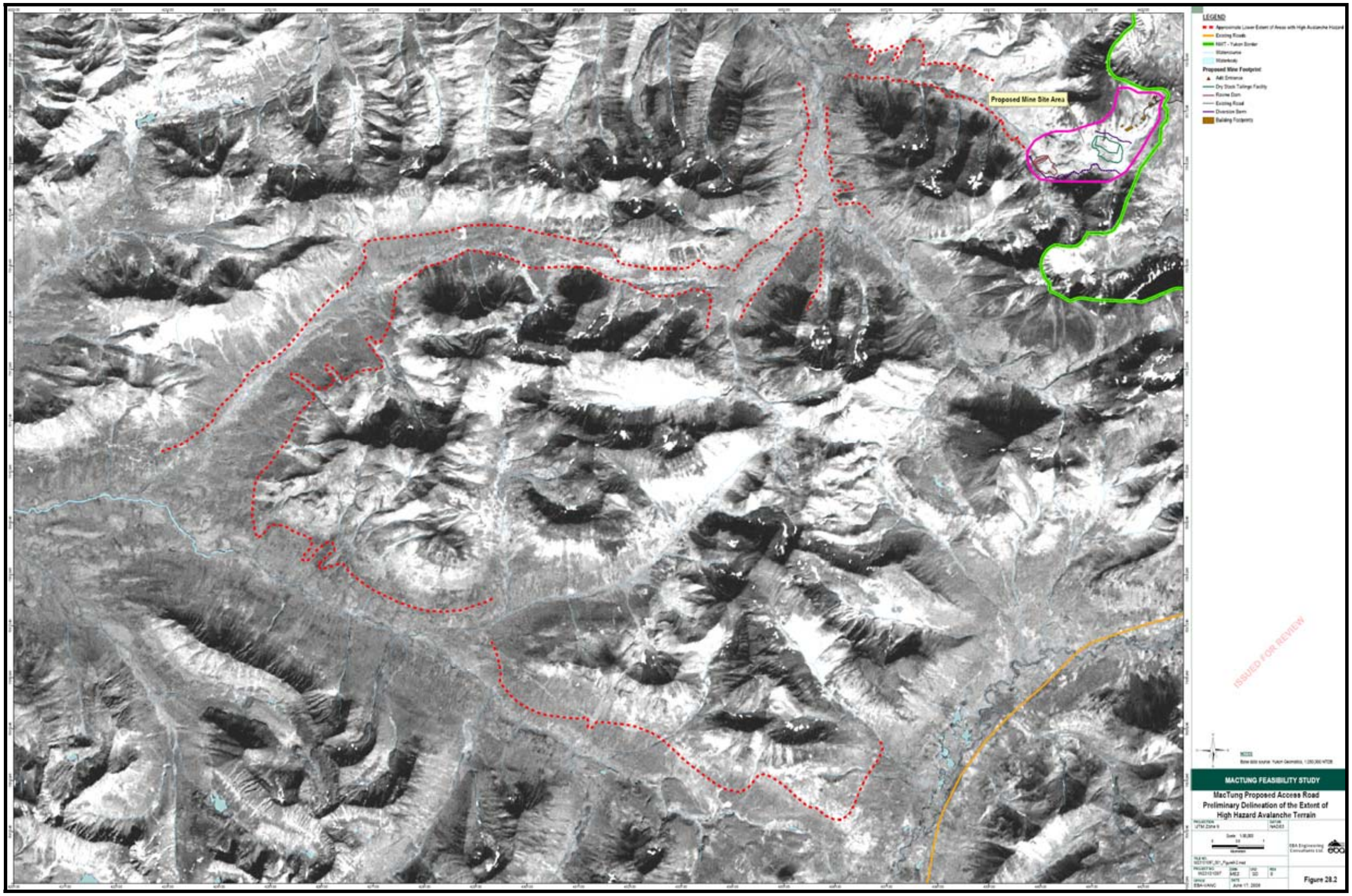
Photo 28.2 Looking Northeast to the Area of the Proposed Mill & Powerhouse



Note: Photo taken by Chris Dixon

The assessment of avalanche hazard identified a number of areas along the proposed access road with potentially moderate to high avalanche potential (Figure 28.2). Additional assessment work was recommended to determine whether the exposure of the proposed road to avalanche hazard could be avoided through routing. Further avalanche assessment work was completed to assist in a Road Avalanche Hazard Program which is expected to be utilized in the development phase of the project.

Figure 28.2 Mactung Proposed Access Road – Preliminary Delineation of the Extent of High Hazard Avalanche Terrain



29.0 PROJECT OPPORTUNITIES

29.1 GEOLOGY

RPA has recommended further drilling to improve the reliability of the mineral resource estimates in the upper 3 zones (3D, 3E, and 3F) as well as on the periphery and northerly portions of all zones. Initial efforts should focus on the areas of the deposit where the higher grades have not been entirely closed off by peripheral drilling.

By converting the inferred mineral resources to indicated categories, Wardrop sees a potential positive economic benefit to the project in terms of extending the life of mine and possibly, complementing the higher grade ore.

Wardrop has proposed a 3D block model at 6 m x 6 m x 6 m size to potentially improve selective mining and hence improve grade control, which would result in increased head grade during the initial years of mining.

29.2 MINING

The viability of decreasing the mine breakeven cut-off grade from 0.616% WO_3 will be investigated based on the feasibility study economics. This would potentially increase the mineral reserves to lengthen the life of the underground mine.

The underground excavation support requirement is designed based on current understanding of Mactung ground conditions. A detailed investigation of the site's permafrost conditions may provide essential mine design parameters. With a better understanding of the ground conditions, the support system and its requirements can be optimized to possibly result in cost savings to the project.

Evaluate backfill characteristics as frozen fill in concurrence with the investigation of the permafrost conditions at Mactung will further assist mine design and pillar configuration. A mix design for treated mill tailings and development waste rock to obtain frozen fill should also be investigated. A better understanding of the backfill behaviour will promote confidence on the mining method and achieve optimal extraction sequences that would improve productivity and ore grades in the initial years of mining.

Investigate the potential for an open pit to mine shallow higher grade ore to replace lower grade ore in the underground mine, to further enhance project economics. This could potentially add an additional 17 years of mine life. A scoping level study was undertaken that indicates the potential for an open pit mine to contribute financial value to the project after the depletion of the underground mine.

Investigate underground mining of the shallow high grade ore behind the north pit highwall in conjunction with open pit mining. The additional higher grade ore to complement the open pit ore would add more value to the project economics.

29.3 METALLURGY

The recommendation for future testwork is to gather environmental data and to confirm the results obtained in the pilot plant tests of 1985. There are some opportunities for possible cost reductions with regards to the process equipment selected.

29.4 PERMIT REQUIREMENTS

The majority of the permits required for a producing mine are now issued by the Yukon Government. The centralized permit process certainly would be beneficial to the project in obtaining timely permits in ensuring timely start of the construction process in 2010.

29.5 TAILINGS DISPOSAL

Further optimization of the dry stacked tailings disposal relative to materials for its construction, the conveyor, and trucking system and method of handling and placement would possibly reduce costs and enhance confidence in the design.

29.6 SUSTAINABILITY AND ENVIRONMENTAL MATTERS

EBA has recommended the following items:

- The development of an EMS for the project will make available detailed information on spill and disaster response, waste materials handling and emergency contact information for the project.
- The underground mining methods to be used at the Mactung mine are favourable to the effective management of waste materials to reduce potential long-term risks at the site. The use of backfill in the mine development process will create the opportunity for underground disposal of dry stack tailings thus minimizing deposition on surface of waste rock materials identified as having acid-rock drainage concerns. The mining and primary crushing processes will also occur underground, minimizes the

potential for releases to the receiving environment based on accidents or spills from these processes.

- A production-phase geochemical characterization program will be developed to track materials being stored. This production phase program will include field and laboratory static and kinetic testwork to confirm the assumptions made during the application and permitting phases and also to provide information for verification of the post closure site environmental model.

29.7 INFRASTRUCTURE

Construction cost of the site facilities such as the mill building layout may be reduced during detailed engineering. As more site data is made available, re-design of buildings would focus on minimizing excavations, structures and concrete quantities.

Further optimization of the power plant configuration would be an area under consideration during detailed engineering to reduce initial project capital.

Pre-construction of buildings off-site are certainly issues to be investigated to reduce high cost of site work.

Conduct further optimization on conveyor layouts for the ore and tailings handling systems.

29.8 ALTERNATIVE ENERGY

Evaluate the site specific wind resource data currently being accumulated to develop definitive wind energy assessment. This supplementary renewable energy source would reduce project operating cost substantially over the long term due to projected high diesel fuel costs.

30.0 INTERPRETATION AND CONCLUSIONS

30.1 FINANCIAL ANALYSIS

Wardrop developed three scenarios with varying economic parameters to evaluate the Mactung project on a pre-tax basis. The base case scenario for the Feasibility Study indicates an NPV of CDN$276.8 M at an 8% discount rate and an IRR of 23.5%. These figures were obtained using the forecast metal prices as recommended by GBRM.

Sensitivity of the project was evaluated based on metal price, exchange rate, grades, operating costs, and capital costs. The project NPV is most sensitive to the foreign exchange rate and, in decreasing order, head grade, price, operating, and capital costs.

30.2 GEOLOGY

The following interpretations and conclusions are made with respect to the Mactung mineral resource estimated completed by Scott Wilson RPA.

- Scott Wilson RPA completed a solid model, or wireframe, and multiple-seam 2D GSM block model for Mactung. The 2D model contains two separate estimates of grade (Kriged and Polygonal) and a single estimate of vertical thickness for each block based on the solid model. Scott Wilson RPA considers the kriged estimate more appropriate for this deposit. A minimum thickness of 4.5 m has been applied to the model and grades have been diluted to the minimum thickness where necessary.
- The kriged estimate contains an indicated mineral resource of 33 million tonnes grading 0.88% WO3, or 290 kt of contained WO3. An additional resource of 11.9 million tonnes grading 0.78% WO3, or 92 kt WO3, has been estimated for the inferred category. These estimates, which are based on assays capped at unique levels for each zone, are reported at a block cut-off of 0.5% WO3, which Scott Wilson RPA considers appropriate for the location and cost profile that can be expected for Mactung.
- CIM definitions (December 2005) were followed for the classification of the mineral resources. Scott Wilson RPA estimates an average drill spacing of 50 m based on the average distance between each composite and its four nearest neighbours. Scott Wilson RPA considers the spacing close enough to classify approximately 76% of the estimated resources as indicated.

- Both the kriged and polygonal estimates are virtually identical in terms of contained metal at the stated cut-off, although the polygonal estimates are lower in tonnage and higher in grade.

30.3 MINERAL PROCESSING AND METALLURGICAL TESTING

The following conclusions have been made regarding mineral processing:

- The mill has been designed to treat 2,000 t/d of material with a tungsten grade of 1.30% WO3 present as the mineral scheelite, a calcium tungstate.
- Two scheelite concentrates will be produced, namely a gravity concentrate with a tungsten grade of 67% WO3 and a flotation concentrate with 55% WO3. The tungsten recovery will be 55% for the gravity concentration process, and 27% for the flotation process for an overall recovery of 82%.
- The mill design relies on conventional recovery processes including crushing, grinding, size classification, gravity concentration, flotation, magnetic separation, thickening, filtration drying, and bagging. Operational plant experience from the Cantung Mine has also been incorporated into the design.
- Fresh water usage will be controlled and reliance has been placed on the re-use of process waters.

30.4 MINING

Two underground stoping methods will be used: LH stoping and MCF. The use of these methods is determined by the geometry, dip, thickness, and strength of rock. LH will mine 89% of the ore and the remaining 11% will be extracted by MCF.

Mining operations will be carried out with trackless mine equipment, diesel-powered loaders and trucks. Electric-hydraulic units will be used for lateral and long-hole drilling.

To optimize ore recovery and stabilize the ground, mined-out stopes will be backfilled with dewatered mill tailings and waste rock from mine development activities. The dewatered mill tailings will have similar characteristics to hydraulic backfill for the material to be free draining when placed.

Permanent 4 m rib pillars, between the 17 m wide stopes, and a 4 m thick transverse pillar, at 60 m along the strike length of stope lines, will remain in place to support the stope back. Recovery of these pillars is not feasible as the pillars are designed to provide containment to the unconsolidated backfill.

Mill tailings will be transported as backfill material to a truck loading station located close to the crusher station, using a conveyor belt located beside the ore conveyor.

Both this backfill material conveyor and the ore conveyor will hang from the back of the decline to allow personnel and equipment to pass under them.

To ensure stable ground conditions, all major mine infrastructure (such as the crusher station, ramps, and raises) will be located in low grade ore areas of the hanging wall rock of the deposit (3C rock zone). The rock formation in the hanging wall is classified to be geotechnically-competent rock.

The mine will operate without heating of ventilation air flow to preserve the marginal permafrost conditions. In the event permafrost conditions are not sustained during the summer, pumps will be installed to dewater the underground workings.

30.4.1 GEOTECHNICAL

The rock mass quality at Mactung was reported to be very good. The geotechnical data collection made in this investigation is based on RQD information provided by Mactung mainly on MS series drill holes; rock mass parameters were determined based on visual interpretation of core photographs. Initial estimates indicate good ground conditions.

The ground support requirement provided is based on the current understanding of Mactung ground conditions. The ground support systems design does not take into consideration the effect of permafrost. Ground movement monitoring is recommended so that the support system and its requirements can be optimized using the experience obtained during operations.

An open stope with dimensions of 20 to 40 m H with an opening of 17 m W x 60 m L is stable based on the stability graph analysis. These are non-man entry stopes. The mining of the Upper and Lower 2B Zone is recommended from the bottom up (from lower to upper) when favourable higher grade is encountered in the lower panel. The thickness of the waste rock in between the Upper and Lower 2B Zone is recommended to be greater than 15 m for mining from top to bottom. Wardrop suggests mining from the lower to upper zone.

Preliminary numerical modelling indicated that 4 m skin pillar yields a factor safety greater than 1.0 and relaxation of the backs. Relaxation of the backs is anticipated thus cable bolting is required.

The permafrost conditions at Mactung are reported to be marginal (D. Stewart, September 1982) and heat conduction through the rock is good (H. Heinicke, September 1979, revised April 1982). Thus consolidated backfill induced by freezing is currently not considered. The AMAX report indicated that at location XC1N (Amax, May-October 1973) that the heat generated by diesel power diamond drilling has caused thawing of the gouge zone.

The backfill is currently considered to be passive. The mill tailings particle size at Mactung satisfies the requirements for hydraulic fill material based on available data.

30.4.2 MINE VENTILATION

An efficient ventilation system and adequate fresh, clean air is paramount to the safety of personnel underground. It is important that personnel be aware of underground ventilation and have adequate training in terms of air supply and dangerous fumes. Basic information about the ventilation system and safety features of the mine should be thoroughly communicated with each employee as required by law and mining regulation.

30.5 YUKON'S ASSESSMENT AND PERMITTING REGIME

The following conclusions have been made regarding environmental baseline considerations and risks:

- The information required for completion of the general physiography section of the YESAA application and regulatory documentation is considered to be available. It should, however, be noted that characterization of the general physiography was not available at the time of preparation of the section for the access road from the North Canol Road to the mine site.
- Information required for the completion of the surficial geology/soils portion of the YESAA application and regulatory documentation was available for the project area; however, more information would be required on the surficial geology and soil for the mine access road from the North Canol Road to the mine site.
- Rock samples were submitted to provide information on acid rock drainage potential and metals leaching; however, the age of the core may be of concern.
- The tailings for the project, based on analysis of data by Wardrop and EBA have been assumed to be similar in composition to the tailings produced as a result of mining at the nearby Cantung Mine located in the NWT. Comparative studies of Mactung and Cantung have been undertaken by EBA and Wardrop to support this assumption.
- Terrain Hazards:
  - Rockfall is the primary erosional process in the area and results in the formation of extensive talus slopes.
  - Small, periodic debris flows are probably an ongoing process within valley sideslope stream channels and contribute to colluvial fan deposits mapped in the study area.
  - Avalanches are an annual occurrence in this region and probably play a part in some downslope transport of colluvial material in the area.
  - A number of rock glaciers are mapped in the study area, mostly on north-facing slopes but are not located within areas impacted by

proposed development and there is a low probability that the project will be impacted by large rock glaciers.

- The terrain hazards mapping being completed for the mine access road from the North Canol Road to the mine site will be incorporated into the YESAA application.

- Hydrology, Water Quality, and Hydrogeology:
  - The construction of the Mactung Mine will decrease the flows in Tributary C only during the period of filling of the Ravine Dam. At other times during the production phase of the project the flows in this tributary will be generally unaffected with the exception of the winter months due to discharge of water from the Ravine Dam during this period.
  - The addition of a reservoir in the upper drainage basin is comparable to a lake headed drainage system. Temperature effects on the downstream portion of Tributary A are anticipated to be minimal based on the elevation and size of the reservoir.
  - For the production period, discharge of process water from the Aging Pond has potential for effects on receiving water chemistry in the project area. For the closure period, metal leaching from the tailings storage facility at the mine has potential for long term effects on receiving water chemistry in the project area.
  - Effluent chemistry is not anticipated to have significant negative effects on the surface water receiving environment, however water quality guidelines have changed since the initial metallurgical study was conducted.
  - Groundwater at the Mactung property is anticipated to exist in unfrozen overburden deposits and bedrock fractures.

- Climate and Air Quality:
  - Precipitation could pose a potential risk with respect to increased potential for erosion and sediment transport in ditches and diversion channels at the site, including the ditch-lines along the proposed access road to the site. This impact can be mitigated through proper ditch and diversion design relating to anticipated flow volumes and rip-rap requirements. Revegetation, where moisture conditions allow, can also be conducted to mitigate impacts from erosion and provide sediment control.
  - Cold winter temperatures pose a risk with respect to freezing water lines used for plant feed and discharge lines, potable water and septic systems. Ground temperatures readings from the site will be used to assist in service designs. The use of adequately insulated or heat-traced water lines will need to be installed as appropriate to mitigate this concern. Utilidors and covered passageways between buildings will be installed to minimize the exposure of personnel to the adverse winter conditions.

- Based on the remoteness of the mine site location, it is not believed that air quality will pose a risk to the feasibility of the proposed Mactung project.

- Environmental Health and Aesthetics:
  - Health and safety issues related to cold temperatures for mine operation personnel will be mitigated through development of operational procedures.
  - Based on the remoteness of the mine site location, it is not believed that noise will form a risk to the feasibility of the proposed Mactung project.
  - There are no full-time residents in the area, and there is limited use of the area for recreational purposes. The project's impact on human or environmental health conditions are expected to be minimal.
- Archaeology:
  - Subject to the results of the 2008 study, no conflicts between the Mactung development and archaeological sites have been found, and there appears to be no archaeological issues that will impact the feasibility of this project.
- Vegetation:
  - The vegetation cover is highly variable within the Mactung Vegetation Local Study Area due to elevation, aspect, microtopography, and soil conditions.
  - In review of the existing vegetation baseline requirements for the YESAA project proposal, as well as the existing data there appears to be low risks to the project proposed.
- Wildlife and Fisheries:
  - EBA conducted an extensive baseline study program at the Mactung project area with the objective of documenting and characterizing wildlife within the study area. Both ground and aerial surveys were carried out starting in October 2005 and continued annually through to the fall of 2008.
  - Wildlife use of the primary footprint area was found to be consistent with other areas in the region and project footprint effects on wildlife are expected to be primarily minimal and localized in spatial scope and reversible nature. Limited secondary effects on wildlife may result from increased access to the area, but can be addressed through mitigation measures.
  - The mine footprint is at high elevation and above natural barriers that exclude fish thus local risks to fisheries resources resulting from the mine development are expected to be low and manageable through the implementation of appropriate and effective mitigation measures for the release of deleterious substances and maintenance of site hydrology. Other project infrastructure including the access road and water intake do intersect several watercourses with fisheries values, however their

potential impacts are expected to be low and controllable through the implementation of proper mitigation and standard industry Best Management Practices (BMPs).

- Based on a review of YESAB's "Proponent's Guide", information requirements can be met with the current baseline information.

30.5.1 Socio-economic Baseline Considerations and Risks

The affected communities have had considerable experience with the mining industry in the past; as a result, some residents have worked or are currently working in the mining industry. Given this experience and knowledge of the industry, the affected communities will have a fairly sophisticated approach to new mining development. They can be expected to be generally supportive of new mining development and will wish to maximize the potential economic benefits to their communities; however, it is also expected that these communities will be highly aware of and concerned about long-term environmental effects.

The relationship that NATC develops and maintains with the affected First Nations will have a material effect on the feasibility of the project. In practical terms, this relationship will likely be shaped by the negotiation and signing of socio-economic participation agreements. These agreements are not a legal or permitting requirement but can be critical in creating community support, easing the passage of the project through the permitting process, and providing a framework for positive relationship throughout the life of the project.

30.5.2 Non-traditional Land Use

Based on the limited existing non-traditional land use as well as the relatively short temporal scope associated with the proposed project, conflicts between non-traditional land use and this project are expected to be minimal.

30.5.3 Consultation

The consultation process ensures that community members are provided an opportunity to understand the project and participate in the project planning, effects identification, and the potential formation of mitigations. These consultation activities are a requirement of the proponent for the development of the project proposal in advance of the submission to YESAB. Consultation activities were underway at the time of finalization of the Feasibility Study.

30.5.4 Assessment and Permitting Requirements

The project will require an assessment pursuant to the Yukon Environmental and Socio-economic Assessment Act and two main permits: a QML and a Type A Water Licence; and several ancillary permits. The QML adheres to the Government of

Yukon's Quartz Mining Act, and is administered by Government of Yukon's Department of Energy, Minerals, and Resources. The QML is issued in two parts to advance non-water related works associated with the project. No significant obstacles are anticipated with the issuance of this licence.

The second major permit, the Type A Water License, is issued through the Yukon Water Board in accordance with the Yukon's Water Act. The process to obtain this permit is not expected to pose a significant hurdle for the project.

30.6 TAILINGS DISPOSAL

Three tailings disposition systems — conventional, thickened, and dry-stacked — were considered for use for the Mactung project. Of the three deposition systems dry-stacked tailings was selected for use at the Mactung Mine.

The DSTF will be constructed from the dewatered tailings, which will be hauled to the site using a combination of conveyor belts and haul trucks, spread out into maximum 600 mm thick lifts using a bulldozer, and then compacted to 95% of the maximum dry density as determined by ASTM D698 using a steel-drum vibratory compactor. The facility will be constructed as a sidehill fill structure with 4H:1V slopes. The facility has been designed to withstand a 1:500 year seismic event. A diversion berm will be required uphill of the facility to direct surface runoff away from the tailings, prevent sediment mobilization, and improve stability.

A dam will be required downstream of the mine footprint to collect surface runoff from the tailings facility and to age process water, which will be reused in the mill. This Ravine Dam has been classified into the Significant Consequence Category according to the 2007 Canadian Dam Association (CDA, 2007) and was designed using the 1:100 year inflow design flood and the 1:1000 year seismic event.

30.7 INFRASTRUCTURE

30.7.1 ACCESS ROAD

Forty-eight kilometres of upgraded/new roads will provide access from the existing Upper Canol Road to the mill site. A road to the pump house at the Hess River Tributary C will branch from the main access road. All access roads will be designed for the high snowfall and long winters typical of this area.

Wardrop evaluated two possible access road routes. The first option is an approximately 25-km route that follows through a narrow valley with steep slopes. The second option is an approximately 35-km route through a wider and flatter valley. The second option was selected for its flatter terrain.

The design of the access roads assumed the following:

- an average vehicle speed of 30 to 50 km/h
- an overall 8% maximum grade, increasing to 12% grade in some sections
- a 0.2 m surface of 25 mm crushed gravel/rock
- an 8 m wide main access road
- a 5 m wide pumphouse road
- 1.0 m deep ditch lines
- 2H:1V cut slopes
- 3H:1V fill slopes.

Based on fieldwork conducted in May 2008, it was recommended to undertake a follow up assessment of snow melt in the area and to develop a hazard management plan for the access road and mine site. As no avalanche protection was incorporated into the road design, subsequent engineering work will plan to re-align the road route to minimize potential avalanche hazards.

30.7.2 AIRSTRIP

The MacMillan Pass airstrip is reported to be 460 m long by 15 m wide (1,500' x 50'). The airstrip is owned and maintained by the Government of Yukon. NATC will upgrade and maintain the airstrip during the life of the project.

The airstrip will be upgraded to 1,375 m long by 30 m wide, to accommodate a 19-person Beechcraft 1900 or similar aircraft. A total of 25.4 ha will be cleared around the strip to provide the necessary space for the airstrip, apron, and obstruction clearances.

Construction will be a compacted granular sub-base with a crushed rock cover. Longitudinal and transverse slopes will be limited to 2%. The finished elevation of the airstrip will be above the flood level of the MacMillan River and, if required, the airstrip will be provided with an armoured berm to protect against erosion during river flooding.

The airstrip will be designed for visual flight rules; no runway lights will be provided, similar to the operating runway at Cantung Mine.

30.8 CAPITAL COSTS

The capital cost for the initial development of the mining, processing, and infrastructure facilities as described in this report is estimated at CDN$402.1 M. The capital cost has been developed to an accuracy of ±15% and has been prepared in third quarter 2008 Canadian dollars.

30.9 OPERATING COSTS

On site operating costs are estimated to be CDN$103.65/t of ore mined including mining, processing, general and administrative, and surface operations. The unit costs are based on annual ore production of 730,000 t/a and 365 days of operation. The operating cost estimate has an accuracy of ±15% and is categorized as labour, power, and consumables.

30.10 PROJECT EXECUTION PLAN

A detailed execution schedule has been produced which reflects work required from the completion of the feasibility study through to commissioning of the project. The schedule includes the development of the underground mine as well as the associated mining and processing facilities and infrastructure. Scheduled milestones are:

- Receipt of environmental approvals and permits:
  - YESAA Application submission – Q4 2008
  - YESAA Approval – Q1 2010
  - Quartz Mining Licence Part 1 & Type B water licence issued – Q1 2010
  - Detailed Engineering completed – Q2 2010
  - Type A Water license – late Q2 2010
  - Quartz Mining Licence Part 2- early Q3 2010
- Procurement of long lead items:
  - Order crushers/grinding mills – Q3 2009
  - Order other process equipment – Q1 2010
  - Order major mining equipment – Q1 2010
- Construction activities:
  - Mobilize to site – Q2 2010
  - Complete airstrip upgrade – Q3 2010
  - Complete access road – Q4 2010
  - Complete camp – Q4 2010
  - Commence pre-production mine development – Q1 2011
  - Complete site development – Q1 2012
  - Complete power plant/power lines – Q2 2012
  - Complete underground conveyor – Q2 2012
  - Complete tailings area/dam – Q3 2012
  - Plant start-up/commissioning – Q4 2012
  - Commercial production – Q1 2013

31.0 RECOMMENDATIONS

31.1 FINANCIAL ANALYSIS

From the favourable economic results of the base case scenario with GBRM market projections, Wardrop recommends proceeding with basic/detailed engineering, procurement, construction, and commissioning to target full production in Q1 2013.

31.2 GEOLOGY

Scott Wilson RPA makes the following recommendations:

- Further drilling is required to improve the reliability of the mineral resource estimates in the upper 3 zones (3D, 3E, and 3F) as well as on the periphery and northerly portions of all zones. Initial efforts should focus on the areas of the deposit where the higher grades have not been entirely closed off by peripheral drilling. Assessments of grade variability indicate a maximum drill spacing in the range of 120 m to classify any portion of the resource as indicated, although this must be confirmed by further drilling for the upper 3 zones.
- Subject to a positive decision to proceed with the EPCM work recommended in Section 31.1, a comprehensive delineation drilling program should be developed to increase the density of drilling to a nominal 50 m to 60 m spacing in that area of the deposit contained in the first 5 years of the mining plan. The program should also serve to provide further delineation of the three main faults and “Z-fold” that subdivide the four main zones into 12 individual lenses.

31.3 MINERAL PROCESSING AND METALLURGICAL TESTING

The following testwork is recommended:

- Conduct a mineralogical evaluation detailing the relative abundance of minerals and the liberation characteristics of scheelite.
- Characterize the sulphide mineral behaviour during the flotation and magnetic separation processes in order to determine the efficiency of these processes.

- The crushability, grinding, and abrasion indices need to be verified in order to confirm the crushing and grinding circuit design.
- The thickening and filtration design parameters of each operation will require confirmation.
- The tailings require full characterization with respect to particle size analysis and sulphide mineral content for environmental permitting.

31.4 Mining

Wardrop recommends the following:

- Investigate the potential for an open pit to mine shallow higher grade ore to replace lower grade underground ore towards the tail end of the underground mine, which will further enhance project economics.
- Investigate the potential for an ore pass from the open pit to the underground crusher.
- Conduct geotechnical drilling for the potential open pit in order to collect data to incorporate the open pit with the underground mine during detailed engineering.

31.4.1 Geotechnical

Although initial estimates of rock mass quality indicate good ground conditions, detailed geotechnical drill information and data collection is required on the main infrastructure to make a more complete evaluation. Additional geotechnical drill holes are recommended to better define the rock mass quality surrounding major infrastructure and to determine the conditions of major faulting.

Wardrop recommends the installation of ground support in all permanent openings as the support guarantees safety of workers, equipment, and major infrastructure and to avoid the potential of future rehabilitation due to deteriorating ground conditions. The permafrost condition of the mine is reported to be marginal.

Ground movement monitoring is recommended so that the support system and its requirements can be optimized using the experience obtained during operations.

It is recommended that stope back (roof) of 17 m wide be supported with cable bolts to support possible wedges formed in the back. This enables production drilling and blasting activities to be performed under support ground conditions. The supported back also reduces additional dilution from the back.

The mining of the Upper and Lower 2B Zone is recommended from the bottom up (from lower to upper) when favourable higher grade is encountered in the lower panel. The thickness of the waste rock in between the Upper and Lower 2B Zone is recommended to be greater than 15 m for mining from top to bottom. Additional numerical modelling in 3D is recommended to define the extraction sequence and pillar dimension in detail. During the geotechnical drill investigation, it is recommended to test samples of cores for engineering properties to increase the accuracy of the numerical modelling.

Wardrop supports Amax (1984) recommendations in that additional studies and test work be conducted to determine the extent of permafrost. Sprayed-on liners or foams are recommended to be applied on frozen faults to provide an insulation blanket and as a support component. On site investigations on the types of liners or foam are recommended to be performed on site.

A detailed investigation of the site's permafrost conditions will provide information essential to reinforce the mine design parameters. This will allow confirmation of dewatering requirements and provide additional fill strength if conditions are favourable to freezing.

Testing is recommended to confirm the percolation rate of the mill tailings to ensure that the backfill drains at the percolation rate above 10 cm/h.

Underground testing of frozen, treated mill tailings is recommended once the mine is in operation to determine moisture content, the time required to freeze treated mill tailings, and frozen fill strength. The technology of frozen fill research is currently being performed, which will assist Mactung in defining the mix design, moisture content, and conditions. Cooled intake air (ice house) may be required during the summer months to counterbalance heat generated from the equipment, personnel, blasting, and air intake to maintain freezing conditions underground.

Detailed investigation on the tailings backfill will be conducted prior to detailed engineering to have a better understanding of permafrost at Mactung. To be included in the investigation as earlier noted will be the percolation tests for drainage to meet or exceed the 10 cm/h rate.

31.4.2 MINE VENTILATION

Regular monitoring of air quality is mandatory and should follow the rules and regulations of the Yukon Territory. Regular measurements of air quality, air volume, speed, and gaseous contaminants should be included in the mine safety and standard procedures and should be meticulously followed.

The fumes from blasting operations must be cleared before production and activity resumes in mining heading. Calculations need to be undertaken to provide clearing delay times for smoke and fumes produced at the production level by excessive mobile equipment activity or production blasting.

A higher air velocity is required to clear the air. Two strategies that may be considered to clear the air in the headings are to:

- Reverse the fan direction so the negative pressure created removes the fumes and smoke.
- Push the air towards the headings using local utility fans so the fumes can travel to the exhaust ramp.

The ventilation system should be modelled in case of fire in several key locations and several remote locations in the mine. This can be done by introducing a constant source of smoke into the model.

31.5 Yukon's Assessment and Permitting Regime

EBA makes the following recommendations:

- Surficial Geology/Soil:
  - More information would be required on the surficial geology and soil for the mine access road from the North Canol Road to the mine site.
- Water Quality and Hydrogeology:
  - Groundwater information to be collected at the site will aid in determining the quality and quantity of potential groundwater inflow to underground workings; and the potential impact to groundwater and surface water from proposed mining activities.
- Climate and Air Quality:
  - The use of adequately insulated or heat-traced water lines will need to be installed as appropriate to mitigate risks to water lines due to cold winter temperatures. Utilidors and covered passageways between buildings will be installed to minimize the exposure of personnel to the adverse winter conditions.
- Environmental Health and Aesthetics:
  - Health and safety issues related to cold temperatures for mine operation personnel will be mitigated through development of operational procedures.

31.6 Infrastructure

31.6.1 Access Road

Particular attention will be given to vertical grades at the next stage of the project provided that detailed survey, geotechnical, etc. information will be available.

Building roads along sloping terrain such as along the mountain sides involves cut-and-fill excavation. Cut-and-fill quantities for the Mactung project will be minimized by making the best possible use of the existing ground contours. This will require the most accurate contour information possible during the more detailed design phases.

Wardrop recommends that the Civil/Transportation team undertake a site visit as early as possible prior to the design process. The purpose of the site visit is for the designer to familiarize him/herself with the area and get first-hand knowledge about the site and ground level terrain issues. The overall site visit could be from a helicopter but ground access whenever possible, in the vicinity of streams and rivers, environmentally sensitive areas, or areas deemed to have design/constructability issues, would help in putting together a much more accurate overall picture of the site/project and its design and constructability issues. Wardrop also recommends that a road building company with extensive expertise in northern resource road development be involved in the design process.

Ground work for terrain mapping took place in July 2008. Large scale aerial photography is required to complete terrain mapping, hazard mapping and to identify and delineate sources of suitable granular materials for road construction. This baseline information is critical to determine a favourable road alignment based on grade, surficial material, drainage, proximity to suitable borrow sources, terrain hazards, favourable stream crossings and potential for permafrost.

The need for truck escape ramps will have to be evaluated during more detailed design phases for sections of the road with long descending grades.

Wardrop will work closely with the Regulatory and Environmental team in the early stages of the initial permitting schedule. Necessary permits and authorizations/approvals as well as environmental requirements will have to be communicated to Wardrop prior to undertaking the design.

It is anticipated that borrow pit(s) will be required throughout the project area. Special consideration will have to be given related to location, sediment control, and drainage.

A detailed geotechnical assessment is required to determine ground support and to optimize ground support for road design. The assessment will include the identification of permafrost and muskeg areas so that these can be avoided in the route alignment.

Field investigation and sub-base and base foundation characteristics (e.g. soil classification, California Bearing Ratio) are required prior to the preliminary design stage. A site investigation to identify suitable borrow sources for road construction materials is also required in the next phase.

Collection of LiDAR survey information is considered to be necessary in order to obtain adequate surface model data for further detailed design. This task is very important and it is critical to meet the project deadlines. Wardrop recommends engaging a consultant for this process as early as possible to ensure sufficient time to complete all the design tasks before construction.

A ground survey is required to obtain cross-sections along the proposed centerline of the bridges and crossings and the respective approaches starting 50 m from each end. A ground survey also needs to be conducted to obtain cross sections at approximately 10 m intervals, up and down stream of the crossings for at least 25 m both sides, and centerline, including depth sounds and bottom profiles if possible.

A water management plan to control erosion and sedimentation will be needed in order to prevent the loss of soil and reduce the possibility of slope failure. The water management plan will also be used to maintain water quality and aquatic habitat in downstream water bodies and riparian areas.

Hydrological inspections and recommendations will be required for stream/river crossings.

The side slopes of the road are subject to adjustments, depending on the results of assessing terrain stability, terrain stability hazards, and snow avalanche hazards.

Regular road maintenance activities will be necessary and are anticipated to consist of practices such as:

- annual (or more frequent) surface grading and spot-resurfacing where required
- snow ploughing and sanding during the winter
- inspection and maintenance of culverts and ditches where required. The thawing of ice in culverts during spring will be required to prevent streams from cutting through the road before the culverts thaw.

Larger occasional maintenance activities such as local slope stabilization or erosion control projects may be necessary due to unusual weather, unforeseen subsoil conditions, or other factors.

Upon completion of mining activities it is anticipated that the access road may have to be deactivated and restored to a more natural condition. Activities to be undertaken are expected to include the removal of all culverts and bridges, the restoration of natural drainage patterns where necessary, and resloping, scarification and revegetation of the access road surface.

31.6.2 AIRSTRIP

In order to proceed with the next phase of the airstrip design, hydrology input is required to determine the risk of flooding from the adjacent stream, and riprap design will be required if flooding is an issue.

The airstrip will also be designed to avoid permafrost and muskeg areas.

A detailed topographic survey of the existing airstrip and surrounding area is also required, as well as a geotechnical investigation. However, four test pits along the length of the runway are anticipated to be sufficient.

32.0 REFERENCES

32.1 GEOLOGY

Abraham, B., 2007: Opinion Letter relating to the Mactung property of North American Tungsten Corporation Ltd., May 14, 2007.

Abraham, B., 2007: Opinion Letter relating to the Mactung property of North American Tungsten Corporation Ltd., May 17, 2007.

Allan, J.F., 1063: MacMillan Pass Tungsten Showing Geological Survey, August 1-9, 1963.

Amax Exploration, 1973: Underground Development and Bulk Sampling Program 1973.

Amax Northwest Mining Co. Ltd., 1983: Initial Environmental Evaluation of Mactung Project, 1983.

Anderson, R.G., 1982: Geology of the Mactung pluton in the Niddery Lake map area and some of the plutons in the Nahanni map area, Yukon Territory and District of Mackenzie; in Current Research, Part B, GSC, Paper 82-1A, pp 229-304.

Anderson, R.G., 1983: Selwyn plutonic suite and its relationship to tungsten skarn mineralization, southeastern Yukon and District of Mackenzie: in Current Research, Part B, GSC, Paper 83-1B, pp 151-163.

Atkinson, D., and Baker, D.J., 1979: Recent Developments in the Understanding of Mactung, in Deposits of the Cordillera, pp. 234-244.

Atkinson, D., and McNeil, W.H., 1983: MacMillan Tungsten Geological and Mineable Ore Reserves as at February 1983.

CIM Definition Standards – on Mineral Resources and Mineral Reserves, adopted by CIM Council on December 11, 2005.

Davies, D.J., et al., 1981: Preliminary Fisheries Report for the North Canol Road, December 1981.

Dick L.A and Hodgson C.J, 1982: The Mactung W-Cu(Zn) Contact Metasomatic and Related Deposits of the Northeastern Canadian Cordillera; Econ. Geol., Vol. 77, pp 845-867.

Gill D., 1978: Large Mammals of the MacMillan Pass Area, NWT and Yukon. September 1978.

Gotthard, R.M., 1981: North Canol Road Archaeological Inventory, December 1981.

Harris, F.R., and Godfrey, T.J.R., 1975: Geology and Ore Reserves -Macmillan Tungsten property; unpublished report, Amax Northwest Mining Co. Ltd., 1975.

Hayes, R., and Mossop, D.H., 1981: Nesting Raptor Studies in the North Canol MacMillan Pass Development Area, 1981.

International Environmental Consultants, 1983: Mactung Meteorology and Air Quality Analysis and Assessment, November 1983.

Kershaw, G.P., 1976: The Periglacial Environment and its Limitations to development, May 1976.

Kershaw, G.P., and Gill, D., 1979: Growth and Decay of Palsas and Peat Plateaus in the MacMillan Pass-Tischu River Area NWT, 1979.

Kershaw, G.P., and Kershaw, L.J., 1983: 1981/1983 Mactung Wildlife Studies Yukon/NWT, June 1983.

Kershaw, G.P., and Kershaw, L.J., 1983: Geomorphology and Vegetation of the Mactung Study Area Yukon/NWT, June 1983 (Private report for Amax Northwest Mining Co. Ltd.).

Mactung Project Scope Book, 3 vols., 1984: Amax Northwest Mining Company Oct 1, 1984.

Murray, P.R., et.al., 1982: Study of the Overwintering Habits of Juvenile Chinook Salmon Along the North Canol Road, Yukon Territory, April 1982.

Olin, E., and J. McAndrew, Mactung Flowsheet Development, March 30, 1980; Report 1980-07.

Orecklin, L., et al., 1981: 1975 – 1981 Small Stream Investigation, 1981.

Roscoe Postle Associates Inc. (RPA), 1992: Discussion on the Undeveloped/Exploration Properties of Canada Tungsten Mining Corporation Limited, Canamax Resources Limited, Minerex Resources Ltd., for First Marathon securities Limited, November 1992.

Roscoe Postle Associates Inc. (RPA), 2000: Report on the Tungsten Assets of North American Tungsten Corporation Ltd., for North American Tungsten Corporation Ltd., June 29, 2001.

Selby, D., Creaser, R.A., Heaman, L.M. and Hart, C.J.R., 2003: Re-Os and U-Pb geochronology of the Clear Creek, Dublin Gulch and Mactung deposits, Tombstone Gold Belt, Yukon, Canada: absolute timing relationships between plutonism and mineralization; CJES, Vol. 40(12), pp 1839-1852.

Steininger, R., Climax Mine Evaluation Group, 1980, Internal report on resources and reserves.

Strathcona Mineral Services, 1982: Ore Reserve Study for the Mactung Project, for Canada Tungsten Mining Corporation, April 1982.

Todd, J.W., 1982: Avifauna of the Mactung Area. November 1982.

Werner, A.B.T., Sinclair, W.D., and Amey, E.B., 1998: International Strategic Mineral Issues Summary Report—Tungsten, USGS Circular 930-O, 1998.

32.2 MINING

AMAX Northwest Mining Company Limited (1984). Scope Book, *Mactung Project*.

AMEC 2000. Rock Mechanics Audit for Cantung Mine.

Bieniawski Z.T. 1976. Rock mass classification in rock engineering. In Exploration for Rock Engineering, Proc. of the Symp., (ed. Z.T. Bieniawski) 1, 97-106. Cape Town, Balkema.

Brown, E.T. 1970. Strength of models of rock with intermittent joints. J. Soil Mech. Foundn Div., ASCE 96, SM6, 1935-1949.

F.Hassani and J. Archibald (editors), Minefill, 1998.

Hoek, E. 1968. Brittle failure of rock. In Rock Mechanics in Engineering Practice. (eds K.G. Stagg and O.C. Zienkiewicz), 99-124. London: Wiley.

Hoek, E. and Brown, E.T. 1980 (b). Underground Excavations in Rock, London, Instn Min. Metall.

Hoek, E. and Brown, E.T. 1980. Empirical strength criterion for rock masses. J. Geotech. Engineering Division, ASCE 106 (GT9), 1013-1035.

Hoek, E., Wood D. and Shah S. 1992. A modified Hoek-Brown criterion for jointed rock masses. Proc. Rock Characterization, Symp. Int. Soc. Rock Mech.: Eurock '92, (ed. J.A. Hudson), 209-214. London, Brit. Geotech.Soc.

Lacroix, P.A., & Cook, R.B. (2007) Scott Wilson RPA Report, NI 43-101: Technical report on the Mactung Tungsten deposit, Macmillan Pass, Yukon: prepared for North American Tungsten Corporation Limited.

Lakefield Research , An Investigation of the Recovery of Tungsten from a Sample of Mactung Ore, Jan 08, 1985.

H.Matt Bolu, Preliminary Gravity Testing of Cantung Tailings Samples by Falcon UF Concentrator, July 2007.

G.Farquharson, Mactung Project Mine Design Progress Report, Strathcona Mineral Services Limited, Aug. 21, 1982.

H.Heinicke, Rock Temperature Measurements at Mactung, Climax Molybdenum Company, Sep. 1979 & Revised Apr. 1982.

D.Stewart, Summary of 1982 Rock Temperature Measuements, Climax Molybdenum Company , Sept.,1982.

Macmillan Tungsten Property Underground Development and Bulk Sampling Program, E&ME Division, AMAX, May-Oct., 1973.

K.Dewing et al., Exploration History and Mineral Potential of the Central Arctic Zn-Pb District, Nunavut, Dec. 1, 2006.

Rocscience Inc. 2007. Phase 2 Software. Version 1.

Rocscience Inc. 2007 (b). Rocklab Software. Version 1.

Rocscience Inc. 2007. Phase 2 Software. Version 6.

Steward, D.R. 1980. Room and Pillar Analysis for the Mactung Project.

AMAX Northwest Mining Company Limited, Mactung Project – Scope Book, Vol 1, Revised Oct 1, 1984.

Canada Occupational Health and Safety Regulations.

Yukon Occupational Health and Safety Regulations.

Yukon Mine Safety Regulations.

32.3 MINERAL PROCESSING AND METALLURGICAL TESTING

Amax E&ME Division (1974): MacMillan Tungsten Property Underground Development and Bulk Sampling Program.

Amax Exploration Inc. (1974): Mactung Project.

Amax Northwest Mining Company Limited (1984): Mactung Project Scope Book.

Atkinson, O. (1984): 1983 Bulk Samples Collected for Ore Sorting Potential Tests.

Banning, W.R. (1974): MacMillan Tungsten Report No. 3 Metallurgical Results and Flowsheet Development, Amax Extractive Metallurgy Laboratory.

Flett, M.E. (1982): Mactung Project Overview of Feasibility.

Lakefield Research of Canada Limited (1985): An Investigation of the Recovery of Tungsten from a Sample of Mactung Ore submitted by Amax Northwest Mining Company Limited, Progress Report No. 1.

Lindberg, T. (1982): Introductory Flotation Tests on Three Scheelite Ore Samples Designated MacMillan Pass Ore Lots 1, 2 and 5, Report No. 2, *Mineral Processes AB.*

Olin, E.J. (1979a): Mactung Flowsheet Development 1978 Progress Report; Extractive Metallurgy Laboratory.

Olin, E.J. (1979b): Mactung Flowsheet Development Semi-Continuous Pilot Circuit Evaluation; Extractive Metallurgy Laboratory.

Roco, J.N., Partridge, M. and Smit, F.J. (1978): Mactung Project, 1977 Flowsheet Development Summary; Climax Molybdenum Division Mine Evaluation.

Scobie, A.G. and Wyslouzil, D.M. (1973): The Recovery of Tungsten from MacMillan Pass Project Samples Submitted by Amax Exploration Inc. Progress No. 1, Lakefield Research of Canada Limited.

Scobie, A.G. and Wyslouzil, D.M. (1974): The Recovery of Tungsten from MacMillan Pass Project Samples Submitted by Amax Exploration Inc. Progress Report No. 3, Lakefield Research of Canada Limited.

Svensson, J. and Olsson, P. (1977): Introductory Flotation Tests on Three Scheelite Ore Samples Marked MacMillan Pass Ore Lots 1, 2 and 5; Report No. 1, Gränges Mineral Processes.

Tenney, D. (2008): Personal Communication, July 26, 2008.

Weyler, P (1974), Telephone Conversations.

32.4 Tailings Disposal

Canadian Dam Association, 2007. "Dam Safety Guidelines 2007". Canadian Dam Association, Ottawa, ON.

EBA Engineering Consultants Ltd., 2007. Mactung Preliminary Geotechnical Investigation, Mactung Mine, YT. EBA File Number: W14101014.002.

PGI Technical Bulletin, March 1997.

32.5 INFRASTRUCTURE

BC Forestry Road Guidelines, Ministry of Forests BC.

BC Mines Act, Regulations and Codes.

Handbook for Mineral and Coal Exploration in BC, The Mining Association of BC, Ministry of Energy, Mines and Petroleum Resources, Ministry of Environment.

Low Volume Roads – Interim Guidelines, Highway Engineering Design Manual BC.

Design of Surface Mine Haulage Road Manual – Department of Interior USA.

Guidelines for Mine Haul Road Design, Department of Civil and Environmental Engineering, University of Alberta.

Resource Road Rehabilitation Handbook, Ministry of Environment, Land and Parks and Ministry of Forests BC.

Highway Geometric Design Guide, Transportation Association of Canada (TAC).

Best Management Practices Handbook: Hillslope Restoration in BC, Ministry of Forests BC.

Other Government of Yukon Design Guidelines.

32.6 YUKON ASSESSMENT & PERMITTING REGIME/RECLAMATION, DECOMMISSIONING & CLOSURE

AMAX E&ME Division. MacMillan Tungsten Property. Underground Development and Bulk Sampling Program. May – October 1973.

AMAX Exploration Inc. MacTung Monitor Well Program – 1982. Report prepared for AMAX Northwest Mining Co. September 1982.

AMAX Northwest Mining Company Limited. Initial Environmental Evaluation of the MacTung Project Yukon and Northwest Territories. 1983.

AMAX Inc. Western Operations. Rock Temperature Measurements at MacTung. September 1979, revised April 1982.

AMAX Inc. Western Operations. Summary of 1982 Rock Temperature Measurements. MacTung Project, Northwest and Yukon Territories. September 1982.

Canadian Council of Ministers of the Environment. Canadian Environmental Quality Guidelines. Canadian Council of Ministers of the Environment, Winnipeg. July 2006 Update. 1999.

Cody, W.J. Flora of the Yukon Territory. National Research Council of Canada, Ottawa. 1996.

Colorado School of Mines and Research Institute (CSMRI). Study of Downstream Effects of Effluent from Proposed Tailings Impoundment. MacTung Project. Prepared for AMAX Northwest Mining Company Limited. February 20, 1976. 44 pp. 1976.

Douglas, G.W., G.W. Argus, H.L. Dickson and D.F. Brunton. The Rare Vascular Plants of the Yukon. Syllogeus No. 28. National Museum of Natural Sciences, Ottawa. 1981.

EBA Engineering Consultants Ltd. (Hayco & Company Consultants). MacTung Project: 2006 Hydrometeorological Survey. Prepared for North American Tungsten Corporation Ltd. January 2007. Vancouver, BC. 2007.

EBA Engineering Consultants Ltd. (Hayco & Company Consultants). MacTung Project: 2007 Hydrometeorological Survey. Prepared for North American Tungsten Corporation Ltd. February 2008. Vancouver, BC. 2008.

EBA Engineering Consultants Ltd. MacTung Project 2006 Environmental Baseline Studies: Fisheries and Aquatic Resources. Prepared for North American Tungsten Corporation Ltd. April 2007. Whitehorse, YT. 2007.

EBA Engineering Consultants Ltd. MacTung Project 2006 Environmental Baseline Studies: Terrain and Surficial Geology. Prepared for North American Tungsten Corporation Ltd. June 2007. Whitehorse, YT. 2007.

EBA Engineering Consultants Ltd. MacTung Project 2006 Environmental Baseline Studies: Vegetation and Ecosystem Land Classification. Prepared for North American Tungsten Corporation Ltd. April 2007. Vancouver, BC. 2007.

EBA Engineering Consultants Ltd. MacTung Project 2006 Environmental Baseline Studies: Wildlife Report. Prepared for North American Tungsten Corporation Ltd. May 2007. Yellowknife, NWT. 2007.

EBA Engineering Consultants Ltd. MacTung Project 2006 Water Quality Sampling Program. Prepared for North American Tungsten Corporation Ltd. April 2007. Yellowknife, NWT. 2007.

EBA Engineering Consultants Ltd. MacTung Project 2007 Environmental Baseline Studies: Fisheries and Aquatic Resources. Prepared for North American Tungsten Corporation Ltd. December 2007. Whitehorse, YT. 2007.

EBA Engineering Consultants Ltd. MacTung Project: 2007 Environmental Baseline Studies: Rare Plant Survey and Ecosystem Land Classification Update. Prepared for North American Tungsten Corporation Ltd. December 2007. Vancouver, BC. 2007.

EBA Engineering Consultants Ltd. MacTung Project: 2007 Environmental Baseline Studies: Wildlife Report. Prepared for North American Tungsten Corporation Ltd. March 2008. Yellowknife, NWT. 2008.

Gill, D. Large Mammals of the Macmillan Pass Area, Northwest Territories and Yukon. AMAX Northwest Mining Company Ltd, Vancouver, British Columbia. 59 pp. 1978.

Government of Canada. 2003. Yukon Environmental and Socio-economic Assessment Act. Ottawa, Canada. Assented May 13, 2003.

Kershaw, G.P. and Kershaw, L.J. 1983. Geomorphology and Vegetation of the MacTung Study Area, Yukon/N.W.T. Prepared for AMAX Northwest Mining Co. Ltd., Vancouver. 85 p.

McJannet, C.L., G.W. Argus and W.J. Cody. 1995. Rare Vascular Plants in the Northwest Territories. Syllogeus No. 73. Canadian Museum of Nature, Ottawa.

Miller, S.J.; N. Barichello, and D. Tait. The Grizzly Bears of the Mackenzie Mountains, Northwest Territories. Completion Report No. 3. N.W.T. Wildlife Service. 188 pp. 1982.

Natural Resources Canada. National Building Code of Canada Seismic Hazard Values. 2005.

Parks Canada. Nahanni National Park Reserve Resource Description and Analysis. Natural Resource Conservation Section, Parks Canada, Prairie Region, Winnipeg, Manitoba. 1984.

Points West Heritage Consulting Ltd. MacTung Project: 2006 Archaeological Investigations. Prepared for EBA Engineering Consultants Ltd. December 2006. Langley, BC.

Points West Heritage Consulting Ltd. MacTung Project: 2007 Archaeological Investigations. Prepared for EBA Engineering Consultants Ltd. November 2007. Langley, BC. 2007.

RWED (Resources, Wildlife and Economic Development). Population status reports. Government of the Northwest Territories (GNWT), RWED, Yellowknife NT. Weblink: http://www.nwtwildlife.rwed.gov.nt.ca/Publications/populationstatusreports/populationrpts.htm (January 2006). 2002.

Smith, C.A.S., Meikle, J.C., and Roots, C.F. (editors). Ecoregions of the Yukon Territory: Biophysical properties of Yukon landscapes. Agriculture and Agri-Food Canada, PARC Technical Bulletin No. 04-01, Summerland, British Columbia, 313 p. 2004.

YESAB. 2005. Proponent's Guide to Information Requirements for Executive Committee Project Proposal Submissions. Version 2005.11.

33.0 DATE AND SIGNATURE PAGE

The effective date of this Technical Report, titled “Amended Technical Report on the Mactung Property”, is April 3, 2009.

Signed,

“Original document signed and sealed by Honorio Narciso, P.Eng.”

Honorio Narciso, P.Eng.
Wardrop Engineering Inc.

“Original document signed and sealed by Iouri Iakovlev, P.Eng.”

Iouri Iakovlev, P.Eng.
Wardrop Engineering Inc.

“Original document signed and sealed by André de Ruijter, P.Eng..”

Marinus André de Ruijter, P.Eng.
Wardrop Engineering Inc.

“Original document signed and sealed by Guy Impey, P. Eng.”

Guy Impey, P.Eng.
Wardrop Engineering Inc.

“Original document signed and sealed by Scott Cowie, MAusIMM”

Scott Cowie, MAusIMM
Wardrop Engineering Inc.

“Original document signed and sealed by Adrian Tanase, P.Eng.”

Adrian Tanase, P.Eng.
Wardrop Engineering Inc.

"Original document signed and sealed by Andy Nichols, P.Eng."

Andy Nichols, P.Eng.
MAN Mining Inc.

"Original document signed and sealed by Jay Collins, P.Eng."

Jay Collins, P.Eng.
Merit Consultants International Inc.

"Original document signed and sealed by Nigel Goodall"

Nigel Goodall
Goodall Business Resource Management Ltd.

"Original document signed and sealed by Peter Lacroix, P.Eng.

Peter Lacroix, P.Eng.
Scott Wilson Roscoe Postle Associates Inc.

"Original document signed and sealed by Richard Trimble, P.Eng."

Richard Trimble, P.Eng.
EBA Engineering Consultants Ltd.

APPENDIX A

CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE OF QUALIFIED PERSON

I, Honorio Narciso, Professional Engineer of Vancouver, British Columbia, do hereby certify:

- I am a Senior Mining Engineer with Wardrop Engineering Inc. with a business address at #800-555 West Hastings St., Vancouver, BC V6B 1M1;
- This certificate applies to the technical report entitled "Amended Technical Report on the Mactung Property", with an effective date of April 3, 2009 with an amendment date of April 30, 2010 (the "Technical Report");
- I am a graduate of the University of the Philippines (BSEM, 1963);
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #32946;
- My relevant experience with respect to this project includes project management in an engineering company and overseas mines, various engineering and mine operations positions in Canadian and US mines, as well as various senior positions in corporate offices for both mining and consulting companies over the past 40 years;
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument");
- I have not conducted a personal inspection of the Property;
- I am responsible for Section(s) 1.0 to 3.0, 17.2, 18.1 to 18.2, 18.6, 18.8 to 18.10, 20.0 (excluding 20.2 and 20.7), 23.0, 24.0 (excluding Construction Management and Mining Capital Costs), 25.0 (excluding 25.2 and 25.3), and 28.0 to 32.0 of the Technical Report. I am independent of North American Tungsten Corporation Ltd. as defined by Section 1.4 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 technical report has been prepared in compliance with the Instrument;
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated as of the 7th day of May, 2010 at Vancouver, British Columbia.

"Original document signed and sealed by Honorio Narciso, P.Eng."

Honorio Narciso, P.Eng.
Senior Mining Engineer
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Iouri Iakovlev, Professional Engineer of Vancouver, British Columbia, do hereby certify:

- I am a Senior Mining Engineer with Wardrop Engineering Inc., with a business address at #800-555 West Hastings St., Vancouver, BC V6B 1M1;
- This certificate applies to the technical report entitled "Amended Technical Report on the Mactung Property", with an effective date of April 3, 2009 with an amendment date of April 30, 2010 (the "Technical Report");
- I am a graduate of the Siberian State Industrial University, Novokuznetsk, Russia (M.Sc. Honours, Mining Engineering, 1983);
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #32213;
- My relevant experience with respect to mine engineering includes over 15 years of mine engineering and mine operations experience, including mine capital and operating cost estimates;
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument");
- I have not conducted a personal inspection of the Property;
- I am responsible for Section(s) 18.11 to 18.17, 25.2, and the mining capital cost component of Section 24.0 of the Technical Report;
- I am independent of North American Tungsten Corporation Ltd. as defined by Section 1.4 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 technical report has been prepared in compliance with the Instrument;
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated as of the 7th day of May, 2010 at Vancouver, British Columbia

"Original document signed and sealed by Iouri Iakovlev, P.Eng.."

Iouri Iakovlev, P.Eng.
Senior Mining Engineer
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Marinus André de Ruijter, Professional Engineer of Delta, British Columbia, do hereby certify:

- I am a Senior Metallurgical Engineer with Wardrop Engineering Inc. with a business address at #800-555 West Hastings St., Vancouver, BC V6B 1M1;
- This certificate applies to the technical report entitled "Amended Technical Report on the Mactung Property", with an effective date of April 3, 2009 with an amendment date of April 30, 2010 (the "Technical Report");
- I am a graduate of the University of Witwatersrand, Johannesburg, South Africa, (B.Sc. – Physics, Mathematics, 1970; B.Eng., 1973; M.Eng., 1979).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #31031;
- My relevant experience with respect to the Technical Report includes sulphide mineral flotation and gravity concentration and flotation research, and development work on cassiterite, wolframite, and chromite ores. I have 25 years experience in research and development, process engineering and plant design, and operations management in the gold industry with experience with base metals and industrial minerals;
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument");
- I have not conducted a personal inspection of the Property;
- I am responsible for Section(s) 16.0 and 25.3 of the Technical Report;
- I am independent of North American Tungsten Corporation Ltd. as defined by Section 1.4 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 technical report has been prepared in compliance with the Instrument;
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated as of the 7th day of May, 2010 at Vancouver, British Columbia

"Original document signed and sealed by Marinus André de Ruijter, P.Eng.."

Marinus André de Ruijter, P.Eng.
Senior Metallurgical Engineer
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Guy Impey, Professional Engineer of Vancouver, British Columbia, do hereby certify:

- I am a Senior Electrical Engineer with Wardrop Engineering Inc., with a business address at #800-555 West Hastings St., Vancouver, BC V6B 1M1;
- This certificate applies to the technical report entitled "Amended Technical Report on the Mactung Property", with an effective date of April 3, 2009 with an amendment date of April 30, 2010 (the "Technical Report");
- I am a graduate of Lakehead University, Ontario (1988);
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #17368;
- My relevant experience with respect to electrical design work includes multiple electrical power and control designs for new industrial facilities and upgrades to existing facilities. I have more than 24 years of applied experience in the electrical field as a member of, or directing engineering teams on power, control and instrumentation projects.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument");
- I have not conducted a personal inspection of the property;
- I am responsible for Section 20.7 Power Supply and Distribution of the Technical Report;
- I am independent of North American Tungsten Corporation Ltd. as defined by Section 1.4 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 technical report has been prepared in compliance with the Instrument;
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated as of the 7th day of May, 2010 at Vancouver, British Columbia

"Original document signed and sealed by Guy Impey, P.Eng.."

Guy Impey, P.Eng.
Senior Electrical Engineer
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Scott Cowie, of London, United Kingdom, do hereby certify:

- I am a Senior Mining Engineer with Wardrop Engineering Inc., with a business address at Ground Floor, Unit 2, Apple Walk, Kembrey Park, Swindon, United Kingdom, SN2 8BL;
- This certificate applies to the technical report entitled “Amended Technical Report on the Mactung Property”, with an effective date of April 3, 2009 with an amendment date of April 30, 2010 (the “Technical Report”);
- I am a graduate of the University of Queensland (Bachelor of Mining Engineering, 2001);
- I am a member in good standing of the Australian Institute of Mining and Metallurgy (Member #206253) ;
- I have eight (8) years of experience as a mining engineer in both engineering and operating environments including mineral reserve estimation, mine optimization, mine cost estimation, and various operational roles.
- My relevant experience with respect to financial modelling includes completion of scoping study to feasibility study mineral project evaluations for base, industrial, and precious metals.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”);
- I have not conducted a personal inspection of the Property;
- I am responsible for Section 27.0 Financial Analysis of the Technical Report;
- I am independent of North American Tungsten Corporation Ltd. as defined by Section 1.4 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 technical report has been prepared in compliance with the Instrument;
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated as of the 7th day of May, 2010 at Swindon, United Kingdom

“Original document signed and sealed by Scott Cowie, MAusIMM”

Scott Cowie, MAusIMM
Senior Mining Engineer
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Adrian B. Tanase, Professional Engineer of Calgary, Alberta, do hereby certify:

- I am a Senior Transportation Engineer with Wardrop Engineering Inc. with a business address at #2200-500 4th Avenue SW, Calgary AB, T2P 2V6;
- This certificate applies to the technical report entitled “Amended Technical Report on the Mactung Property”, with an effective date of April 3, 2009 with an amendment date of April 30, 2010 (the “Technical Report”);
- I am a graduate of the Technical University of Civil Engineering (Faculty of Roads, Bridges and Railways), Bucharest, Romania (B.Sc. in Civil Engineering, 1995; M.Sc. in Civil Engineering, 1996);
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #31344;
- My relevant experience with respect to the Technical Report includes over 14 years of experience as a transportation engineer in functional planning, preliminary and detailed design, cost evaluation, and project management of urban roadways, highways, and interchanges. I have substantial Canadian and International experience working on major transportation projects for both public clients such as Alberta Infrastructure and Transportation and the City of Calgary, and private sector clients;
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”);
- I have not conducted a personal inspection of the Property;
- I am responsible for Section 20.2 Roads of the Technical Report;
- I am independent of North American Tungsten Corporation Ltd. as defined by Section 1.4 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 technical report has been prepared in compliance with the Instrument;
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated as of the 7th day of May, 2010 at Calgary, Alberta

“Original document signed and sealed by Adrian Tanase, P.Eng.”

Adrian Tanase, P.Eng.
Senior Transportation Engineer
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Michael Andrew Nichols, Professional Engineer of Maple Ridge, British Columbia, do hereby certify:

- I am owner with MAN Mining Inc., with a business address at 20729 125th Ave, Maple Ridge, BC V2X 8N9;
- This certificate applies to the technical report entitled “Amended Technical Report on the Mactung Property”, with an effective date of April 3, 2009 with an amendment date of April 30, 2010 (the “Technical Report”);
- I am a graduate of Camborne School of Mines (ACSM, 1973);
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #23632;
- My relevant experience with respect to Mine Engineering includes over 30 years of mine engineering, mine operations, and project experience at mining properties in Canada, Africa, Asia and Central America. My experience includes mechanised and conventional underground mining of narrow vein, tabular and massive ore deposits. I also have experience in pillar, caving, and backfill mining methods including sub-level, room and pillar, long hole, cut and fill, and open stoping.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”);
- My most recent personal inspection of the Property was on August 8, 2007;
- I am responsible for Section(s) 18.3, 18.4, 18.5 and 18.7 of the Technical Report;
- I am independent of North American Tungsten Corporation Ltd. as defined by Section 1.4 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 technical report has been prepared in compliance with the Instrument;
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated as of the 7th day of May, 2010 at Vancouver, British Columbia

“Original document signed and sealed
by M. Andrew Nichols, P.Eng..”

M. Andrew Nichols, P.Eng.
Owner
MAN Mining Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Jay Collins, Professional Engineer of Vancouver, British Columbia, do hereby certify:

- I am President of Merit Consultants International Inc., with a business address at #401-750 West Pender St., Vancouver, BC V6C 2T8;
- This certificate applies to the technical report entitled “Amended Technical Report on the Mactung Property”, with an effective date of April 3, 2009 with an amendment date of April 30, 2010 (the “Technical Report”);
- I am a graduate of Portsmouth Polytechnic (B.Sc., 1974);
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #12741;
- My relevant experience with respect to our work associated with this report includes many other mining reviews, and developments of mining capital projects around the world, including capital costs over the past 30 years;
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”);
- I have not conducted a personal inspection of the property.
- I am responsible for the construction management costs included in Section 24.0 Capital Cost Estimate of the Technical Report;
- I am independent of North American Tungsten Corporation Ltd. as defined by Section 1.4 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 technical report has been prepared in compliance with the Instrument;
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated as of the 7th day of May, 2010 at Vancouver, British Columbia

“Original document signed and sealed by Jay Collins, P.Eng..”

Jay Collins, P.Eng.
President
Merit Consultants International Inc.

Nigel J. Goodall
13A Rossmoyne Drive
Rossmoyne, Western Australia 6148
Telephone Number 011 61 8 9457 7665
Fax Number 011 61 8 9457 2544
Email Address: nigel@gbrm.com.au

CERTIFICATE OF AUTHOR

I, Nigel J. Goodall, Principal with Goodall Business Resource Management Ltd. of 13A Rossmoyne Drive, Rossmoyne, Western Australia, do hereby certify that:

1. I am the author of that section entitled "Preliminary Market Review of Tungsten" of the report with an effective date of April 3, 2009 entitled "Report on the Mactung Property" by Wardrop Engineering Inc. (the "Technical Report") as amended by the Amended Technical Report on the Mactung Property dated April 30, 2010 to which this Certificate applies.

2. I am a graduate of London University (Bachelor of Science (Engineering) in Mining/Mineral Processing 2.1 honours), August 1971, and University of Western Australia (Master of Business Administration), September 1985.

3. I am a member in good standing of The Australasian Institute of Mining & Metallurgy (member no. 300804).

4. I have practised my profession continuously since graduation in 1971. My experience includes the following:

   (a) From 1974 to 1985, I was with Allied Eneabba Pty Ltd. in the following capacities starting with Metallurgist, Senior Metallurgist, Plant Manager, Business Manager and then General Manager Operations. As Business Manager, I was involved in marketing of the Company's products in Canada, the United States, Europe, Southeast Asia and Japan, as well as assessment of new business opportunities.

   (b) From 1985 to 1987, I was the General Manager of Ravensthorpe Mining responsible for the feasibility study of a mineral sands project in Western Australia which was subsequently sold to a Japanese titanium oxide producer.

   (c) In 1988, I became the Managing Director of Goodall Business and Resource Management Pty Ltd. to provide consulting services to industry. The company offers services such as operational and project development, strategic planning, international market assessment, pre-feasibility studies, project assessment, project management, assistance with company / government interface, development of research centres and science / technology parks and business planning. Since the company was established, it has successfully finalized several hundred projects for its clients.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.

6. I have no prior involvement with the Property that is the subject of the Technical Report.

7. I am independent of North American Tungsten Corporation Ltd. as defined by Section 1.4 of NI 43-101.

8. I have read NI 43-101, NI 43-101 CP and NI 43-101 F-1 and the Preliminary Market Review section of the Technical Report has been prepared in compliance therewith.

9. As of the date of this Certificate, to the best of my knowledge, information and belief, the Preliminary Market Review section of the Technical Report with an effective date of April 3, 2009 as amended April 30, 2010 contains all scientific and technical information that is required to be disclosed to make the Preliminary Market Review section of the Technical Report not misleading.

Dated at Rossmoyne, Western Australia 30 April, 2010.

__*"signed"*________________________
Qualified Person's Signature
Name: Nigel J. Goodall
(Seal, if applicable)

CERTIFICATE OF QUALIFIED PERSON

I, Peter A. Lacroix, Professional Engineer of Surrey, British Columbia, do hereby certify:

- I am an Associate Mining Engineer at Scott Wilson Roscoe Postle Associates Inc., with a business address at #388-1130 West Pender St., Vancouver, BC V6E 4A4, as well as Princial, Lacroix & Associates with a business address at 1931 128 St., Surrey, BC V4A 3V5;
- This certificate applies to the technical report entitled “Amended Technical Report on the Mactung Property”, with an effective date of April 3, 2009 with an amendment date of April 30, 2010 (the “Technical Report”);
- I am a graduate of the University of Alberta (Bachelor of Science degree in Mining Engineering with Distinction, 1983);
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #22528;
- My relevant experience with respect to the technical report includes
    - mineral resource and reserve estimation, mine planning, feasibility studies, economic analysis, due diligence, independent review and audit on numerous mining projects and operations worldwide
    - various engineering and mining-related positions at three Canadian mines
    - various senior positions at the corporate offices of a middle tier base metal and gold producer including Manager Engineering, Manager Operations, and Manager Acquisitions and Project Development
    - Principal Mining Consultant for two international consulting firms
    - Associate Mining Consultant for various mining consulting firms on numerous mining projects and operations worldwide
    - Principal, Lacroix & Associates, an independent wholly-owned mining consulting firm providing mining consulting services since 1997.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”);
- My most recent personal inspection of the Property was on August 8, 2007;
- I am responsible for Section(s) 4.0 through to 15.0 (inclusive) and Section 17.1 of the Technical Report;
- I am independent of North American Tungsten Corporation Ltd. as defined by Section 1.4 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 technical report has been prepared in compliance with the Instrument;

- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated as of the 7th day of May, 2010 at Surrey, British Columbia

"Original document signed and sealed by Peter A. Lacroix, P.Eng.."

Peter A. Lacroix, P.Eng.
Associate Mining Engineer
Scott Wilson Roscoe Postle
Associates Inc.

CERTIFICATE OF QUALIFIED PERSON

I, James Richard Trimble, Professional Engineer of Whitehorse, Yukon, do hereby certify:

- I am a Professional Engineer with EBA Engineering Consultants Ltd. with a business address at Unit 6-151 Industrial Road, Whitehorse, YT Y1A 2V3;
- This certificate applies to the technical report entitled “Amended Technical Report on the Mactung Property”, with an effective date of April 3, 2009 with an amendment date of April 30, 2010 (the “Technical Report”);
- I am a graduate of Queens University (B.Sc., M.Sc. (Eng) 1977);
- I am a member in good standing of the Association of Professional Engineers of Yukon Territory, License #541;
- My relevant experience with respect to the Technical Report includes site work and review of technical components of the study as well as includes over 33 years of geotechnical engineering for northern minesite development.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”);
- I conducted a personal inspection of the Property in 2006
- I am responsible for Section(s) 19.0, 21.0 and 22.0 of the Technical Report;
- I am independent of North American Tungsten Corporation Ltd. as defined by Section 1.4 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 technical report has been prepared in compliance with the Instrument;
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated as of the 7th day of May, 2010 at Whitehorse, Yukon

“Original document signed and sealed
by J. Richard Trimble, FEC, P.Eng..”

J. Richard Trimble, FEC, P.Eng.
Principal Consultant
Office Manager
EBA Engineering Consultants Ltd.
Whitehorse, YT