

# CANADIAN NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT

ON THE

# Sisson Project

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### List of Abbreviations

Above mean sea level .....	amsl
Ampere .....	A
Annum (year) .....	a
Bank cubic meter .....	BCM
Barren Rock Storage Area .....	BRSA
British thermal unit .....	BTU
Copper.....	Cu
Canadian Dollar .....	C\$
Cubic meter.....	m <sup>3</sup>
Cubic meters per day .....	m <sup>3</sup> /d
Cubic meters per hour.....	m <sup>3</sup> /h
Day .....	d
Days per week.....	d/wk
Days per year (annum).....	d/a
Degree.....	°
Degrees .....	deg
Degree Celsius .....	°C
Degree Fahrenheit.....	°F
Diameter.....	∅
Diameter.....	dia.
Dead-weight ton.....	dwt
Dry metric tonne.....	dmt
Gold .....	Au
Gram.....	g
Grams per cubic centimeter .....	g/cc
Grams per liter .....	g/L
Grams per tonne.....	g/t
Greater than.....	>
Hectare (10,000 m <sup>2</sup> ).....	ha
Hertz.....	Hz
Horsepower .....	hp
Hour.....	hr
Hours per day .....	h/d
Hours per week.....	h/wk
Hours per year (annum).....	h/a
Imperial gallon.....	gal
Inch.....	in
Internal Rate of Return.....	IRR
Joule (Newton-meter).....	J
Kilometer .....	km
Kilopascal.....	kPa
Kilovolt .....	kV
Kilovolt-ampere .....	kVA

Kilowatt.....	kW
Kilowatt-hour .....	kWh
Kilowatt-hours per short ton (US).....	kWh/st
Kilowatt-hours per tonne (metric tonne) .....	kWh/t
Kilowatt-hours per year (annum).....	kWh/a
Kilowatts adjusted for motor efficiency .....	kWe
Lead .....	Pb
Less than .....	<
Life-of-Mine .....	LOM
Liter.....	L
Liters per day .....	L/d
Liters per minute.....	L/m
Liters per second.....	L/s
Loose cubic meters.....	LCM
Megawatt.....	MW
Megawatt-hour .....	MWh
Megabytes per second .....	Mb/s
Meter.....	m
Meters above sea level.....	masl
Meters per second.....	m/s
Metric tonne .....	t
Metric tonne .....	mt
Metric tonne unit.....	mtu
Micrometer (micron).....	µm
Microsiemens (electrical).....	µS
Miles per hour .....	mph
Million.....	M
Million metric tonnes (megatonne).....	mmt
Minute (plane angle).....	'
Minute (time).....	min
Molybdenum .....	Mo
Month .....	mo
Newton.....	N
Net Present Value.....	NPV
Ohm (electrical).....	Ω
Ounce (troy).....	ozt
Parts per billion.....	ppb
Parts per million .....	ppm
Pascal (Newtons per square meter).....	Pa
Pascals per second .....	Pa/s
Percent .....	%
Percent moisture (relative humidity).....	%RH
Phase (electrical).....	Ph
Potential of Hydrogen (i.e. acidity or alkalinity level).....	pH
Pound .....	lb

Pound per square inch absolute .....	psia
Pound per square inch gauge.....	psig
Power factor.....	pF
Preliminary Assessment.....	PA
Professional Engineer.....	PE
Relative elevation.....	RL
Revolutions per minute .....	rpm
Run-of-Mine.....	ROM
Samuel Engineering, Inc.....	SE
Second (plane angle).....	"
Second (time).....	s
Short ton (2,000 lb).....	st
Short ton (US).....	st
Short tons per annum .....	stpa
Short tons per day (US).....	stpd
Short tons per hour (US) .....	stph
Short tons per year (US).....	stpy
Silver .....	Ag
Specific gravity .....	sg
Square meter.....	m <sup>2</sup>
Tailings Storage Facility.....	TSF
Tonne (1,000 kg) .....	t
Tonnes per annum.....	t/a
Tonnes per annum.....	tpa
Tonnes per day .....	t/d
Tonnes per day .....	tpd
Tonnes per hour.....	t/h
Total dissolved solids .....	TDS
Total suspended solids.....	TSS
United States dollar .....	US\$
United States gallon.....	USg
United States gallon per minute .....	USgpm
Volt .....	V
Volt-Ampere .....	VA
Watt (Joules per second).....	W
Week .....	wk
Weight percent.....	w%
Weight/weight .....	w/w
Wet metric ton .....	wmt
Yard .....	yd
Year (annum) .....	a
Year (US).....	yr

### Measurement Units and Symbols

The units used in this report are the International System of Units (SI). The reference conditions for gas volume are 0°C and 101.325 kPa, corresponding with a molar (ideal) gas volume of 22.414 m<sup>3</sup>/ (kg-mol). This is shown as “m<sup>3</sup> (normal)” or abbreviated to (non-SI) “Nm<sup>3</sup>.” The unit “t” rather than Mg is used for 1,000 kilograms mass. The dimensionally independent SI base units are shown in Table I. The permitted base units are shown in Table II.

Table I SI Base Units		
Quantity	Unit	Symbol
Length	Meter	m
Mass	Kilogram	kg
Time	Second	s
Electric Current	Ampere	A
Thermodynamic Temperature	Kelvin	K
Amount of Substance	Mole	mol
Luminous Intensity	candela	cd

Table II Permitted Base Units			
Quantity	Unit	Symbol	Definition
Time	Minute	min	60 seconds
	Hour	h	60 minutes
	Day	d	24 hours
	Calendar Year	y	365 days
Mass	Metric Tonne	t	1,000 kg

SI prefixes, as listed in Table III, are used only with SI base units. It is incorrect to use these prefixes with the permitted base units shown in Table II



Table III SI Base Units			
Power	Prefix	Symbol	Decimal Equivalent
10 <sup>24</sup>	yotta-	Y	1,000,000,000,000,000,000,000,000
10 <sup>21</sup>	zeta-	Z	1,000,000,000,000,000,000,000,000
10 <sup>18</sup>	exa-	E	1,000,000,000,000,000,000,000
10 <sup>15</sup>	peta-	P	1,000,000,000,000,000
10 <sup>12</sup>	tera-	T	1,000,000,000,000
10 <sup>9</sup>	giga-	G	1,000,000,000
10 <sup>6</sup>	mega-	M	1,000,000
10 <sup>3</sup>	kilo-	k	1,000
10 <sup>2</sup>	hecto-	h	100
10 <sup>1</sup>	deca-	da	10
10 <sup>0</sup>			1
10 <sup>-1</sup>	deci-	d	0.1
10 <sup>-2</sup>	centi-	c	0.01
10 <sup>-3</sup>	milli-	m	0.001
10 <sup>-6</sup>	micro-	μ	0.000 001
10 <sup>-9</sup>	nano-	n	0.000 000 001
10 <sup>-12</sup>	pico-	p	0.000 000 000 001
10 <sup>-15</sup>	femto-	f	0.000 000 000 000 001
10 <sup>-18</sup>	atto-	a	0.000 000 000 000 000 001
10 <sup>-21</sup>	zepto-	z	0.000 000 000 000 000 000 001
10 <sup>-24</sup>	yocto-	y	0.000 000 000 000 000 000 000 001

The prefixes and prefix symbols are used with the SI base and derived units – with the exception of kg. The base mass unit, kg, already has a prefix; hence the SI prefixes are applied to the unit gram (g). In this manner, the symbol for metric tonne is Mg; however, in this report the permitted alternate, t as listed above, is used.

## 1.0 Summary

Northcliff Resources Ltd. (Northcliff), a mineral exploration and development company listed on the Toronto Stock Exchange (TSX), engaged the services of Samuel Engineering, Inc. (SE) to complete a Canadian National Instrument (NI) 43-101 Technical Report for the Sisson Project (Sisson Project), New Brunswick, Canada based on a Feasibility Study that has recently been completed. The Feasibility Study confirmed the viability of Sisson as a long-life mining operation.

Northcliff entered into contract with the companies listed in Table 1.1 to advance the process and engineering designs as well as capital and operating cost estimates in order to complete the economic analysis for the Feasibility Study.

<b>Table 1.1 Companies Providing Input to the Feasibility Study</b>	
<b>Company</b>	<b>Contributions</b>
RPA, Inc. (RPA)	Resource Estimate
Knight Piésold Ltd. (KP)	Waste and Water Management
Moose Mountain Technical Services (MMTS)	Mine Design and Production Planning
Bolu Consulting Engineering, Inc. (Bolu)	Metallurgy, Process, and Mechanical Design and Plant Layouts for the Concentrator and APT Plants
Gene Greskovich	Review of Ammonium Paratungstate (APT) Plant Design including: Metallurgy, Process, Layout and Mechanical Design
Stantec Consulting	Environmental Baseline and Assessment
Samuel Engineering, Inc. (SE)	Infrastructure, Civil, Electrical, Mechanical Equipment Specification and Project Economics
SRK Consulting Ltd	Geochemical Characterization and ML/ARD Potential Assessment

## 1.1 Property Description, Location and Access

The Sisson property is located on Crown land approximately 10 km southwest of the community of Napadogan, New Brunswick, and approximately 60 km directly northwest of the city of Fredericton. The site is readily accessible, 100 km by road from Fredericton, with local access provided by numerous secondary and forestry roads (Figure 1.1).



Figure 1.1: Project Location Map

### 1.2 Land Tenure

Tenure for the mineral rights is held via five contiguous claim groups comprising a total of 850 units encompassing 18,880 ha. In October 2010, Northcliff completed an earn-in obligation to secure a 70% interest in June 2012. In April 2012, Northcliff and Geodex entered into an agreement by which Northcliff could acquire the remaining 30% interest in the property, which closed on June 21, 2012. Northcliff, through its wholly-owned subsidiary, Northcliff Canada (Holdings) Ltd, holds 100% of the Sisson Project as of the date of this report.

### 1.3 Property History

The first significant work in the Sisson area was carried out in the late 1950s by Nashwaak Pulp and Paper Co. Twelve holes were completed in 1955 and 43 holes in 1959-1960 which resulted in the discovery of the Nashwaak polymetallic vein deposit.

From 1967 to 1969, Penarroya Canada Ltée conducted geological mapping, a ground magnetic survey, and soil sampling mostly south of the Sisson deposit. Texasgulf Inc. and Kidd Creek Mines Ltd. carried out

exploration work from 1973 to 1983 comprising soil sampling, geological mapping, trenching, ground geophysical surveys, and drilling. Relatively limited work was conducted by various operators between 1977 and 2001. From 2004 to 2009, Geodex, initially in joint venture with Champlain Resources Inc., carried out ground and airborne geophysical surveys, compilation of historical data, trenching, reanalysis of historical drill core, geological mapping and prospecting, and extension of previous soil and till sampling grids over and around the Sisson deposit. Approximately 210 drill holes were completed. Preliminary economic assessments with positive conclusions were completed by Wardrop Engineering Inc. in 2007 and Geodex in 2009. Northcliff signed a joint venture agreement with Geodex in October 2010. Northcliff has conducted diamond drilling and excavated some small pits to investigate overburden characteristics.

#### **1.4 Geology Setting and Mineralization**

The Sisson deposit can be defined as an intrusion related, structurally controlled, bulk tonnage tungsten-molybdenum deposit. Deposits of this type have general hydrothermal similarities to porphyry copper deposits.

The Sisson deposit is centered on a north-trending contact between Acadian intrusions to the west and older metavolcanic and metasedimentary rocks to the east. Mineralization occurs in four contiguous zones. Zones I and II are narrow, structurally controlled zones that extend north from Zone III, which hosts the bulk of the deposit. The Ellipse Zone extends northwest from the southwest corner of Zone III. Metavolcanic and metasedimentary host rocks at Sisson formed during the Taconic Orogeny and are of Cambrian to Ordovician age. They include the predominantly clastic sedimentary sequences in the Miramichi Group overlain by Ordovician felsic to mafic volcanic strata and clastic sedimentary rocks of the Tetagouche Group.

Mineralization in the Sisson deposit is hosted by:

- The quartz diorite and gabbro phases of the Howard Peak Granodiorite
- Felsic, mafic, and mafic crystal tuffs in the western part of the Turnbull Mountain Formation
- Biotite wacke with minor interbeds of tuff in the eastern part of the Turnbull Mountain Formation
- Volumetrically minor granite dykes and very rare mafic dykes

#### **1.5 Project Status**

With completion of the Feasibility Study, Northcliff is now focused on the Environmental Impact Assessment (EIA) for the Sisson Project, to facilitate regulatory/public review and permitting. The Company expects to submit an EIA Report to federal and provincial regulatory agencies in 2013. Following regulatory and public review of the Sisson Project EIA report, Northcliff will apply for the broad range of construction and environmental permits necessary to take the Sisson Project into production.

In addition to filing the EIA, Northcliff will continue to collect baseline environmental data from the site and will complete field programs to advance engineering. Field work is expected to include geotechnical drilling in the immediate vicinity of the planned infrastructure as well as test pits to supplement the current characterization of the overburden and bedrock.

## 1.6 Mineral Resource Estimate

In June 2012, Roscoe Postle Associates Inc. (RPA) conducted an audit of an updated mineral resource estimate for the Sisson Project prepared by Northcliff personnel. The effective date of this estimate was February 29, 2012. The following is an extract from that audit report.

'Mineral Resources were estimated using a block model constrained with wireframe models of the principal geological domains. Values for bulk density, WO<sub>3</sub>, and Mo were interpolated into the blocks using Inverse Distance Squared (ID<sup>2</sup>) weighting.

The Mineral Resource estimate for the Sisson Project is based primarily on information from surface drilling, supplemented in part by historical surface mapping and geophysical data to assist in the interpretations. The database contained collar records for 304 holes.

Samples were capped at 1.1% WO<sub>3</sub> and 0.65% Mo prior to compositing. Samples were composited in down-hole intervals of four meters, starting at the wireframe pierce-point for each zone and continuing to the point at which the hole exited the zone.

The Mineral Resources were reported at a NSR cut-off value of US\$9/t. The estimate was constrained by a Lerchs-Grossmann pit shell.'

For the purpose of this report, RPA was requested to revise the NSR calculation of the February 29, 2012 resource estimate to reflect the processing parameters derived during the Feasibility Study. This most recently revised estimate is considered to be current at December 31, 2012 and is summarized in Table 1.2.

Table 1.2 Mineral Resource Estimate December 31, 2012							
Category	Tonnage (Mt)	WO <sub>3</sub> (%)	Mo (%)	WO <sub>3</sub> (M mtu)	Mo (M lb)	WO <sub>3</sub> Eq (%)	Avg NSR (\$/t)
Measured	108	0.072	0.023	7.70	55.3	0.096	26.67
Indicated	279	0.065	0.020	18.0	122	0.086	23.42
<b>Measured + Indicated</b>	<b>387</b>	<b>0.067</b>	<b>0.021</b>	<b>25.7</b>	<b>178</b>	<b>0.089</b>	<b>24.33</b>
Inferred	187	0.050	0.020	9.41	82.6	0.074	18.63
Notes:							
1. CIM definitions were followed for Mineral Resources.							
2. Mineral Resources are estimated at a net smelter return (NSR) cut-off grade of \$US9.00/t.							
3. Mineral Resources are estimated using a long-term metal prices of US\$350 per mtu WO <sub>3</sub> and \$US15/lb Mo, and a US\$/C\$ exchange rate of 0.9:1.							
4. Metallurgical recoveries for the NSR calculation were 82% for Mo and averaged 77% for WO <sub>3</sub> over the life of mine. WO <sub>3</sub> recovery is a function of mill head grade.							
5. Numbers may not add due to rounding.							

## 1.7 Mineral Reserve Estimate

The mineral reserves for the Sisson deposit have been estimated by Moose Mountain Technical Services (MMTS). This estimate is based on several series of pit designs, beginning with conceptual Lerchs-Grossman (LG) pit optimization and culminating with detailed phased pit designs.

All mine planning work have been carried out on a 3D block model, and unsmoothed pit limits have been developed using a MineSight® software LG algorithm. Prices, recoveries and operating costs were then used to estimate the value of each block in the model and only blocks which contained measured or indicated resources were considered as revenue generating. A series of nested pit limits were developed using varying revenue factors between the base price for each metal and 10% incremental change in price. The nested pit shells were used to determine ultimate pit shell limits and to guide phase design for the detailed pit phase designs.

The mineral reserves, contained within the detailed ultimate pit, are expressed in terms of NSR cut-off grade (CoG), where the NSR is calculated and stored in the 3D block model. The CoG for the detailed ultimate reserve calculations is the incremental break even CoG which is the sum of Processing, Ammonium Paratungstate (APT) Plant and Tailings Storage Facility (TSF) costs. The reserves for the detailed ultimate pit design economic pit limit are presented in Table 1.3.

Table 1.3 Mineral Reserves for the Series 2 Detailed Ultimate Pit Design Economic Pit Limit							
Category	Cut-off Grade	Ore Above CoG	Average Grade Above CoG			Contained Metal Above CoG	
	NSR (\$/t)	ROM kt	NSR	WO <sub>3</sub> %	Mo%	Tungsten (M mtu)	Molybdenum (M lb)
Proven	8.83	105,415	25.48	0.069	0.023	7.3	53.1
Probable	8.83	228,948	23.54	0.065	0.020	14.9	101.7
Total	8.83	334,363	24.15	0.066	0.021	22.2	154.8

A mining loss and dilution factor, based on the waste contact edges was estimated for each pit phase. This factor has been applied to the data in Table 1.3.

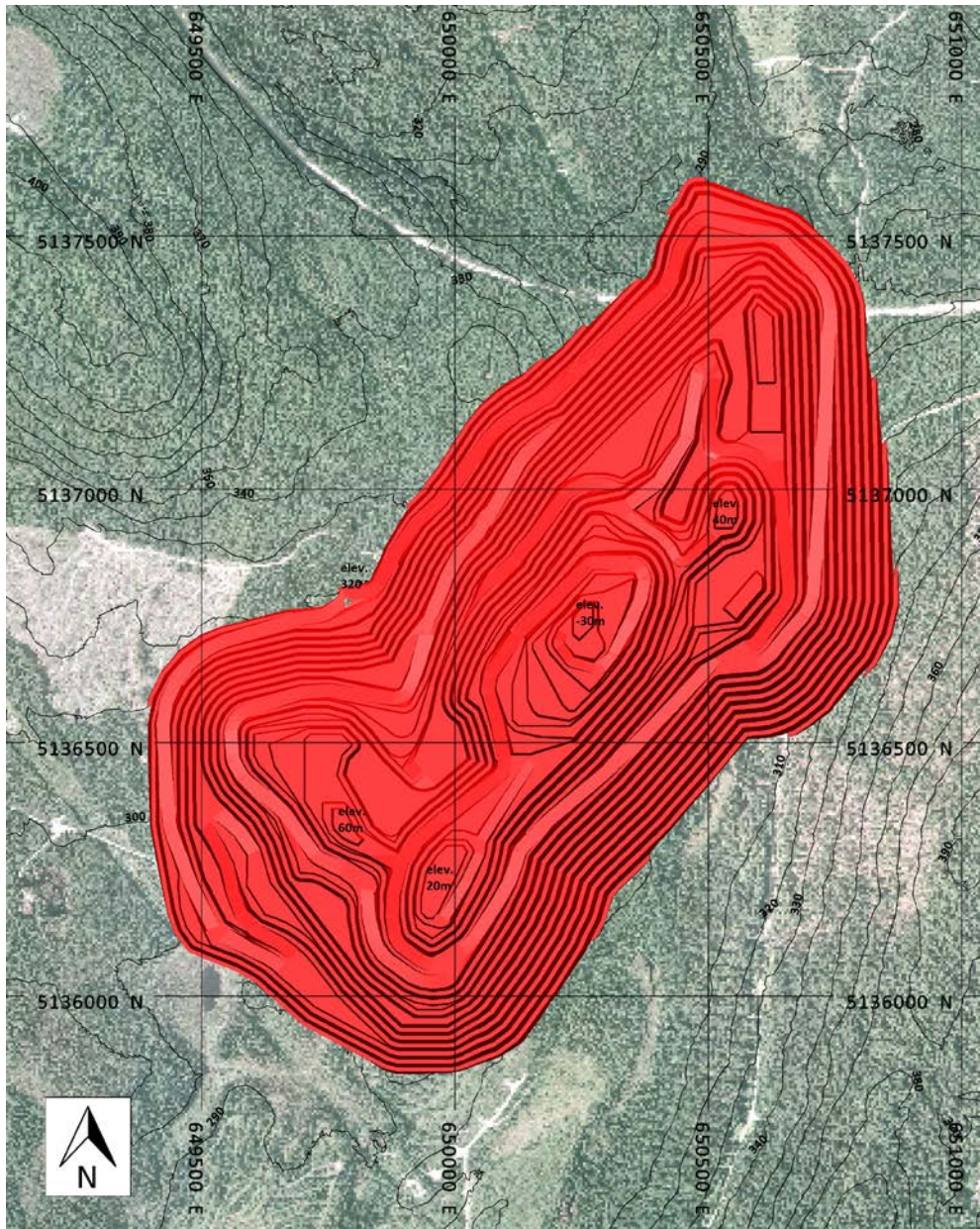
## 1.8 Mining

Mining operations for the Sisson Project are typical of open pit, truck-and-shovel mining methods for year round operations in Canadian climates. The mining fleet consists of 136-t trucks, 16.5 m<sup>3</sup> hydraulic shovels and support equipment. The mining fleet will excavate, on average, approximately 23.7 million tonnes per year (Mt/a) over the life of the Sisson Project, including 10.5 Mt/a of mill feed, nominal 30,000 tonnes per day (tpd), 10.6 Mt/a of barren open pit rock and mid-grade stockpile and 2.6 Mt/a of quarry material for construction of the tailings facility embankments. The mine life is estimated to be 27 years.



**1.8.1 Pit Design**

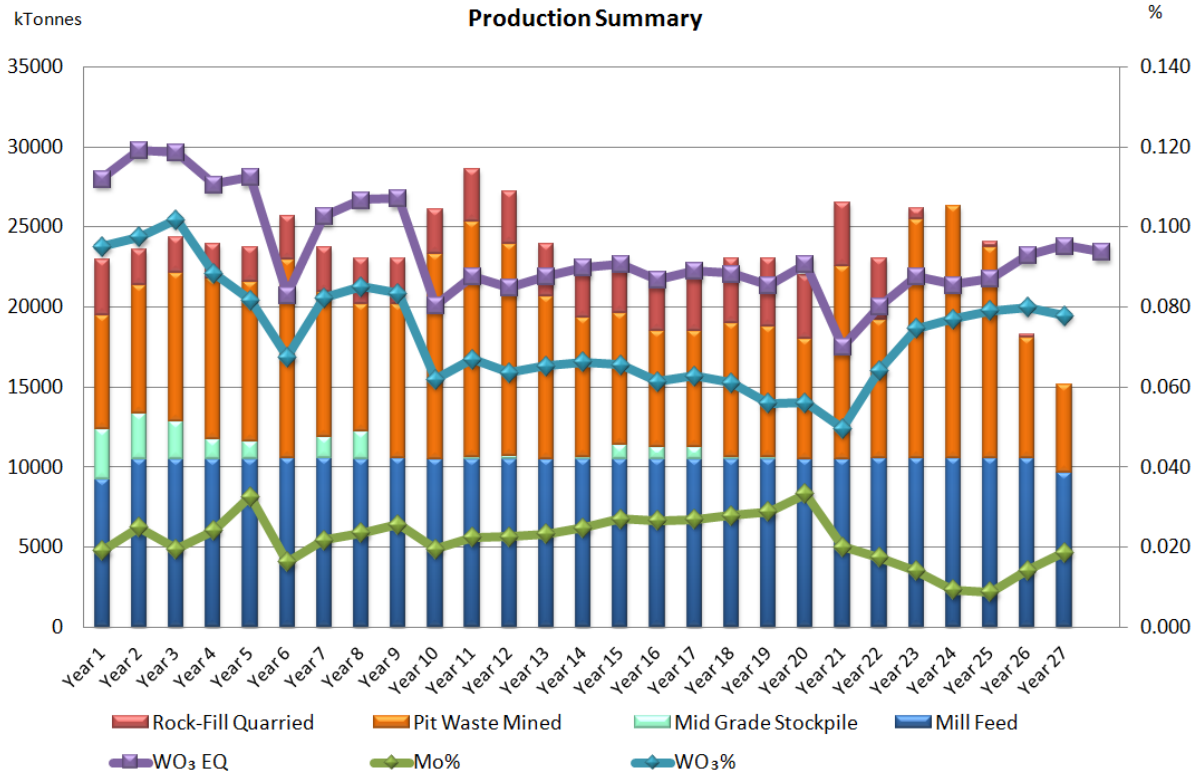
A 3D block model, originating from RPA, was used as the basis for conceptual pit optimization using MineSight® software LG algorithm. This software, in conjunction with economic, metallurgical and geotechnical criteria were used to develop a series of economic pit shells which formed the basis for the detailed pit phased design and production schedule. The detailed pit design of the ultimate pit is presented in Figure 1.2.



**Figure 1.2: Final Detailed Ultimate Pit Design**

**1.8.2 Open Pit Production Schedule**

The open pit production schedule that forms the basis of the economic analysis is summarized in Figure 1.3.



**Figure 1.3: Summary of Production Schedule**

**1.9 Mineral Process and Metallurgical Testing**

Planned mineral processing operations for the Sisson Project include crushing, grinding and flotation to generate both molybdenum and tungsten concentrates. These unit operations for concentration of the minerals of interest are industry standard techniques. Northcliff engaged senior tungsten process engineers to assist with the testing and design of an APT plant which will be used to further treat tungsten concentrates. Northcliff will then market APT directly to the tungsten industry while molybdenum concentrates will be sold to third parties for further refining.

Both the concentrator and APT unit operations have been tested extensively at the SGS Lakefield laboratory using a metallurgical sample collected from Sisson in 2011. This well respected laboratory, along with Northcliff’s consulting engineers, provided the basis for both the concentrator and APT plant design and performance estimates. Recoveries for tungsten and molybdenum are predicted to be 77% (life-of-mine average) and 82%, respectively. Recovery of tungsten in the APT plant is estimated to be 97%.



### 1.9.1 Concentrator Design

Based upon the results of the 2012 Feasibility Study metallurgical test program, the process flowsheet for the Sisson Concentrator includes the following major processing steps:

- Three stage crushing
- Single stage, dual-line grinding and classification
- Molybdenum rougher-scavenger and bulk sulphide flotation
- Molybdenum regrind and four-stage cleaner flotation
- Molybdenum concentrate dewatering and packaging
- Tungsten rougher-scavenger flotation
- Tungsten three-stage cleaner flotation
- Reagent preparation and utilities

A general, simplified block flow diagram for the concentrator is provided in Figure 1.4 below.

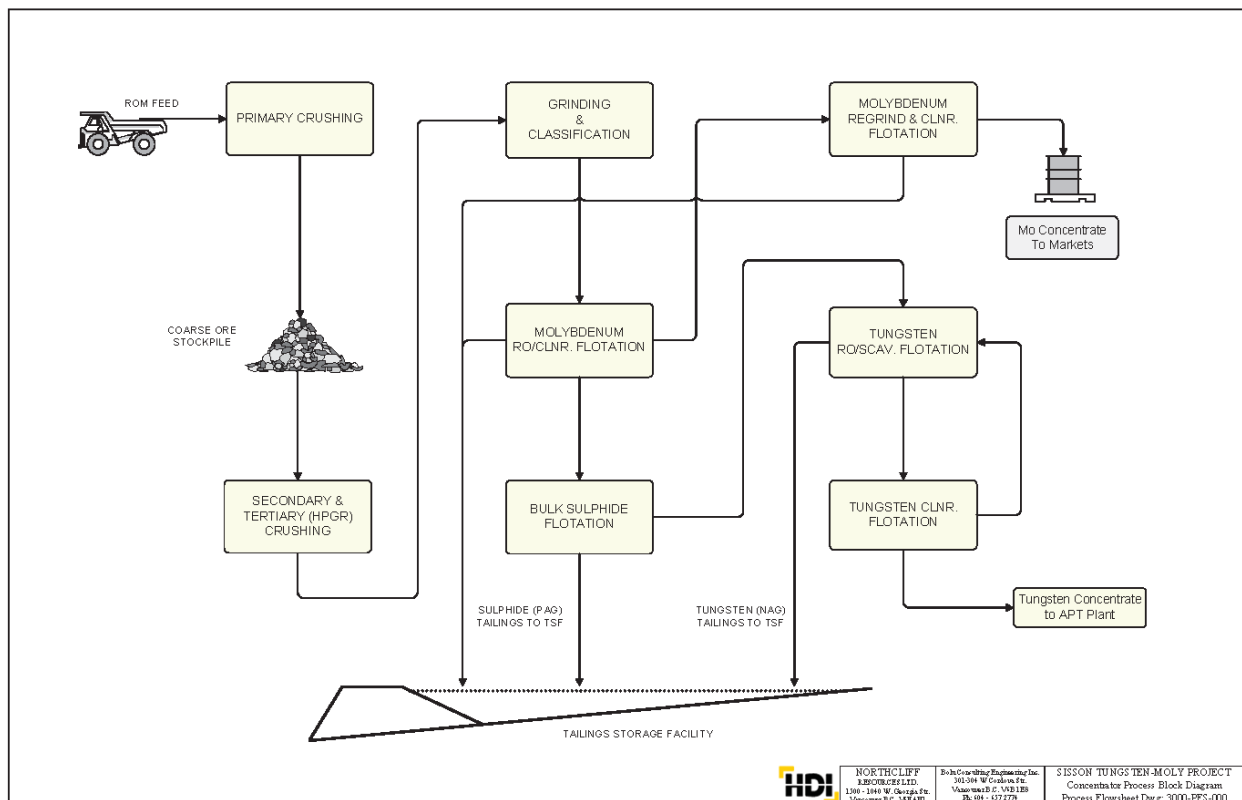


Figure 1.4: Concentrator Simple Block Flow Diagram

### 1.9.2 APT Plant Design

The process flowsheet for Sisson Project APT plant will include the following major processing steps:

- Feed Preparation
- Digestion and Residue Filtration
- Alkali Recovery and Solution Purification
- Conversion to Ammonium Tungstate
- Ammonium Paratungstate (APT) Crystallization
- APT drying and packaging
- Reagent preparation and utilities

A simplified block flow diagram (Figure 1.5) shows the main process steps as designed.

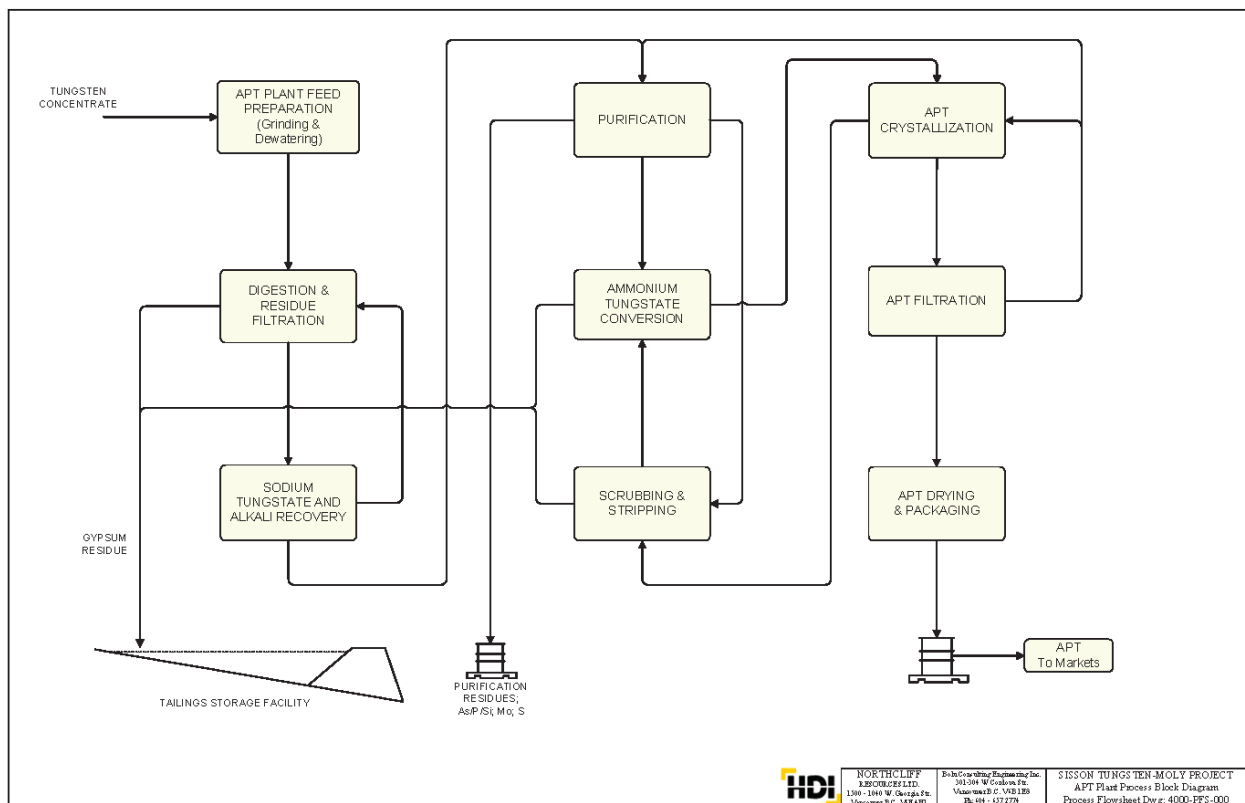


Figure 1.5: APT Simple Block Flow Diagram

### 1.10 Local Resources and Infrastructure

The Sisson property is located 60 km directly northwest of the City of Fredericton in the Province of New Brunswick, Canada, and is readily accessible by 100 km of existing roadways from the south and east. The Sisson Project enjoys ready access to modern transportation and power infrastructure and a skilled labor force. Situated in an area of rolling topography, local access to the site is provided by numerous

secondary and forestry roads. Power lines from the provincial grid cross the property. A rail line and an existing rail siding is located 15 km east of the deposit and connects to deep-sea ports at Saint John and Belledune. Road access to these ports is also readily available.

The Sisson Project's power requirements will be satisfied by tying into the existing New Brunswick electrical grid. Although an existing 345 kV line crosses the Sisson claim block, a new 42 km, 138 kV transmission line will be constructed by NB Power alongside the existing line. NB Power will own the line and switchgear but Northcliff will own the mine site terminal station. Power costs are estimated at \$0.065 per kWh in the Feasibility Study.

Sufficient water exists from precipitation and runoff collected within the project footprint to support the mine operations.

Infrastructure and facilities at the Sisson Project mine site will include an open pit, crushing, conveying, ore stockpile, ore concentrator, APT plant, tailings storage facility, water clarification facility, process buildings, and ancillary buildings including offices, maintenance shops and warehouses.

## **1.11 Environmental and Permitting**

In April 2011, a project description for the Sisson Project was accepted by the Canadian Environmental Assessment Agency to launch the federal Environmental Impact Assessment (EIA) process. The Sisson Project is undergoing a harmonized federal/provincial EIA review. The Terms of Reference (TOR) for the EIA were finalized in April 2012, and Northcliff is completing an EIA Report to meet the TOR requirements.

Northcliff expects to submit an EIA Report to a joint federal and provincial Technical Review Committee (TRC) in 2013. Provincial and federal review and approval processes differ somewhat, but are expected to result in EIA decisions in 2014. A comprehensive permitting plan is being developed so that the individual permits and authorizations required for the Sisson Project will be obtained in a timely fashion to allow the start of construction shortly after the EIA decisions are made.

Northcliff has initiated studies of air quality, acoustics, surface and groundwater resources, environmental geochemistry, terrestrial and aquatic habitats, fish and wildlife, wetlands, land and resource uses, heritage resources, socio-economics, and traditional Aboriginal land uses. The purpose of these studies is to provide information for use in project planning and design and in preparing the EIA Report.

An approved reclamation and closure plan and a financial security held by the province for the associated costs are required before approval to construct the Sisson Project is granted. The reclamation and closure plan undertaken will be to establish self-sustaining physical, chemical and biological stability of the site, and to meet desired end land uses, all as required under provincial and federal legislation and regulations.

Since early 2011, Northcliff has conducted numerous discussions, presentations, workshops and open houses with interested individuals, local communities, First Nations, and stakeholder groups to disseminate project information and to learn about and help address specific concerns.

At this stage, SE is not aware of any issues that may prevent the Sisson Project from advancing.

### 1.12 Tailings Storage Facility

A Tailings Storage Facility (TSF) will manage all tailings and barren rock, including the subaqueous disposal of potentially acid generating materials to mitigate the onset of acidic conditions. Care was taken to minimize the footprint of the TSF and keep the facility in a single watershed. Material for TSF embankment construction will be sourced from a non-potentially acid generating rock quarry located within the footprint of the TSF. The primary aspects of the TSF design include:

- Zoned embankments constructed of earthfill and rockfill
- Upslope TSF diversion channels
- Access roads and haul roads for embankment construction
- Seepage and embankment runoff collection ditches and ponds
- Tailings transport and deposition system
- Reclaim water system
- Tailings beaches
- Supernatant water pond, and
- Barren Rock Dump and Mid-Grade Ore stockpile.

### 1.13 Operating and Capital Cost Estimate

For the Sisson Project Feasibility Study, the operating costs were estimated based on labor productivity, supervision and management for mine operations, equipment consumables, and equipment reliability, availability and productivity. The life-of-mine average operating costs, which include the cost of mining, processing, waste management and general and administrative (G&A) services for the Sisson Project, are summarized in Table 1.4.

<b>Table 1.4 Life of Mine Average Annual Operating Costs</b>		
<b>Average Annual Operating Costs</b>	<b>\$/t</b>	<b>\$/mtu</b>
Mining (per tonne milled)	4.15*	77.7
Milling	7.11	133.2
Waste Management	0.59	11.1
G&A	0.47	8.8
<b>Total Operating Cost to Concentration</b>	<b>12.32</b>	<b>230.8</b>
Moly By-Product Credits	(5.76)	(107.8)
<b>Total Operating Costs</b>	<b>6.56</b>	<b>123.0</b>
APT Costs (including offsite costs)	1.62	30.3
<b>Total APT Cash Costs</b>	<b>8.18</b>	<b>153.3</b>

\*Mining cost per tonne mined is \$2.09/t

The capital cost estimate for the Sisson Project was developed by SE and includes costs from Consultants, within their specialized scope of work, and from SE. The estimate has an accuracy of minus 5% plus 15% with a base date of Q3 2012. The total estimated cost to design, procure, construct and start-up the

facilities described in this report is \$579 million in Canadian currency, as summarized in Table 1.5. The estimate has a contingency allowance of approximately 15%.

<b>Table 1.5 Capital Cost Summary</b>	
<b>Cost Area</b>	<b>\$M</b>
Mine	34.1
Concentrator & APT Plant	247.9
Site Infrastructure & Ancillary	55.4
Owner's Costs & Indirects	168.4
Contingency (approximately 15%)	73.0
<b>Total</b>	<b>578.8</b>

## 1.14 Project Economics

SE prepared a pro forma, un-levered cash flow model for the Sisson Project. Technical and cost inputs for the economic model were developed by SE with specific inputs from Northcliff and other consultants. A summary of the results is provided in Table 1.6.

<b>Table 1.6 Economic Summary</b>		
<b>Financial Results*</b>	<b>Pre-Tax</b>	<b>Post-Tax</b>
Net Present Value (8%)	\$714 M	\$418 M
Internal Rate of Return (IRR)	20.4%	16.3%
Payback	4.1 years	4.5 years
Total Capital Costs	\$579 M	

\*Financial results are as at commencement of construction. Exchange rate assumptions for US\$:C\$ are:  
Capital cost 1.0:1; Years 1-4 0.98-0.92:1; Years 5+ 0.90:1

The metal prices used in the financial analysis are average long-term prices of \$350/mtu for WO<sub>3</sub> and \$15/lb for molybdenum.

## 1.15 Qualified Persons Recommendations and Conclusions

The Feasibility Study confirmed the viability of the Sisson Project as a long-life mining operation. The Sisson Project hosts a Mineral Reserve of 334 Mt at average grades of \$24.14/t (NSR), 0.066% (WO<sub>3</sub>) and 0.021% (Mo) at an \$8.83/t cut-off grade. Financially speaking, the Sisson Project is robust with a post-tax NPV of \$418M and a 16.3% IRR. Technically, the Sisson Project presents no fatal flaws. It is recommended that the Sisson Project proceed to the next level of engineering to enhance the design to pre-construction levels. A summary of the interpretations, conclusions and recommendations to advance the Sisson Project are provided in Sections 25 and 26.

## **2.0 Introduction**

### **2.1 Purpose of the Technical Report**

Samuel Engineering, Inc. (SE) was commissioned by Northcliff Resources Ltd. (“Northcliff”, the “Company” or the “Owner”) to complete a Feasibility Study on the Sisson Project located in New Brunswick, Canada, to prepare a 43-101 Technical Report based on the results of the Study, and to report an updated estimate of mineral resources that has been developed since the last Technical Report on the Sisson Project prepared by Roscoe Postle Associates, Inc. (RPA), dated June 29, 2012. This Technical Report is intended to conform to the standards and reporting requirements set forth in National Instrument 43-101 Standards of Disclosure for Mineral Projects, including Companion Policy 43-101CP and Form 43-101F1. The Technical Report supports Northcliff’s January 29, 2013 news release announcing the results of the study.

Northcliff is a publicly traded (stock symbol TSX: NCF) mineral resource company associated with Hunter Dickinson Inc. (HDI), a diversified, global mining group. Northcliff owns the Sisson Project, located near Fredericton, New Brunswick. Northcliff completed earn-in obligations in exploration, feasibility, and other project costs to secure a 70% interest in June 2012. Northcliff entered into an agreement to acquire the remaining 30% interest in April 2012, which was completed in June 2012. The Sisson tungsten-molybdenum deposit is comprised of disseminated scheelite and molybdenite occurring in sheeted and shear-hosted quartz veins associated with Devonian-aged granitic intrusions.

### **2.2 Sources of Information**

The authors of this report have relied on published and unpublished reports and literature for information provided in this Technical Report. The documentation reviewed and sources of information referenced are listed in Item 27 of this report.

This Technical Report is prepared with contributions from Northcliff and various consulting firms including, RPA, Inc. (RPA), Knight Piésold Ltd. (KP), Moose Mountain Technical Services (MMTS), Bolu Consulting Engineering, Inc. (Bolu), Stantec Consulting, SRK Consulting Ltd and Samuel Engineering, Inc. (SE).

Table 2.1 summarizes the section responsibilities of Qualified Persons contributing to this Technical Report. All persons and their respective companies listed are independent of Northcliff, as defined by NI 43-101.

Table 2.1 Qualified Persons Section Responsibilities		
Qualified Persons	Consultant	Section Responsibility
David Rennie, P. Eng.	RPA, Inc.	Sections: 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0 and 14.0 Portions of: 1.0 and 25.0
Daniel Friedman, P. Eng.	Knight Piésold Ltd.	Sections: 18.12, 18.13 and 18.14 Portions of: 1.0, 5.2, 20.6, 21.0, 25.0 and 26.0
Jim Gray, P. Eng.	Moose Mountain Technical Services	Sections: 15.0 and 16.0 Portions of: 1.0, 18.0, 21.0, 25.0 and 26.0
Matt Bolu, M.Sc., P. Eng.	Bolu Consulting Engineering, Inc.	Sections: 13.0 and 17.0 Portions of: 1.0, 18.0, 21.0, 25.0 and 26.0
Gene Greskovich, PE	Gene Greskovich	Sections: none APT Portions of: 1.0, 13.0, 17.0, 21.0 and 26.0
Steven Pozder, PE	Samuel Engineering, Inc.	Sections: 2.0, 3.0, 4.0, 5.1, 5.3, 5.4, 18.1, 18.2, 18.3, 18.4, 18.5, 18.6, 18.7, 18.8, 18.9, 18.10, 18.11, 19.0, 20.1, 20.2, 20.3, 20.4, 20.5, 21.0, 22.0, 23.0, 24.0 and 27.0. Portions of: 1.0, 5.2, 20.6, 25.0 and 26.0.

### 2.3 Personal Inspection of the Sisson Property

David Rennie, P. Eng., RPA Principal Geologist completed a site visit to the Sisson Project on February 3, 2012. The purpose of the site visit was to solicit information required to conduct an audit of an updated mineral resource estimate for the Sisson Project. The result of this audit is reported in the June 29, 2012 Technical Report presented by RPA and revised for this report (see Section 14.0).

Mr. Rennie held discussions with the following personnel from Northcliff and HDI:

- Mr. David Gaunt, P. Geo., Vice President Resource and Database, HDI
- Dr. James Lang, P. Geo., Senior Vice President Geology, HDI
- Mr. Yuri Likhtarov, GIS/Database Specialist, HDI
- Mr. Drew Takahashi, Operations Manager, Northcliff
- Mr. Eric Titley, P. Geo., Senior Manager Resource Geology, HDI

Daniel Friedman, P. Eng., Senior Engineer, KP visited the Sisson Project site on September 13, 2011. The purpose of the site visit was to gain a firsthand understanding of the site characteristics. Geotechnical site investigation work was ongoing at the time of the visit, which allowed a brief review of the equipment and procedures, as well as some of the samples and test results.

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James H Gray, P. Eng., Principal Mining Engineer, MMTS visited the Sisson Project site on November 7, 2011. The purpose of the site visit was to review the project site and gain an understanding of the project's surface characteristics, potential layout, and logistics of the general area.



### 3.0 Reliance on Other Experts

This report has been prepared by SE for Northcliff. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to SE at the time of preparing this report,
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and other information supplied by SE, Northcliff and other third party sources.

For the purpose of this report, SE and the other QP contributors have relied on ownership information provided by Northcliff (the Client). The Client has relied on an opinion by the law firm, Barry Spalding, dated March 24, 2011 (Barry Spalding, 2011), and this opinion is relied on in Section 4.0 and the Summary of this report.

SE has relied upon Northcliff, through a letter from Trevor Thomas, Northcliff's Secretary and Legal Counsel, dated February 26, 2013, confirming that title to the claims comprising the Sisson Project is held in the name of Northcliff and these are in good standing. SE has not researched property title or mineral rights for the Sisson Project and expresses no opinion as to the ownership status of the property.

Northcliff provided the information in Sections 20.1, 20.2, 20.3, 20.4 and 20.5, and parts of the information in Section 20.6. SE has not been involved in the EIA, permitting, or social aspects of the project and has not verified the information provided.

SE engaged the services of PricewaterhouseCoopers LLP (PwC), an Ontario limited liability partnership, to conduct a review of the income and mining tax requirements of the Sisson Project. PwC's findings have been presented to SE in a January 22, 2013 letter distribution titled, PwC Tax Report, and its review of these matters are relied on in Section 22.0 of this report.

Except for the purposes legislated under provincial securities laws any use of this report by any third party is at that party's sole risk.

## 4.0 Property Description and Location

### 4.1 Location

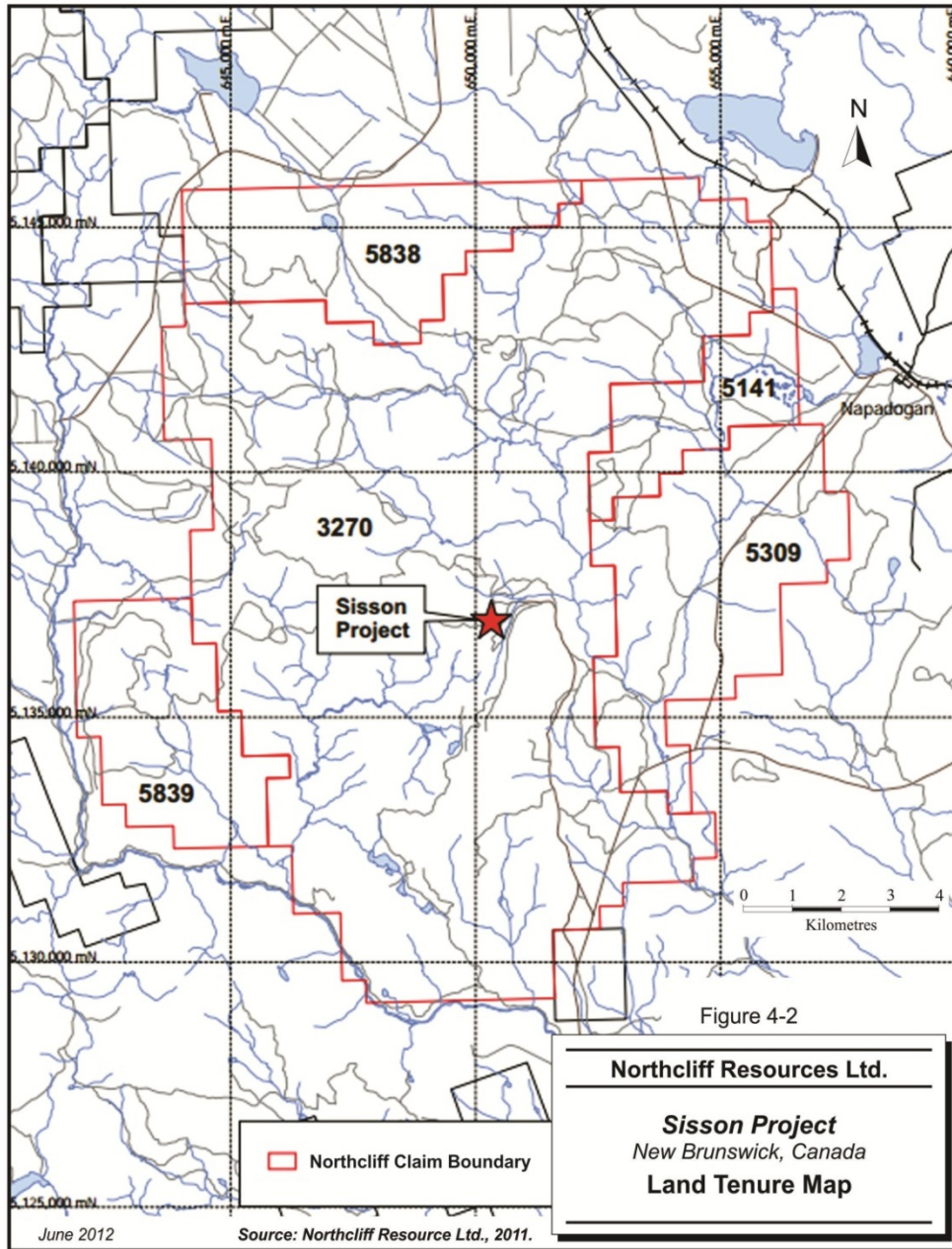
The Sisson property is located 60 km directly northwest of the City of Fredericton in the Canadian province of New Brunswick, as shown in Figure 4.1.



**Figure 4.1: Project Location Map**

## 4.2 Claim Boundaries

The claims lie within National Topographic Series map sheets 21J2, 3, 6, and 7, and the approximate center of the property is at UTM grid coordinates 5135500N, 650000E, as shown in Figure 4.2.



**Figure 4.2: Claim Boundary and Land Tenure Map**

### 4.3 Land Tenure

Tenure for the mineral rights is held via five contiguous claim groups comprising a total of 850 units (Figure 4.2) encompassing 18,880 ha. In New Brunswick, claims are staked online as blocks of units which measure 500 m x 500 m each. The list of claim groups is provided in Table 4.1. SE accepts the data as reported and cannot guarantee that the information is either accurate or current.

Northcliff reported that the claims were originally staked prior to the implementation of the map-based system now in place. The claims were converted to the newer system on February 3, 2011.

Northcliff provided a number of documents supporting the land tenure. These documents include a letter dated March 24, 2011, from the legal firm Barry Spalding of Saint. John, New Brunswick, which confirmed the ownership of the claims, and a letter from Trevor Thomas, Northcliff's Secretary and Legal Counsel, dated February 26, 2013, confirming that title to the claims comprising the Sisson Project is held in the name of Northcliff and are in good standing. Also provided were copies of the digital confirmations of the most recent assessment work filed on the claims to maintain them in good standing.

The Mineral Resources are all located within claim group number 3270.

Table 4.1 Mineral Claims List							
Right No.	Mineral Claim Name	Mineral Claim Type	Mineral Claim Sub Type	Issue Date	Expiration Date	Status	Units
5141	Turnbull Mountain	Mineral	Claim	2007-06-14	2013-06-14	Active	40
5839	Barker Brook West Branch	Mineral	Claim	2010-08-17	2013-08-17	Active	66
5838	Napadogan	Mineral	Claim	2010-08-17	2013-08-17	Active	77
5309	Napadogan Brook	Mineral	Claim	2007-11-28	2013-11-28	Active	106
<b>3270</b>	<b>Sisson Brook</b>	<b>Mineral</b>	<b>Claim</b>	<b>1997-09-04</b>	<b>2013-09-04</b>	<b>Active</b>	<b>561</b>
<b>Total</b>							<b>850</b>

Northcliff acquired the claims through a joint venture (JV) agreement with Geodex, which was signed on October 21, 2010. Under the terms of the JV, Northcliff could acquire a 70% interest in the Sisson Project by funding up to C\$17 million in exploration and development costs. Northcliff was appointed the operator of the Sisson Project. Northcliff completed the earn-in obligations to secure a 70% interest in June 2012.

In April 2012, Northcliff and Geodex entered into a binding letter agreement by which Northcliff could acquire the remaining 30% interest in the property. Geodex agreed to sell its interest for 16,003,700 common shares of Northcliff, \$1 million in cash, and the return of 3,333,333 Geodex common shares which had been purchased by Northcliff in 2010. Northcliff shareholders unanimously approved the acquisition at the company's Annual General Meeting held on May 30, 2012, and Geodex shareholder approval was received on June 18, 2012. The transaction closed on June 21, 2012. Northcliff owns a 100% interest in the Sisson Project as of the date of this report.

#### **4.4 Royalties, Back-in Rights, Payments or Other Agreements**

There are no royalties on the property, or back-in rights.

Northcliff does not hold any surface rights. New Brunswick mining law allows for access and use of the surface for mining through the permitting process.

#### **4.5 Environmental Liabilities and Permits, Other Factors**

Northcliff is currently compiling an Environmental Impact Assessment (EIA) Report in preparation for permitting, but Sisson Project permits have not yet been obtained.

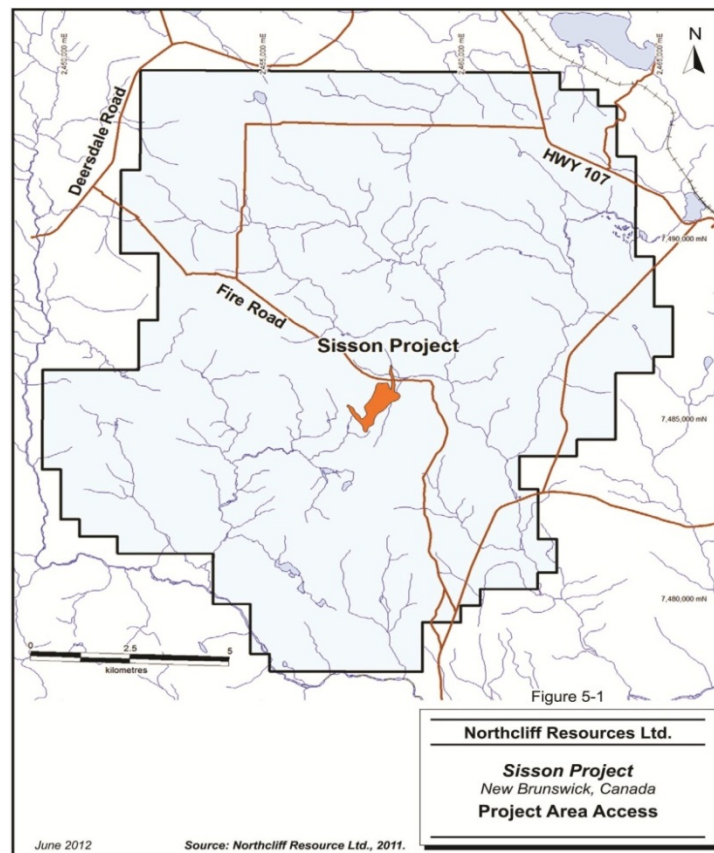
SE is not aware of any factors or risks that may affect access, title, or the right to perform work on the property.



## 5.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

### 5.1 Accessibility

Access to the Sisson property is gained via paved highway and good quality gravel Forestry Service Roads (FSR) as shown on Figure 5.1. The nearest large population center is Fredericton, which is the provincial capital of New Brunswick. From Fredericton, the property is accessible by travelling west for approximately 60 km along Highway 1 (Trans-Canada Highway) to the town of Nackawic, and then north via approximately 45 km of secondary highway and Valley Forest Products forestry roads. Access via Highways 8 and 107 (through the villages of Stanley and Napadogan) is an alternative route to the Sisson property from Fredericton. Paved road from Fredericton to just north of Napadogan is approximately 75 km. Another 10 km of gravel forest road heading west, accesses the Sisson property.



Note: The orange outline is the outline of the ore deposit.

**Figure 5.1: Project Area Access**

### 5.2 Climate

Climate records for Fredericton can be obtained for the period 1971 to 2000 from the Environment Canada website (<http://climate.weatheroffice.gc.ca>). However, since the Sisson property is somewhat

higher in elevation than Fredericton, temperatures are likely to be somewhat lower and precipitation higher.

More recent climate data have been collected in the Sisson Project area since December 2007 using an automated meteorological station located south of the deposit area at an elevation of 305 masl. Hydrometric monitoring was initiated in April 2008 at seven gauging stations and continued until late 2010. Seven new stations were installed in 2011 and are currently active. In addition to the site specific climate and hydrometric data, data from regional climate stations operated by the Meteorological Services of Canada (MSC) branch of Environment Canada were incorporated in the climate analyses to predict long-term regional climate patterns at the Sisson Project site. The closest active MSC station is Juniper (8102275), which is located approximately 23 km northwest of the Sisson Project area climate station at an elevation of 259 masl. There are 36 complete years of record at the Juniper station.

#### Temperature

The mean annual temperature for the Sisson Project area is estimated to be 3.3°C, with minimum and maximum monthly temperatures of -16.6°C and 20.0°C occurring in January and July, respectively.

#### Precipitation and Evaporation

The Mean Annual Precipitation (MAP) for the Sisson Project area is estimated to be 1350 mm, with 1013 mm falling as rain and 337 mm falling as snow. The estimated mean annual lake evaporation is 500 mm at the TSF.

Snow can generally be expected from November to March, with accumulations remaining on the ground from December to February. In spite of these conditions, exploration and mining activities can be carried out year round.

### **5.3 Local Resources and Infrastructure**

Fredericton is the closest major population center, the third largest city in New Brunswick, and is the capital city of the province with a population in 2011 of just over 56,000. The city is able to supply all necessary supplies and commercial services for exploration, mining and construction consumables. Daily international commercial air service operates out of Fredericton, as well as rail, bus, courier and truck transport.

Several smaller towns and villages are located in the Sisson Project area, and these can provide labor and minor services. The largest is Nackawic, a town of approximately 7,000 people, located 35 km from the property. Nackawic has been the site of a pulp mill since 1970, and is also a likely source of skilled labor and heavy industrial services.

Power lines from the provincial grid cross the property and, as previously stated, the site is readily accessible by existing paved and all-weather gravel roads. The CN rail line is located 15 km east of the deposit and connects to deep-sea ports at Saint John and Belledune. Road access to these ports is also readily available. There are currently no existing structures at the Sisson Project site.

Sufficient water exists from precipitation and runoff collected within the Sisson Project footprint to support the mine operations. The Feasibility Study includes an allowance in the capital cost estimate to drill 5 to

10 groundwater wells and to install pumping systems and pipelines for fresh water supply to the processing facility and truck maintenance shop. However, the principle water requirements for the Sisson Project will be supplied by recycling and reusing water captured within the tailings storage facility.

The Feasibility Study has identified areas for tailings storage, barren rock storage, topsoil and organic stockpiles, sewage leach fields, ore processing facilities, quarry, the open pit mine and other ancillary facilities. The locations of these facilities are shown on the site plan, Figure 5.2.



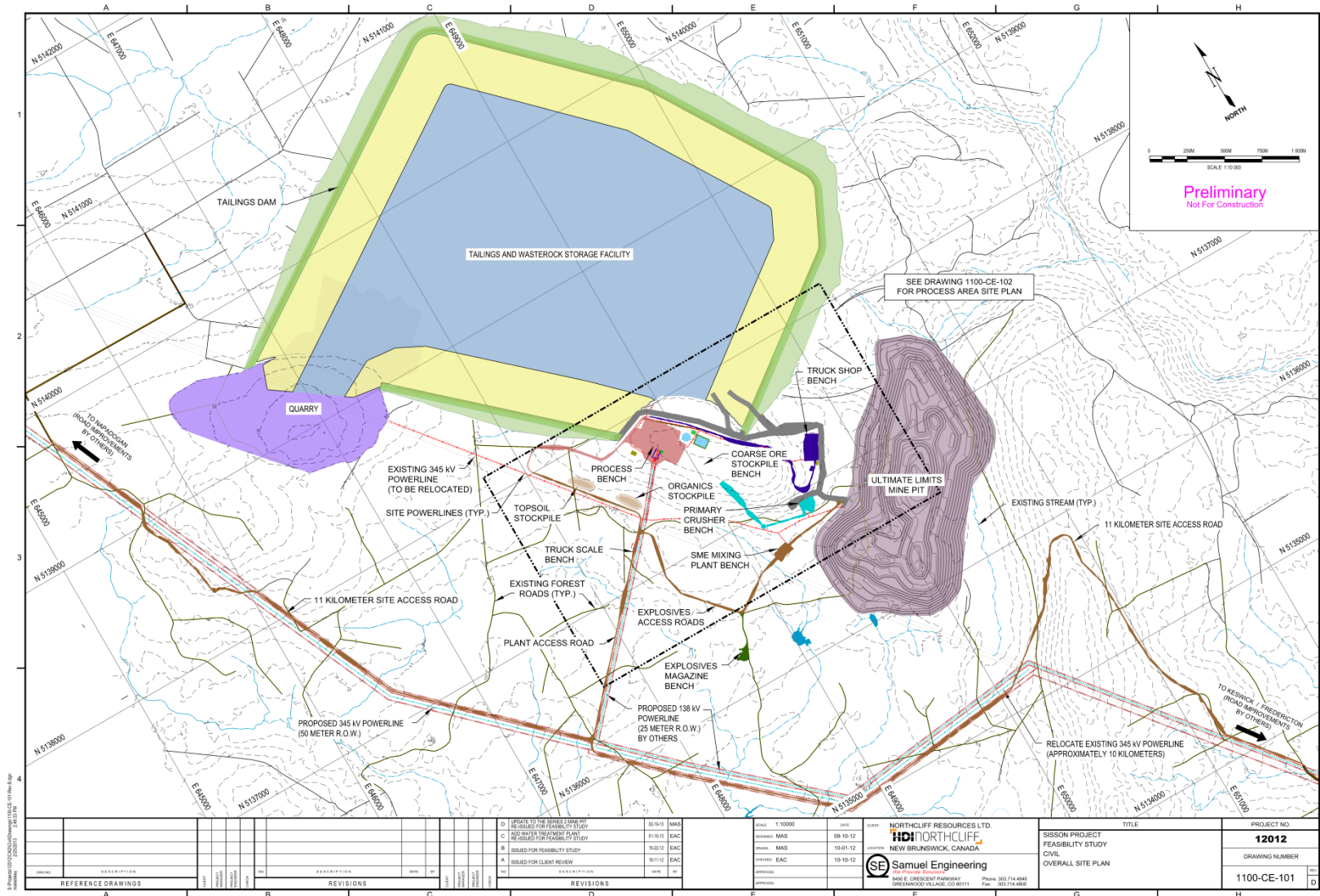


Figure 5.2: Sisson Site Plan

## 5.4 Physiography

The Sisson Project is located in the Miramichi Highlands (also known as the Central or New Brunswick Highlands), which are characterized by rolling topography and broad valley features. Hilltops are strongly glaciated and rounded, and drainage valleys tend to be broad and open. The surface elevation in the Sisson Project area typically ranges from approximately 290 m to 400 m above sea level, with some local hills rising to just over 400 m. The Sisson Project is within the Nashwaak River watershed, which is a tributary of the Saint John River to the south. The Sisson Project is not located in the Miramichi River Watershed.

The site has a mix of deciduous and coniferous forests typical of Canadian boreal environments, and clear-cut logging and reforestation operations are active on the property. Other land uses for the surrounding region include agriculture and recreation.

RPA, in their June 29, 2012 Technical Report, expressed the view that the area and physiography do not present any impediment to exploration or the development of an open pit mining operation and its site infrastructure.

The following are the site data developed for the design of the Sisson Project:

- Latitude 46° 22' North
- Longitude 67° 03' West
- Elevation approximately 385 meters
- Frost Depth 2000 millimeters
- Snow Load (1/50) NBC 2010,  $S_s = 3.6$  kPa;  $S_r = 0.6$  kPa
- Wind Speed Design 88.2 kilometers per hour
- Earthquake Zone Site Class C, PGA = 0.20
- Atmospheric Pressure (mean) 97 kPa
- Maximum Design (dry bulb) Temperature, plus 28°C
- Maximum Design (wet bulb) Temperature, plus 22°C
- Minimum Design (dry bulb) Temperature, minus 27°C
- Annual Rainfall 894 millimeters
- Maximum Snowfall Depth 900 millimeters (estimated)
- Design Maximum Rainfall 24 hours 72 millimeters

## 6.0 History

This section has largely been extracted from the June 29, 2012 Technical Report by RPA.

### 6.1 Exploration History

This section was prepared by Dr. James Lang, P. Geo., Senior Vice President Geology, HDI, and is a compilation from various assessment reports filed with the New Brunswick Department of Natural Resources and from a previous compilation by Giggi (2006).

#### Nashwaak Pulp and Paper Co. (1955-1960)

The first significant work in the Sisson area was completed in the late 1950s. The focus of this early work was east of Sisson and included prospecting, line cutting, geological mapping, trenching, ground (horizontal loop electromagnetic (EM) and magnetometer) and airborne (EM and magnetometer) geophysical surveys and diamond drilling. Twelve holes were completed in 1955 and 43 holes in 1959-1960 which resulted in the discovery of the Nashwaak polymetallic vein deposit.

#### Penarroya Canada Ltée. (1967-1969)

Geological mapping, a ground magnetic survey, and soil sampling were completed mostly south of the Sisson deposit. This work identified local zinc and copper soil anomalies. Four holes were drilled, including one near the Sisson deposit, but results were not encouraging.

#### Texasgulf Inc./Kidd Creek Mines Ltd. (1973, 1978-1983)

The work in 1973 focused on the area of the Nashwaak vein deposit east of Sisson and included soil sampling, geological mapping, trenching, ground geophysical surveys (magnetic and horizontal loop EM), and five short drill holes. In 1978, soil surveys in the area of the Sisson deposit returned strong geochemical anomalies for Zn, Cu, W, Bi, Pb, and Sn. Subsequent drilling of 40 holes between 1979 and 1983 led to discovery of Zones I and II in 1979, and Zone III in 1981. This work is summarized by Mann (1981, 1982).

#### Various Operators (1977-2001)

Between 1977 and 2001, additional, comparatively limited work was completed by Canadian Nickel Company Ltd. (1977-1981), Riocanex Inc., and Rio Tinto Canadian Exploration Ltd. (1980-1982), Shell Canada Resources (1981), K.D.A. Whaley (1981-1982; 1989), Phelps Dodge Corporation of Canada Ltd. (1992-1993), Noranda Exploration Company Ltd. (1992-1996), and Nikon Holdings Ltd./Freegrant Silver Corporation Ltd./Champlain Resources Inc. (1999-2001). The work by these groups was mostly surface exploration and geophysical surveys, accompanied by very limited drilling. Most of this work was completed outside the Sisson deposit and did not result in any significant discoveries of mineralization.

#### Geodex Minerals Ltd. and Champlain Resources Inc. (2004-2009)

The next period of significant exploration in the Sisson deposit was completed by Geodex Minerals Ltd. (Geodex), initially in joint venture with Champlain Resources Inc. Work included ground and airborne

geophysical surveys, compilation of historical data, trenching, reanalysis of historical drill core, geological mapping and prospecting, and extension of previous soil and till sampling grids over and around the Sisson deposit. Approximately 210 drill holes were completed. The results of this work largely outlined the extent of mineralization in Zones I, II and III, and led to discovery of the Ellipse Zone in 2008. Several NI 43-101 compliant technical reports, Mineral Resource estimates, and Mineral Resource updates, using various tungsten trioxide ( $WO_3$ )-equivalent ( $WO_3Eq$ ) cut-off values, were produced between 2007 and 2010. Preliminary economic assessments with positive conclusions were completed by Wardrop Engineering Inc. (Wardrop) in 2007 and Geodex in 2009.

#### Northcliff Resources Ltd. and Geodex Minerals Ltd. (2010 to Present)

Northcliff Resources Ltd. (Northcliff) signed a joint venture agreement with Geodex in October 2010. Under the terms of the agreement, Northcliff would be the operator with the opportunity to earn 70% of the Sisson Project for expenditures of about \$17 million (see details in the section of this report entitled Land Tenure, Section 4.3). The work completed by Northcliff following the JV agreement is described in Sections 9.0 and 10.0 of this report.

## **6.2 Historical Resource Estimates**

The earliest Mineral Resource estimate referenced in the Geodex technical reports was compiled by Kidd Creek Mines Ltd. in or around 1980 (Mercator, 2007). This estimate was carried out for Zone I, and totaled 7.5 million tons of “Drill Indicated” material grading 0.21%  $WO_3$  and 0.35% Cu. RPA noted that this is a historical estimate as defined by NI 43-101 and should not be relied upon. It is provided here for reference only.

Geodex subsequently prepared an estimate for Zone II which totaled 1.6 million tons grading 0.215%  $WO_3$  and 0.41% Cu. RPA noted that this is a historical estimate and should not be relied upon. RPA further noted that the date for this estimate was not reported in the data they were provided but it was apparently compiled prior to 2006.

In 2006, Geodex prepared an estimate for what was then termed a “Potential Mineral Deposit” within Zone III. This estimate comprised 26.0 Mt of Inferred Mineral Resources grading 0.05%  $MoS_2$  and 0.102%  $WO_3$ . Once more, RPA stated that this was a historical estimate and should not be relied upon.

The first public disclosure of a Mineral Resource estimate for the Sisson Project under NI 43-101 was made in March 2007 (Mercator, 2007). The estimate was compiled for Zone III on behalf of Geodex by Mercator. Mercator prepared the estimate using a block model constrained by 3D wireframe solids, and grades were interpolated using  $ID^2$  weighting.

RPA noted in their June 2012 Technical Report that the resources were reported at a range of  $WO_3$ -equivalent ( $WO_3Eq$ ) cut-offs. However, the base-case cut-off grade was not specified, nor were the metal prices used to derive the  $WO_3Eq$  values and cut-offs.

An update of the Mineral Resource estimate, using data from an additional 18 holes drilled that year was released in October 2007. All Mineral Resources were classified as Inferred. This estimate formed the

basis for a preliminary economic assessment (PEA) on the Sisson Project, which was conducted by Wardrop in 2007.

Again, the base-case cut-off grade for the resource estimate was not specified in the Technical Report. RPA noted that in the PEA, cut-off criteria were not explicitly stated either. The financial model was based on a total of 207 Mt of mill feed grading 0.063% WO<sub>3</sub> and 0.024% Mo. In RPA's opinion, this implied that the cut-off grade would be somewhere between the 0.025% WO<sub>3</sub>Eq and 0.075% WO<sub>3</sub>Eq cut-offs.

Geodex continued with diamond drilling, which resulted in discovery of additional Mineral Resources as well as upgrades in classification to both Measured and Indicated categories (Mercator, 2008). Mercator concluded that the increase in tonnes from the previous estimate was due to additional resources discovered by drilling.

Following the 2008 drilling program, Mercator prepared an updated estimate which included both Zone III and the Ellipse Zone (Mercator, January 2009). RPA noted that for the first time, a base-case cut-off was assigned to the estimate. This cut-off grade was 0.125% WO<sub>3</sub>Eq and was derived using metal prices of US\$9.00/lb WO<sub>3</sub> and US\$15.00/lb Mo, plus recoveries of 70% for WO<sub>3</sub> and 85% for Mo.

Compared to the previous estimate, the January 2009 estimate was somewhat smaller in terms of tonnes in some categories and cut-off grades, but larger in others. Grades for WO<sub>3</sub> were moderately to significantly higher in all categories. Molybdenum grades were generally slightly higher with some categories showing a modest reduction. Mercator concluded that the change in the Mineral Resources was due to an increase in drilling information plus changes to the cut-off criteria applied to the model.

The January 2009 Mineral Resource estimate was used by Geodex in an update of the PEA for the Sisson Project, conducted during 2009 (Geodex, 2009). The financial model for the updated PEA used metal prices of US\$10.00/lb WO<sub>3</sub> and US\$15.00/lb Mo, with an exchange rate of C\$1:US\$0.85. Metallurgical recoveries were 73.9% for WO<sub>3</sub> and 70.4% for Mo. The cut-off grade derived from the PEA was reported to be 0.100% WO<sub>3</sub>Eq, and this cut-off was applied to subsequent Mineral Resource estimates.

Geodex continued the drilling in 2009, and released another updated resource estimate in December of that year (Mercator, December 2009). As explained above, the cut-off grade for the December 2009 resource estimate was reduced from 0.125% WO<sub>3</sub> to 0.100% WO<sub>3</sub>. This was reportedly done to reflect the base-case cut-off grade developed by Geodex in the update of the PEA.

The tonnage for Measured and Indicated Mineral Resources increased for all cut-off grades, and decreased for the Inferred. Grades for WO<sub>3</sub> tended to remain unchanged from the January 2009 estimate in Zone III but were generally lower in the Ellipse Zone. This was particularly true for the higher cut-off grades (i.e., 0.175% WO<sub>3</sub> and 0.225% WO<sub>3</sub>). Molybdenum grades were similar for both estimates in the Measured and Indicated categories in Zone III, but significantly higher for the Inferred. In the Ellipse Zone, molybdenum grades varied in both directions, ranging from an increase of 32.8% in the Inferred at the 0.225% WO<sub>3</sub> cut-off grade to a decrease of 20.8% in the Indicated at the 0.025% WO<sub>3</sub> cut-off grade. There were no opinions offered in the Mercator December 2009 Technical Report as to the reasons for the changes, but RPA assumed it was due primarily to the drilling done during 2009.

In 2011, Mercator was retained to prepare a NI 43-101 Technical Report on the property to support a reverse take-over agreement between Northcliff and Cabre Capital Corp. (Mercator, 2011). The Mineral Resource estimate in this report was unchanged from the December 2009 report. The only modification made was the effective date, which was revised to February 4, 2011.



## 7.0 Geological Setting and Mineralization

This section was extracted in its entirety from the June 29, 2012 Technical Report by RPA and was prepared by Dr. James Lang, P. Geo., Senior Vice President Geology, HDI for that report.

### 7.1 Regional Geology

The regional geological setting of southwest New Brunswick is illustrated in Figures 7.1 and 7.2. The summary which follows is based mostly upon work by Fyffe et al. (2008), Fyffe and Thorne (2010), and references contained therein.

Mesoproterozoic to Earliest Cambrian basement rocks are exposed in southwest New Brunswick (Figure 7.1). They include carbonate and clastic sedimentary rocks of Mesoproterozoic to Neoproterozoic age in the Green Head Group and Neoproterozoic to earliest Cambrian mafic and felsic volcanic rocks and clastic sedimentary rocks in the Belleisle Bay Group. The tectonic setting comprised a comparatively stable volcanic and sedimentary continental margin that continued into the Cambrian. Intrusions related to the basement sequences were emplaced between 622 and 528 Ma and have calc-alkalic, continental margin geochemical signatures. These basement rocks are overlain by variably deformed and metamorphosed stratified sequences that were deposited in continental to marine environments between Early Cambrian and Early Devonian time.

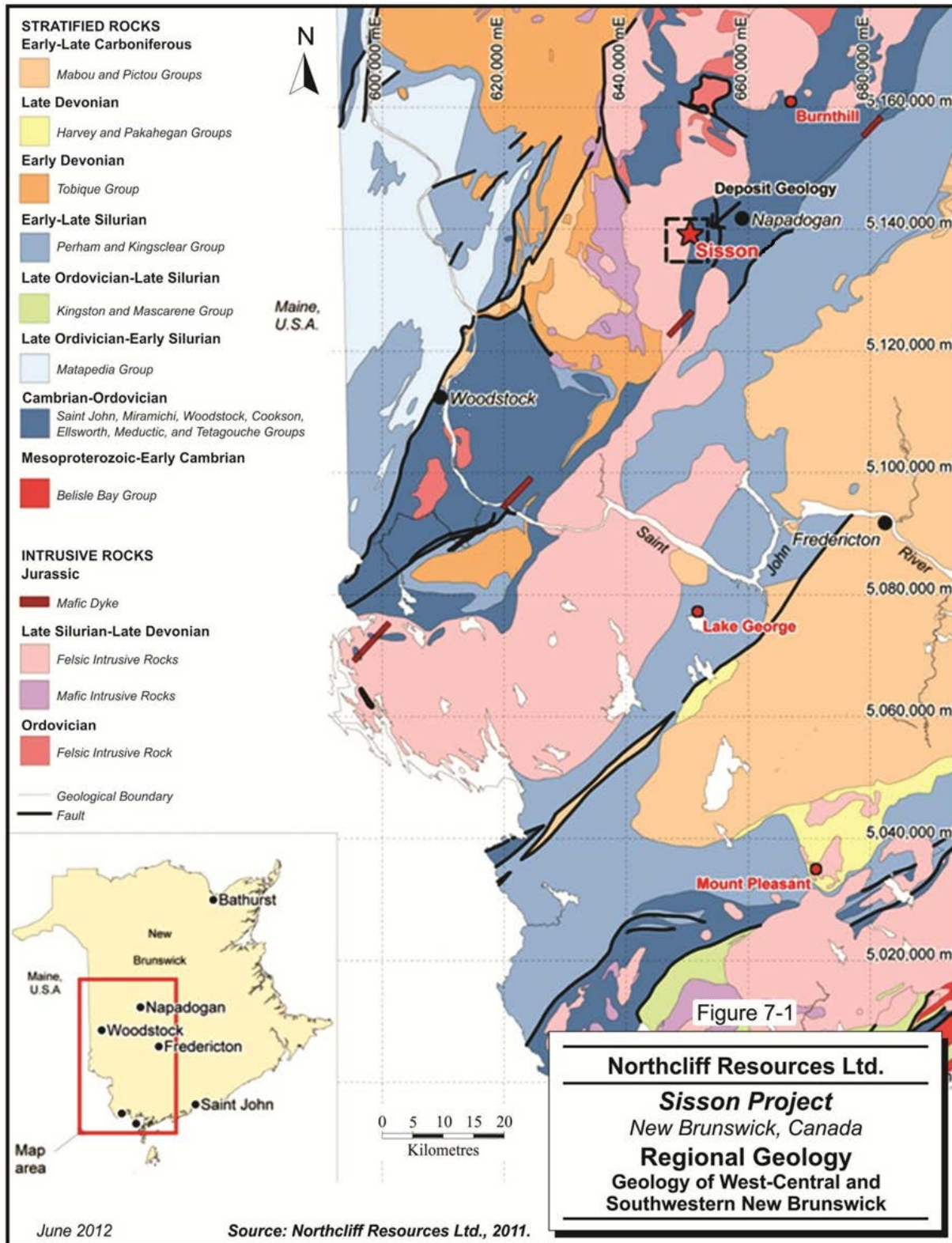
The Taconic Orogeny began by about 490 Ma, in the Early Ordovician, and continued into the Silurian. Tectonism was characterized by amalgamation of volcanic arcs during subduction of Iapetus (proto-Atlantic) oceanic crust. Sedimentary and volcanic rocks were deposited in volcanic arc and/or back-arc tectonic settings. Early Cambrian to Silurian sequences (Figure 7.1) are dominated by fine to coarse grained, clastic sedimentary rocks and lesser carbonate sequences which are, in many places, interlayered with mafic and felsic volcanic rocks. Metavolcanic and metasedimentary host rocks at Sisson formed during the Taconic Orogeny and are of Cambrian to Ordovician age. They include the predominantly clastic sedimentary sequences in the Miramichi Group overlain by Ordovician felsic to mafic volcanic strata and clastic sedimentary rocks of the Tetagouche Group.

The Acadian orogeny occurred from Late Silurian to Late Devonian and was a response to closure of the Iapetus Ocean and subsequent crustal thickening. Stratified sequences include both volcanic and clastic sedimentary rocks. The most important manifestation of the Acadian Orogeny is a north-northeast trending belt of plutons and batholiths that were emplaced between about 425 and 360 Ma (Figures 7.1 and 7.2; Fyffe et al., 2008). These intrusions have geochemical signatures consistent with volcanic arc and, less commonly, collisional tectonic settings (W. Zhang, pers. comm., 2011). Younger intrusions of this group have a spatial, and an interpreted genetic, relationship to base and lithophile element mineralization, including tungsten deposits (Figure 7.2). The most significant deposits in this group include Sisson (Re-Os dates of ~377 Ma on molybdenite; Zhang unpublished), Burnt Hill (Ar-Ar date on muscovite alteration of ~383 Ma; Taylor et al., 1987; MacLellan and Taylor, 1989), Mount Douglas (Ar-Ar date on alteration of ~362 Ma; McLeod, 1991); Lake George (alteration not dated; Seal et al., 1987, 1988) and Mount Pleasant (associated rock types between 360 and 365 Ma; Kooiman et al., 1986; Hunt and Roddick, 1990; Tucker et al., 1998).

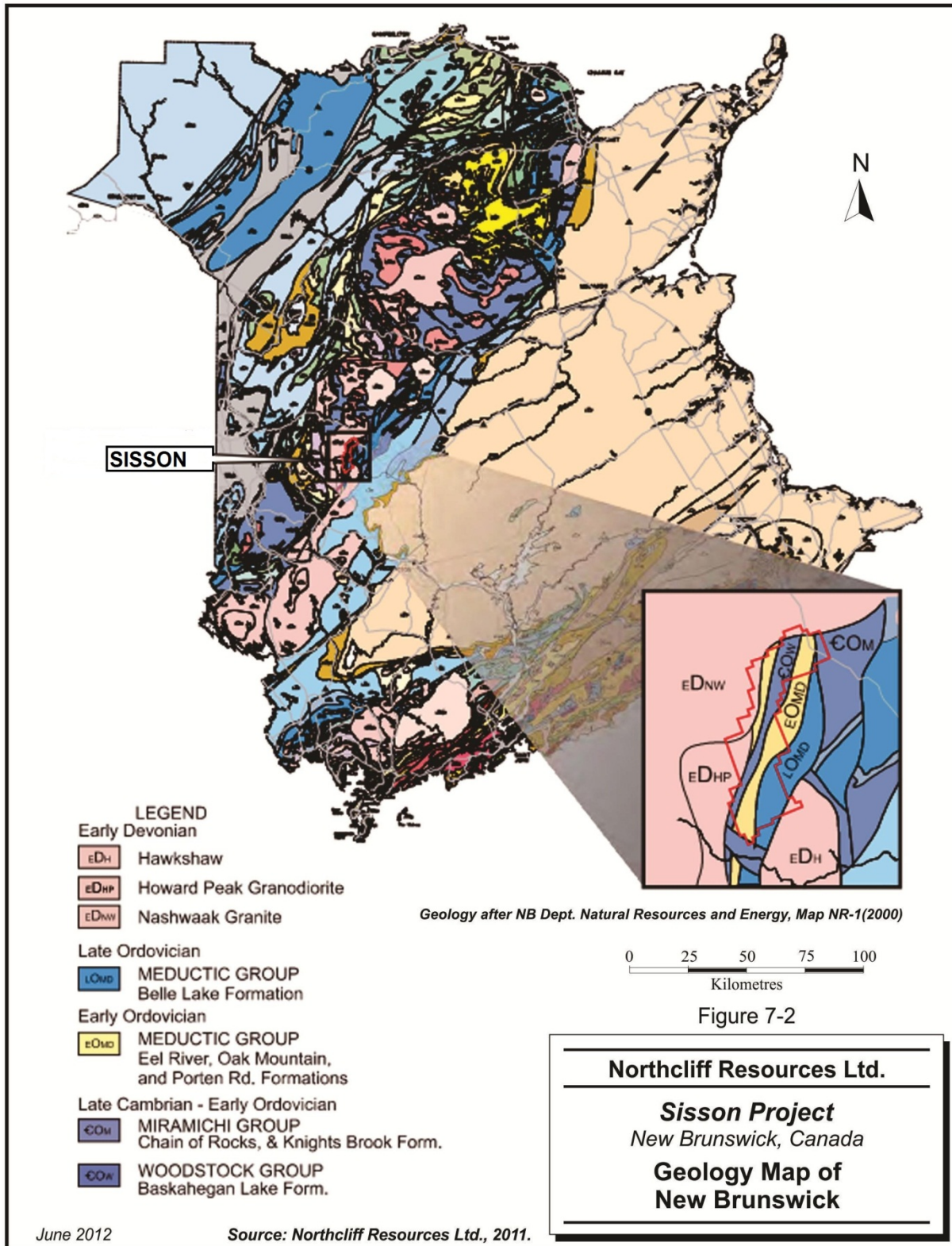
Regional deformation during the Taconic and Acadian orogenies includes folding, thrust faults and normal, reverse and strike-slip faults. Regional metamorphic facies are typically greenschist or lower. Wide contact metamorphic aureoles surround many of the Late Silurian to Late Devonian intrusions.

Youngest rocks in the region form the nearly flat lying, Late Devonian to Carboniferous strata of the “Carboniferous platform”, which extends across most of central and eastern New Brunswick (Figure 7.1).





**Figure 7.1: Regional Geology of West Central and Southwestern New Brunswick**



**Figure 7.2: Geology Map of New Brunswick**

## 7.2 Local and Property Geology

The Sisson deposit is centered on a north-trending contact between Acadian intrusions to the west and older metavolcanic and metasedimentary rocks to the east (Figure 7.3, the “meta” prefix is hereafter omitted for simplicity). Mineralization occurs in four contiguous zones (Figure 7.3). Zones I and II are narrow, structurally controlled zones that extend north from Zone III, which hosts the bulk of the deposit. The Ellipse Zone extends northwest from the southwest corner of Zone III.

There is very little outcrop in the area of the Sisson deposit and the geological interpretation is based primarily on results of drilling, exploration pits, and trenches and regional interpolation. From west to east (Figure 7.3), rock types progress through the following units, which use the formational assignments of Fyffe et al. (2008).

- *Nashwaak Granite*: A massive, probably multiphase, equigranular biotite granite batholith of Acadian age. The batholith is poorly dated with ages of  $422 \pm 4$  Ma (Rb-Sr) and  $386 \pm 5$  Ma (K-Ar) from Whalen and Theriault (1990) and an Ar-Ar date of  $\sim 379$  Ma by Taylor et al. (1987).
- *Howard Peak Granodiorite – granodiorite phase*: An undated, equigranular biotite granodiorite that grades into quartz diorite to the east and is intruded by Nashwaak Granite to the west.
- *Howard Peak Granodiorite – quartz diorite phase*: Hosts mineralization in the western part of the Ellipse Zone at Sisson. It is an undated, medium grained, subporphyritic hornblende quartz diorite.
- *Howard Peak Granodiorite – gabbro phase*: Hosts mineralization in the eastern part of the Ellipse Zone and the western part of Zone III at Sisson. It is an undated, medium grained, weakly porphyritic, pyroxene bearing hornblende gabbro. Its eastern contact with the Turnbull Mountain Formation is a vertical fault.
- *Turnbull Mountain Formation of the Ordovician Tetagouche Group*: This unit comprises bimodal tuffaceous volcanoclastic rocks and biotite wackes. It is the main host rock to mineralization in Zone III at Sisson and is further described in this section.
- *Cambrian to Early Ordovician Miramichi Group*: Dominated by siliceous wackes interbedded with siltstones and quartzites, with minor interbeds of intermediate volcanoclastic rocks. This rock sequence may host low grade mineralization on the eastern margin of the Sisson deposit.
- *Hayden Lake Formation of the Ordovician Tetagouche Group*: This unit contains black shales, flow banded felsic volcanic rocks, and fragmental mafic volcanic rocks which unconformably overlie the Miramichi Group. It is located east of the Sisson deposit.
- *Push and Be Damned Formation of the Ordovician Tetagouche Group*: These are clastic sedimentary rocks located east of the Sisson deposit.

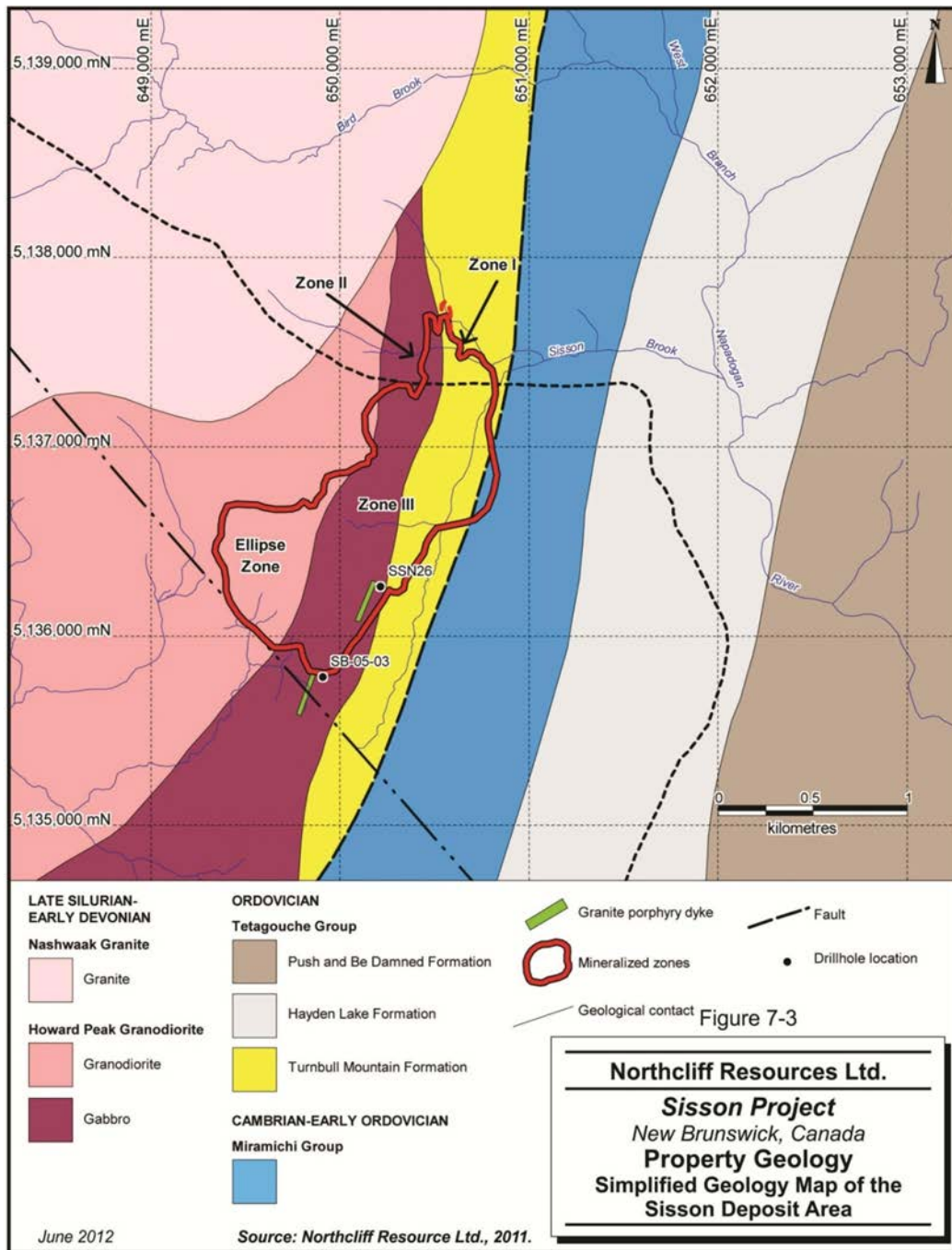
Stratified rocks within and near the Sisson deposit consistently strike north-northeast and dip steeply to the east.

Mineralization in the Sisson deposit is hosted by:

- The quartz diorite and gabbro phases of the Howard Peak Granodiorite
- Felsic, mafic and mafic crystal tuffs in the western part of the Turnbull Mountain Formation
- Biotite wacke with minor interbeds of tuff in the eastern part of the Turnbull Mountain Formation
- Volumetrically minor granite dykes and very rare mafic dykes (Figure 7.3).

Low grade mineralization on the eastern edge of the deposit is hosted by more siliceous biotite-sericite wackes that may be part of the Miramichi Group.

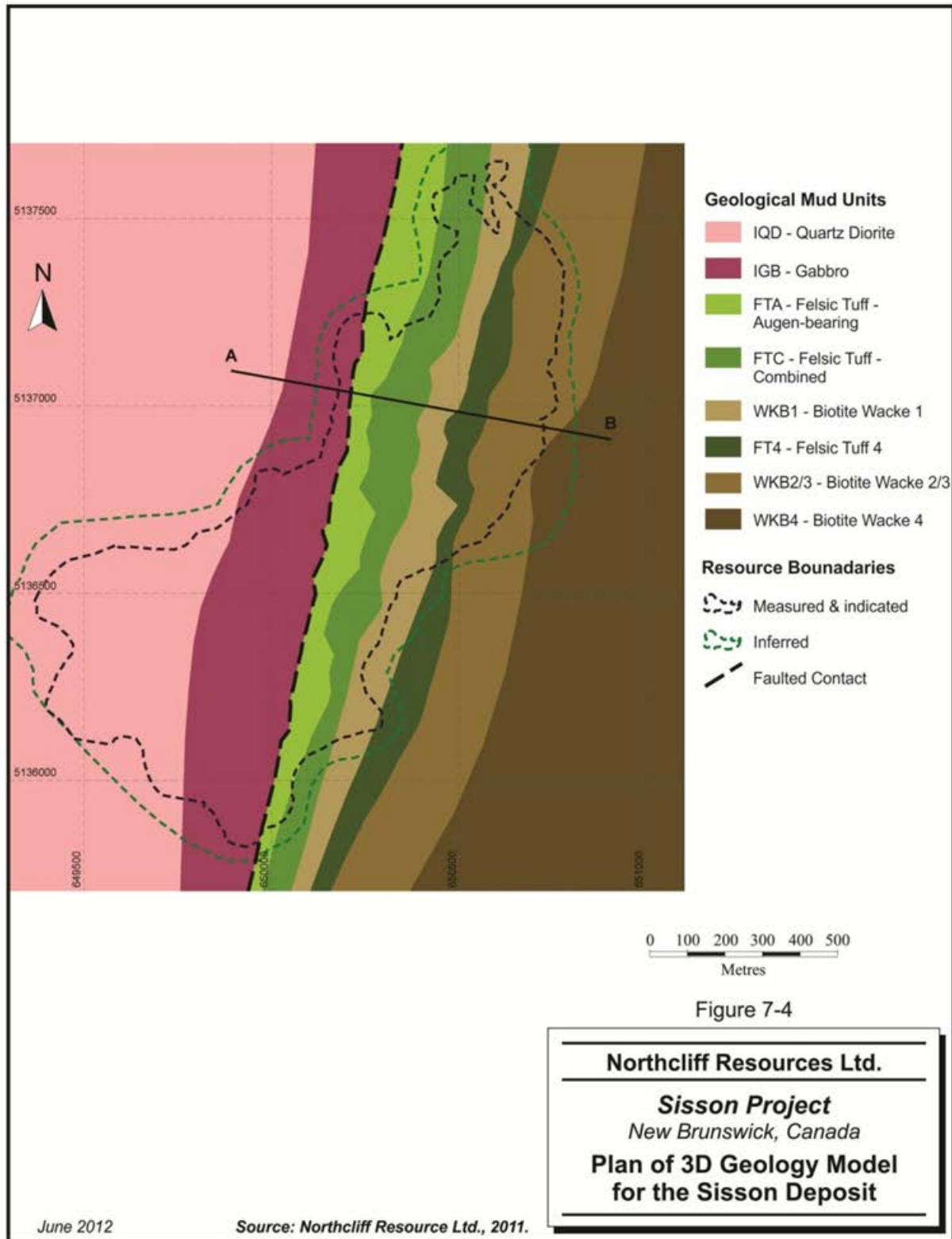




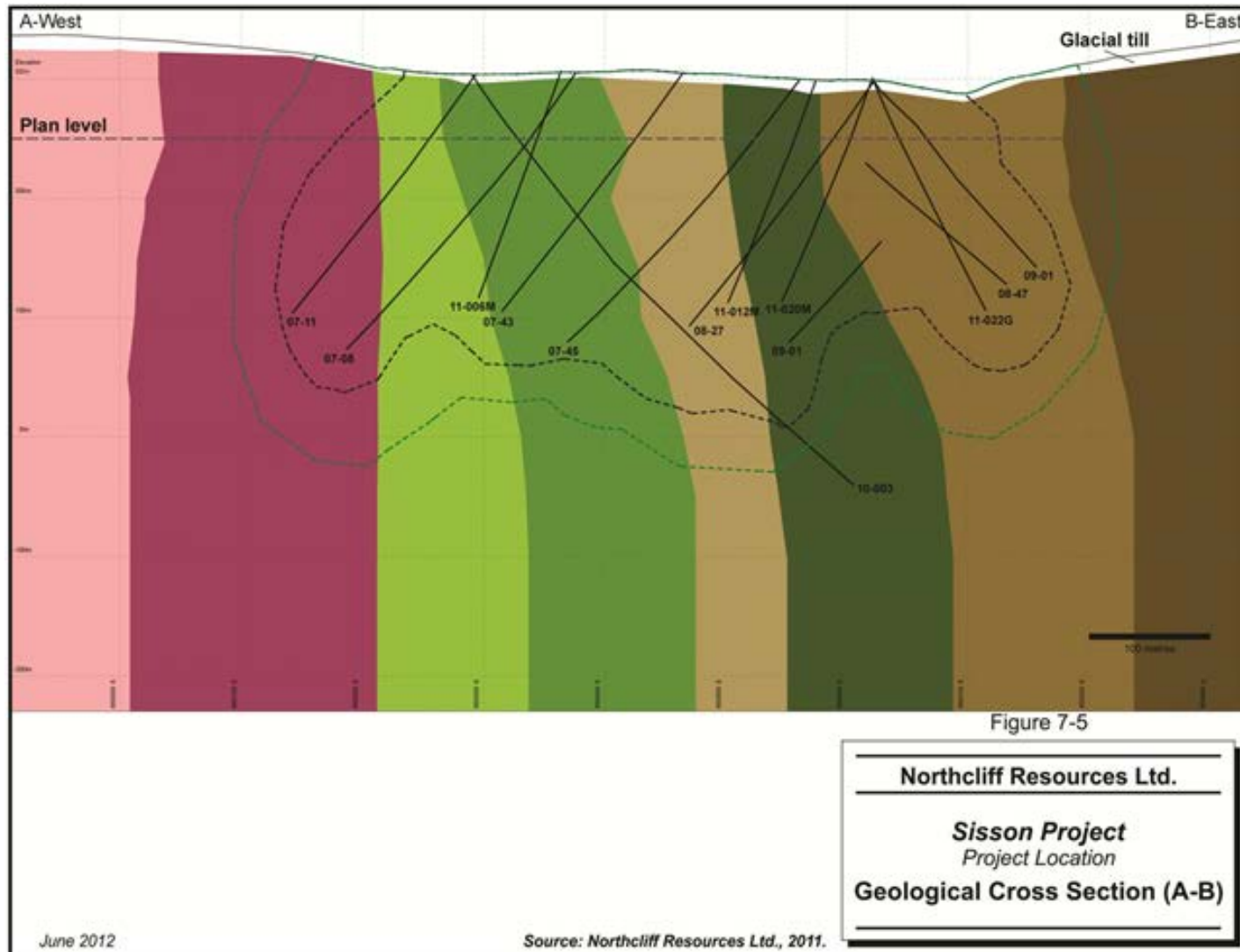
**Figure 7.3: Property Simplified Geology Map of the Sisson Deposit Area**

### 7.2.1 Rock Types Within and Proximal to the Sisson Deposit

Rock types within the Sisson deposit are described in this section in the context of the 3D geological model which has been constructed from drilling information. The geological model for the Sisson deposit comprises eight units, which are summarized in Table 7.1 and shown in plan and section on Figure 7.4 and Figure 7.5. The gabbro and quartz diorite intrusive units and the biotite and biotite-sericite wacke units form comparatively thick, internally coherent units that can be traced laterally across the deposit. In contrast, felsic, mafic, and mafic crystal tuffs in the western part of the deposit comprise narrow, complexly interlayered beds, which may manifest significant lateral facies changes and whose lateral continuity has been further disrupted by deformation. In this light, individual rock types typically cannot be correlated confidently between sections and these sequences therefore have been subdivided into model units which contain distinct combinations of rock types that can be traced across the deposit.



**Figure 7.4: Plan of 3D Geology Model for the Sisson Deposit**

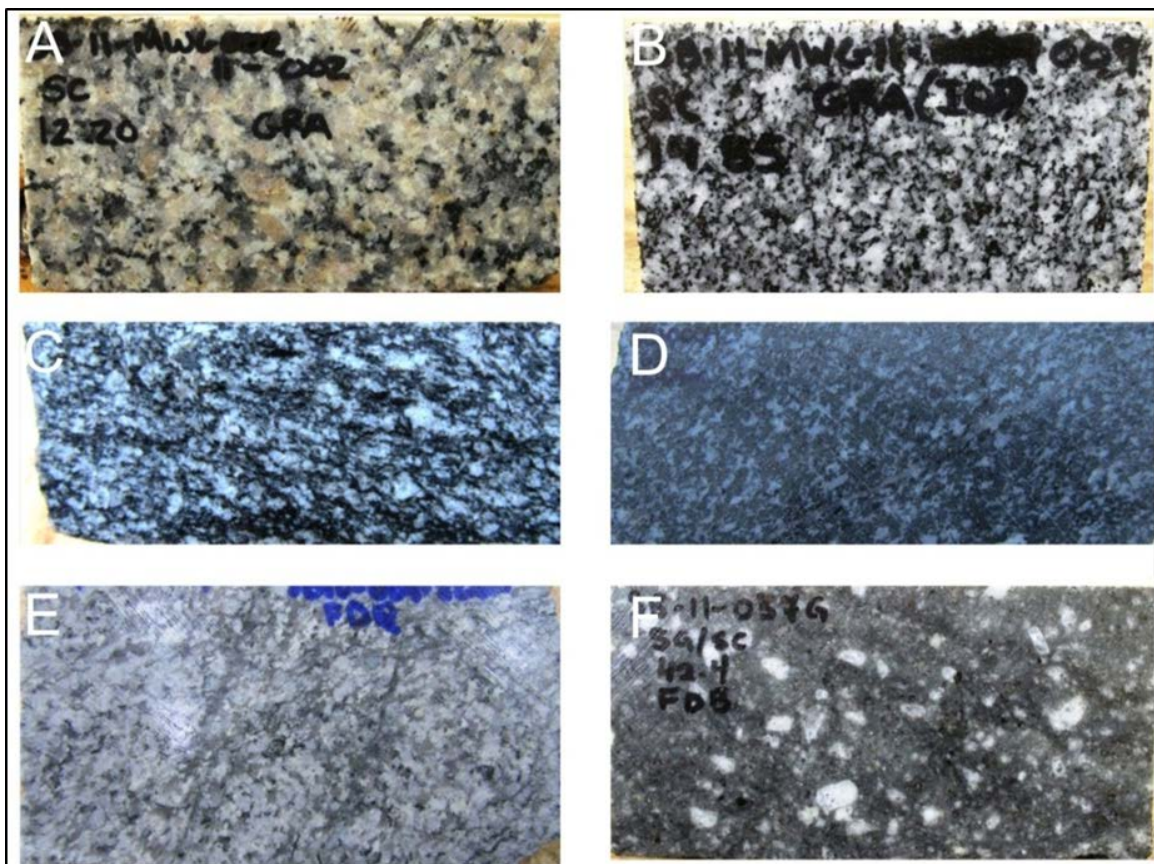


**Figure 7.5: Geological Cross Section A-B**



Quartz Diorite and Gabbro (Model Units IQD and IGB)

The quartz diorite phase of the Howard Peak Granodiorite (Figure 7.6C) is an important host to mineralization in the western part of the Ellipse Zone, whereas the gabbro (Figure 7.6D) hosts the western part of Zone III and the eastern part of the Ellipse Zone. The quartz diorite is dark grey to mottled, subporphyritic, and medium grained. It is dominated by subhedral plagioclase and about 25% to 35% subhedral hornblende, accompanied by up to 5% small, anhedral grains of interstitial quartz, minor K-feldspar, 3% hematized magnetite, and accessory titanite, apatite, and zircon. The gabbro is very similar but can be distinguished from quartz diorite by its higher concentration of hornblende, rare accessory pyroxene, and a near absence of quartz. Both intrusions can be well foliated and marked by segregations of metamorphic biotite. The quartz diorite grades to the biotite granodiorite phase of the Howard Peak intrusion (Figure 7.6B) to the east, and both are intruded by the Nashwaak Granite (Figure 7.6A) Quartz diorite and gabbro are intermingled near their contact. However, possible chilled contacts suggest that the quartz diorite intrudes the gabbro (Oliver, 2010). The contact between gabbro and volcanic rocks of the Turnbull Mountain Formation to the east is faulted and sheared and was a major locus for hydrothermal activity and tungsten mineralization. Dykes of gabbro have not been conclusively identified within the Turnbull Mountain Formation.

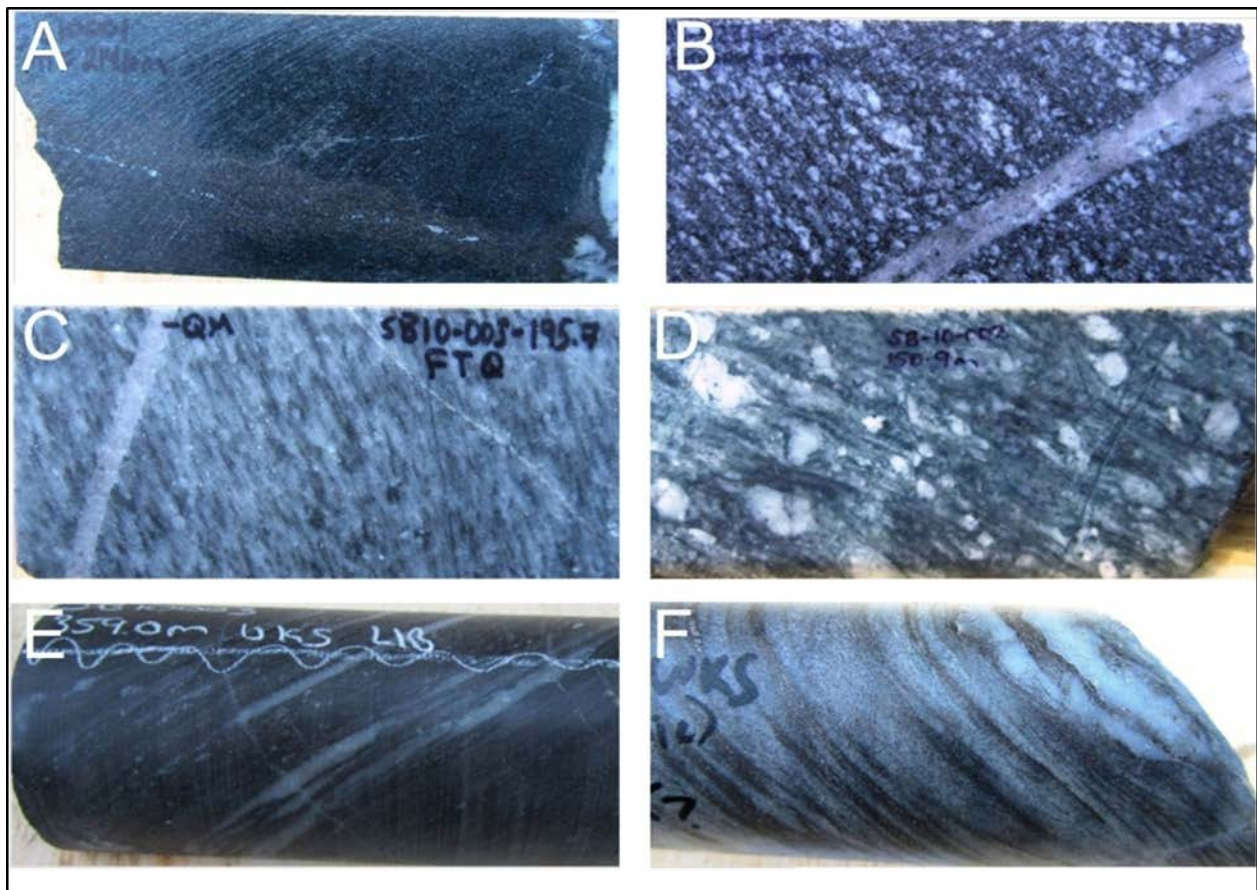


Photographs of igneous rock types within and proximal to the Sisson deposit. A: Nashwaak Granite; B: Howard Peak intrusion, biotite granodiorite phase; C: Howard Peak intrusion, quartz diorite phase; D: Howard Peak intrusion, gabbro phase; E: granite dyke; F: biotite granite porphyry dyke.

**Figure 7.6: Photographs of Igneous Rock Types**

Mafic Tuff

Mafic tuff (Figure 7.7A) is interbedded with felsic (Figure 7.7C) and mafic crystal tuff (Figure 7.7B) throughout the volcanic sequence, and minor interbeds are locally present in biotite wacke intervals in the central part of the deposit. It is present primarily in geology model units FTA, FTC, WKB1, and FT4. Drill intersections of individual mafic tuffs are rarely more than one meter to two meters in length and range down to sub-centimeters scale; as such, individual beds of mafic tuff cannot be confidently correlated between drill holes or cross sections. Mafic tuff is dark grey to dark green and is very fine grained to aphanitic. It is dominated by amphibole and plagioclase, along with accessory titanite. The rock is typically massive but locally has a weak laminar fabric.



Photographs of volcanic and sedimentary rock types within and proximal to the Sisson deposit. A: mafic tuff; B: mafic crystal tuff; C: felsic tuff; D: felsic tuff with augen; E: biotite wacke; F: wacke with biotite- and sericite-rich interbeds.

**Figure 7.7: Photographs of Volcanic and Sedimentary Rock Types**

Mafic Crystal Tuff

Mafic crystal tuff (Figure 7.7B) is a texturally variable rock type that forms the largest part of the volcanic sequence within the deposit. It occurs primarily in geology model units FTC and FT4 and, to a lesser extent, in units FTA and WKB1. It is black, dark brown, or mottled in color and is typically interbedded at all scales with mafic and felsic tuffs. Narrow intersections of this rock type are locally present in biotite wacke

intervals. The rock contains plagioclase, quartz, amphibole, minor alkali feldspar, and accessory magnetite and/or ilmenite, zircon and apatite; metamorphic biotite is abundant and replaces hornblende. The abundance of quartz indicates an intermediate composition. This rock type ranges from massive to strongly foliated.

#### Felsic Tuff and Augen-Bearing Felsic Tuff

Felsic tuff (Figure 7.7C) occurs throughout the volcanic portions of the stratigraphy and is an important host to mineralization. It occurs primarily in geology model units FTA, FTC, and FT4 and, to a lesser extent, in unit WKB1. It is typically interbedded with mafic tuff and mafic crystal tuff but internally homogenous beds locally range up to 20 m or more in true thickness. Narrow interbeds are locally present within wacke intervals. The rock is white to light grey and is dominated by quartz and feldspars. It ranges from massive and medium grained to strongly foliated and fine grained. Augen bearing felsic tuff (Figure 7.7D) occurs only in the western part of the volcanic stratigraphy. Geology model unit FTA is defined by the western and eastern extents of augen bearing felsic tuff. It is distinguished from felsic tuff by up to 15% feldspar crystals or crystal aggregates up to two centimeters in size. These features probably represent volcanoclastic lapilli, but the term “augen” has been used historically and is retained here.

#### Biotite Wacke (Model Units WKB1, WKB2/3 and WKB4)

The eastern part of the Sisson deposit is hosted predominantly by three geological model units defined by the extent of biotite wacke (Figure 7.7E). Intersections of biotite wacke range from nearly black, to dark brown to medium grey-brown. They are locally medium grained but are mostly fine grained and have a laminar fabric. All are characterized by high concentrations of metamorphic biotite, abundant feldspar, and lesser granular quartz; metamorphic sericite variably replaces feldspar. Quartz segregations occur parallel to the laminar fabric. The westernmost zone of biotite wacke, defined as geology model unit WKB1, contains andalusite formed during contact metamorphism. Central model unit WKB2/3 is similar to WKB1 but lacks andalusite. The easternmost intersections, which form geology model unit WKB4, may be part of the Miramichi Group and contain staurolite, have a higher concentration of sericite (Figure 7.7F), and include narrow interbeds of siliceous siltstone and quartzite. Narrow interbeds of mafic, felsic, and mafic crystal tuff occur in the western parts of the biotite wacke section.

#### Minor Intrusions

The Sisson deposit is cut by several types of volumetrically minor dykes.

- *Granite Dykes* - Light grey, fine grained to pegmatoidal, locally granophyric granite dykes (see Figure 7.6E) are very common within the gabbro and quartz diorite intrusions and, to a much lesser extent, in volcanic rocks immediately east of the gabbro. True width for most dykes is less than three meters. All of these dykes contain mineralization, veins, and/or alteration. Paragenetic relationships of hydrothermal features suggest that some of these dykes may be syn-hydrothermal; this is supported by overlap in their preliminary U-Pb ages from some dykes and Re-Os dates on molybdenite (Zhang, pers. comm., 2011).
- *Biotite Granite Porphyry Dykes* - These intrusions (Figure 7.6F) have been intersected in several drill holes, mostly from the south part of Zone III. They are grey, massive and unfoliated, fresh and commonly truncate mineralized veins. They are distinguished by phenocrysts of plagioclase, alkali



feldspar, quartz, and 8% biotite in a fine grained matrix. Fyffe et al. (2008) obtained a U-Pb date on zircon of  $364.5 \pm 1.3$  Ma, which is significantly younger than the  $\sim 377$  Ma dates on molybdenite (Zhang, unpublished data).

- *Mafic Dykes* - These are rare at Sisson and occur mostly in gabbro in the southwest part of the deposit. They are altered and mineralized but are volumetrically insignificant.

Table 7.1 Rock Types Used to Model the Sisson Deposit			
Model Unit	Unit Name	Mineralization	Constituent Rock Types
IQD	Quartz diorite	Western part of the Ellipse Zone	Quartz diorite phase, Howard Peak Granodiorite; internally homogenous
IGB	Gabbro	Eastern part of the Ellipse Zone, Zone II and western part of Zone III	Gabbro phase, Howard Peak Granodiorite; internally homogeneous
FTA	Felsic tuff – augen-bearing	Western part of Zone III	Thinly interbedded felsic > mafic > mafic crystal tuff; defined by eastern limit of augen bearing felsic tuff
FTC	Felsic tuff combined	Central part of Zone III	Moderately to thinly interbedded mafic crystal > felsic > mafic tuff
WKB1	Biotite wacke 1	Zone II and central part of Zone III	Medium grained biotite wacke, commonly with contact metamorphic andalusite; contains minor mafic and felsic tuff near eastern and western contacts
FT4	Felsic tuff 4	Central and eastern parts of Zone III	Interbedded mafic crystal > mafic > felsic tuff; increasing biotite wacke interbeds toward eastern and western margins
WKB2/3	Biotite wacke 2/3	Eastern part of Zone III	Fine grained biotite wacke with minor interbeds of mafic tuff; lacks andalusite
WKB4	Biotite wacke 4	Low-grade mineralization near the eastern margin of Zone III	Fine grained biotite wacke with interbeds of siliceous siltstone and quartzite; may be the western margin of the Miramichi Group; locally contains staurolite

### 7.2.2 Structure

The Miramichi Group is believed to occupy the core of a north-northeast trending, south plunging anticline (Lutes, 1981) and to be bounded to the east and west by younger volcanic and sedimentary rocks of the Ordovician Tetagouche Group (Figure 7.3). The interpretation and/or specific characteristics of a major anticline cored by Miramichi Group, however, continue to be debated. Fyffe et al. (2008) note that rocks of the Tetagouche Group do not correlate on the east and west sides of the Miramichi Group, and Mann (1980, 1981) suggests that the volcanic rocks to the west of the Miramichi Group may young to the east, which precludes an antiformal structure. Small isoclinal folds, however, have been observed in drill core, and measurements in oriented core confirm that they plunge steeply to the south (Duncan, 2011), which is consistent with the interpretation of Lutes (1981). The western contact of the Miramichi Group against the Turnbull Mountain Formation may be a fault (Figure 7.3; Fyffe et al., 2008), which may further complicate

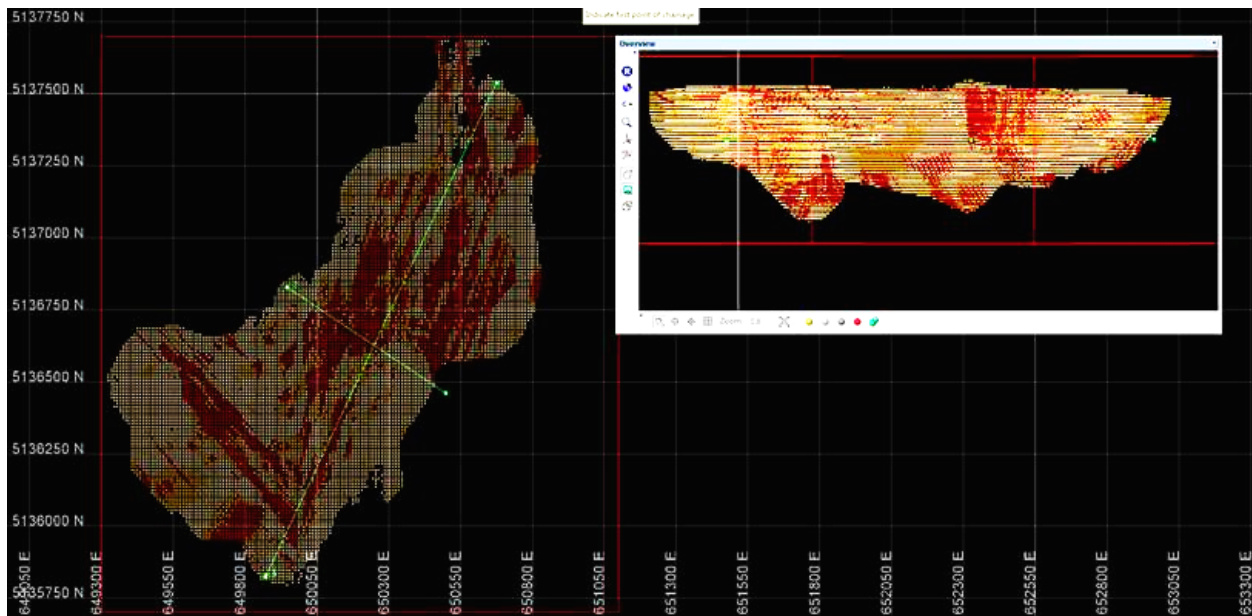
local structural interpretation, although a definite structure has not been identified during the work described in this report. To the east, the Miramichi Group is overlain unconformably by the Hayden Lake Formation of the Tetagouche Group, which yields to the younger Push and Be Damned Formation of the Tetagouche Group. Regardless of the uncertainties in structural interpretation and regional correlation of rock types, these effects occur at scales larger than the zone of mineralization and do not impact on exploration and development considerations.

The north trending contact between the gabbro and the Turnbull Mountain Formation is a vertical fault zone, which ranges from a few to about 20 m in width. It is marked by strong fracturing, local brecciation, and minor gouge seams. The strongest alteration and tungsten mineralization and most abundant quartz-sulphide veins occur along and adjacent to this important fault, although it did not apparently influence later molybdenum mineralization. Kinematic indicators are compatible with sinistral strike-slip movement, but absolute displacement is not constrained. Large quartz-sulphide veins in a trench in Zone I, described in a later section, also formed along north-trending, sinistral faults. Brittle faults with gouge are largely absent from the rest of the deposit but a few rubble zones are present.

Most rock types exhibit a weak to strong foliation, defined by biotite and/or muscovite, which is parallel to stratigraphic orientation. A northwest trending, steeply dipping cleavage is also commonly present (Fyffe and Thorne, 2010).

### 7.3 Mineralization

Mineralization in the Sisson deposit is related to several types of alteration and veins, each of which has a distinct relationship to metal introduction (Figure 7.8). The mineralization has an approximate strike length of 1,900 m, average width of 650 m and average depth of 350 m.



**Figure 7.8 Mineralization Dimensions**

Hydrothermal features of the Sisson deposit are described below from early to late effects, to the extent that their timing has been constrained by paragenetic relationships. To summarize, early barren to weakly mineralized alteration was followed sequentially by an early stage of tungsten mineralization, an intermediate stage of molybdenum  $\pm$  tungsten mineralization, and a late stage of polymetallic mineralization in which tungsten is accompanied by numerous base and trace elements. Most alteration and nearly all mineralization can be directly related to specific types of veins. The overall vein density at Sisson is only about 3% and pervasive alteration zones are spatially limited, which indicates that the hydrothermal system at Sisson had a low fluid flux.

#### Early Sodic-Calcic Alteration and Amphibole Veins

The earliest hydrothermal feature at Sisson comprises amphibole veins and associated albite-actinolite (sodic-calcic) alteration (Figure 7.9A). These features occur almost exclusively in gabbro and, to a lesser extent, in quartz diorite. They are most abundant in the west-central part of Zone III and extend into the Ellipse Zone. The AA veins and sodic-calcic alteration contain a few percent pyrrhotite-pyrite but completely lack molybdenum and contain only trace scheelite. The veins are mostly between two and 20 mm in width, have diffuse contacts with host rocks and contain mostly actinolite, lesser albite, minor calcite, and trace to minor pyrrhotite and/or pyrite. Alteration envelopes are mineralogically similar and can be up to several times the width of the associated vein. Where vein density is high, the alteration envelopes coalesce to pervasive alteration.

#### Early Biotite and Biotite-Sulphide Alteration

This alteration mostly spans the contact between gabbro and volcanic rocks on the west side of Zone III. This alteration is commonly intense and texture-destructive (Figure 7.9B). Similar alteration is erratically distributed through gabbro west of the contact, but is rare in quartz diorite. Sulphide concentration ranges from trace to 10% and manifests cubiform pyrite locally accompanied by pyrrhotite. Biotite-sulphide alteration typically contains neither scheelite nor molybdenite.

#### Calc-Silicate Alteration

Drilling has intersected mineralized calc-silicate alteration (Figure 7.9C) over lengths between 0.5 m and three meters. This alteration occurs both in gabbro and in volcanic rocks but mostly at depths greater than about 300 m. The timing of this alteration relative to other hydrothermal effects has not been constrained. The alteration is massive and pervasive and associated veins have not been observed. The mineralogy comprises various combinations of red-brown garnet, green pyroxene, calcite, epidote, and possibly amphibole, and up to a few percent pyrrhotite and lesser pyrite. Disseminated scheelite is ubiquitous but molybdenite has not been observed. Relicts of granite dyke within this alteration suggest that it may be endoskarn.

#### Quartz-Scheelite Veins (Type QW)

Quartz-scheelite (QW) veins represent the earliest stage of significant tungsten mineralization at Sisson. These veins are typically less than five millimeters in width, have sharp to locally diffuse contacts with host rocks, and range from planar to strongly curvilinear (Figure 7.9D). They contain clear to milky quartz, scheelite, low concentrations of pyrite and/or pyrrhotite, and rare molybdenite. Alteration envelopes are

mostly less than one centimeter in width and comprise biotite in mafic host rocks (Figure 7.9D) and, more rarely, sericite in felsic host rocks. The alteration envelopes commonly contain disseminated scheelite, particularly when hosted by gabbro or quartz diorite. The density of these veins diminishes markedly below about 300 m depth in the center of Zone III, and at shallower and deeper levels in the north and south parts of the deposit, respectively.

#### Quartz-Molybdenite (Type QM) and Quartz-Feldspar (Type QF) Veins

Quartz-molybdenite (QM, Figure 7.9E) veins consistently cut QW veins. They represent the main stage of molybdenum mineralization and, except in the north part of Zone III, also contain high concentrations of scheelite. They are most abundant in the center of Zone III, of intermediate abundance in the south part of Zone III and the Ellipse Zone, and are rare to absent in Zones I and II, the north part of Zone III, and in gabbro from the west-central part of Zone III. Type QM veins extend to much greater depth than QW veins and, for example, remain abundant at 525 m true depth in DDH-SB10-004.

The QM veins are mostly one centimeter to five centimeters in width, but veins greater than 20 cm wide are common. They are planar to curvilinear and have sharp contacts with wall rocks (Figure 7.9E). The QM veins are dominated by clear, locally epitaxial quartz. Molybdenite can form up to 10% of these veins but has a clotty distribution along their length. Most QM veins contain less than 3% pyrite, rarely accompanied by pyrrhotite. They contain low to locally high concentrations of scheelite (Figure 7.9F), although in some cases tungsten mineralization is related to younger type SX veins which exploited fractured QM veins. Sericite alteration envelopes less than one centimeter in width are only locally present and rare biotite envelopes occur in mafic host rocks.

Type quartz-feldspar (QF) veins are very similar to QM veins but occur mostly at depths below about 250 m. They are distinguished by selvages of white K-feldspar, contain a dark green mineral that may be grossular garnet, have a significantly lower concentration of molybdenite and scheelite than normal QM veins, and are mostly less than three centimeters in width. Vein types QM and QF are generally contemporaneous.

#### Quartz Shear (Type QS) and Sulphide-Rich (Type SX) Veins

Late quartz-shear (QS, Figure 7.9G) and sulphide-rich (SX, Figure 7.9H) veins are closely related and represent the youngest stage of significant mineralization at Sisson. The QS veins are the main style of mineralization in Zones I and II and in the strongly mineralized contact between gabbro and volcanic rocks in Zone III. They are common throughout the Turnbull Mountain Formation in Zone III but are less common in gabbro in Zone III and in the Ellipse Zone. The QS veins and associated alteration commonly attain grades between 0.25%  $WO_3$  and 1.00%  $WO_3$ , although they constitute only a very small portion of the overall deposit. Drill hole intersections of QS veins are mostly less than two meters in length, although intersections up to ten meters have been observed and swarms of smaller QS veins in rare cases span zones 20 m or more in apparent width. They exhibit crack-seal textures and formed by repeated reopening. As such, their formation was plausibly protracted and may have spanned much of the life of the Sisson hydrothermal system.

The QS veins range from planar to curvilinear and contacts with host rock are typically sharp (Figure 7.9G). Lensoidal forms occur in outcrop in Trench 1. The QS veins contain mostly white quartz cut by

myriad brittle fractures that reflect structural disruption during and after vein formation. Sulphide concentration can attain 15% and comprises mostly pyrrhotite and lesser pyrite, 1% to 3% chalcopyrite, and highly variable concentrations of arsenopyrite, galena, sphalerite, and bismuth minerals. These veins host almost all of the wolframite in the deposit (Figure 7.9G), which occurs as black, prismatic crystals that are variably to completely replaced by scheelite. Molybdenite has not been observed.

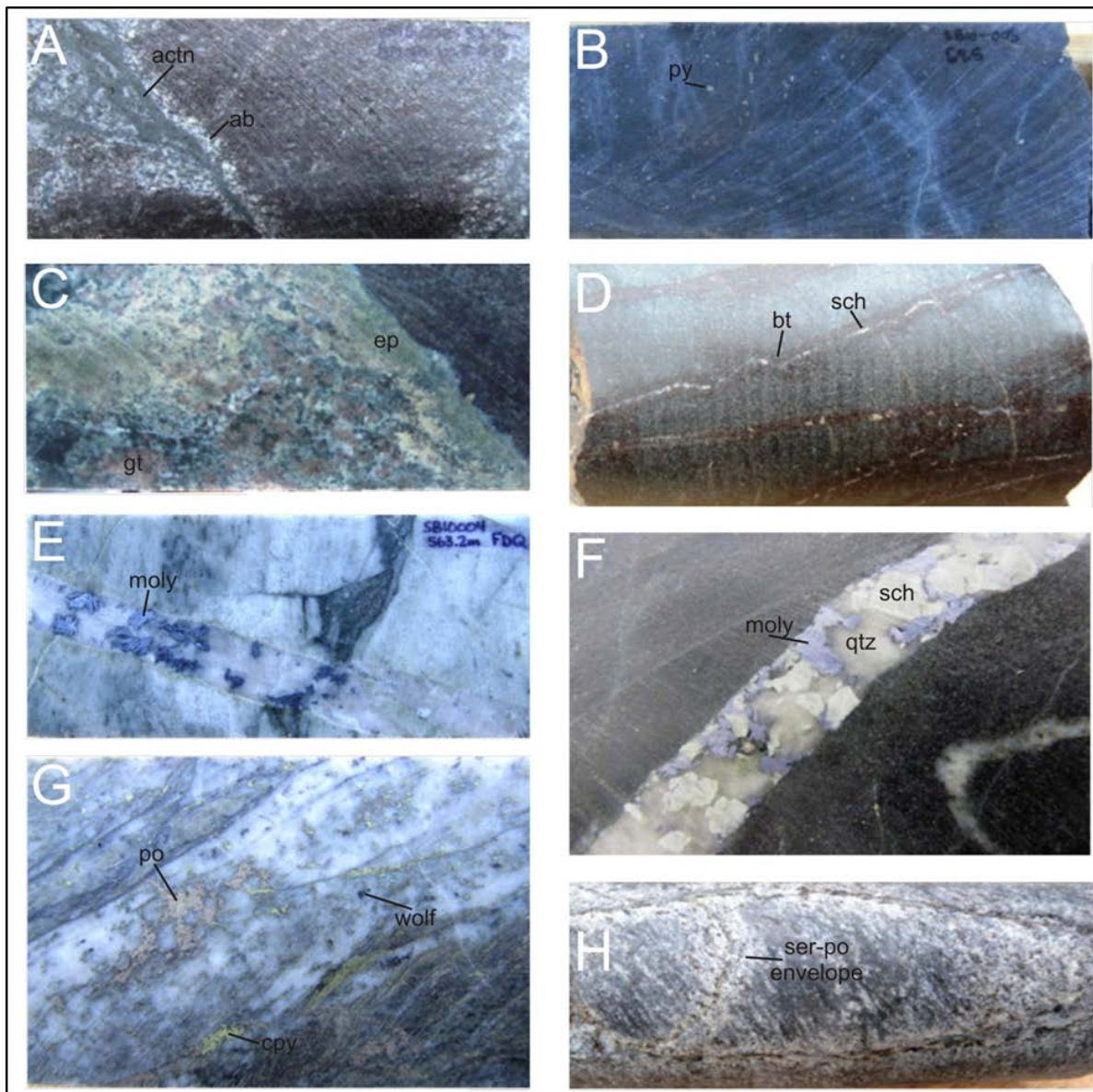
The QS veins are enclosed by envelopes of intense, texture-destructive sericite-quartz-sulphide alteration. The envelopes coalesce where multiple QS veins are closely spaced. The envelopes are dominated by sericite and quartz and contain up to 15% pyrrhotite, which is locally intergrown with lesser pyrite. The metal assemblage in veins and envelopes is similar. A few larger QS veins, mostly from the far eastern side of the deposit, have an outer, poorly mineralized carbonate envelope. These envelopes manifest swarms of narrow fractures that are filled with ankerite or ferroan dolomite, which cut and replace the proximal sericite-sulphide envelopes but extend up to ten meters from the associated veins.

The SX veins (Figure 7.9H) are widely distributed through the Turnbull Mountain Formation but are rare in the Ellipse Zone and gabbro in Zone III. Relationships exposed in Trench I suggests that SX veins may occur as sheeted swarms of planar fractures oriented at conjugate angles to and surrounding QS veins. Individual SX veins are fracture fills that rarely exceed three millimeters in width and which are enclosed by intense sericite-quartz-sulphide alteration envelopes (Figure 7.9H) identical to those related to QS veins. The SX veins are mineralogically similar to QS veins.

#### Carbonate Veins (Type QC)

Late carbonate veins are planar, up to one centimeter in width, and lack alteration envelopes. They contain calcite or dolomite, quartz, pyrite and minor green to purple fluorite. In a few rare cases, they contain minor scheelite, but molybdenite has not been observed. This same assemblage locally forms open-space infill of small voids in late SX and QS veins. These veins are widespread but volumetrically and economically insignificant.





A: actinolite veins with albite envelopes in gabbro. B: Pervasive biotite-pyrite alteration in mafic tuff. C: Calc-silicate alteration in either granite dyke and/or gabbro. D: Early quartz-scheelite (QW) veins with biotite alteration envelopes in mafic tuff. E: Intermediate stage quartz-molybdenite (QM) veins in felsic tuff. F: High-grade type QM or QW vein in mafic tuff. G: Late quartz-shear (QS) vein. H: Late sulphide-rich (SX) veins with prominent sericite-pyrrhotite alteration envelopes in mafic crystal tuff.

**Figure 7.9: Examples of Alteration and Vein Types in the Sisson Deposit**

### 7.3.1 Styles of Mineralization

Mineralization at Sisson occurs almost exclusively in quartz veins, fractures, and their alteration envelopes (Figure 7.10). Tungsten and molybdenum are the metals of principal economic interest, whereas several other metals, including copper, zinc, lead, arsenic, and bismuth, occur in geochemically anomalous but subeconomic concentrations.

Mineralization occurs in contiguous Zones I, II, and III and in the Ellipse Zone (Figure 7.3). Zones I and II trend approximately north from the northern end of Zone III. Zone I is hosted by the Turnbull Mountain Formation, whereas Zone II occurs in gabbro. Zones I and II are structurally controlled, up to tens of meters in width, and extend several hundred meters along strike. They are dominated by QS and SX veins and contain tungsten, copper, and associated trace elements, but lack significant molybdenum. Zone III contains the bulk of the tungsten and molybdenum resource at Sisson. It is an ovoid zone that obliquely spans the contact between gabbro and the Turnbull Mountain Formation and mineralization occurs in gabbro, volcanic, and sedimentary rocks. The highest grades of molybdenum occur in both volcanic and sedimentary rocks in the central part of Zone III, whereas the highest grades of tungsten occur on either side of the contact between gabbro and volcanic rocks in Zone III. The Ellipse Zone and the southern part of Zone III are very similar to each other and contain moderate grades of both tungsten and molybdenum. Host rocks in the southern part of Zone III are gabbro and volcanic rocks, whereas the Ellipse Zone is hosted by gabbro and quartz diorite. Mineralization diminishes very abruptly at the south end of Zone III.

The minerals of economic interest at Sisson are molybdenite, scheelite, and minor wolframite. Scheelite precipitated during formation of QW, QM, QS, and SX veins, whereas most molybdenite is found in QM and, to a lesser extent, QF veins. Overall, the deposit only contains very minor wolframite, which occurs almost exclusively in QS and SX veinlets where it is replaced, in whole or in part, by scheelite (Figure 7.10C). The highest concentration of copper, zinc, lead, arsenic, bismuth, and other trace elements is directly related to late QS and SX veins and their sericite-sulphide envelopes.

Tungsten and molybdenum mineralization precipitated mostly within quartz-sulphide veins (Figure 7.10A) and, in some cases, in their alteration envelopes (Figure 7.10B). Scheelite ranges from very fine (greater than 100  $\mu$  m) to coarse grained (less than one centimeter). Scheelite occurs both as selvages and along the central axes of veins. Fine grained scheelite is also commonly disseminated through biotite alteration envelopes to QW veins (Figure 7.10B), particularly in gabbro and quartz diorite host rocks. Disseminated scheelite is less common in sericite alteration envelopes to QW, QS, and SX veins. Molybdenite occurs almost primarily within QM veins where it commonly forms coarse grained selvages (Figure 7.9E).

Metal zoning has only been established empirically at Sisson. The concentration of scheelite decreases with depth and grades diminish rapidly below about 250 m and 400 m in the north and south parts of Zone III, respectively. Above average grades of tungsten are most consistently present along the contact between gabbro and volcanic rocks and, to a much lesser extent, in narrow, steeply dipping zones on the eastern side of the deposit. The highest concentration of molybdenum occurs in the center of Zone III, where type QM veins are most abundant. Moderate grades of molybdenum occur in the south part of Zone III and in the Ellipse Zone. Zones I and II and the north part of Zone III contain almost no molybdenum. Significant intersections of molybdenum mineralization extend to a much greater depth than tungsten mineralization. The distribution of both tungsten and molybdenum mineralization are consistent with a south plunge to the Sisson deposit. The highest concentrations of copper and associated elements

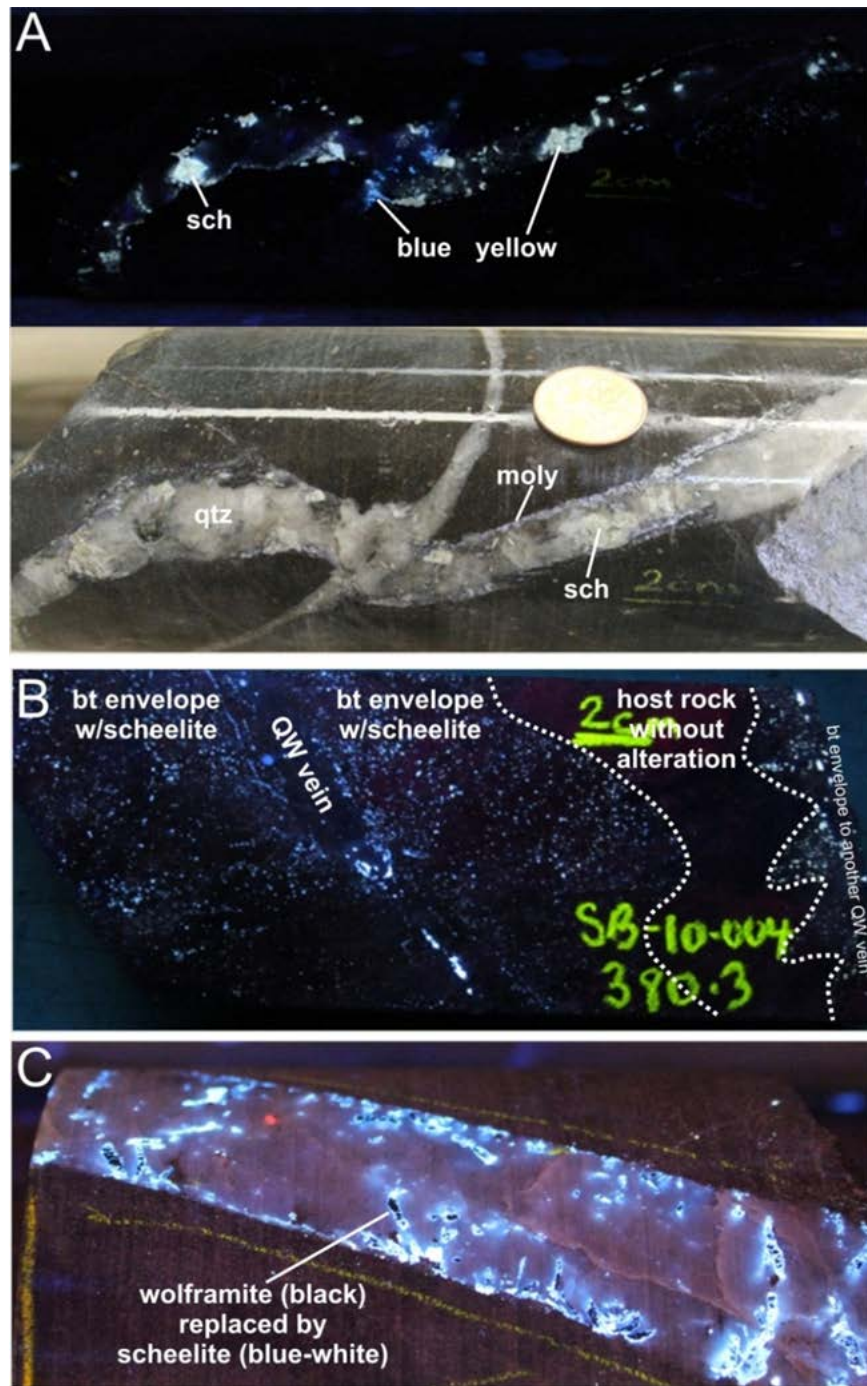
are most consistently present in Zones I and II, whereas elevated concentrations of these elements elsewhere are related to more widely distributed QS and SX veins.

The grade of tungsten correlates only weakly with the composition of host rocks. Nast (1985) and Nast and Williams-Jones (1991) have suggested that the paucity of pervasive alteration and weak development of alteration envelopes to veins is compatible with a low flux of hydrothermal fluids. In such a rock-buffered hydrothermal system, they further suggest that scheelite preferentially precipitated in more mafic, calcium-rich host rocks. Empirically, there is a tendency for higher concentrations of scheelite to occur in mafic tuff, gabbro, and mafic crystal tuff than in adjacent felsic tuffs. The biotite wacke located along the central axis of Zone III (geological model unit WKB1; Figure 7.5) commonly has lower tungsten grade than adjacent volcanic host rocks. Host rock does not exhibit any apparent control on the grade of molybdenum.

Scheelite exhibits both blue and yellow fluorescence (Figure 7.10A) when exposed to shortwave ultraviolet light. Yellow fluorescence can indicate anything from end-member powellite ( $\text{CaMoO}_4$ ) to minor substitution of molybdenum for tungsten in scheelite (referred to as molybdoscheelite). Electron microprobe analyses indicate that yellow fluorescence at Sisson reflects molybdoscheelite which contains a maximum of about two weight-percent molybdenum (Gregory, 2011). These results are consistent with Nast and Williams-Jones (1991) who report a maximum of about four weight-percent molybdenum in scheelite. These results indicate that less than three percent of the total molybdenum inventory at Sisson might be sequestered in molybdoscheelite, using average deposit grades and a concentration of one weight-percent molybdenum in molybdoscheelite.

Oxidation related to surface weathering is mostly weak to moderate and confined to the upper 20 m of the deposit. Oxidation along fractures locally extends to greater depths, particularly along the structurally disrupted contact between gabbro and the Turnbull Mountain Formation. Where oxidation is strongest, primary rock textures remain visible and hypogene pyrite and pyrrhotite are, at most, only partially converted to iron oxides. Scheelite, wolframite, and molybdenite are essentially non-reactive in surface settings and are not significantly affected.





A. Vein-hosted scheelite and molybdenite mineralization. Scheelite includes both blue fluorescent grains of pure scheelite and yellow fluorescent grains of molybdo-scheelite which contain a maximum of about 2 wt% molybdenum. Note the absence of fluorescent grains outside the vein itself. Drill hole SB11-019M. B. A narrow QW vein with blue fluorescent scheelite surrounded by a biotite alteration envelope with abundant, very fine-grained, disseminated scheelite. This effect is most common in gabbro. Drill hole SB10-004 at 380.3 m depth. C. A late QS vein which contains wolframite (dark, elongate grains) partially to completely replaced by fluorescent scheelite. Drill hole SB11-019M.

**Figure 7.10: Examples of Mineralization Styles in the Sisson Deposit**

## 8.0 Deposit Types

This section was extracted in its entirety from the June 29, 2012 Technical Report by RPA.

The Sisson deposit can be defined as an intrusion related, structurally controlled, bulk tonnage tungsten-molybdenum deposit. Deposits of this type have general hydrothermal similarities to porphyry copper deposits. These types of deposit form in convergent margin to collisional tectonic environments and are related to highly evolved granitic melts formed from continental crust.

### 8.1 Provisional Magmatic-Structural-Hydrothermal Model

Tungsten-molybdenum deposits in New Brunswick have been linked genetically and temporally to hydrothermal fluids generated by Late Devonian felsic batholiths and intrusions formed during the Acadian Orogeny (e.g., Fyffe and Thorne, 2010; dating cited on p. 42 and p. 46 in this report). The overlap between U-Pb ages on granite dykes and Re-Os dates on molybdenite support a similar link at Sisson (W. Zhang, pers. comm., 2011). A larger granitic pluton as a source for mineralizing hydrothermal fluids at Sisson has not been intersected by drilling thus far, but the numerous, albeit narrow, granitic dykes allow the presence of such a body to be confidently inferred at depth below the deposit.

Structures focused the ascent from depth of the intrusion derived hydrothermal fluids which formed the Sisson deposit. The physical relationship between QS and SX veins exposed in exploration Trench 1 in Zone I and the concentration of mineralization along the disrupted contact between gabbro and the Turnbull Mountain Formation suggest the possible nature of this control. Trench 1 is cut by several broadly north trending, approximately vertical QS veins that are between 0.3 m and 1.5 m in width. Kinematic indicators support long-lived to episodic, sinistral movement along the structures which host the QS veins. Sheeted arrays of northwest trending, steeply dipping, narrow SX veins, as well as fewer but larger, lensoidal quartz extension veins, form an envelope at approximately conjugate angles to the larger QS veins. Orientations obtained on 1,631 veins of all types in seven oriented geotechnical drill holes (Duncan, 2011) also show that most veins throughout the deposit form a sheeted array with northwest strike and steep southwest dip.

A provisional, highly simplified genetic model for Sisson therefore comprises: (1) emplacement of granitic magma at depth; (2) magmatic fractionation and release of hydrothermal fluids; (3) ascent of these fluids along north trending, sinistral fault zones; and (4) lateral distribution of mineralizing fluids along northwest trending, sheeted fracture arrays.

## 9.0 Exploration

This section was extracted in its entirety from the June 29, 2012 Technical Report by RPA and was prepared by Dr. James Lang, P. Geo., Senior Vice President Geology, HDI.

Early exploration programs, conducted prior to Northcliff's acquisition of the property, are discussed in Section 6.0, History, of this report.

### 9.1 Northcliff Exploration History

#### 2011 Test Pit Program

The following is largely an excerpt from a Northcliff Assessment Report, filed with the New Brunswick government, but not yet available to the public.

In 2011, Northcliff conducted a test pitting program comprising 349 temporary pits averaging 2 m in diameter and 4 m in depth. The pits were dug to bedrock using a Caterpillar 320 excavator and were backfilled after logging and sampling were completed. The purpose of the program was to assess exploration potential in areas of possible infrastructure development. A total of 409 till samples and 276 bedrock samples were collected from these pits for geochemical analysis. The locations of the pits were chosen by measuring approximately equidistant sections along pre-existing roads at spacings of between 50 m and 100 m. In addition to the samples collected in 2011, 147 till and 44 bedrock samples collected by Geodex in 2008 were also submitted for analysis. These sites were located between 2011 sample locations to ensure geochemical coverage of the target areas.

The physical characteristics of the till profile were documented as each pit was dug. Where bedrock was encountered, a sample was broken off by the excavator and rock type, grain size, foliation, mineralization, weathering, oxidation, veining, alteration, and color were described. Bedrock samples were only collected where the material was angular and interpreted to be in place.

Other information recorded at each test pit site included UTM coordinates, the depth of the pit, depth to where the till began (below BC horizon), depth of till sample(s) taken, depth of bedrock sample taken, photo number, and whether any water seepage within the pit occurred. UTM coordinates were marked/saved in a GPS unit (Garmin 62s).

Till and bedrock samples were prepared for analysis by Activation Laboratories Ltd. in Fredericton, New Brunswick. Full chain of custody control was maintained for all analytical samples from collection through delivery to the analytical laboratory. The till samples were dried and screened to 230 mesh. The concentration of tungsten was analyzed by instrumental neutron activation analysis (INAA). A suite of 58 additional elements was analyzed by a combination of INAA and inductively coupled plasma-mass spectrometry (ICP-MS) methods following four-acid digestion (Code UT-5 INAA(INAAGEO)/Total Digestion ICP/MS). All geochemical analyses were completed by Activation Laboratories Ltd. in Ancaster, Ontario. Bedrock samples were prepared and analyzed using protocols identical to those described in Section 11.0 of this report for drill core geochemical analysis.

Principal geological observations from the 2011 till and bedrock sampling program include the following.

- Analytical results for both bedrock and tills yield only a few scattered values for various elements at very low concentrations throughout the area of infrastructure.
- Slightly higher, but still weak, results were encountered over and immediately to the north of known mineralization in Zones I and II. These values decrease rapidly to the north.
- The results are fully consistent with the lack of mineralization and alteration in the hydrogeological and engineering drill holes completed in the area of potential infrastructure.
- The rock types encountered in bedrock samples generally conform to the distribution of major units as defined by published DNR regional geology maps.
- The results indicate that exploration potential for significant deposits of tungsten, molybdenum, or other metals in the area of potential infrastructure is very low.

## **9.2 Exploration Potential**

The Sisson deposit has been essentially closed off by drilling to the northeast, northwest, and south. At the present time, Northcliff considers the exploration potential in the immediate vicinity to be limited and has no plans for additional exploration work.

## 10.0 Drilling

This section has largely been extracted from the June 29, 2012 Technical Report by RPA and was prepared by Eric Titley, P. Geo., Senior Manager Resource Geology, HDI.

Since 1979, some 64,768 m of drilling has been completed on the property in 304 drill holes. Average core recovery for this drilling was 97.6%. A drilling summary by year is shown in Table 10.1.

Table 10.1 Drilling Summary – All Years					
Operator	Year	Drill Hole ID	No. of Holes	Core Size	Meters
Kidd Creek Mines Ltd.	1979	SSN01 to SSN11	11	BQ	1,663
	1980	SSN12 to SSN23	12	BQ	2,419
	1981	SSN24 to SSN32	9	BQ	1,727
	1982	SSN33 to SSN40	8	BQ	1,697
Geodex Minerals Ltd.	2005	SB-05-01 to SB-05-09	9	NQ	1,219
	2006	SB-06-02 to SB-06-30	29	NQ	7,653
	2007	SB-07-01 to SB-07-75 <sup>1</sup> ; SBM-07-01 SE-07-01 to SE-07-05 SP-07-01	83	NQ	20,194
	2008	SB-08-01 to SB-08-47 <sup>2</sup>	49	NQ	12,122
	2009	SB-09-1 to SB-09-28	28	NQ	4,899
Northcliff Resources Ltd.	2010	SB-10-001 to SB-10-005	5	HQ	2,701
Northcliff Resources Ltd.	2011	SB-11-006M to SB-11-037G	61	PQ & HQ	8,474
		SB-11-MW-001D to SB-11-MW-006D			
		SB-11-MW-001S to SB-11-MW-006S			
		SB-11-MWG-001 to SB-11-MWG-018			
<b>Total</b>			<b>304</b>		<b>64,768</b>
Notes:					
1. Including hole SB-07-10A.					
2. Including holes SB-08-7A and SB-08-34A.					

The following describes the drilling activities carried out by Kidd Creek Mines Limited (Kidd Creek), Geodex, and Northcliff on the Sisson property.

### 10.1 Geodex and Kidd Creek Programs (1979 – 2009)

Diamond drilling to delineate the W-Mo ± Cu mineralization in Zones I, II and III on the Sisson property was completed by Kidd Creek during the 1979 to 1982 period and Geodex in the 2005 to 2009 period.



Kidd Creek completed 40 drill holes on the property in three campaigns. Geodex drilled 197 additional holes totaling 45,943 m in five subsequent campaigns to the end of 2009.

The initial digital compilation of the historic drilling data was carried out by Geodex staff. Drilling information, such as lithologic and sampling logs, assay results, collar survey data and downhole survey data, was compiled from hard copy assessment reports filed with the New Brunswick government and from in-house records and reports. The Geodex compilation was provided to Northcliff and additional information was added where available.

Ideal Drilling Ltd. provided contract drilling services to Kidd Creek for the 1979 through 1981 drilling campaigns and Petro Drilling Limited provided services in 1982. In all cases, BQ-size equipment, producing drill core measuring approximately 36.5 mm in diameter, was used. Kidd Creek staff supervised the on-site geological work and also carried out core logging, sampling, and interpretive and reporting functions. Kidd Creek programs were coordinated from the company's Fredericton exploration office.

The 2005 drilling program by Geodex was contracted to Maritime Diamond Drilling Limited (Maritime) of Hilden, Nova Scotia. Maritime recovered NQ-size drill core measuring approximately 47.6 mm in diameter. In 2006, Maritime initiated the Zone III program using two drill rigs, both recovering NQ core. Geodex also contracted Lantech Drilling Limited (Lantech) of Moncton, New Brunswick, to assist in the completion of the 2006 drill program. Lantech also recovered NQ core. Geodex staff supervised all aspects of the Sisson Project, including on-site supervision, core logging, sampling, interpretive and reporting functions. Lantech was retained by Geodex to carry out all 2007 and 2008 drilling on the Sisson property and Maritime carried out the 2009 program. The Geodex programs were directed by staff operating out of their Fredericton office.

With the exception of one track-mounted drill rig operated by Maritime in 2006, conventional skid-mounted drilling equipment was utilized.

Drill core from all programs completed prior to 2006 is stored at the New Brunswick Department of Natural Resources core storage facility near Sussex, New Brunswick. Core from 2006 through 2009 programs was held in a secure storage facility by Geodex. Northcliff took over this storage facility in 2011.

Drill hole collar locations for all Kidd Creek holes were originally surveyed and coordinated to Mining Lease survey pins in the area. Most of these drill collars in the Zone III area were relocated by Geodex in 2006 and subsequently resurveyed along with all Geodex holes of the 2006 through 2009 programs. All Geodex surveying was carried out by suitably certified technical staff employed by an independent, third party surveying firm. UTM Zone 19 grid coordinates based on North American Datum 83 (NAD 83) were generated from this program. Holes were typically tested for inclination and azimuthal variation using downhole survey instruments and these data were incorporated in the Sisson Project database for use in the deposit model. Early holes were tested using mechanical instrumentation (Tropari) and those subsequent to 2005 used FlexIt (Reflex Instruments) downhole orientation tools. Mercator Geological Services Limited (Mercator) independently checked drill hole collar location coordinates for selected drill holes completed by Geodex through September 2009 and consistently found excellent agreement with the Geodex coordinates.

## 10.2 Northcliff Drilling

### 10.2.1 2010 Program

Northcliff contracted Lantech to carry out a seven-hole drilling program that was initiated in October 2010. Conventional skid-mounted drilling equipment recovering HQ-size core (63.5 mm diameter) was used and the program was terminated in mid-December 2010 after completion of only five holes (2,700 m). Professional and technical staff acting on behalf of Northcliff managed all aspects of the Sisson Project, including on-site supervision, core logging, sampling, interpretive, and reporting functions. Field activities were coordinated from offices in Fredericton, New Brunswick, and the Geodex core logging facility located at Keswick Ridge, New Brunswick, was used for logging and sampling purposes.

Northcliff hole locations were set out by suitably certified technical staff and locations were surveyed by an independent, third party surveying firm. UTM Zone 19 grid coordinates based on NAD 83 were generated from this program and incorporated in the Sisson Project drilling database. Holes were tested for inclination and azimuthal variation using electronic (Reflex) downhole survey instruments and these data (Table 10.2) were incorporated in the Northcliff database. Remaining half-core and coarse rejects are stored at the Northcliff core logging facility located on Kingsley Road north-northwest of Fredericton. The sample pulps are stored at a secure warehouse separate from the core logging facility.

RPA inspected the Northcliff core logging, sampling, and storage facility and found it to be well configured, properly equipped, and secure.

Table 10.2 Northcliff 2010 Drill Holes Coordinates and Orientations							
Hole ID	Northing <sup>(1)</sup>	Easting	Elevation	Length (m)	Azimuth (°) <sup>(2)</sup>	Dip (°)	Purpose
SB-10-001	650334.1	5136881.7	303.2	528	270	-70	Exploration
SB-10-002	650161.7	5137146.5	327.8	356	90	-50	Exploration
SB-10-003	650286.7	5137002.4	303.7	570	90	-55	Exploration
SB-10-004	649827.4	5136215.4	300.8	674	90	-59	Exploration
SB-10-005	650092.4	5136660.2	305.5	573	90	-55	Exploration
Notes:							
1. Coordinates are in NAD83 Zone19.							
2. Azimuth and dip are for the collar.							

### 10.2.2 2011 Program

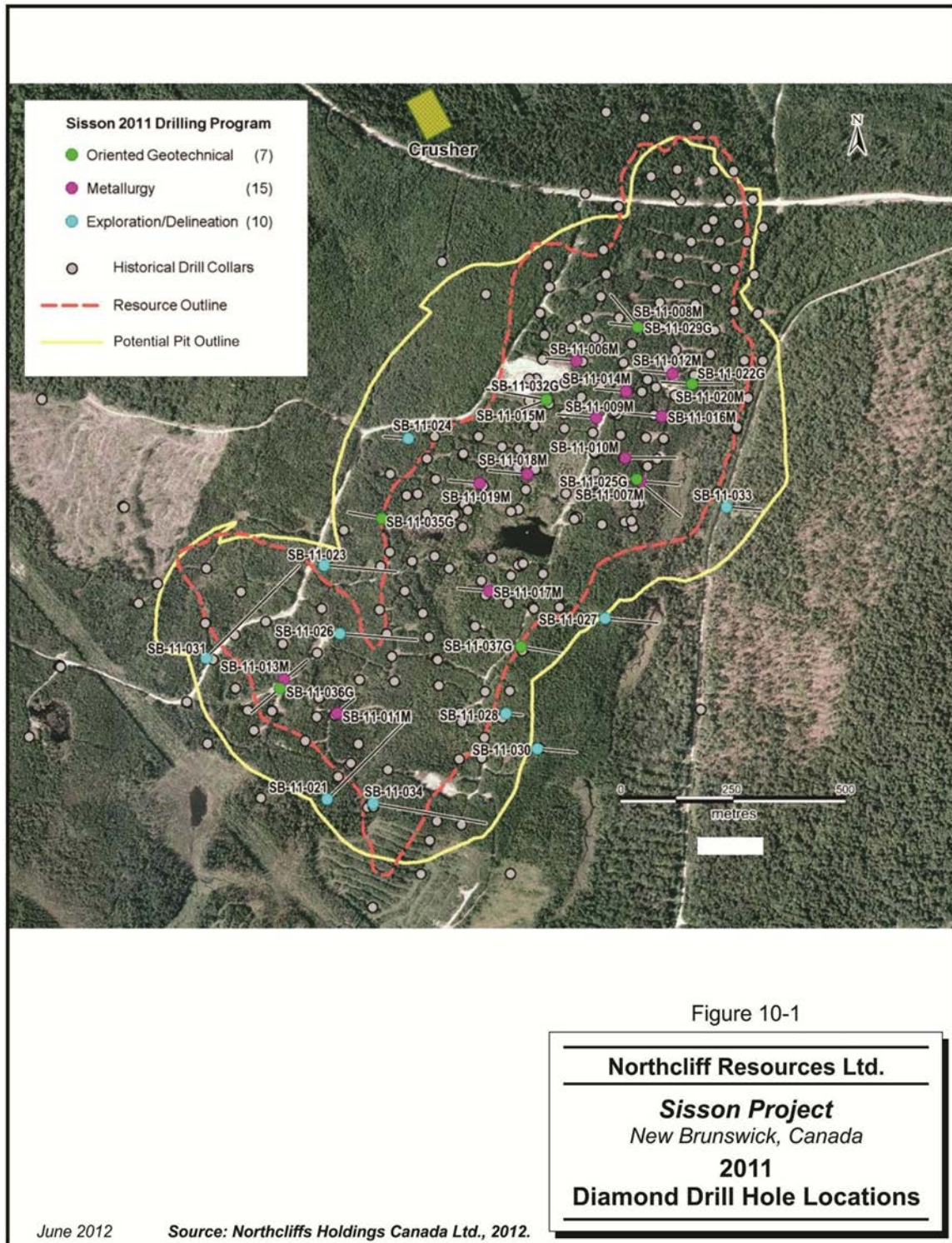
In 2011, 61 drill holes totaling 8,474 m were completed by Northcliff. The purpose of this program was to provide additional geological, metallurgical, hydrological, and geotechnical information for the Feasibility Study. The following information was obtained:

- Metallurgical and comminution sample material (15 holes, 3,048 m)
- Hydrogeological information (12 holes, 275 m)
- Engineering data in areas of proposed infrastructure (17 holes, 608 m)

- Pit wall geotechnical data in oriented drill holes (seven holes, 1,650 m)
- Additional deposit delineation and exploration to depth (ten holes, 2,901 m)

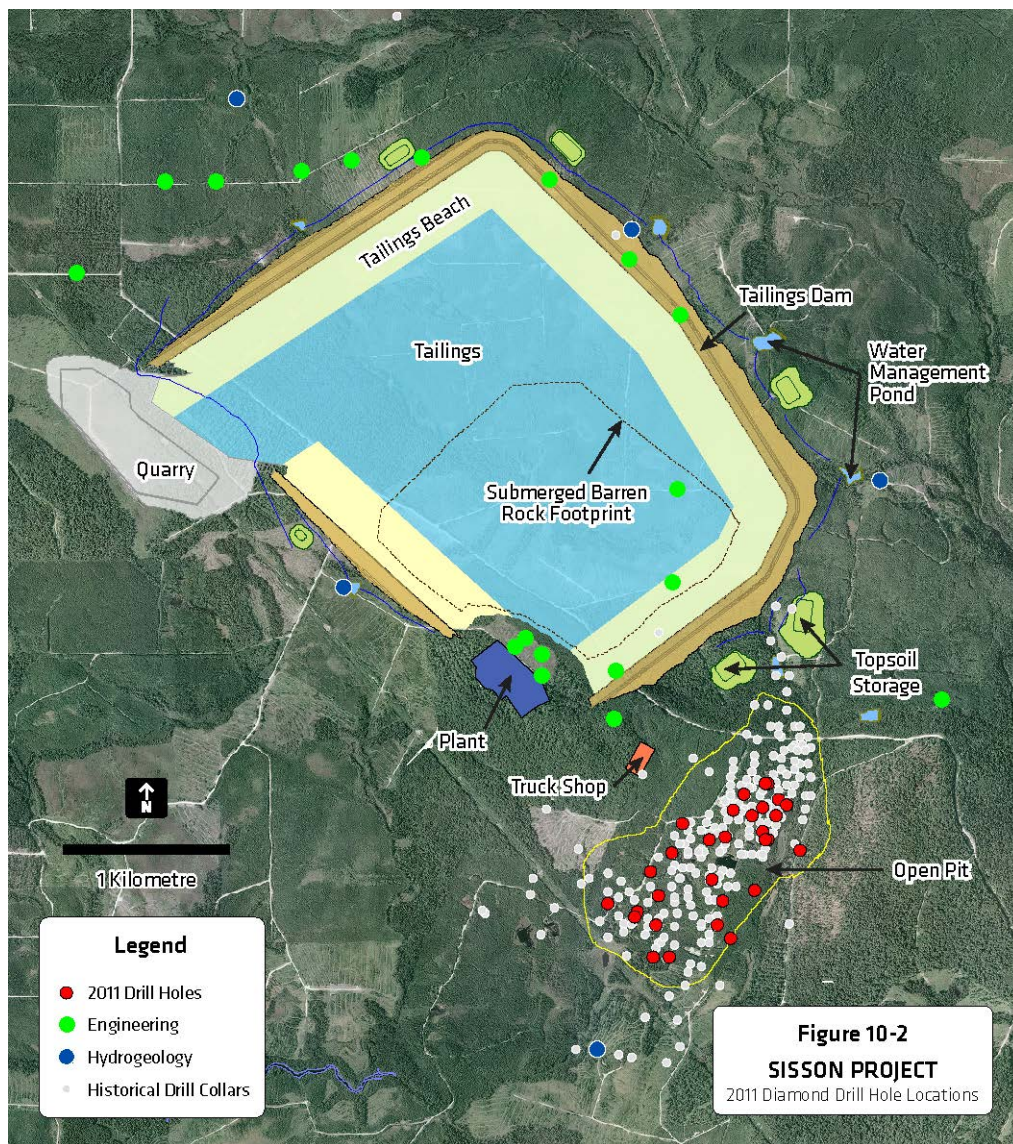
Northcliff contracted Layne Christiansen Canada Ltd. to carry out the 2011 drilling program initiated in June of that year. Conventional skid-mounted drilling equipment recovering both PQ size drill core measuring approximately 85.0 mm in diameter and HQ size core was used and the program was successfully completed in October 2011. Professional and technical staff acting on behalf of Northcliff managed all aspects of the Sisson Project, including on-site supervision, core logging, sampling, and interpretive and reporting functions. Field activities were coordinated from offices located in Fredericton, New Brunswick.

Drill hole locations were set out by suitably certified technical staff and locations were surveyed by an independent, third party surveying firm. UTM coordinates (NAD 83 zone 19) generated from this program were incorporated in the Sisson Project drilling database developed by Northcliff. Holes were tested for downhole inclination and azimuth variation using Reflex electronic survey instruments and these data were also incorporated in the Northcliff database. The 2011 drill hole locations are highlighted in Figure 10.1 and their coordinates and orientations are listed in Table 10.3. The location of the drill holes in relation to the proposed site infrastructure is shown in Figure 10.2.



**Figure 10.1: 2011 Diamond Drill Hole Locations**





**Figure 10.2: 2011 Diamond Drill Hole Locations Relative to Proposed Infrastructure**

**Table 10.3  
Northcliff 2011 Drill Hole Coordinates and Orientations**

Hole ID	Northing <sup>(1)</sup>	Easting	Elevation	Length (m)	Azimuth (°) <sup>(2)</sup>	Dip (°)	Purpose
SB-11-006M	650,361.5	5,136,996.5	305.7	201.0	275	-70	Metallurgy
SB-11-007M	650,502.6	5,136,730.7	300.1	150.0	95	-55	Metallurgy
SB-11-008M	650,498.5	5,137,074.5	305.3	189.1	275	-70	Metallurgy
SB-11-009M	650,405.1	5,136,871.3	306.7	252.0	275	-70	Metallurgy
SB-11-010M	650,469.1	5,136,782.0	301.1	198.0	90	-55	Metallurgy
SB-11-011M	649,832.2	5,136,217.7	300.8	147.0	50	-70	Metallurgy
SB-11-012M	650,574.0	5,136,968.5	299.2	201.0	275	-70	Metallurgy
SB-11-013M	649,714.0	5,136,291.9	298.8	200.7	50	-70	Metallurgy
SB-11-014M	650,471.0	5,136,928.3	307.1	201.0	275	-70	Metallurgy
SB-11-015M	650,296.2	5,136,910.7	304.3	201.0	250	-70	Metallurgy
SB-11-016M	650,550.3	5,136,874.6	299.2	252.0	275	-70	Metallurgy
SB-11-017M	650,168.3	5,136,487.7	304.9	201.0	275	-70	Metallurgy
SB-11-018M	650,252.7	5,136,746.6	304.0	252.0	275	-70	Metallurgy
SB-11-019M	650,147.1	5,136,725.8	302.9	201.0	275	-70	Metallurgy
SB-11-020M	650,617.2	5,136,944.0	299.4	201.0	275	-70	Metallurgy
SB-11-021	649,810.5	5,136,027.3	293.2	508.4	45	-60	Exploration
SB-11-022G	650,617.2	5,136,944.8	299.4	216.0	90	-65	Geotechnical
SB-11-023	649,803.2	5,136,543.8	313.0	282.0	95	-55	Exploration
SB-11-024	649,990.0	5,136,825.0	319.6	129.1	275	-65	Exploration
SB-11-025G	650,495.4	5,136,734.6	300.8	303.0	130	-65	Geotechnical
SB-11-026	649,838.4	5,136,393.8	303.6	405.0	95	-65	Exploration
SB-11-027	650,424.6	5,136,427.5	301.6	204.0	95	-55	Exploration
SB-11-028	650,205.2	5,136,216.3	310.6	141.0	95	-75	Exploration
SB-11-029G	650,497.1	5,137,071.7	305.4	219.0	320	-65	Geotechnical
SB-11-030	650,274.9	5,136,139.7	311.9	150.0	95	-55	Exploration
SB-11-031	649,542.5	5,136,338.9	300.5	507.0	45	-55	Exploration
SB-11-032G	650,294.9	5,136,910.4	304.3	321.0	280	-65	Geotechnical
SB-11-033	650,693.5	5,136,673.4	308.9	135.0	95	-55	Exploration
SB-11-034	649,912.0	5,136,018.1	296.4	441.0	100	-55	Exploration
SB-11-035G	649,931.5	5,136,648.5	310.2	180.0	280	-65	Geotechnical
SB-11-036G	649,706.1	5,136,272.1	298.3	201.0	230	-65	Geotechnical
SB-11-037G	650,238.4	5,136,364.8	314.8	210.0	100	-65	Geotechnical

**Table 10.3  
Northcliff 2011 Drill Hole Coordinates and Orientations**

Hole ID	Northing <sup>(1)</sup>	Easting	Elevation	Length (m)	Azimuth (°) <sup>(2)</sup>	Dip (°)	Purpose
SB-11-MW-001D	647,944.0	5,138,245.0	300.0	38.0	0	-90	Hydrogeology
SB-11-MW-001S	647,944.4	5,138,252.5	345.5	24.4	0	-90	Hydrogeology
SB-11-MW-002D	649,477.8	5,135,470.4	293.3	37.2	0	-90	Hydrogeology
SB-11-MW-002S	649,476.1	5,135,475.1	293.6	8.0	0	-90	Hydrogeology
SB-11-MW-003D	651,193.7	5,138,895.3	287.5	33.2	0	-90	Hydrogeology
SB-11-MW-003S	651,200.0	5,138,893.7	287.8	6.0	0	-90	Hydrogeology
SB-11-MW-004D	651,551.8	5,137,567.0	261.3	35.0	0	-90	Hydrogeology
SB-11-MW-004S	651,550.1	5,137,567.6	262.0	9.0	0	-90	Hydrogeology
SB-11-MW-005D	647,315.3	5,141,200.4	308.8	35.4	0	-90	Hydrogeology
SB-11-MW-005S	647,312.0	5,141,200.1	309.0	9.1	0	-90	Hydrogeology
SB-11-MW-006D	649,690.8	5,140,410.8	305.9	28.9	0	-90	Hydrogeology
SB-11-MW-006S	649,687.2	5,140,414.2	306.1	9.2	0	-90	Hydrogeology
SB-11-MWG-001	648,977.7	5,137,896.8	382.4	39.7	0	-90	Engineering
SB-11-MWG-002	649,135.0	5,137,847.7	386.3	29.9	0	-90	Engineering
SB-11-MWG-003	649,040.2	5,137,948.9	382.2	39.7	0	-90	Engineering
SB-11-MWG-004	649,149.8	5,137,718.7	388.4	29.8	0	-90	Engineering
SB-11-MWG-005	646,872.2	5,140,695.7	328.3	42.9	0	-90	Geotechnical
SB-11-MWG-006	647,692.3	5,140,769.7	320.0	35.3	0	-90	Geotechnical
SB-11-MWG-007	648,414.3	5,140,845.2	323.5	50.5	0	-90	Geotechnical
SB-11-MWG-008	649,184.6	5,140,705.9	332.8	32.5	0	-90	Geotechnical
SB-11-MWG-009	649,672.4	5,140,233.9	314.7	23.1	0	-90	Geotechnical
SB-11-MWG-010	649,978.2	5,139,892.0	340.6	22.5	0	-90	Geotechnical
SB-11-MWG-012	649,960.8	5,138,850.0	306.0	33.9	0	-90	Geotechnical
SB-11-MWG-013	649,934.3	5,138,292.2	317.6	39.6	0	-90	Engineering
SB-11-MWG-014	649,600.4	5,137,748.0	336.6	36.9	0	-90	Engineering
SB-11-MWG-015	649,577.4	5,137,472.3	358.6	30.8	0	-90	Geotechnical
SB-11-MWG-016	646,341.6	5,140,156.2	354.3	35.3	0	-90	Geotechnical
SB-11-MWG-017	647,996.0	5,140,821.4	321.7	38.3	0	-90	Geotechnical
SB-11-MWG-018	647,178.0	5,140,700.6	320.8	39.7	0	-90	Geotechnical

Notes:

- Coordinates are in NAD83 Zone19.
- Azimuth and dip are for the collar.

## 11.0 Sampling Preparation, Analysis and Security

This section has been extracted from the June 29, 2012 Technical Report by RPA and was largely prepared by Eric Titley, P. Geo., Senior Manager Resource Geology, HDI.

Table 11.1 is a summary of the sampling and sample preparation work performed on the Sisson Project since 1979. The analytical history is summarized in Table 11.2.

Table 11.1 Sampling and Sample Preparation Summary						
Year	Drill Holes	Operator	Method	Sample Length		Sample Preparation Laboratory
				Range	Typical	
1979	11	Kidd Creek Mines Ltd	Half Split	0.05 - 4.1 m Selected Intervals	2.0 - 4.1 m Selected Intervals	Bondar-Clegg Ottawa, ON
1980	12					
1981	9					
1982	8					
2005	9	Geodex Minerals Ltd	Half Split	0.12 - 2.56 m Continuous Intervals	1.5 m Continuous Intervals	Actlabs Ancaster ON or Fredericton, NB
2006	29		Half Split Mechanical	0.5 - 1.5 m Continuous Intervals	1.5 m Continuous Intervals	
2007	82					
2008	49					
2009	28					
2010	n/a	Northcliff Due Diligence	Pulp Split	1.5 m Selected Intervals	1.5 m Selected Intervals	Acme Vancouver, BC
			Reject Split			
			¼ Core Split			
2010	5	Northcliff	Half Split Diamond Saw	0.7 – 5.0 m Continuous Intervals	3.0 m Continuous Intervals	Actlabs Fredericton, NB
2011	61	Northcliff	Half Split Diamond Saw	0.7 – 5.0 m Continuous Intervals	3.0 m Continuous Intervals	Actlabs Fredericton, NB



**Table 11.2  
Assay Summary**

Year	DDH	Assay Laboratory	Elements Assayed	Method & Code	Digestion or Activation	Finish
1979	11	Bondar-Clegg Ottawa, ON	WO <sub>3</sub> , Cu, Mo	Unknown	Unknown	Unknown
1980	12					
1981	9					
1982	8		WO <sub>3</sub> , Cu, Mo, Ag, As, Bi	Colorimetric; titration; AAS; FA	Thiocyanate for WO <sub>3</sub> ; HF + HNO <sub>3</sub> for Cu and Bi; FA for Ag; Na <sub>2</sub> O <sub>2</sub> fusion for As	Colorimetric for WO <sub>3</sub> ; I2 titration for As; AAS for Cu, Mo, Ag, Bi
2005	9	Actlabs Ancaster, ON	W, Mo, Au, Ag, Cd, Cu, Mn, Ni, Pb, Zn, S, As, Ba, Hg, Sb	Multi- element analytical package IEPI AQUAGEO	Neutron activation for W, Au, As, Ba, Sb & Hg; AR digestion for Ag, Cd, Cu, Mn, Mo, Pb, Zn & S	W, Au, As, Ba, Sb & Hg by INAA; Ag, Cd, Cu, Mn, Mo, Pb, Zn & S by ICP
2006	29		W, Mo, Cu	INAA INAAGEO	Neutron activation for W, Mo; AR digestion for Cu	W & Mo by INAA; Cu by ICP
2007	82					
2008	49					
2009	28					
2010	Due Dil.	Acme Vancouver, BC	W, Mo, Cu and 38 additional elements	1EX 7KP2	Phosphoric acid digest for W and 4 additional elements, 4 acid digest for Cu & 40 additional elements	ICP-AES for W and 4 additional elements, ICP-MS for Cu & 40 additional elements.
2010	5	Actlabs Ancaster, ON	W, Mo, Cu & 61 additional elements	Geodex- INAA (INAAGEO) UT-2-0.5 g	Neutron activation for W, Mo; Aqua Regia digest for Cu & 61 Elements	W & Mo by INAA; Cu & 61 elements by ICP- OES/MS
2011	61	Actlabs Ancaster, ON	W, Mo, Cu & 61 additional elements	Geodex- INAA (INAAGEO) UT-2-0.5 g	Neutron activation for W, Mo; Aqua Regia digest for Cu & 61 Elements	W & Mo by INAA; Cu & 61 elements by ICP- OES/MS

### 11.1 Kidd Creek Samples (1979-1982)

According to Mercator (2011), documentation of the Kidd Creek drilling programs does not provide a lot of detail on the sample preparation methodologies, analytical procedures, or security considerations. The bagged samples of half core material were shipped from Kidd Creek's Fredericton office to Bondar-Clegg laboratories in Ottawa, Ontario, where sample preparation and assay were conducted. Assays from 1979 to 1981 were done for copper and WO<sub>3</sub> and in 1982 for copper, WO<sub>3</sub>, silver, arsenic, and bismuth. The analytical methods used were as follows:

- WO<sub>3</sub> - thiocyanate digestion with colorimetric finish
- Mo – nitric and hydrochloric acid (HNO<sub>3</sub> + HCl) digestion with atomic absorption (AAS) finish
- Cu – hydrofluoric and nitric acid (HF + HNO<sub>3</sub>) digestion with AAS finish

- Ag fire assay (FA) fusion with AAS finish
- Bi - HF + HNO<sub>3</sub> digestion with AAS finish
- As - Na<sub>2</sub>O<sub>2</sub> fusion and then titrated by I<sub>2</sub> solution.

In addition, certain samples were selected for geochemical analysis. Table 11.3 lists the analytical methods used in the 1982 program.

Table 11.3 Analytical Methods 1982 Program				
Type	Element	Fraction	Digestion or Extraction	Finish
Assay	WO <sub>3</sub>	-200 Mesh	Thiocyanate	Colorimetric
Assay	Cu	-200 Mesh	HF + HNO <sub>3</sub>	Atomic Absorption
Assay	Ag	-200 Mesh	Fire Assay	Atomic Absorption
Assay	As	-200 Mesh	Na Peroxide Fusion	Idiometric Titration
Assay	Bi	-200 Mesh	HF + HNO <sub>3</sub>	Atomic Absorption
Geochemical	Pb, Zn, Bi	-200 Mesh	HNO <sub>3</sub>	Atomic Absorption
Geochemical	Mo	-200 Mesh	HNO <sub>3</sub> + HCl	Atomic Absorption
Geochemical	Sb, Sn	-200 Mesh		X-Ray Fluorescence
Geochemical	F	-200 Mesh	Fusion	Specific Ion Electrode
Geochemical	As	-200 Mesh	HNO <sub>3</sub> + HClO <sub>4</sub>	Atomic Absorption

No mention is made of sample or site security measures in the Kidd Creek reports. It is assumed that security measures consistent with industry standards of the day were in place during the 1979 through 1982 drilling programs.

## 11.2 Geodex Samples (2005-2009)

Activation Laboratories Ltd. (Actlabs) performed sample preparation and analytical work for the 2005 through 2009 drilling programs. Two Actlabs facilities, one in Ancaster, Ontario, and the other in Fredericton, New Brunswick, were used. Core samples were placed in plastic sample bags, checked for sample sequence integrity, and shipped by courier to the laboratory. Core samples were crushed, split, and then pulverized to 95% passing 105 µm (150 mesh).

In 2005, INAA was used to determine gold, arsenic, barium, antimony, mercury, and tungsten and Inductively Coupled Plasma Atomic Emission Spectroscopy (ICP-AES) was used to determine silver, cadmium, copper, manganese, molybdenum, nickel, lead, zinc, and tin after aqua regia digestion.

From 2006 and onward, Geodex focused on tungsten and molybdenum, as the main metals of economic interest. Tungsten and molybdenum were determined by the INAA method with included tungsten and molybdenum standards. Copper in selected samples was determined by aqua regia digestion with ICP-AES finish. In addition, some samples were selected for multi-elements analysis using a variety of assay methods (Table 11.4).

In addition to the mainstream samples analyzed by Actlabs, check duplicates from the 2006 drilling program were analyzed by ALS Chemex (ALS) in Vancouver, British Columbia, using X-ray fluorescence (XRF) on pressed pellets. In the 2007, 2008, and 2009 drilling programs, check duplicate analyses were carried out at Global Discovery Laboratory in Vancouver, British Columbia (acquired by Acme Analytical Laboratories in 2008).

Mercator (2009) reported that a system of drill core security was implemented during the 2006 program that began at the drill site and continued through transportation to the sampling and logging facility, and then onward through shipment of samples to the laboratory. At the drill site, the security of core was the responsibility of the contractor under the direction of Geodex. Drill crews were instructed that only they and designated Geodex personnel were to access core at the drill sites and that core boxes should be covered immediately after drilling. Designated field transfer points were established at which responsibility for core security passed from the drilling contractor to Geodex. This was coordinated daily between the drilling contractor crews and Geodex personnel. Core delivered to the facility was placed in a secure location prior to logging and sampling, and Geodex personnel were responsible for security of all core and samples at this facility. Prior to sampling, a digital photograph of each box of core was taken and archived for future reference. After core logging and sampling were completed, core boxes containing half-core archives were labeled with weather-proof aluminum tags and were bound with steel strapping and stored at the logging facility on pallets. Core sample numbers were recorded on sample shipment forms, and after checking against sample listings, were placed in sealed plastic buckets for shipment to the laboratory. Details of the shipment were recorded; including date shipped, sample numbers, client account number, laboratory name, and name of the individual responsible for preparation of the documents and security of the sample shipment preparation process.

### **11.3 Northcliff Due Diligence (2010)**

As part of a due diligence review, a sampling program of core, rejects, and pulps was carried out by Darrel Johnson, P.Geol., of HDI. Eighteen core samples, 25 coarse rejects, and 237 pulp samples from site were collected on behalf of Northcliff and submitted to Acme Analytical Laboratories (Acme) in Vancouver, British Columbia. The samples were prepared at Acme. All core samples were crushed to 75% passing a 2 mm screen (10 mesh) and a 250 g subsample was taken. Pulverizing was performed on the 250 g splits to 85% passing 75  $\mu$ m (200 mesh).

Molybdenum, copper, and 39 additional elements were determined by four acid digestion with an ICP-MS finish (Acme code: 1EX). Molybdenum, tungsten, and three additional elements were reassayed by phosphoric acid leach with ICP-AES finish (Acme code: 7KP2). In addition, 27 external standards and five blanks were inserted and assayed along with the mainstream samples.

### **11.4 Northcliff Samples (2010)**

Sampling during Northcliff's 2010 drilling program was carried out under the supervision of Dr. James Lang, P.Geol., of HDI.

Core logging and sampling activities were conducted by Northcliff at the Geodex logging facility located in Keswick Ridge, New Brunswick. Procedures for core handling and processing were generally similar to those employed by Geodex. Drill cores were transported from the Sisson Project site to the core logging

facility by company truck. Core received from the field was washed and prepared for initial inspection under ultra-violet (UV) light to define major rock units and associated mineralized intervals. This information was used to provide a preliminary assessment of mineralization intensity. Core was then stacked for subsequent detailed logging and sampling.

After logging by a Northcliff geologist, but prior to sampling, all core was photographed using a standardized layout format and digital camera to provide a permanent pre-sampling record of core from each drill hole. Continuous rock quality and core recovery determinations were also recorded for all cores and entered into the digital project database. Conventional core logging procedures were standardized through use of a project lithological legend and lithocode system with log information entered directly into a digital logging utility accessed via the logging geologist's laptop computer and later integrated into the digital project database.

Core sampling by Northcliff was conducted using diamond saws. One half of each split core sample was placed in a pre-labeled plastic bag along with a corresponding sample book tag and then sealed and laid out for checking prior to insertion of quality assurance/quality control (QA/QC) materials into the associated sample sequence. A corresponding tag was secured to the core box for archive purposes. Tracking of sample intervals and QA/QC materials was integrated with the logging function and thereby within the digital project database.

The core samples were collected by site geological staff. Most sample intervals were from continuous three meter lengths of core. Some interval lengths varied depending on the geologic features encountered. The sample intervals were marked and then the core samples were cut in half lengthwise with a diamond saw. The samples were sealed in plastic bags and sent to Actlabs sample preparation laboratory in Fredericton, New Brunswick.

Core samples and associated QA/QC samples were prepared at Actlabs sample preparation laboratory in Fredericton, New Brunswick, and analyzed at Actlabs in Ancaster, Ontario.

At Actlabs in Fredericton, the samples were sorted, dried, weighed, and then crushed to 75% passing 2 mm (10 mesh). After crushing, an 800 g subsample was split and pulverized to 95% passing 105  $\mu\text{m}$  (150 mesh). The pulverized subsample pulps were sent to Actlabs in Ancaster, Ontario, for assay. Actlabs in Ancaster is ISO 17025 accredited for specific registered tests. In addition, it is also accredited to CAN-P-1579, specific to mineral analysis laboratories.

At Actlabs in Ancaster, tungsten and molybdenum were determined by the Geodex INAA method (Actlabs code: INAAGEO); copper, silver, sulphur, and an additional 59 elements were analyzed by aqua regia digestion with Inductively Couple Plasma Optical Emission Spectroscopy/Mass Spectrometry (ICP-OES/MS) finish (Actlabs code: UT-2-0.5g).

In-line duplicates were assayed by Actlabs along with the mainstream samples using the same method. Inter-laboratory check assays were performed by Acme. Acme is ISO certified (since 1996) and is currently registered to the ISO 9001:2000 quality system standards. Tungsten, molybdenum, and three additional elements were determined by phosphoric acid leach with ICP-AES finish (Acme code: 7KP2). Molybdenum, copper, and 39 additional elements were determined by four-acid digestion with ICP-MS finish (Acme code: 1EX).

A similar system of drill core security, as used by Geodex in 2005 through 2009, was implemented by Northcliff for the 2010 drill program.

### **11.5 Northcliff Samples (2011)**

Sampling during the 2011 drilling program was carried out under the supervision of Dr. James Lang, P.Geo., of HDI, using essentially the same procedures as those used in 2010. Mainstream sample preparation and assay work continued to be performed by Actlabs using the same procedures as for the 2010 drill program. Sample preparation and test work on metallurgical samples was performed by SGS of Lakefield, Ontario.

Figure 11.1 is a flow chart illustrating the 2011 drill core sampling, sample preparation, and assay analysis.

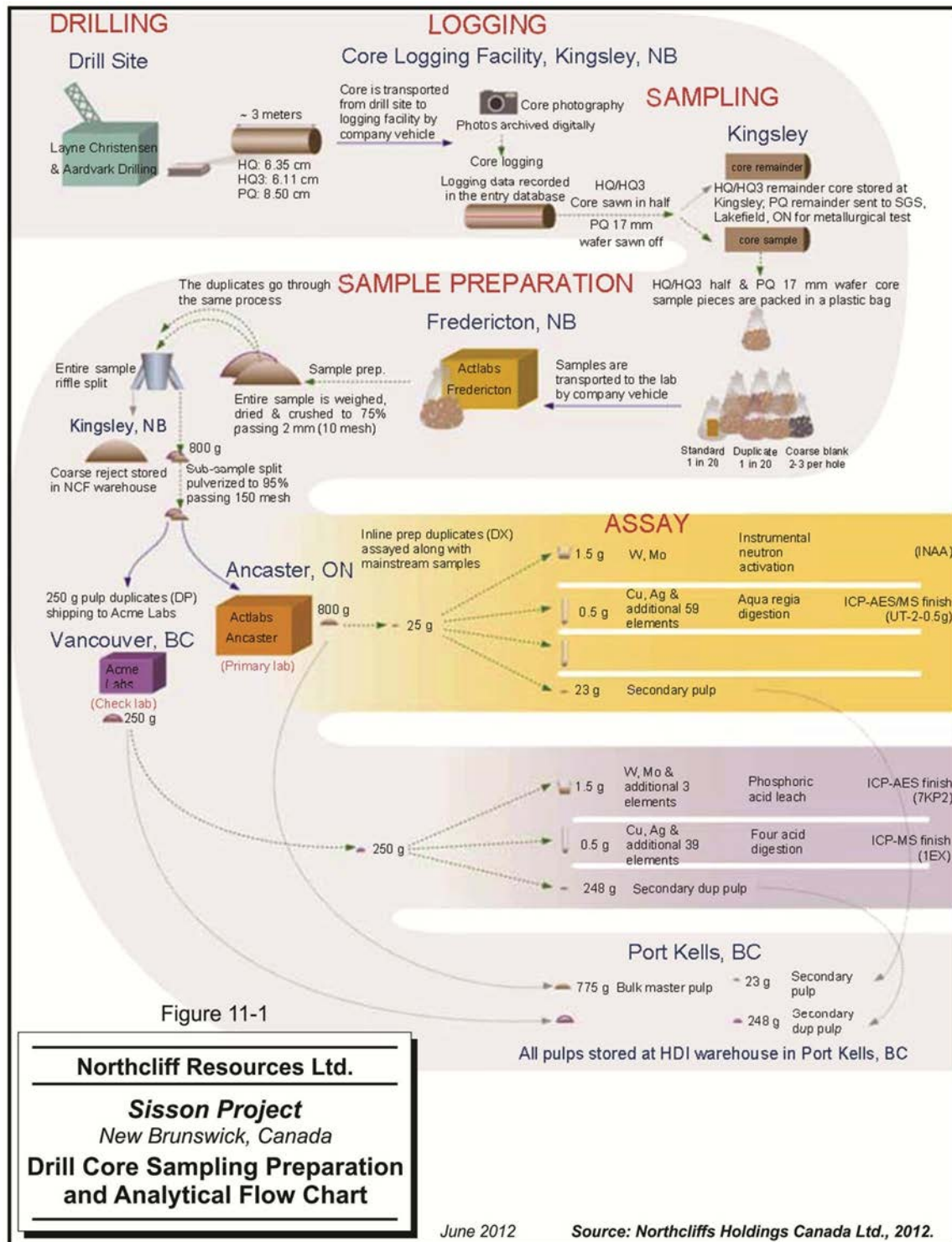


Figure 11.1: 2011 Drill Core Sampling Preparation and Analytical Flow Chart

## 11.6 Core Recovery

Table 11.4 summarizes the core recovery for the drilling conducted from 2006 onwards. RPA noted that the recovery appears to be satisfactory.

Table 11.4 Core Recovery		
Year	Number of Intervals	Mean Core Recovery Percent
2006	2,341	98.0
2007	6,632	97.1
2008	3,871	97.6
2009	1,561	97.4
2010	894	98.0
2011	2,415	98.6
<b>Total</b>	<b>17,714</b>	<b>97.6</b>

## 11.7 Discussion and Conclusions

In RPA's opinion, the sampling and core handling was carried out in an appropriate manner consistent with common industry practice. The assaying was performed at independent commercial accredited laboratories, using conventional approved methodologies. The holes were drilled and samples taken in such a way that they would be representative of the deposit mineralization.

RPA noted that the low grade ranges encountered for the Sisson Project prompted the selection of INAA as the primary analytical technique which was not as common a technique as more typical assay methods such as ICP or AAS, and was not generally offered by commercial laboratories. At the time, RPA noted that Actlabs was one of the few labs in Canada that offered this service. Consequently, pulp duplicate comparisons with other laboratories, using different protocols, would likely to have resulted in apparent biases. RPA believes that this could have created some difficulties in the evaluation of assay QA/QC results, but does not anticipate it would adversely affect the Mineral Resource estimate.



## 12.0 Data Verification

This section is reproduced from the June 29, 2012 Technical Report by RPA and was largely prepared under the direction of Eric Titley, P. Geo., Senior Manager Resource Geology, HDI, with assistance from Weidong Yang, Ph.D., and Catherine Sidwell.

### 12.1 Earlier Operators

#### Kidd Creek (1979-1982)

Reports documenting the Kidd Creek Mines drilling programs do not specifically address QA/QC issues and no evidence was noted of any external QA/QC samples inserted with core samples. However, it was reported that 67 original sample pulp splits from drill hole SSN-33 were reassayed and overall reproducibility of assay values between sample splits was found to be consistent and acceptable.

The overall impact of the Kidd Creek drill holes on the current Sisson resource estimate is limited as many of the holes are outside the current study area. Of the 31,900 core samples assayed on the Sisson Project from 1979 to the end of 2011, only 2,700, or about 8.5% of the total, are from the Kidd Creek drill holes. In 2007 and 2008, Geodex sampled and assayed 769 intervals from the historic Kidd Creek drill core, which had never previously been assayed.

#### Geodex (2005-2009)

During the 2005 drill program, no external QA/QC samples were applied by Geodex. Only internal standards, blanks, and duplicates administered by Actlabs were used to assess the analytical results. From 2006 onward, Geodex implemented a QA/QC protocol which included the use of external standards and blanks, which were inserted and analyzed with the mainstream core samples. Table 12.1 is a summary of the regular mainstream samples and additional QA/QC samples analyzed in the years 2005 through 2009.

In the 2006 drill program, Geodex completed 29 holes and collected 4,876 core samples. The company adopted a QA/QC program that included systematic insertion of certified standards and barren rock blanks and analysis of duplicate samples. As shown in Table 12.1, 52 standards and 193 blanks were inserted and assayed along with mainstream samples. A total of 161 check duplicates including 147 mainstream duplicates and 14 blanks were submitted to the ALS Chemex laboratory in Vancouver, British Columbia, for check analysis.

<b>Table 12.1 Assay QA/QC Summary for 2005-2009</b>					
<b>Year</b>	<b>MS</b>	<b>DP</b>	<b>ST</b>	<b>BL</b>	<b>ST %</b>
2005	515	-	-	-	-
2006	4,876	147	52	193	4
2007	11,940	320	498	522	4
2008	5,114	186	213	219	4
2009	2,945	121	156	129	4
<b>ALL</b>	<b>25,390</b>	<b>774</b>	<b>919</b>	<b>1,063</b>	<b>4</b>

Notes:

1. MS – mainstream sample.
2. DP - duplicate.
3. ST- standard.
4. BL - blank.
5. ST% - standards as a percent of the total mainstream samples.

In the 2007 drill program, Geodex drilled 83 holes with a total length of 20,194 m. A total of 11,940 core samples were collected and 498 standards and 522 blanks were applied. In addition, 417 check duplicates (320 mainstream duplicates and 97 blank duplicates) were assayed at Actlabs.

Geodex completed 49 holes in 2008 with a total length of 12,122 m and collected 5,114 core samples. They submitted 213 external standards and 219 external blanks with these samples for analysis as Actlabs. A total of 190 check duplicates (186 mainstream duplicates and four blank duplicates) were also assayed.

A 4,899 m, 28-hole program was completed by Geodex in 2009 in which 2,945 core samples were collected, and 156 standards and 129 blanks were included for analysis at Actlabs. In addition, 121 mainstream duplicates were selected for check assay.

In 2007 and 2008, Geodex also assayed 769 core samples from the Kidd Creek program, from core intervals that had not previously been assayed.

### Standards

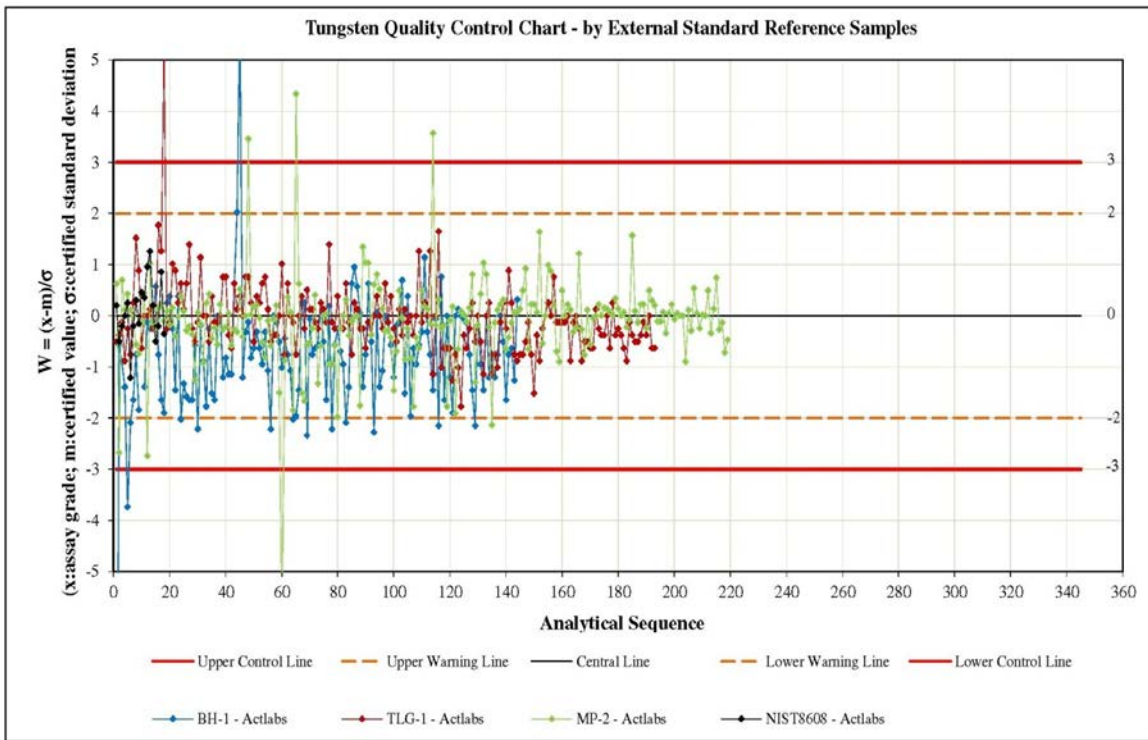
Table 12.2 lists the standard reference samples used in 2006 to 2009 exploration programs.

Table 12.2 Standards Used by Geodex 2006 Through 2009				
Standard	Times Used	W (%)	Mo (%)	Cu (%)
BH-1	144	0.422	<b>0.02</b>	-
HV-2	345	-	0.048	0.57
MP-2	219	0.65	0.281	-
NIST8608	18	0.22	0.098	0.096
TLG-1	193	0.083	-	-

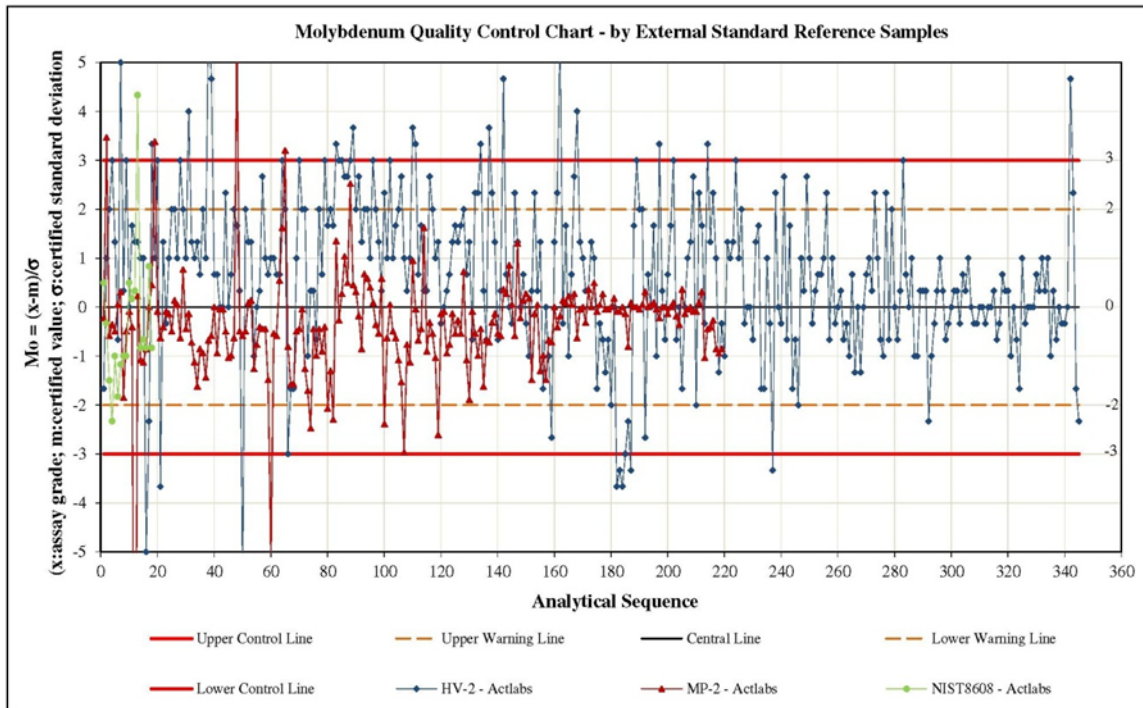
Notes: Value in *italics (bold)* is not certified.

Figure 12.1 and Figure 12.2 show the combined standard control charts based on Geodex 2006 and 2009 assay QA/QC results.

In RPA's opinion, the standards results from the Geodex assays showed a fairly broad scatter, but were generally within acceptable limits and unbiased.



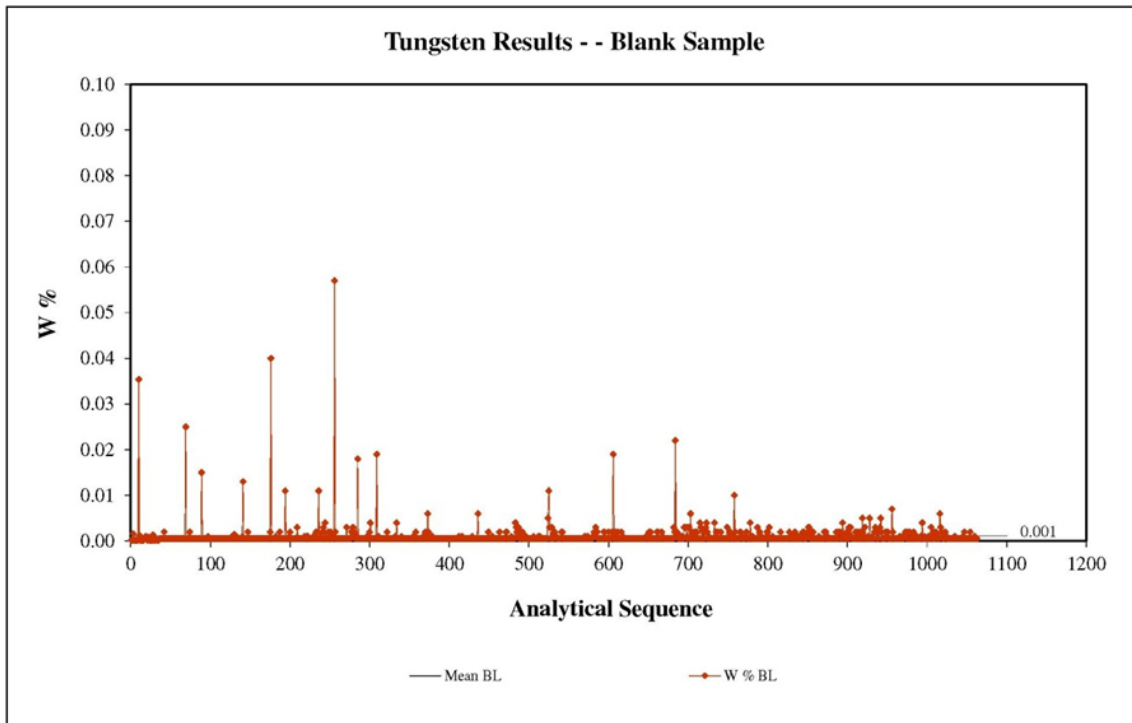
**Figure 12.1: Tungsten Quality Control Chart 2006-2009 Drill Programs**



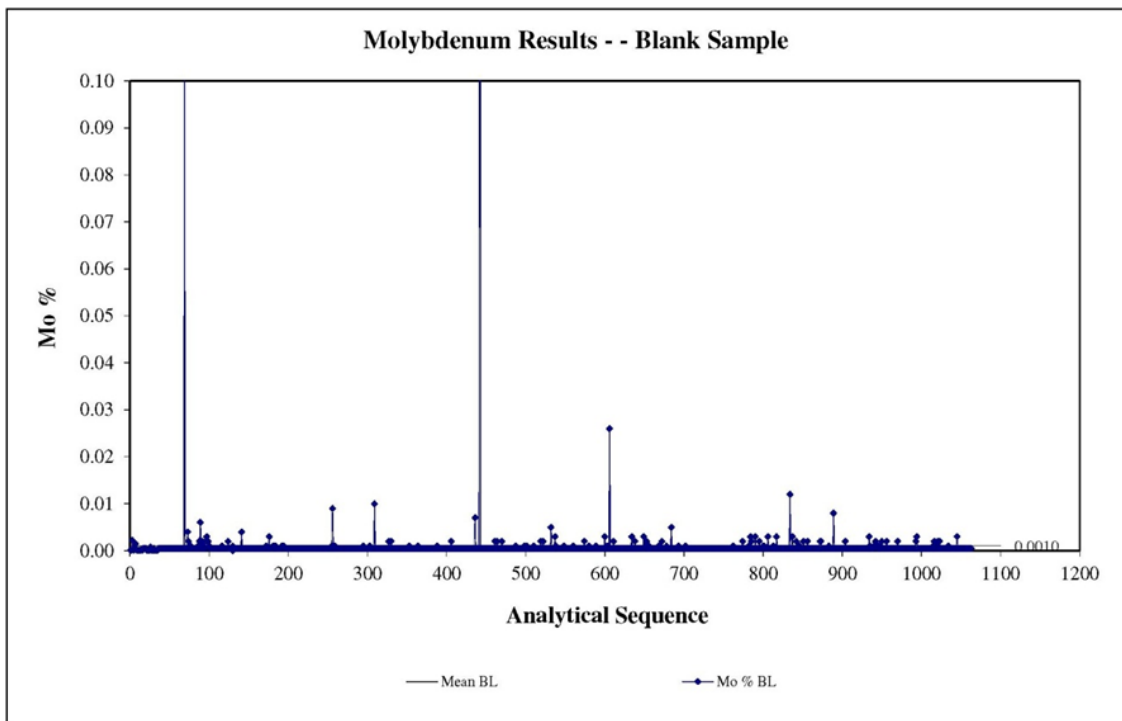
**Figure 12.2: Molybdenum Quality Control Chart 2006-2009 Drill Programs**

Blanks

A total of 1,063 external blanks were included with the regular samples during the 2006 through 2009 drill programs. Figure 12.3 and Figure 12.4 show the results of tungsten and molybdenum respectively for these blanks. The assay grades of the blanks are generally low ( $\leq 0.001\%$ ). However, some are obviously higher than normal, which Northcliff personnel think may be due to some degree of cross-contamination during sample preparation, perhaps during the sample crushing process. In RPA's opinion, this was a plausible explanation. The total number of apparent failures relative to the number of determinations was within an acceptable tolerance and seems to have improved with time.



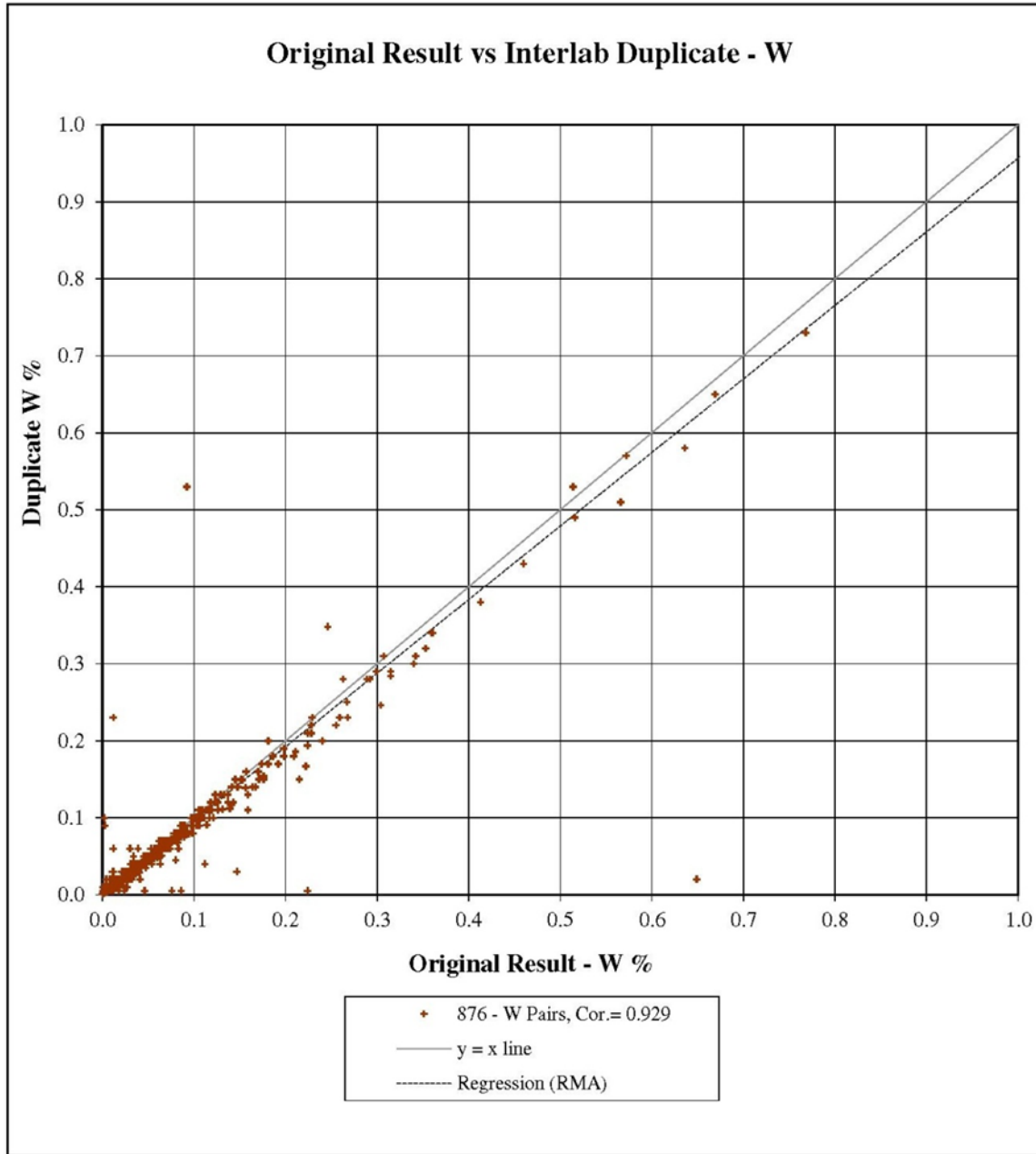
**Figure 12.3: Tungsten Results of Blank Samples 2006-2009 Drill Programs**



**Figure 12.4: Molybdenum Results of Blank Samples 2006-2009 Drill Programs**

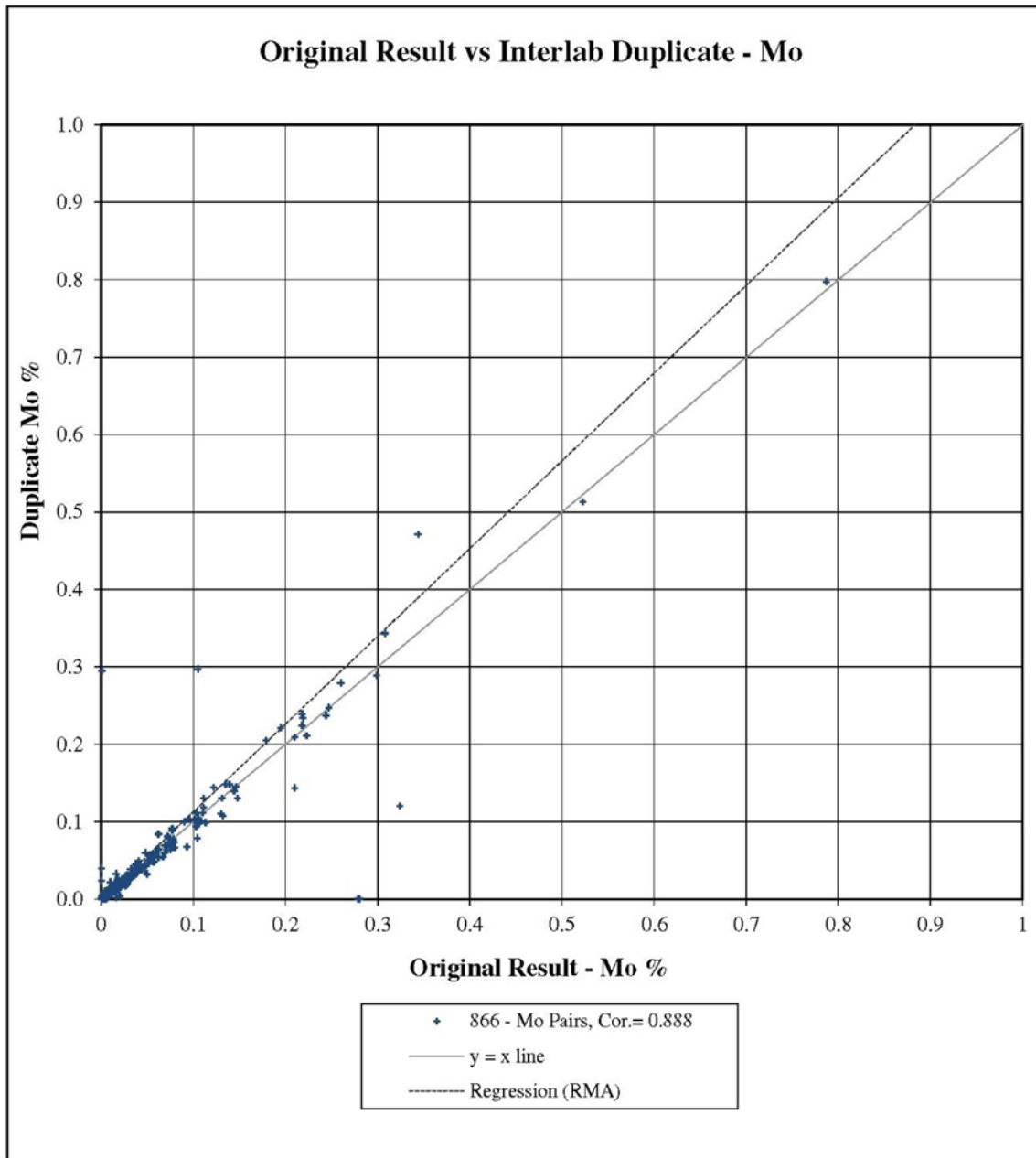
Duplicates

A total of 889 inter-laboratory pulp duplicates (774 mainstream duplicates and 115 blank duplicates) were assayed at a second laboratory during the 2006 through 2009 Geodex drilling programs. Figure 12.5 and Figure 12.6 are scatterplots of the matched pair duplicate results for tungsten and molybdenum respectively comparing Actlabs results on the X axis with the check laboratory results from ALS Chemex (2006) and Global Discovery Labs (2007-2009) on the Y axis.



**Figure 12.5: Tungsten Duplicates Scatterplot 2006 - 2009 Drill Programs**





**Figure 12.6: Molybdenum Duplicates Scatterplot 2006 - 2009 Drill Programs**

In RPA's opinion, the results for molybdenum demonstrated a small bias between the original and duplicate assays. On inspection of the data, this apparent bias was found to be due to one outlier, which plots outside the range of Figure 12.6. The apparent bias disappeared when this outlier was removed.

## 12.2 Northcliff Due Diligence Sampling (2010)

As part of a due diligence review conducted by Northcliff in 2010, 18 core samples, 25 coarse rejects, and 237 pulps were selected and sent for reanalysis. Northcliff included 27 standards and five blanks

with the samples and sent them to Acme in Vancouver for assay. The results from this resampling program confirmed the original results obtained by Geodex.

A total of 27 external standards and five blanks were included with this suite of samples. Table 12.3 is a summary of the due diligence sample types.

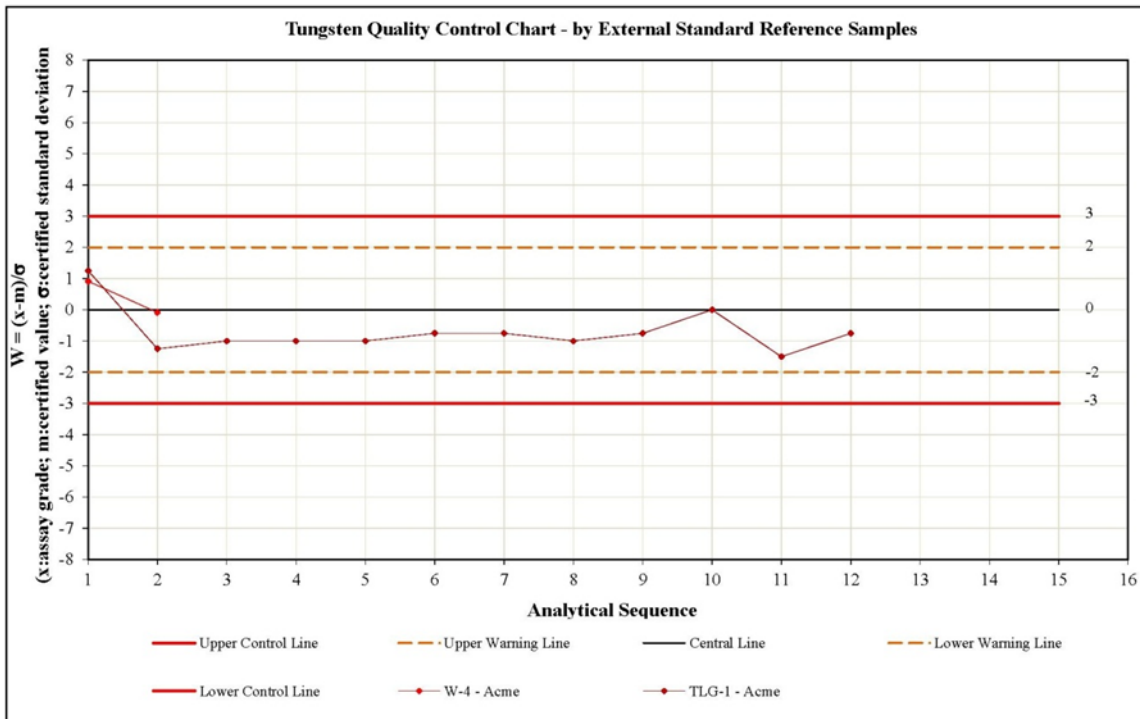
Table 12.3 Summary of 2010 QA/QC Samples				
Type	MS	ST	BL	Sum
¼ Core	18	2	1	21
Reject	25	2	0	27
Pulp	237	23	4	264
<b>ALL</b>	<b>280</b>	<b>27</b>	<b>5</b>	<b>312</b>

### Standards

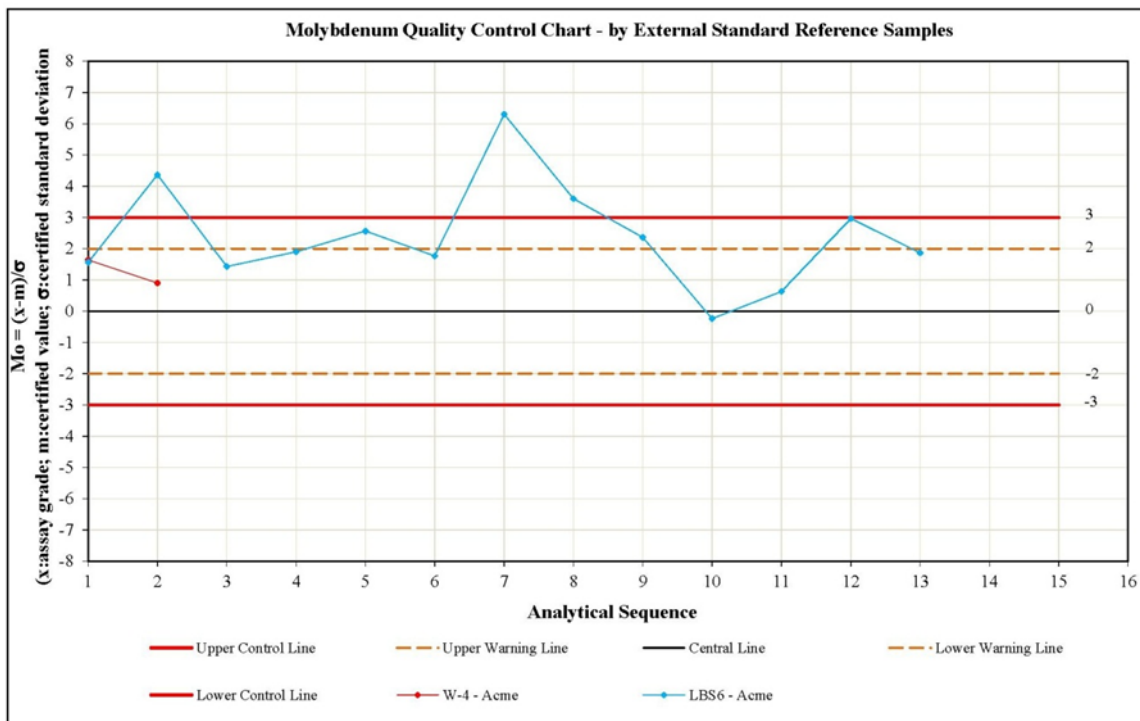
Three different assay standards were submitted as part of the due diligence program. Two commercially available standards W-4 and TLG-1 were submitted along with a proprietary copper-molybdenum standard. Table 12.4 lists the key parameters of these standards.

Table 12.4 Summary of 2010 QA/QC Samples				
Standard	Times Used	W (%)	Mo (%)	Cu (%)
W-4	2	0.366	0.11	0.139
Cu-Mo	13	-	0.0146	1.23
TLG-1	12	0.083	-	-

Figure 12.7 and Figure 12.8 show the assay results of these standards in the due diligence program. The assayed values of tungsten and molybdenum for W-4 and the assayed values of tungsten for TLG-1 are within the control limits. Three of the thirteen molybdenum values for the copper-molybdenum standard returned above of the upper control limit.



**Figure 12.7: Tungsten Quality Control Chart for Due Diligence**

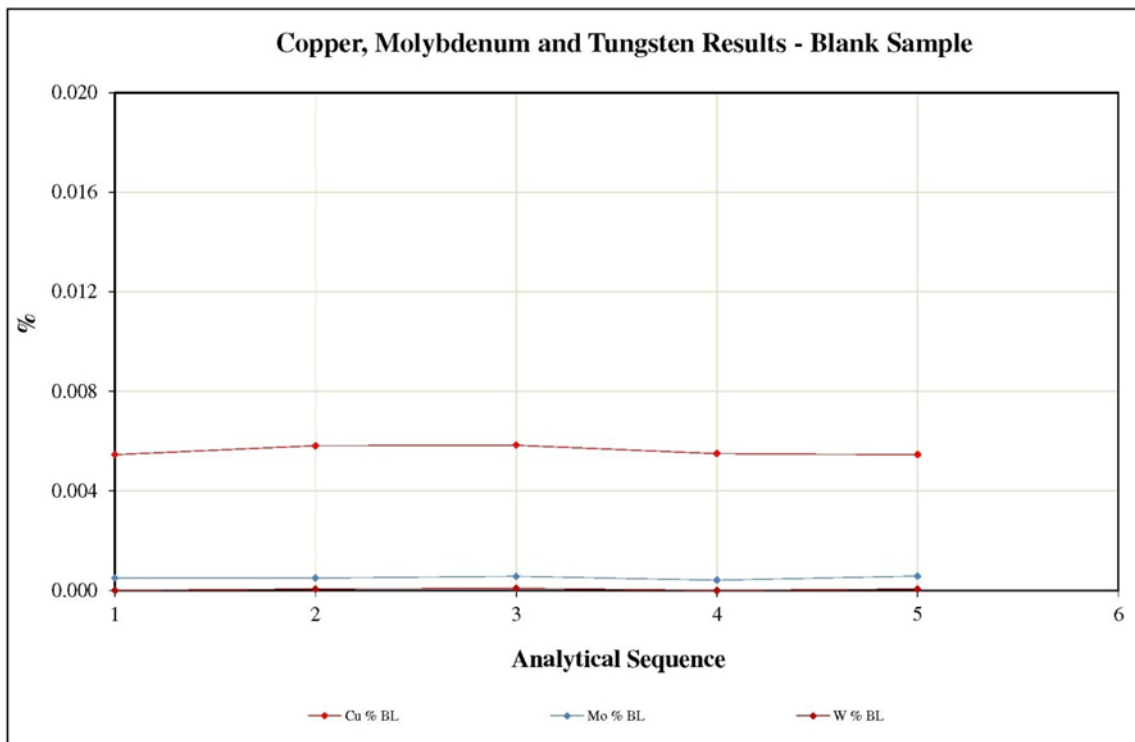


**Figure 12.8: Molybdenum Quality Control Chart for Due Diligence**

RPA noted that the molybdenum standards results showed a persistent positive bias. However, since these samples were not part of the resource estimation database, there was no concern with respect to the block grade interpolations.

Blanks

A total of five pulp blanks, comprising commercial blank BL-6, were included with the due diligence samples to monitor for potential contamination during sample preparation and assay. BL-6 is certified as a standard with very low (<0.01 ppm) precious metal content, however, as it had been analyzed extensively on another HDI project with low tungsten (0.6 ppm) and molybdenum (4.7 ppm) results, it was considered suitable for this purpose. Figure 12.9 shows the copper, molybdenum, and tungsten results. In Northcliff’s opinion, no contamination was apparent based on these results, and RPA concurred.

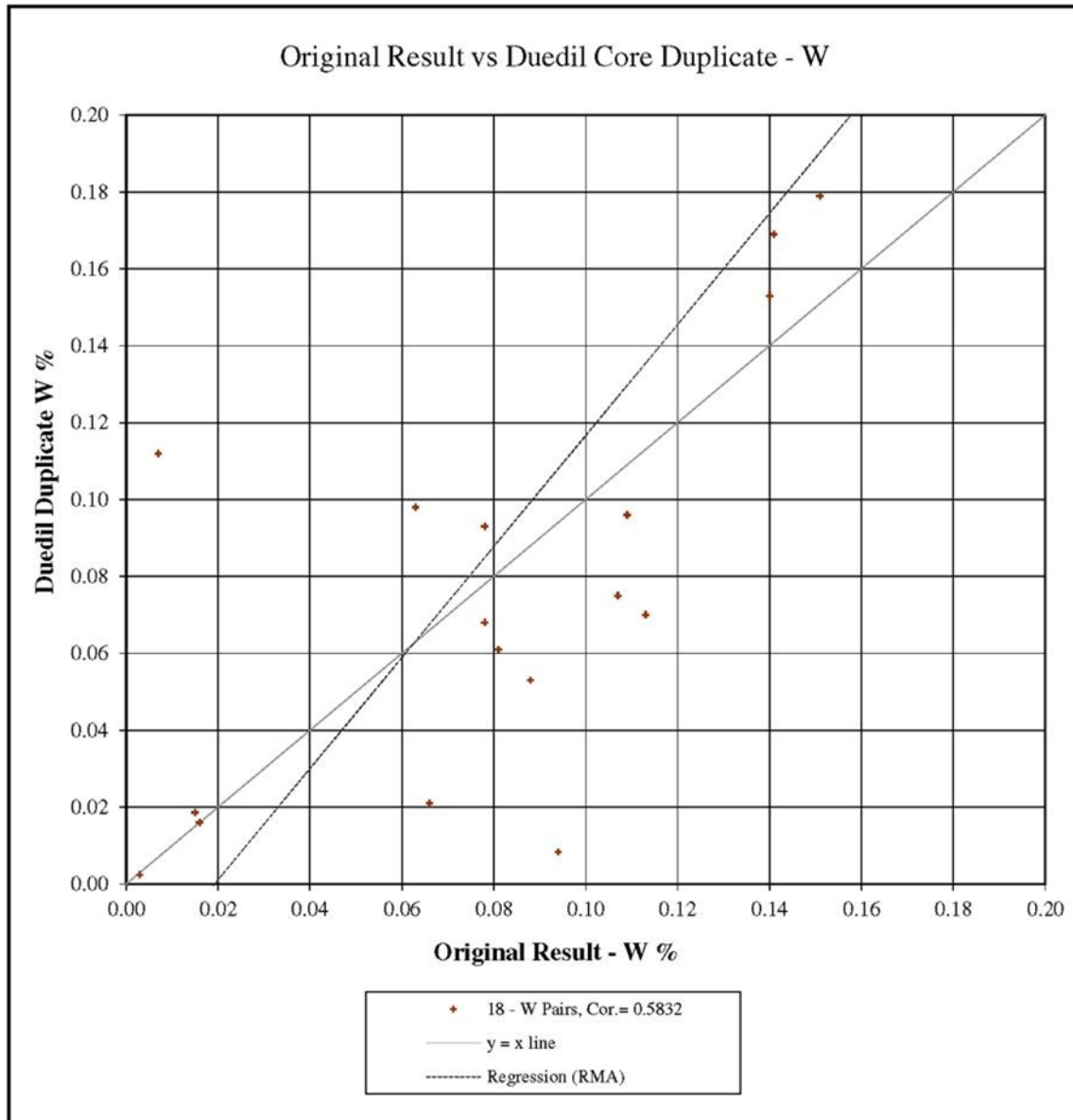


**Figure 12.9: Copper, Molybdenum and Tungsten Results of Blank Sample BL-6 for Due Diligence**

Duplicates

The 280 due diligence samples were compared with the original results obtained by Geodex. Figure 12.10 through Figure 12.15 are comparisons of the matched pairs duplicate results in a series of scatterplots by sample type. Generally speaking, the pulp and reject duplicates matched the original results reasonably well. Although the results of the quarter core duplicates do not match the original half core sample results very well, the presence of tungsten and molybdenum mineralization is confirmed in the samples. Northcliff personnel are of the opinion that it is likely that a significant portion of the difference between these samples is attributable to the fact that it is often not possible to obtain a truly

representative sample of vein-type mineralization in quarter core. In RPA's opinion, this was a plausible explanation.



**Figure 12.10: Tungsten Scatterplot of ¼ Core Duplicates for Due Diligence**

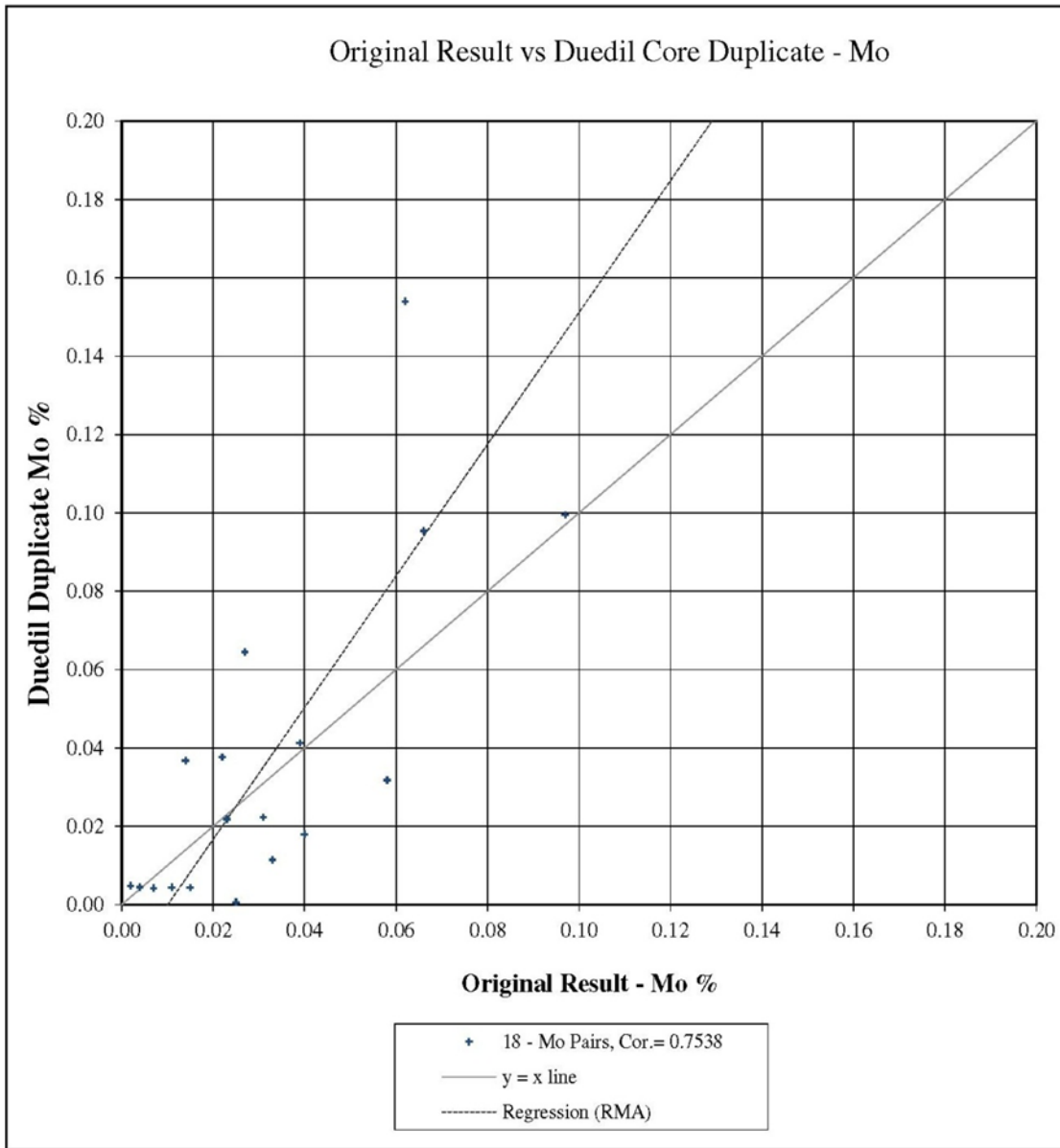


Figure 12.11: Molybdenum Scatterplot of ¼ Core Duplicates for Due Diligence



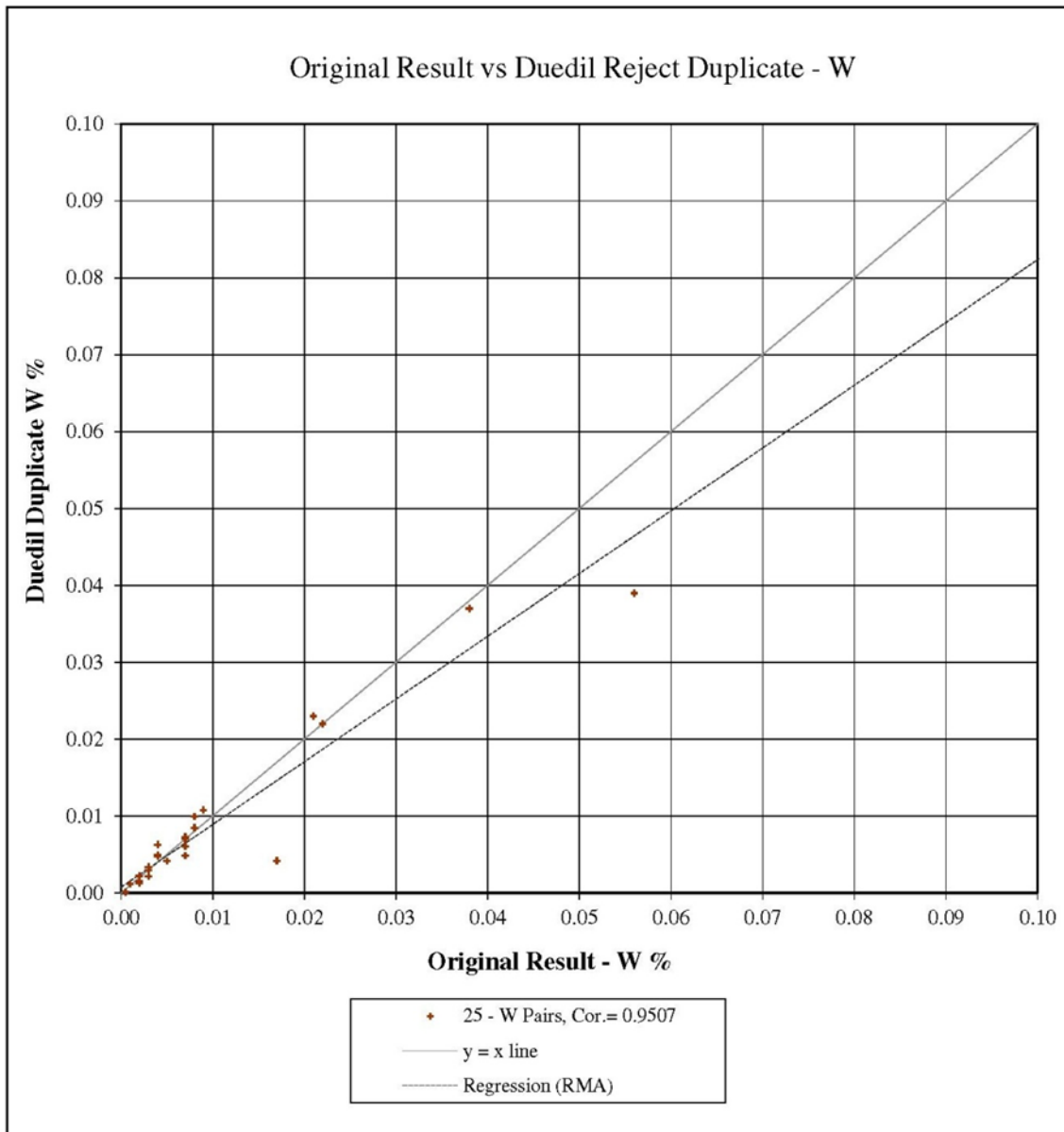


Figure 12.12: Tungsten Scatterplot of Reject Duplicates for Due Diligence

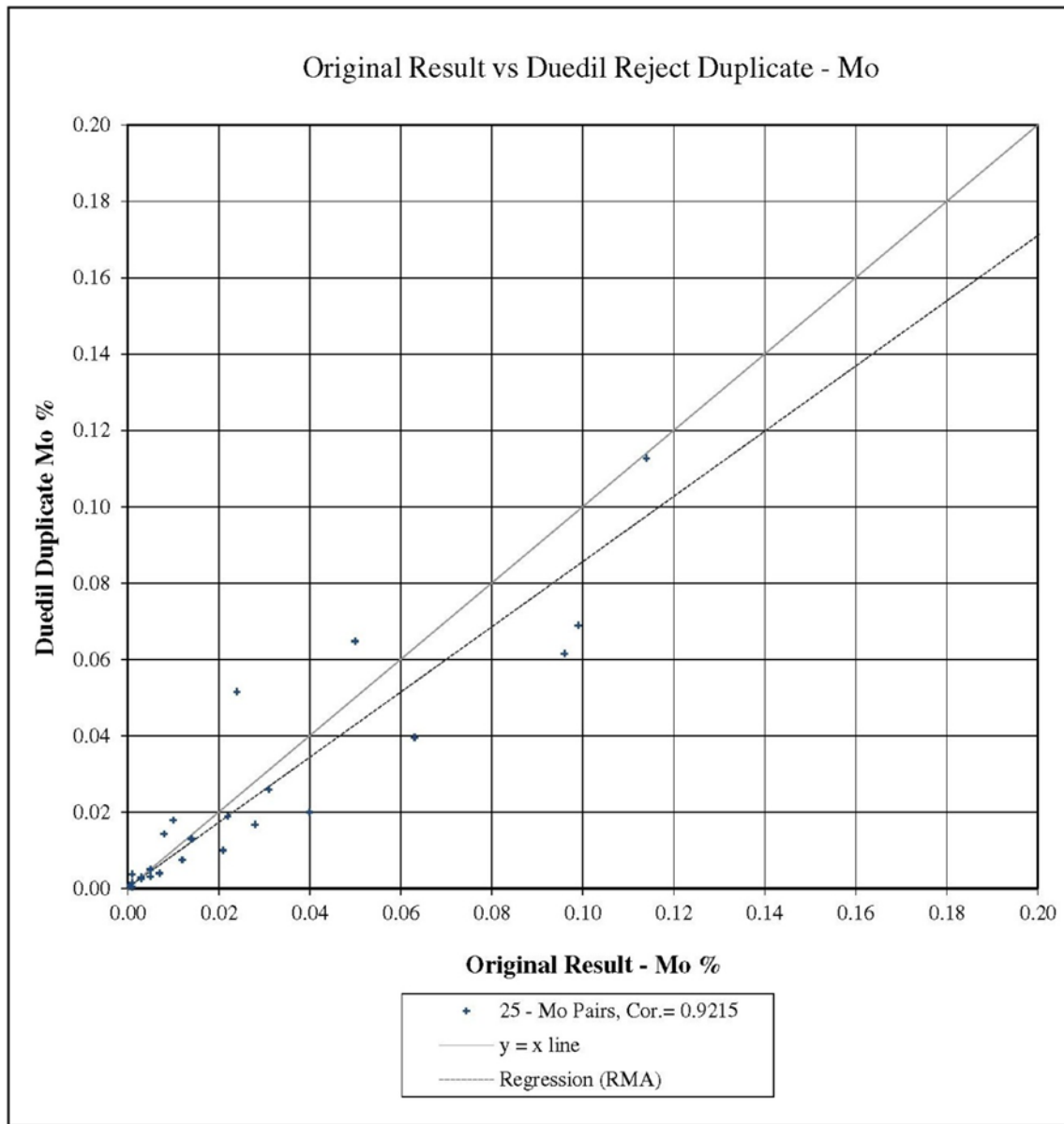
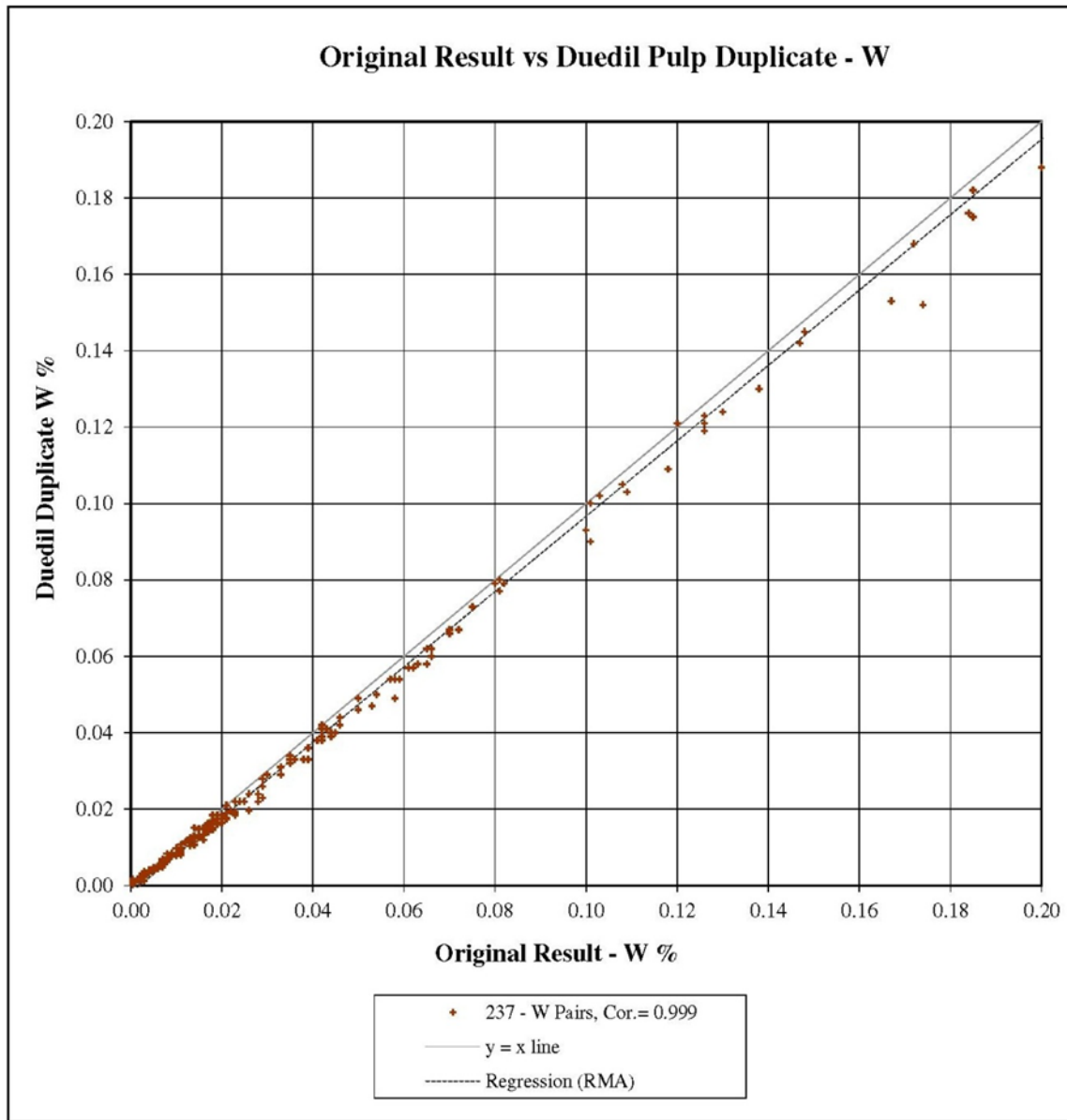
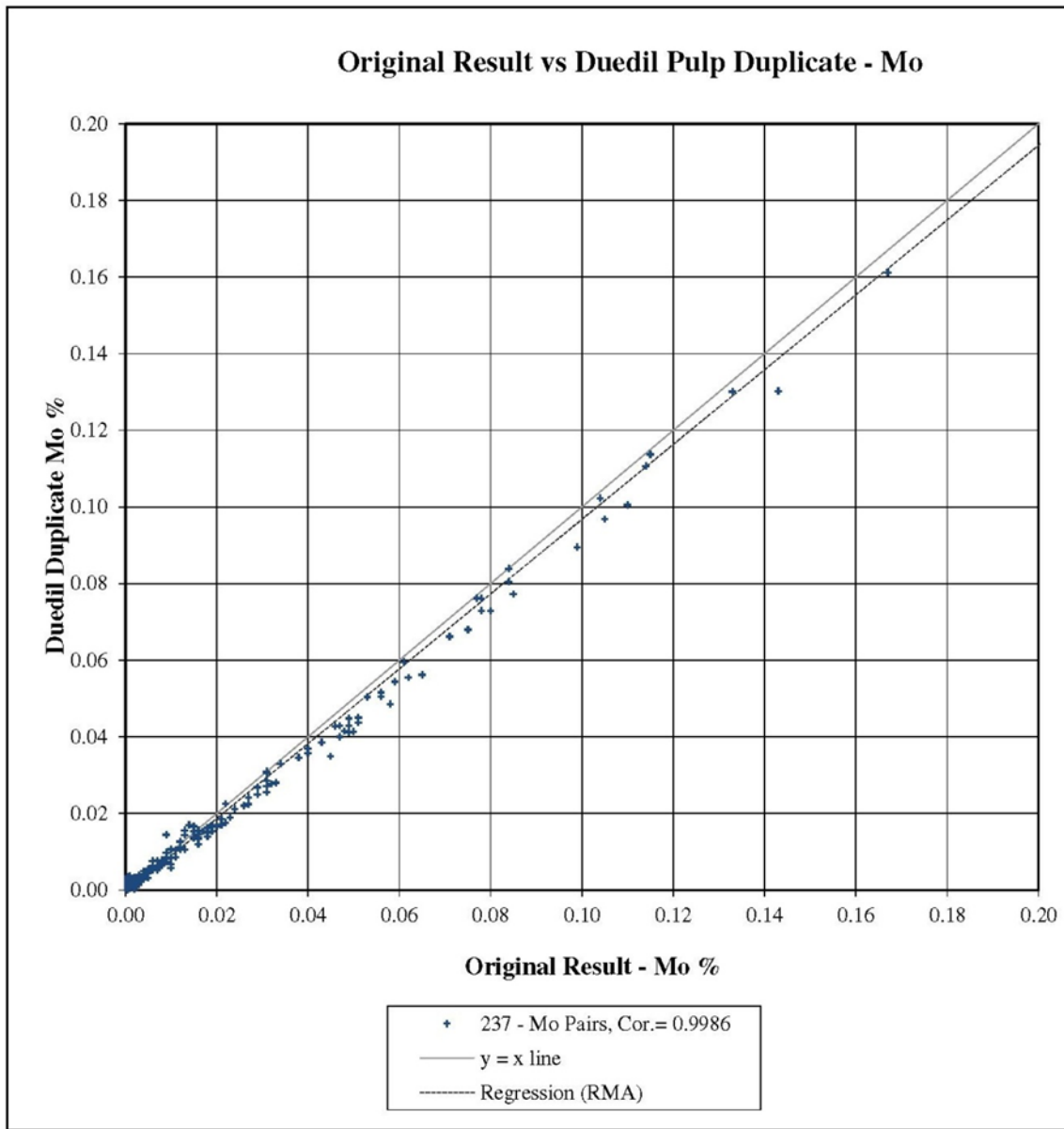


Figure 12.13: Molybdenum Scatterplot of Reject Duplicates for Due Diligence



**Figure 12.14: Tungsten Scatterplot of Pulp Duplicates for Due Diligence**



**Figure 12.15: Molybdenum Scatterplot of Pulp Duplicates for Due Diligence**

### 12.3 Northcliff Drilling (2010)

The commercial standards used by Geodex were reportedly the best available at the time. However, the tungsten and molybdenum grades of these standards are three to ten times higher than the median values for these elements in the area of interest. In addition, the relative standard deviation (a measurement of precision) for these standards is typically between 5% and 10% (see discussion in Section 10.0 on 2011 Northcliff drilling). In Northcliff's opinion, these standards were suitable for exploration, but it was recognized that they needed to be replaced for the next stage of project development. In order to implement a better QA/QC program for the ongoing drilling and engineering programs, six new matrix-matched certified reference materials (MMCRM) were produced. These new standards were made from

the coarse rejects of 2009 drill core from the Sisson property. Material was selected based on typical lithological types and tungsten grades. The six MMCRM's were prepared and certified by CDN Resource Laboratories Ltd. The ten laboratories participating in the round robin analysis are listed below:

- Activation Laboratories Ltd. (Actlabs)
- Acme Analytical Laboratories (Acme)
- ALS Minerals (ALS)
- Genalysis Laboratory Services (Genalysis)
- OMAC Laboratories Ltd. (OMAC)
- Becquerel Laboratories Inc. (Becquerel)
- SGS Canada Inc., Toronto (SGS Toronto)
- SGS Canada Inc., Vancouver (SGS Vancouver)
- AGAT Laboratories Ltd. (AGAT)
- Inspectorate Exploration and Mining Services Ltd. (Inspectorate)

Table 12.5 is a summary of the key values for the MMCRM.

<b>Table 12.5 Summary of Matrix-Matched Certified Reference Materials</b>					
<b>Standard</b>	<b>W ppm</b>	<b>Mo ppm</b>	<b>Cu ppm</b>	<b>Original Rock Type</b>	<b>Source</b>
SBRK-1	1,196	452	223	Volcanic & Sedimentary	Sisson Drill Core Rejects
SBRK-2	670	321	274		
SBRK-3	310	114	295		
SBRK-4	1,201	326	134	Gabbro	
SBRK-5	802	254	151		
SBRK-6	287	80	146		

In the 2010 drill program, a total of 113 external QA/QC samples were applied, including 49 standards, 52 duplicates, and 12 blanks (see Table 12.7). The standards and duplicates were inserted alternately at the frequency of one in every 20 mainstream samples. Blanks were inserted at the rate of two or three per drill hole.

### Standards

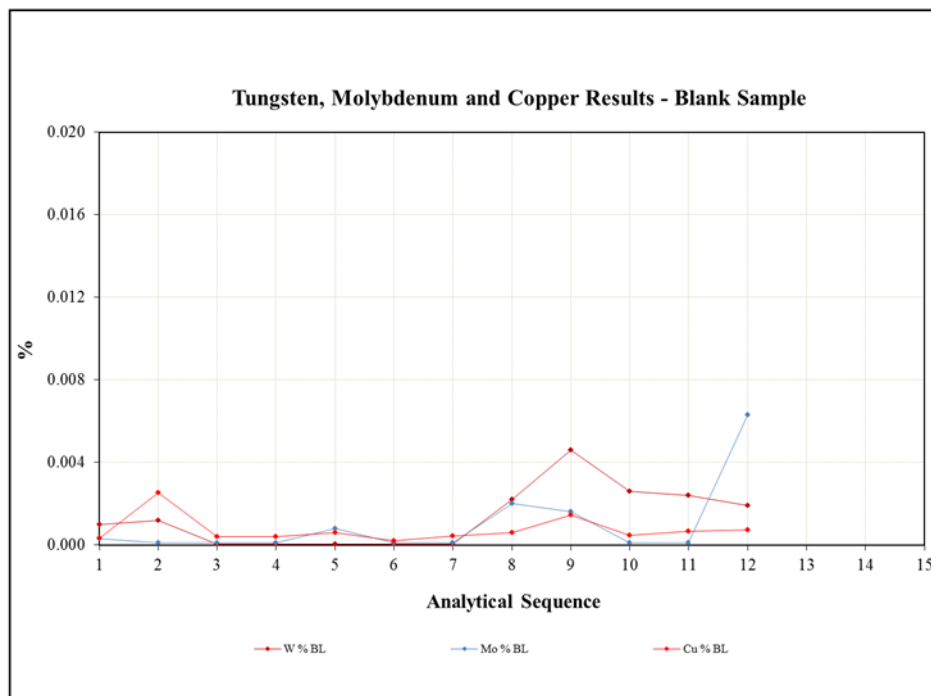
Northcliff implemented the new MMCRM along with the higher grade commercial tungsten molybdenum standard W-4 in the 2010 drill program. Table 12.6 is a summary of standards used to monitor in the 2010 program.

Table 12.6 Standard Materials Used In 2010 Drill Program	
Standard	Times Used
SBRK-1	3
SBRK-2	5
SBRK-3	8
SBRK-4	2
SBRK-5	2
SBRK-6	3
W-4	26

Figure 12.21 and Figure 12.22 show the results of the standards inserted and assayed for both the 2010 and 2011 programs.

Blanks

A total of 12 barren core blanks (granite) were applied to monitor the potential contamination for the 2010 drill program. Figure 12.16 shows copper, molybdenum, and tungsten results of these blanks. No obvious contamination was indicated based on these results.

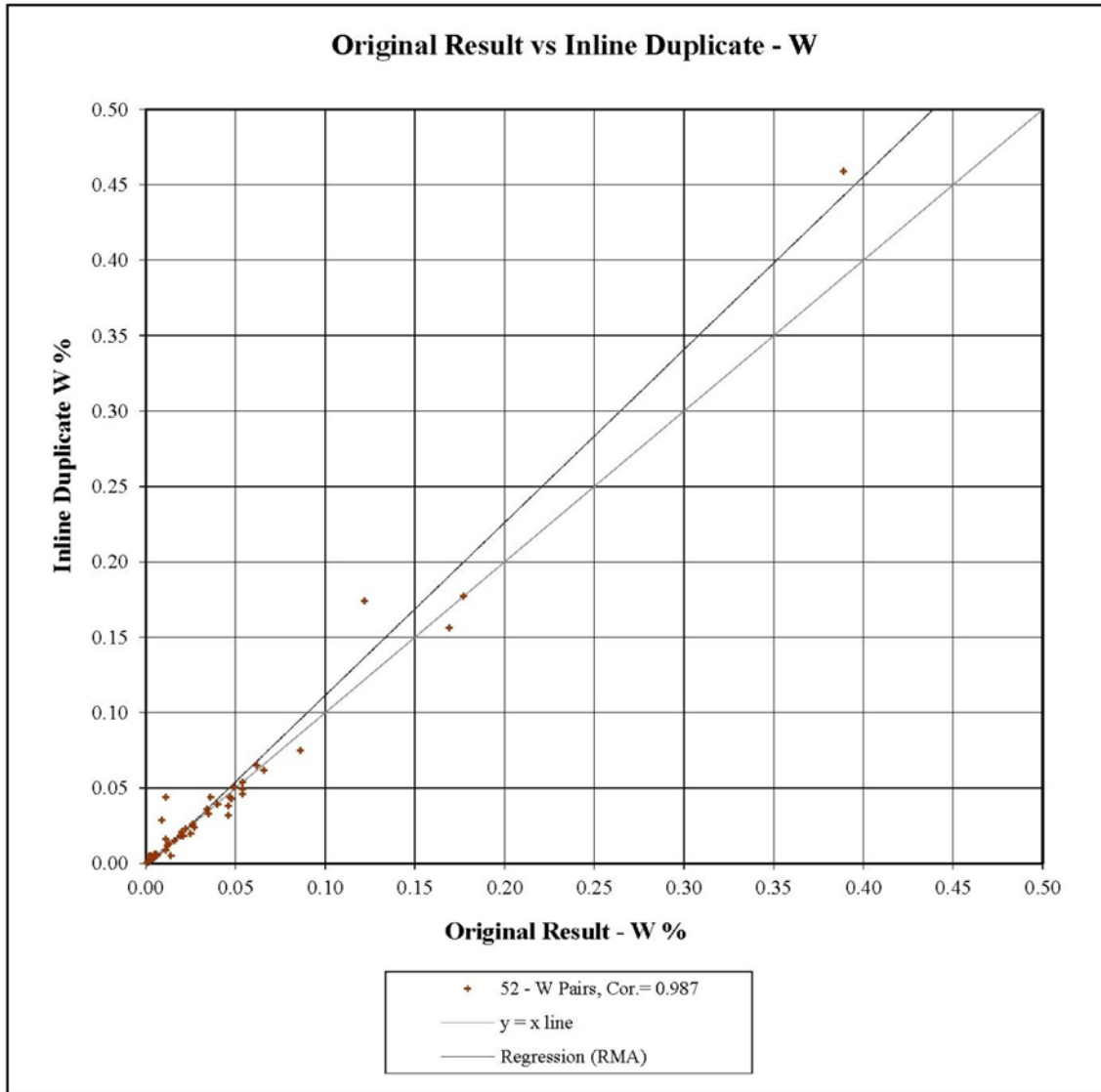


**Figure 12.16: Tungsten, Molybdenum and Copper Results of Blank Samples for 2010 Drill Program**

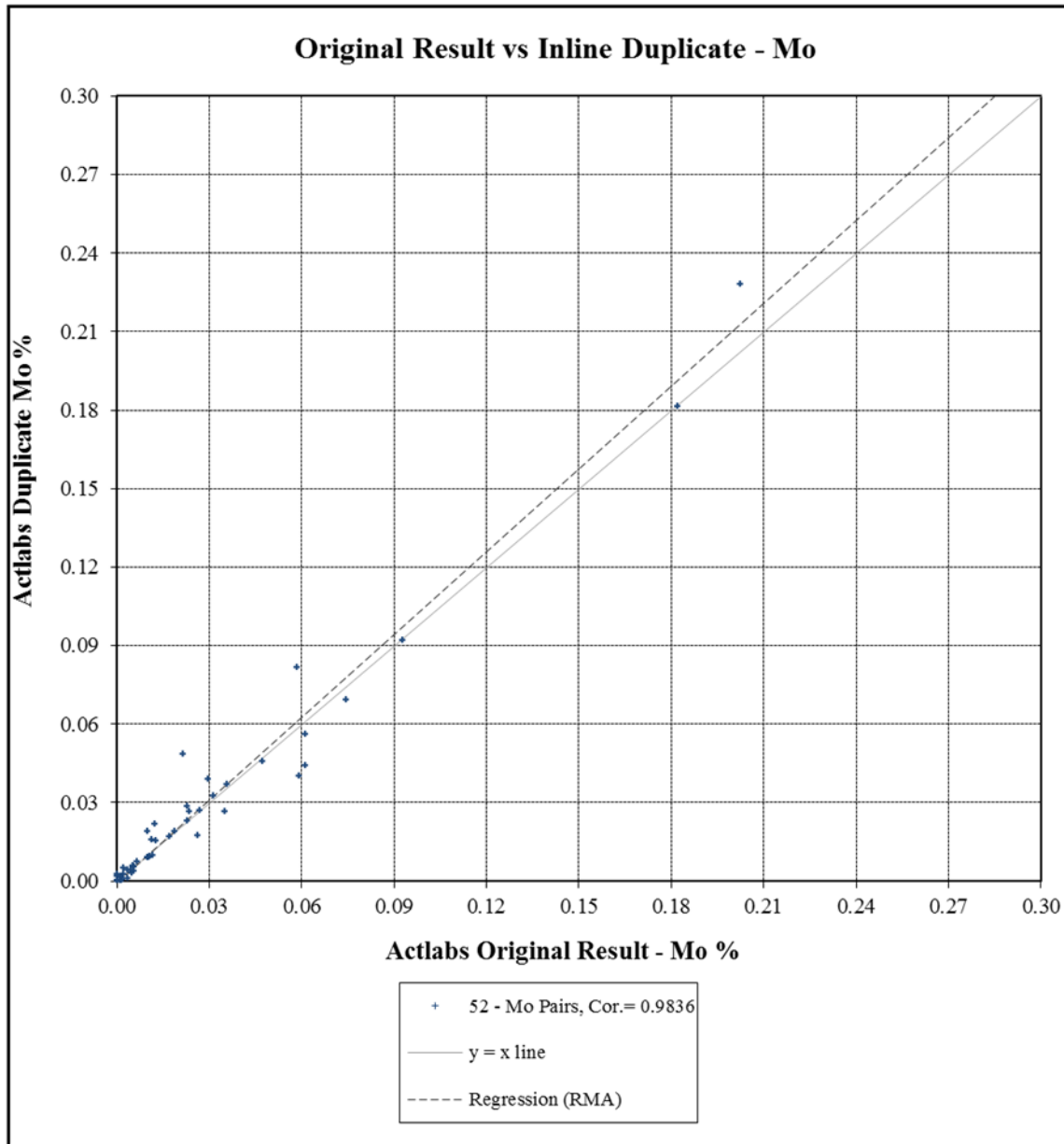


Duplicates

A total of 52 duplicates were applied to monitor the reproducibility of the primary assay laboratory (in-line duplicates) as well as the second check laboratory (inter-laboratory duplicates). In Northcliff's nomenclature, an in-line duplicate is a second split from the reject material, while the inter-laboratory duplicate is a split from the pulp. Figure 12.17 to Figure 12.20 are scatterplots of the duplicate pair results.



**Figure 12.17: Tungsten Scatterplot of In-Line Duplicates for 2010 Drill Program**



**Figure 12.18: Molybdenum Scatterplot of In-Line Duplicates for 2010 Drill Program**

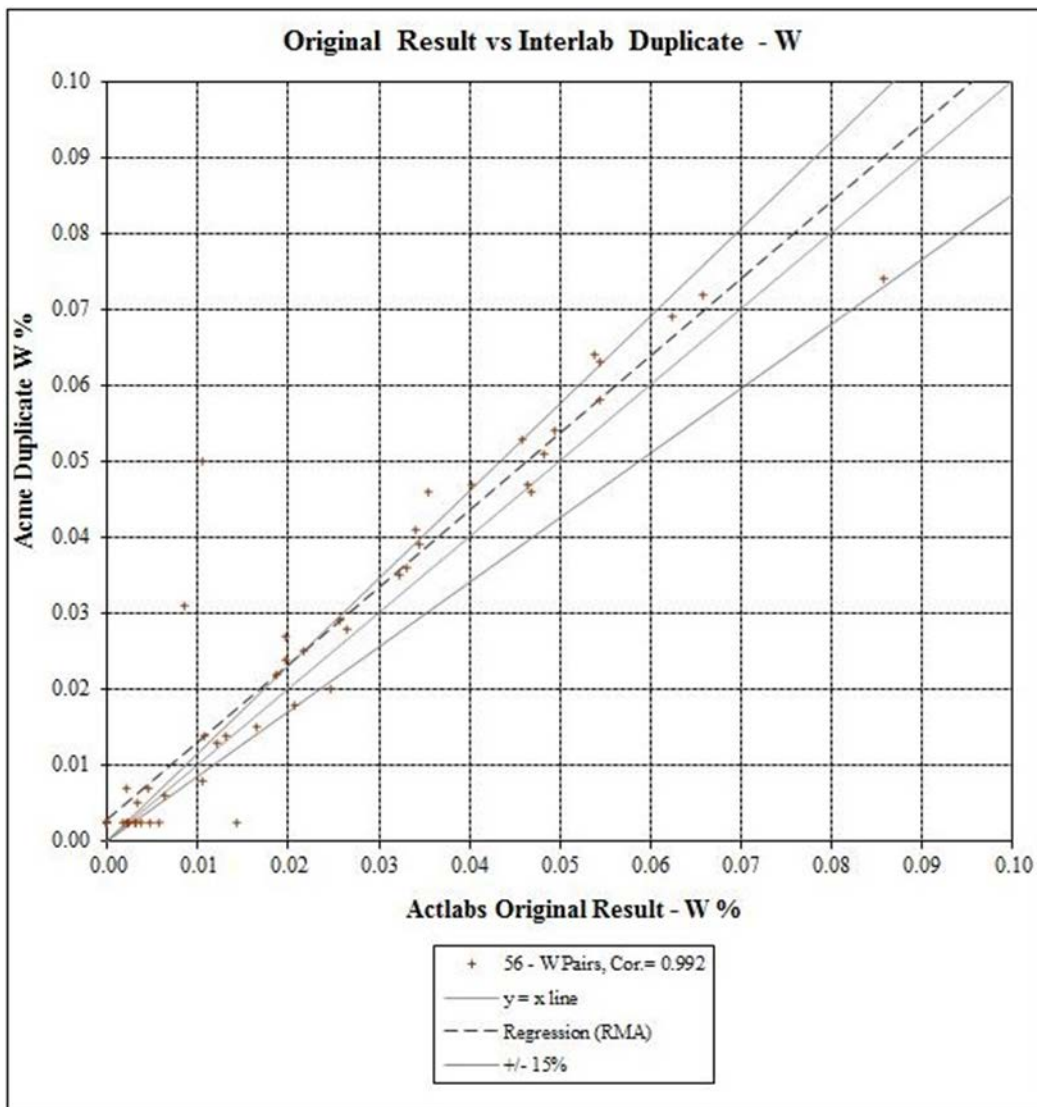
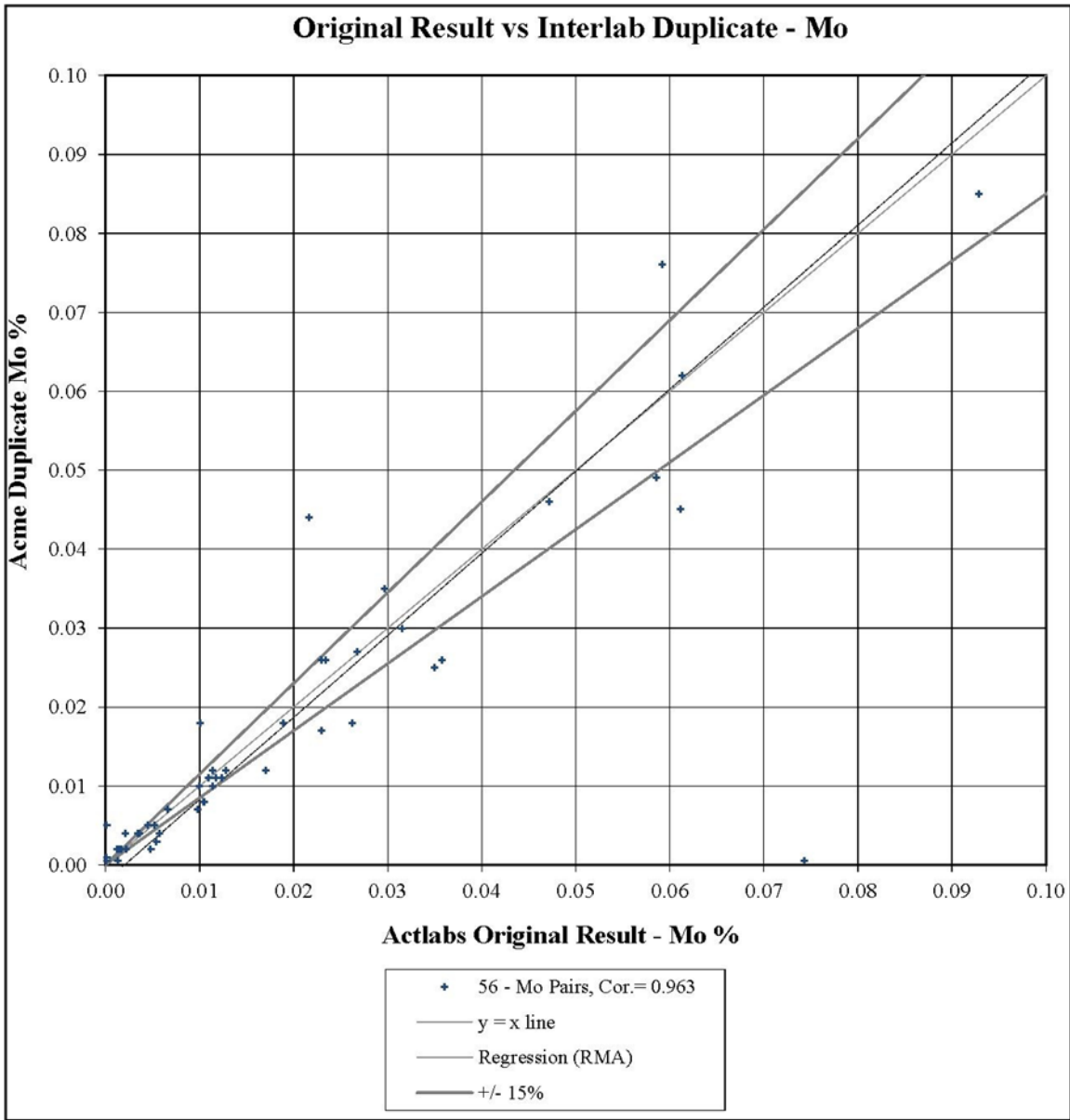


Figure 12.19: Tungsten Scatterplot of Inter-lab Duplicates for 2010 Drill Program



**Figure 12.20: Molybdenum Scatterplot of Inter-Lab Duplicates for 2010 Drill Program**

### 12.4 Northcliff Drill (2011)

In the 2011 drill program, a total of 337 external QA/QC samples were applied, including 149 standards, 144 duplicates, and 44 blanks (Table 12.7). The standards and duplicates were inserted alternately with the frequency of one in every 20 mainstream samples. Blanks were inserted at the rate of two or three for each drill hole.

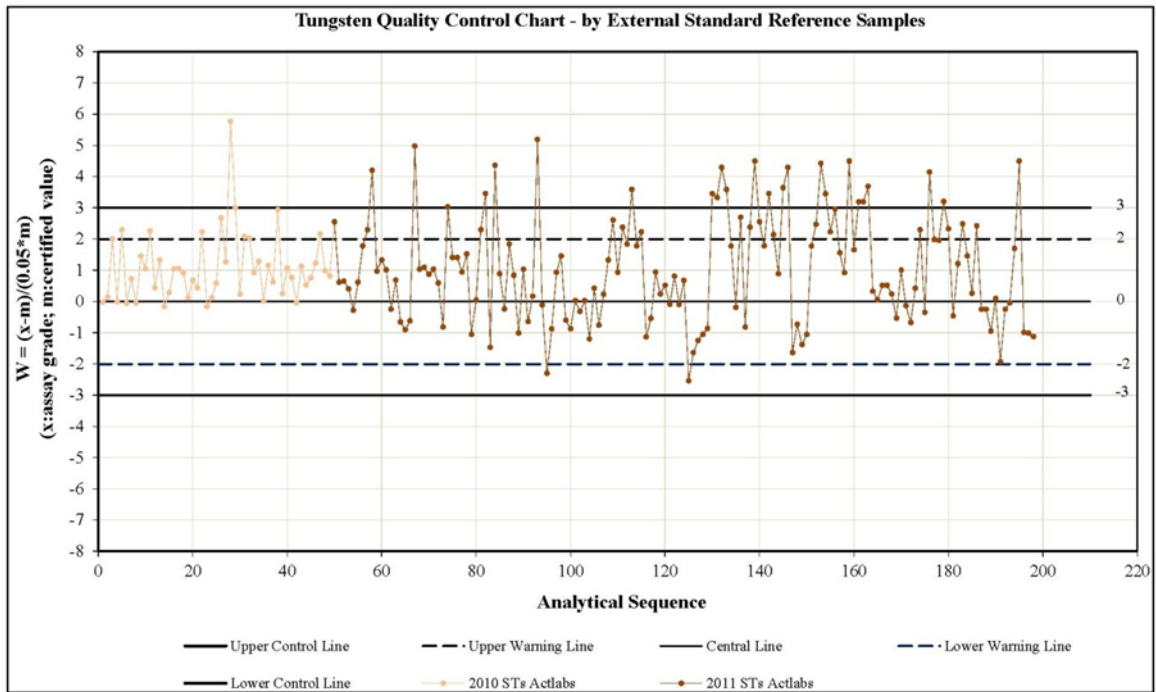
Table 12.7 QA/QC Summary for 2010 - 2011					
Year	MS	DP	ST	BL	ST%
2010	929	52	49	12	5
2011	2,768	144	149	44	5
<b>ALL</b>	<b>3,697</b>	<b>196</b>	<b>198</b>	<b>56</b>	<b>5</b>

### Standards

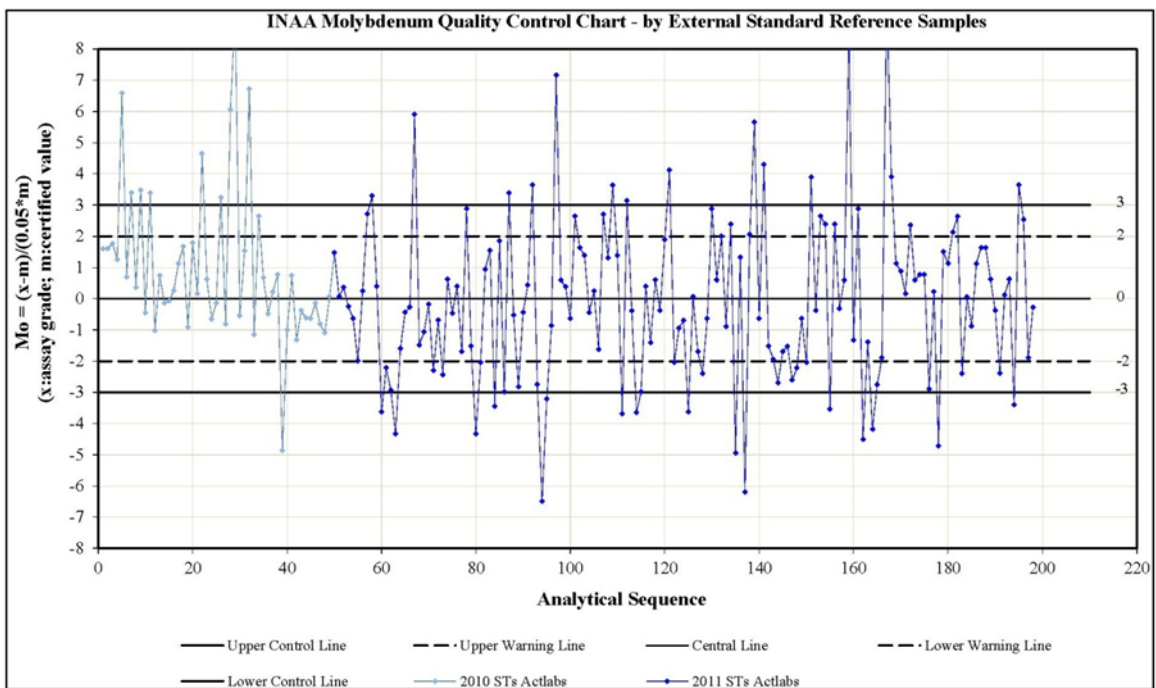
Table 12.8 is the summary of standards used to monitor the tungsten and molybdenum assays in the 2011 drill program.

Table 12.8 Standard Materials Used in 2011 Drill Program	
Standard	Times Used
SBRK-1	8
SBRK-2	21
SBRK-3	49
SBRK-4	4
SBRK-5	14
SBRK-6	45
W-4	8
<b>Overall</b>	<b>149</b>

Figure 12.21 and Figure 12.22 show the results of the standards inserted and assayed along with the regular samples.



**Figure 12.21: Tungsten Control Chart for 2010 & 2011 Drill Program**



**Figure 12.22: Molybdenum Control Chart for 2010 & 2011 Drill Program**

The warning and failure limits for the control diagrams are based on the relative standard deviation (RSD), which is the coefficient of variation (i.e., standard deviation divided by the mean) multiplied by 100%. The RSD values for the standards, as derived from the round robin assays, are shown in Figure 12.23 and



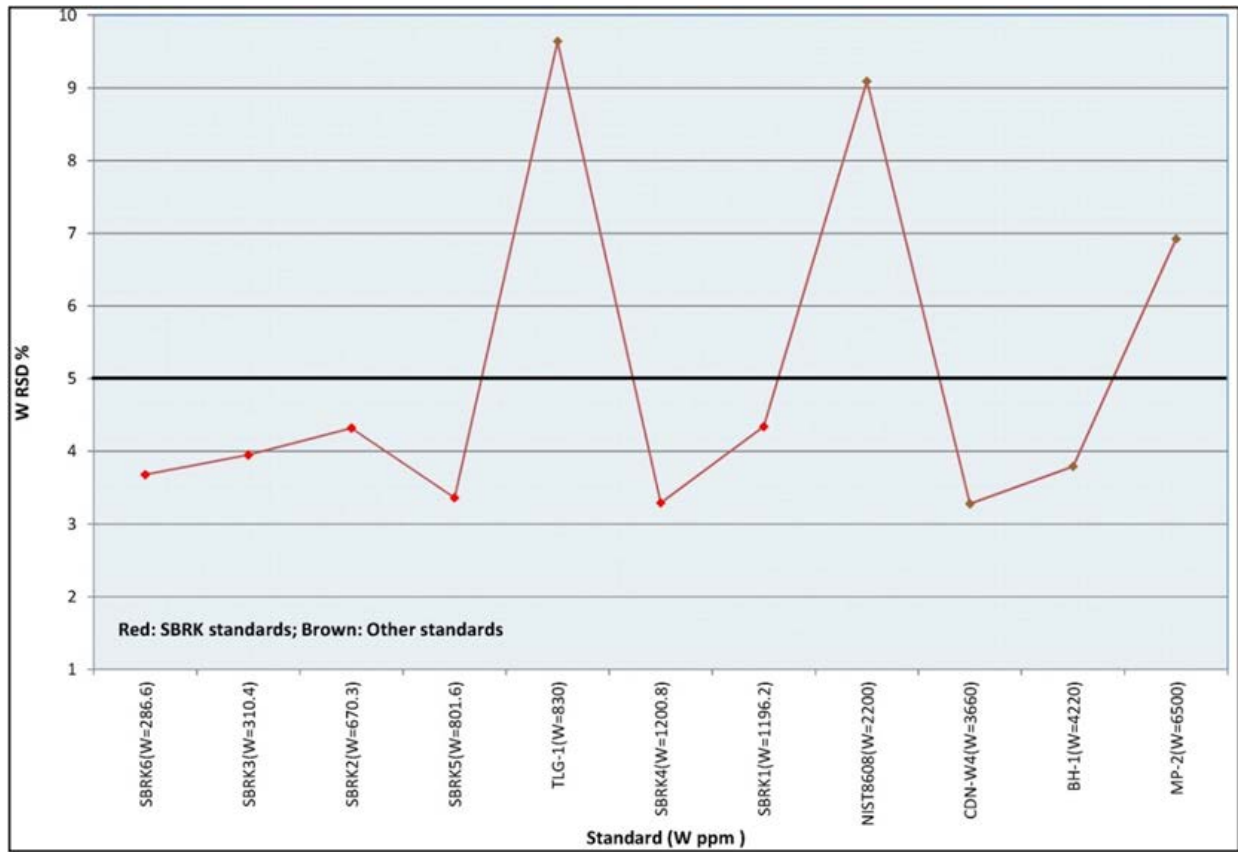
Figure 12.24. The tolerance limits (termed Recommended Control Rules by Northcliff personnel) applied to the standards assays are listed in Table 12.9.

Table 12.9 Control Limits Applied			
Element (commodity)	Standardized RSD	Warning limits	Control limits
Au (Pt, Pd)	5%	Mean*(1 ± 10%)	Mean*(1 ± 15%)
Mo, W	5%	Mean*(1 ± 10%)	Mean*(1 ± 15%)
Ag, Pb, Nb	4%	Mean*(1 ± 8%)	Mean*(1 ± 12%)
Cu, Zn	3%	Mean*(1 ± 6%)	Mean*(1 ± 9%)

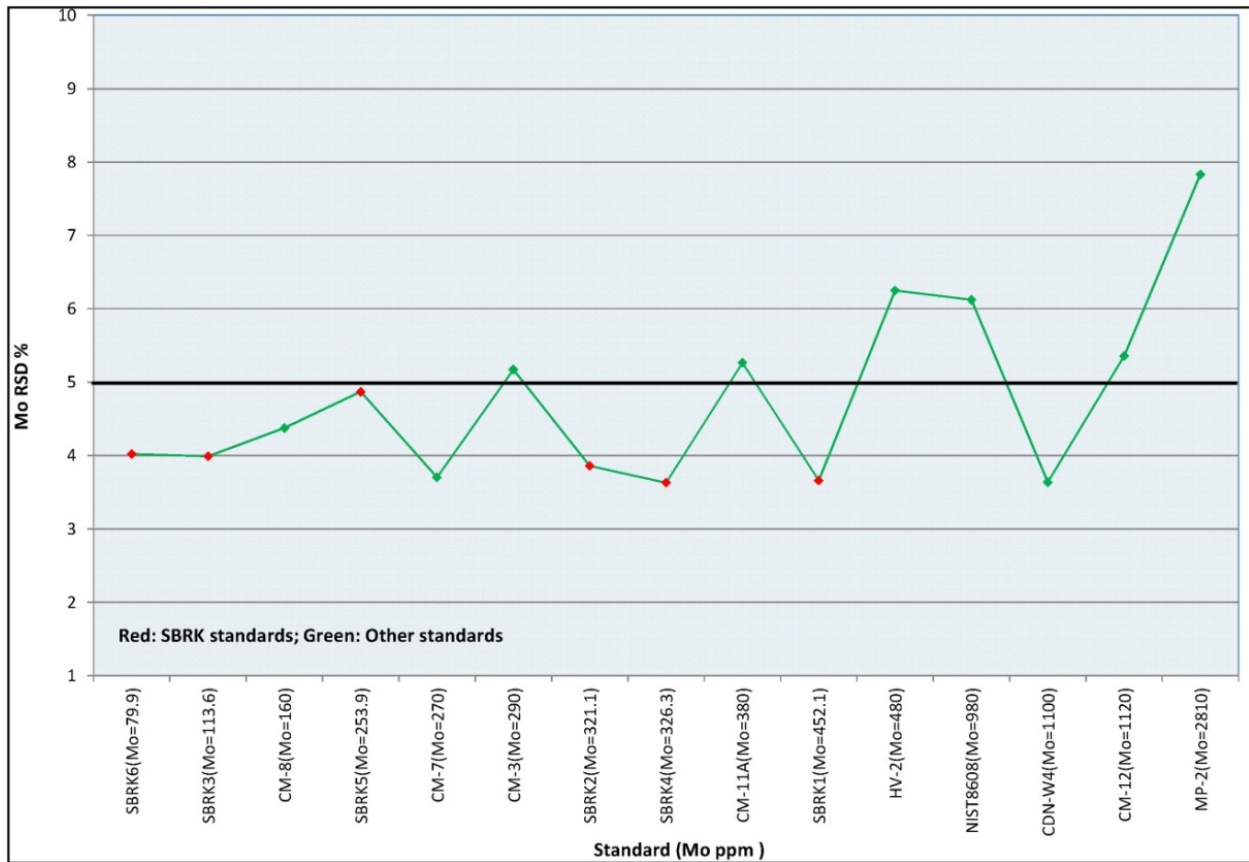
Northcliff personnel found that the MMCRM used in the 2010 and 2011 drilling programs on the Sisson Project had generally smaller RSDs than the commercial standards, which led to a higher failure rate than usual. In Figure 12.23, RSD values for the commercial standards TLG-1, NIST8608, and MP-2 are all significantly higher than for Northcliff's internal standards. This indicates that the precision with which the assay values for these standards are known is less than for the in-house standards. The net effect was that when the conventional  $\pm 2$  and  $\pm 3$  SD error limits were applied, an excessive number of failures were generated for the MMCRM. This resulted in a larger number of re-assays, significant time lost in resolving apparent issues, and increased costs without an attributable improvement in the overall assay database. In response, Northcliff adopted the control limits outlined in Table 12.9 for gauging assay performance.

In RPA's opinion, the approach taken by Northcliff in developing and using the in-house standards were reasonable. The grade ranges for the samples at Sisson, including the MMCRM, were relatively low, which would result in higher variance in assay results. According to RPA, this tendency for assay precision to deteriorate at lower grades is quite common and well documented.

RPA also noted that there was some evidence of a slight positive bias in the results for tungsten over certain periods of time, which should be investigated further. The molybdenum results demonstrated a fairly high degree of scatter, but there did not appear to be any significant bias.



**Figure 12.23: Tungsten RSD (%) of Certified Standards**

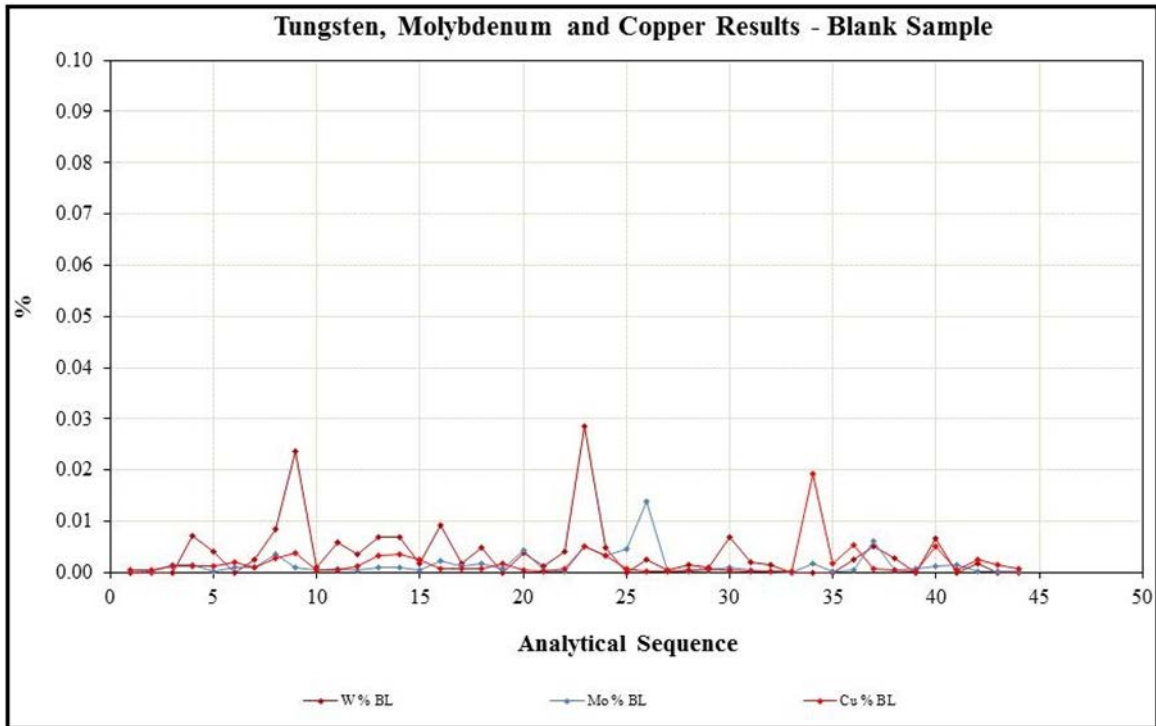


**Figure 12.24: Molybdenum RSD (%) of Certified Standards**

Blanks

A total of 44 barren course (granite) blanks were applied to monitor the potential contamination for the 2011 drill program. Figure 12.25 shows tungsten, molybdenum, and copper results of these blanks. No significant contamination was found based on these results.

In RPA's opinion, the blanks results showed an acceptable failure rate.

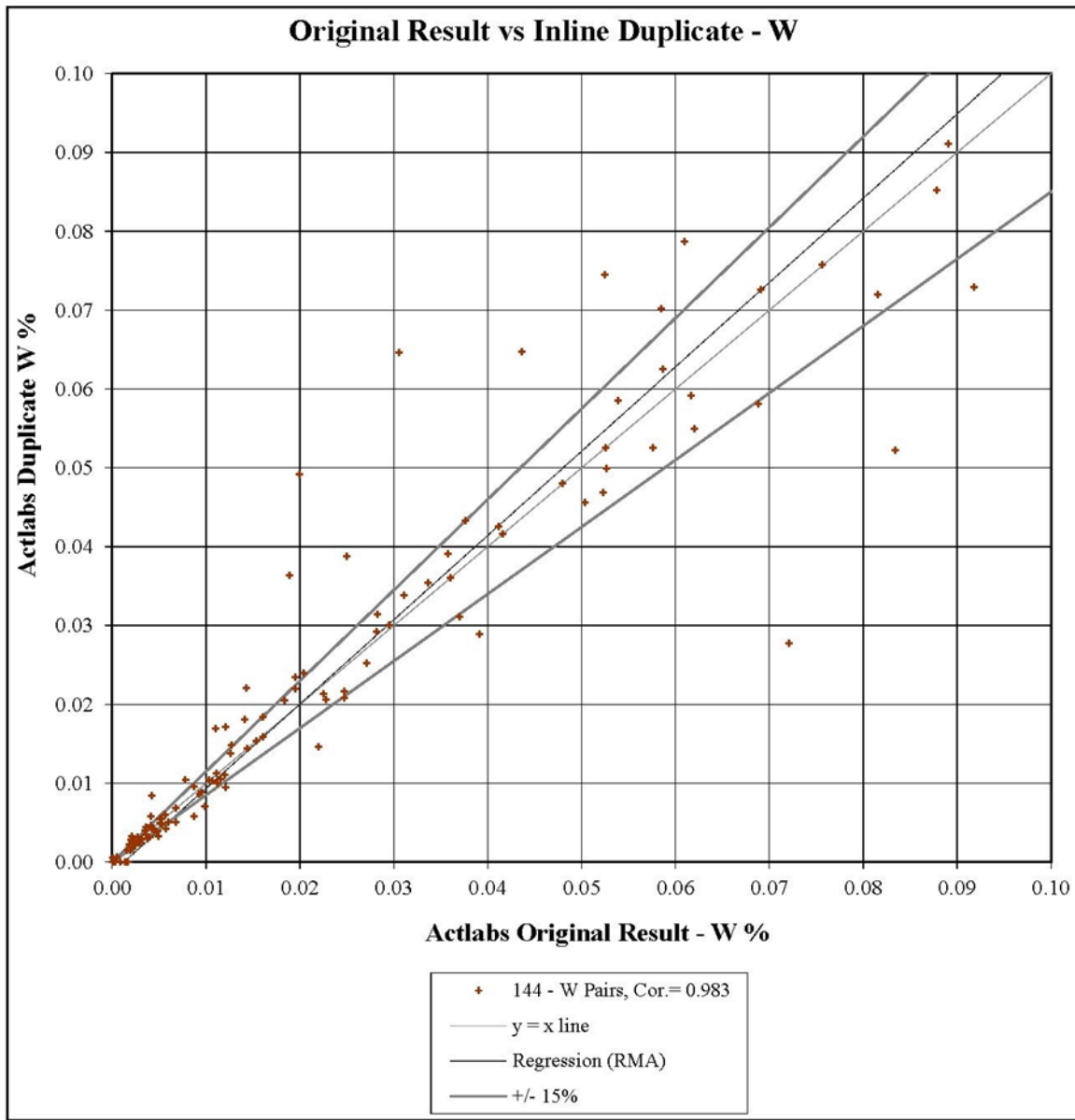


**Figure 12.25: Tungsten, Molybdenum and Copper Results of Blank Samples for 2011 Drilling Program**

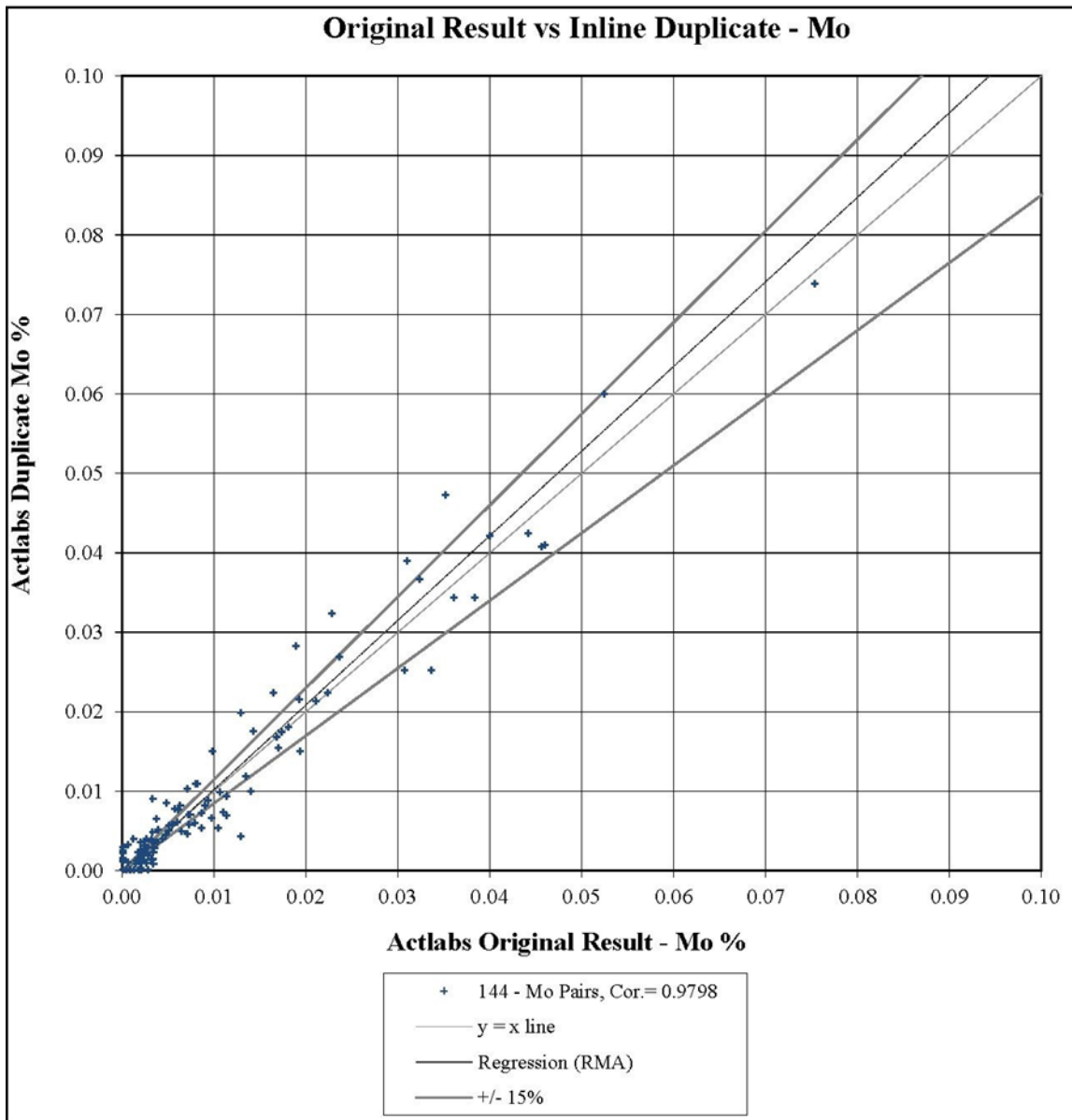
Duplicates

A total of 144 duplicates were applied to monitor the repeatability of the primary assay laboratory (in-line duplicates) as well as the reproducibility of the second check laboratory (inter-laboratory duplicates). Figure 12.26 to Figure 12.29 are scatterplots of the related pairs.

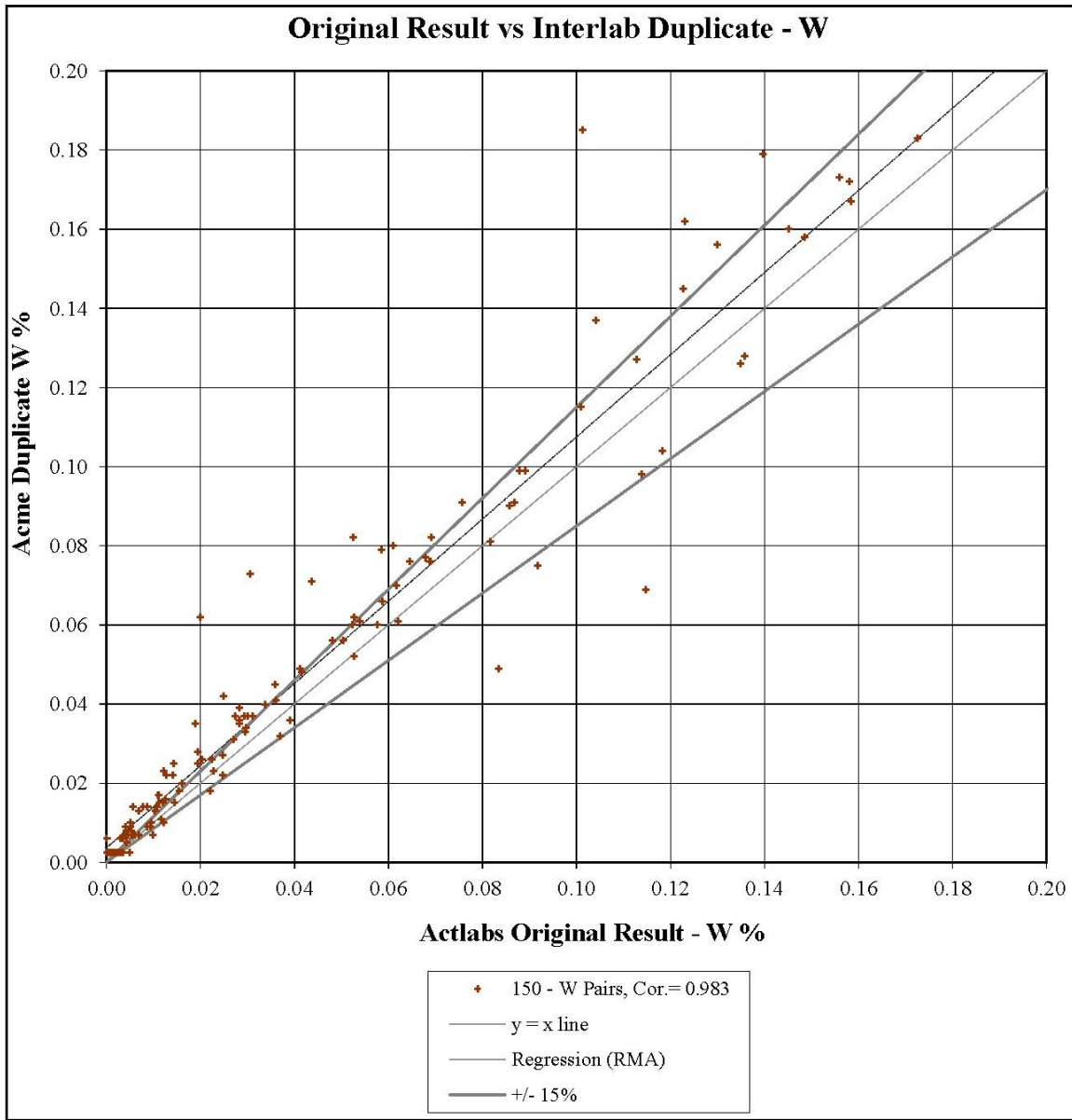
In RPA's opinion, the duplicate results demonstrated a somewhat increased level of scatter which was perhaps related to the low grade issue. The in-line duplicates did not appear to be biased. The inter-laboratory duplicates showed some evidence of bias which, in RPA's opinion, was probably due to the use of different assay techniques.



**Figure 12.26: Tungsten Scatterplot of In-Line Duplicates for 2011 Drill Program**

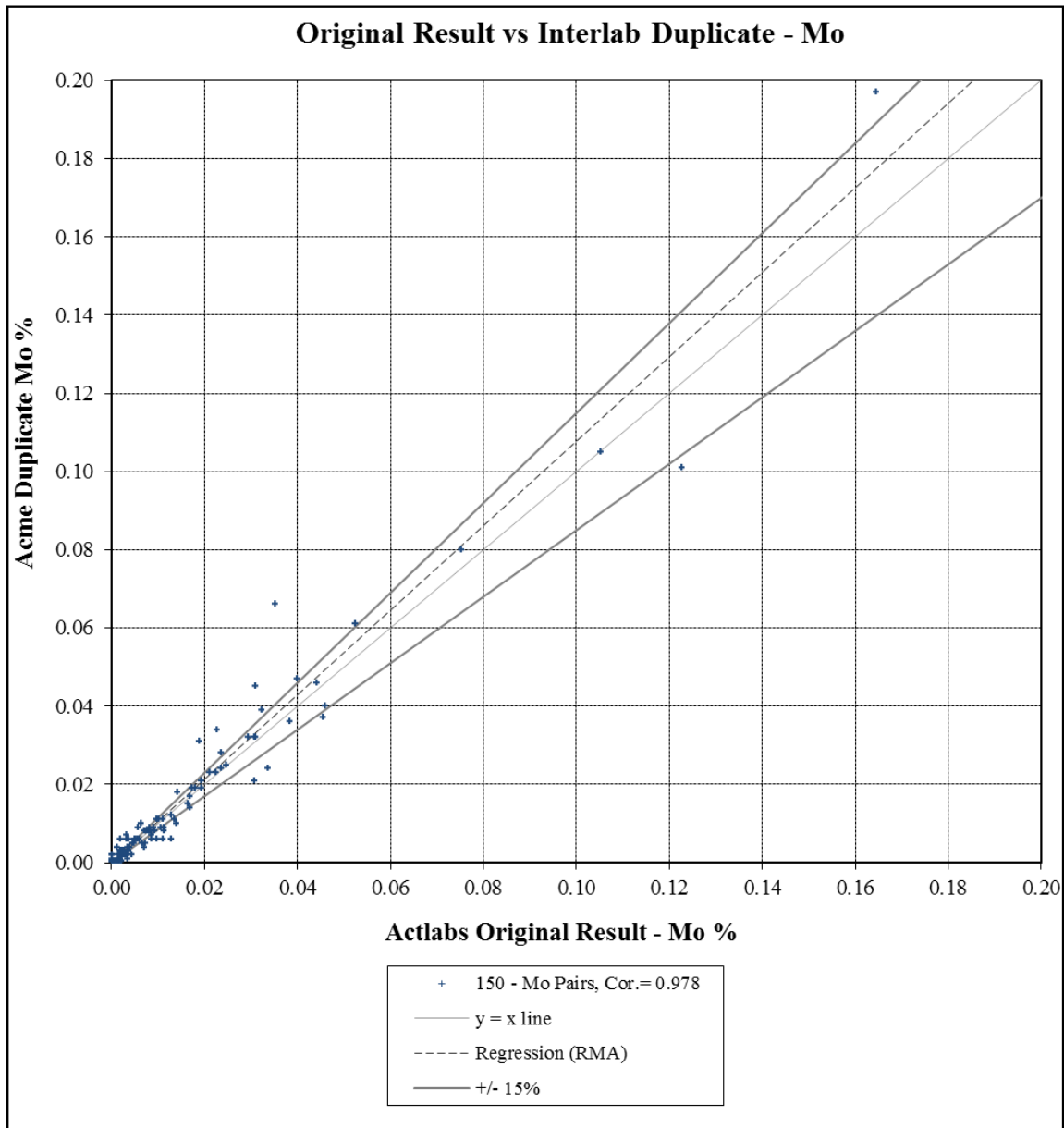


**Figure 12.27: Molybdenum Scatterplot of In-Line Duplicates for 2011 Drill Program**



**Figure 12.28: Tungsten Scatterplot of Inter-Lab Duplicates for 2011 Drill Program**





**Figure 12.29: Molybdenum Scatterplot of Inter-Lab Duplicates for 2011 Drill Program**

### 12.5 Data Collection and Management

All the drill logs and the assay results from 2011 and previous years were compiled in a SQL server into data tables which are compatible with Microsoft® Access relational database. Drill hole logs were entered into notebook computers running the Access data entry module for the Sisson Project in the core logging area at the site. The core logging computers were synchronized on a regular basis with the master site entry database.

Core photographs were transferred to the site geology office on a daily basis. In the geology office, the logs were printed, reviewed, and validated and initial corrections made. The site data were transmitted to the Vancouver office of HDI on a weekly basis where the logging data were imported into the master drill hole database and merged with digital assay results provided by the analytical laboratories. A further printing, validation and verification step followed after this import.

Updates and corrections to the field geological and sampling data were performed at the site office as the project progressed. This was done so that the revised information was reflected in the primary database in the Vancouver head office after the regular weekly transmittals were received and imported. Updates and corrections to the analytical data within the database were made in the Vancouver office. Compiled geological and analytical data were exported from the primary database for use: at site, in resource modeling, and by other users and consultants.

Sisson Project data are processed so that they could be assessed with respect to ongoing requirements for timely disclosure of material information by management. In this regard, compiled drill data and assay results are made available to management, the technical team, and project consultants advancing the project immediately after the initial error trapping and analytical QA/QC appraisal processes are completed, provided there are no significant concerns. The data are then subjected to more extensive, long-term validation, verification, QA/QC, and error correction processes. The findings of these long-term reviews are assessed as to their impact on previous disclosures and the necessity for further disclosure if there is a significant material change.

## 12.6 Verification

### Verification by Mercator

Government assessment reports and internal Geodex files consisting of core sample records, lithologic logs, laboratory reports, and associated drill hole and survey information for all holes used in the Sisson resource estimates were reportedly reviewed by Mercator.

### Due Diligence Verification by Northcliff

Northcliff reports that it conducted independent reviews of much of the analytical data from past work on the Sisson property as part of the due diligence process. This work included:

- Review of historical drilling, sampling, and assay results.
- Review of drilling, sampling, and assay results provided by Geodex.
- Review of certificates of analysis obtained directly from the analytical laboratories.
- Due diligence sampling and reanalysis of materials from the 2007, 2008, and 2009 Geodex drill programs including:
  - 18 quarter core samples
  - 25 coarse reject samples
  - 237 pulps samples
- Review of Geodex analytical quality control data from 2006 to 2009.

### Verification by Northcliff

The 2010 and 2011 drill hole geological and sample data were collected and digitally entered by site geological and technical personnel and sent to the Vancouver office of HDI on a weekly basis. In Vancouver, the digital database was compiled, merged with the analytical results, and reviewed for QA/QC purposes. Verification and validation took place at the site and Vancouver. At the site, the technical personnel responsible reviewed the digitally entered geology, sample and field log data.

At HDI Vancouver, the compiled data from the header, survey, assay, geology, and geotechnical tables were validated for missing, overlapping or duplicated intervals or duplicated sample numbers, and for matching drill hole lengths in each table. Drill hole collars and traces were reviewed in plan and sectional views by a geologist as a visual check on the validity of the location information. As the analytical data were returned from the laboratory, they were merged with the sample logs and printed out.

#### **12.6.1 RPA Verification**

##### Independent Samples

RPA collected eight quartered-core samples from drill core stored at the Northcliff logging facility. These samples were selected more or less randomly, split under RPA's supervision, and kept in the custody of RPA until delivered to a bonded carrier for shipment. The samples were sent to the SGS Canada Inc. assay laboratory in Vancouver, British Columbia, where they were analyzed for tungsten and molybdenum. The analytical method used was ICP after  $\text{Na}_2\text{O}_2$  fusion. Table 12.10 compares the results obtained with the original assays in the Northcliff database.

<b>Table 12.10 Verification Sample Results</b>						
<b>Hole ID</b>	<b>Original No.</b>	<b>W (ppm)</b>	<b>Mo (ppm)</b>	<b>RPA No.</b>	<b>W (ppm)</b>	<b>Mo (ppm)</b>
SB11-026	919572	442	816	921567	320	50
SB11-026	919579	1430	85	921568	180	<10
SB10-004	920202	73	1035	921569	90	860
SB10-004	920223	686	487	921570	260	340
SB11-034	920487	915	130	921571	1020	400
SB11-034	920499	435	610	921572	570	1040
SB10-003	923894	29	728	921573	70	1180
SB10-003	923899	1004	324	921574	530	160

In RPA's opinion, the resampled assays compare reasonably well with Northcliff's results, particularly considering the difference in assay methods used.

##### Database Validation

RPA validated the drilling database using the utilities provided in GEMS. No errors were found.

RPA also compared assay results from the original laboratory certificates to the values stored in the Northcliff database for approximately 12% of the holes used in the resource estimate. Thirty-two holes were selected from those drilled by either Geodex or Northcliff, and the digital laboratory certificates were acquired and compiled into a database. Some 3,532 samples out of the total of 32,319 (note that not all were included in the estimate) were checked in this manner. There were no discrepancies found.

## **12.7 Discussion and Conclusions**

In RPA's opinion, Northcliff demonstrated an adequate level of validation and verification of the database contents. The level of database validation for earlier operators, particularly Kidd Creek, was not well documented, but the proportion of the data collected during this regime was small. Geodex appeared to have also applied a reasonable level of validation and verification effort to the database. Northcliff had been particularly rigorous in external monitoring of assay QA/QC. In RPA's opinion, the database was reasonably free of errors and suitable for use in estimation of Mineral Resources.

## **13.0 Mineral Process and Metallurgical Testing**

Over the years, a considerable amount of scoping, preliminary and feasibility level mineralogical and metallurgical investigations have been conducted on samples from the Sisson property.

In 2006, Micron Geological Ltd. of North Vancouver, BC carried out a petrographic examination of crushed samples from a drill hole in order to gain an indication of how mineralization behaves metallurgically for an upcoming preliminary testing by Process Research Associates (PRA) Ltd, Richmond, BC. The subsequent metallurgical test program was brief, consisting of three scoping tests only.

In 2008, a scoping level metallurgical test program on two composites was conducted by SGS Lakefield Research Ltd, of Lakefield, Ontario. The tests focused on mineralogy, grinding characteristics, gravity separation and flotation separation techniques for scheelite and molybdenum (Mo or 'moly'). Pre-concentration techniques were also evaluated in this program.

In late 2008 and early 2009, six samples from the Sisson deposit were tested at SGS Canada Inc. of Vancouver, BC. The test plan included pre-concentration tests, flotation tests and release analyses. However, the program was interrupted in early 2009 before its completion due to global economic uncertainties impacting Geodex's operating funding at that time.

In 2010, a series of pre-concentration tests were conducted at Inspectorate Metallurgical Division of Richmond, BC on a split of a master composite sample from the above referenced 2008 SGS Canada metallurgical test program. The pre-concentration techniques evaluated included heavy media separation, gravity separation, and electronic sorting.

After gaining ownership of the Sisson Project in late 2010, Northcliff proceeded with the completion of the interrupted 2008 prefeasibility test program in order to increase confidence in the development of the processes that were targeted. In parallel to this metallurgical test program in 2011 at SGS Lakefield Research in Lakefield, Ontario, Northcliff carried out a summer drilling program to collect a metallurgical sample for a Feasibility Study level test program planned for late 2011 and 2012. A total of ~35 t core samples from 15 PQ and HQ drill holes were gathered and shipped to SGS Lakefield Research Ltd., in Lakefield, for a feasibility level metallurgical test program including tungsten concentrate leach and APT conversion.

Summary of historical metallurgical test programs, mineralogical investigations, and the latest 2012 Feasibility Study level test program which formed the basis of process development for Sisson are provided in the following paragraphs.

### **13.1 Historical Mineral Processing Testwork**

#### **13.1.1 Micron Geological (2006) and PRA Test Program (2007)**

Five 9-m sections of drill core were selected from hole SB-0603 at depths from 102 to 317 m for a petrographic investigation in support of a scoping level metallurgical testing by Process Research Associates Ltd (PRA) of Richmond, BC. The mineralogical report was titled "Petrographic Report on five composited crushed core samples from Sisson Brook W-Mo-Cu Property, New Brunswick, Reported to

Geodex Minerals Ltd, December 10, 2006". The metallurgical report was issued in a memo format titled "Sisson Brook Metallurgy" dated June 21, 2007.

Results of the petrographic work confirmed that scheelite and molybdenite are the main minerals of economic interest and trace amounts of sphalerite, galena, chalcopyrite, and wolframite were noted. Scheelite liberation in the 0.86 mm to 0.250 mm size fraction was 80-95%. No powellite was found. Molybdenite is the main carrier of Mo and like scheelite, in the 0.86 mm to 0.25 mm size range it occurs as free grains at 80-90% by surface area. The major minerals in the composite are quartz, feldspar, amphibole, biotite-phlogopite and chlorite. In general, the rocks were found to be poor in sulphides and appear to have little acid generation potential which might be counteracted to some extent by the minor calcite content.

The metallurgical testing by PRA consisted of three tests which concluded that scheelite was amenable to upgrading by flotation separation.

### **13.1.2 SGS Lakefield Research Scoping Level Test Program (2008)**

In 2008, a scoping level metallurgical test program was conducted by SGS Lakefield Research Ltd, of Lakefield, Ontario. The report was titled "An Investigation into The Recovery of Molybdenite and Scheelite from the Sisson Brook Deposit prepared for Geodex Minerals Ltd, October 22, 2008".

The samples were selected intervals of drill core from holes SP-07-01 and SBM-0701 and weighed approximately 1,582 kg. The samples were further grouped into two composites based on their lithology; Volcanics (0.090%  $WO_3$  and 0.036% Mo) and Gabbro (0.071%  $WO_3$  and 0.020% Mo). The purpose of the investigation was to gain preliminary information on the metallurgical response of the samples - particularly the economic minerals of interest; tungsten and molybdenum - in terms of their mineralogical characteristics, grindability, gravity separation and flotation separation.

The two composites were subjected to QEMSCAN mineralogy which showed that scheelite is the predominant tungsten mineral and, trace amounts of wolframite (0.2% and 10% of the total tungsten content in the Gabbro and the Volcanics samples, respectively) was also observed. Molybdenite is the main molybdenum mineral. Combined quartz, plagioclase, K-feldspar, chlorites, amphibole, muscovite and biotite make up 84%-91% of the two samples while sulphides, mainly pyrite and pyrrhotite, make up 1.2-1.7% of the same. In both composites, 92% of the scheelite was 80% liberated. For molybdenite, liberation varied from 73% to 84% between the two samples. Theoretical recoveries, as calculated by QEMSCAN analysis, indicated that moly recoveries could range between 95% and 55% for the two samples. It is noted that the platy nature of molybdenite may influence findings. Theoretical recoveries for scheelite on the other hand were indicated to be in the high nineties percent.

Grindability analysis were made using standard Bond techniques where the Volcanics sample tested medium-hard (15.7 BM Wi) and Gabbro tested hard (19.1 BM Wi).

Gravity recovery potential of tungsten minerals was tested using a size-by-size (Release) analysis and the results indicated that 70% to 91% of tungsten in the +20  $\mu$ m size range could be recovered in each size range while the recoveries for -20  $\mu$ m fraction was ~20% as expected for this very fine size range for

gravity separation. Marketable grade scheelite concentrates were produced by gravity in this test program.

Scheelite flotation on samples after moly and slimes removal recovered +90% of the tungsten in the flotation feed into a rougher concentrate. Scheelite flotation on gravity tailings containing ~40% of the tungsten in the main feed on the other hand, indicated ~70% tungsten recovery into rougher concentrates. Optimization of cleaner flotation conditions was recommended.

Moly rougher flotation recoveries exceeding 93% Mo were achieved using conventional reagent schemes on relatively coarse ground samples (~348 µm P80). Rougher concentrate upgrading efforts required further testing into parameters such as selection of regrind size (liberation) and more selective reagents.

### **13.1.3 SGS Canada Inc. Prefeasibility Level Metallurgical Test Program (2008/2009)**

In September 2008, Geodex Minerals Ltd embarked on a prefeasibility level metallurgical test program on the Sisson Project at the facilities of SGS Canada Inc., Vancouver, BC. The results of this program were provided in a report titled "An Investigation into The Recovery of Molybdenite and Scheelite from the Sisson Brook Resource prepared for Geodex Minerals Ltd, June 22, 2009". The testwork commenced in September 2008 and was interrupted late that year and eventually stopped in February 2009 due to global economic uncertainties at the time.

Six samples shipped to SGS were selected intervals of split drill core from the 2006 and 2007 drilling programs and included holes SB-06-08, 09, 10, 14, 15, 16, 17 and SB-07-06, 10A, 13, 24, 28, 38, 44, 47, 49, 53, 59, 64 and 66. The samples weighed approximately 2,857 kg in total and were reported to represent the resource and mineable areas at the time, as well as a range of varying rock lithologies and spatial locations within the central area. The head assay of the master composite was determined as follows: 0.096% WO<sub>3</sub>, 0.038% Mo and 0.45% S.

Scope of testwork included pre-concentration tests by electronic and gravity techniques, gravity release analysis, gravity separation of tungsten, and flotation recovery of both moly and scheelite.

The results of preliminary bench-top ore sorting test analysis using -25mm +12mm rocks showed that potential existed for a significant upgrade in tungsten grade using dual energy X-ray transmission (DEXRT) or UV fluorescence sorting techniques while rejecting a low grade waste product from the feed samples.

The analysis of test results further showed that the DEXRT test could recover about 92% of the total tungsten in a pre-concentrate product of approximately 40% of the total feed mass whilst the UV sorting technique could recover about 92% of the total tungsten in a pre-concentrate product of approximately 34% of the total feed mass. The results of moly upgrading by the DEXRT technique showed that 86% of the total moly could be recovered into a pre-concentrate product of approximately 40% of the feed mass. Results of the tests using the UV sorting technique showed much less promise where only about 72% of the total moly could be recovered for the same mass pull.

Heavy liquid separation tests conducted on composite samples of -12mm+0.85mm size range and using varying specific gravity liquids showed that tungsten recovery varied 89%-92% between the three fractions of the head sample tested while the mass pull into the pre-concentrate varied 35%-46% for the corresponding size fractions. Moly recoveries were 86%-94% for the same percent mass pull rates.



Release analysis was conducted on composite samples to identify the liberation and upgrading potential of scheelite by gravity techniques. The results indicated that scheelite is likely liberated at 212 µm and further, significant upgrading was achieved in the -212 µm +53 µm fractions where tungsten recoveries varied between 51% and 91%. As expected, the recovery of moly by gravity techniques was much less promising. These results indicated that potential existed to produce a marketable grade tungsten concentrate by gravity techniques.

Combined flotation and gravity tests demonstrated that while the tungsten gravity recovery on samples following moly and sulphides removal could be very efficient in the +74 µm size fraction, it is unacceptably low in the fine size ranges. This analysis and observation brought to focus the criticality of the comminution and classification processes, and the need for a separate tungsten recovery process from the fine fraction such as flotation.

Moly flotation on composite samples using P80 of 300 µm primary grind recovered ~92% of the Mo into a rougher concentrate assaying 2.4% Mo. Bulk sulphide removal after moly rougher flotation contained acceptable levels of tungsten and moly losses in this product. Limited number of scheelite flotation tests prior to the interruption of the test program indicated that tungsten concentrates assaying greater than 46% WO<sub>3</sub> and at over 76% overall recovery (or ~92% stage recovery) could be achieved.

The test program concluded that sufficient process data had been gathered to generate a conceptual flow sheet and that metallurgical investigations should continue to the next level to further develop and optimize the targeted processes.

#### **13.1.4 Inspectorate and Terra Vision Pre-concentration Test Programs (2010)**

In May 2010, Geodex Minerals Ltd retained Inspectorate Exploration and Mining Services Ltd – Metallurgical Division of Richmond, BC to carry out further laboratory testing with respect to pre-concentration of composite samples using gravity separation processes. The results of the gravity based pre-concentration tests are compiled in a report entitled “Pre-Concentration Metallurgical Testing of Samples from Sisson Brook Project for Geodex Minerals Ltd, November 25, 2010”. In parallel to the above program, Terra Vision Ore Sorting Solutions, as representatives of Commodas Ultra-Sort GmbH was contracted to conduct ore sorting tests using a commercial scale sorting unit at their facility in Wedel (Hamburg), Germany. The results of the ore sorting tests were provided in a report entitled “Terra Vision Ore Sorting Solutions – Ore Sorting Tests on Production Scale DEXRT Ore Sorter, June 14-15, 2010”. All ore sorting test products were shipped to Inspectorate for weighing and assaying to allow metallurgical accounting to be completed.

The Inspectorate pre-concentration test program was conducted on a split of composite sample from the above referenced 2008/2009 SGS Canada Inc. metallurgical test program. The -3/4 inch crushed composite sample assayed 0.106% WO<sub>3</sub>, 0.032% Mo, 0.56% S and weighed a total of 390 kg. The ore sorting test samples were selected intervals of split NQ and BTW drill core from the 2006, 2008 and 2009 drilling programs and included holes SB-06-13, SB-08-42, and SB-09-16,03,02,21,23 representing the Zone III Central Volcanics, East Flank, Contact Volcanics and Gabbro. Total sample assayed 0.103% WO<sub>3</sub> and 0.041% Mo, and weighed a total of 497 kg. The samples were crushed and composites of -1.5+1.0 inch and -1+0.5 inch were prepared before shipment to the Commodas testing facility in Germany.

The objective of the Inspectorate pre-concentration testing was to test the amenability of the composite sample to pre-concentration by gravity separation and to determine the potential for low grade waste rejection without significant losses of W and Mo. Three gravity separation processes were included in the test program: spiral concentration, heavy liquid separation, and centrifugal gravity concentration.

Heavy liquid separation on -10mm+0.425mm size fraction achieved ~96% tungsten and 86% molybdenum recoveries at approximately 30% of the test feed material as low grade reject. As expected, increased recoveries were achieved in finer size ranges tested. Both spiral and centrifugal concentration tests achieved reasonable recoveries for tungsten into the pre-concentrate while molybdenum recoveries were not as promising.

The objective of the ore sorting tests was to test the technical viability of upgrading feed samples at high metal recoveries while producing a low grade waste using a commercial scale DEXRT sorting machine.

The ore sorting tests at Commodas-Ultra Sort were conducted on a commercial size and continuous unit fitted with a DEXRT sensing technology. It was deemed at the time that the UV sensing technology was not developed sufficiently in terms of processing speeds to warrant a pilot test. The DEXRT test results indicated that, for both size ranges, tungsten recoveries ranging from 88% to 94.7% and moly recoveries ranging from 88% to 92% were achieved. For the various tests conducted, the amount of low grade (reject) waste removed ranged from 30% to nearly 60% by feed mass. The upgrading ratio achieved of tungsten in the sorted concentrates over the test feeds was as high as 2.1:1.

### **13.2           SGS Lakefield Research Feasibility Study Mineral Processing Testwork (2011/2012)**

A mineral processing test program in support of a feasibility study was completed during late 2011 and 2012 on samples from the Sisson tungsten-molybdenum deposit in New Brunswick, Canada. The program was conducted by SGS Lakefield Research Ltd, Lakefield, Ontario for Northcliff Resources Ltd (Northcliff) Vancouver, British Columbia.

The results of this program were provided in a report titled "An Investigation into The Recovery of Tungsten and Molybdenum from the Sisson Property prepared for Northcliff Resources Ltd, December 30, 2012".

The first phase of the testwork program commenced in July 2011 utilizing the 3,288 kg of unused test samples made available from the test program described in section 13.1.4 above.

Of the six groups of samples received and labeled M08-V1 to V6, two samples were examined, a Master Composite of all samples and a Gabbro sample (M08-G2). Little difference in the two samples was measured in all tests. These samples assayed 0.100% WO<sub>3</sub>, 0.048% Mo, 0.55% S for the master composite and 0.100% WO<sub>3</sub>, 0.036% Mo, 0.37% S for the Gabbro sample.

QEMSCAN mineralogy showed that for a primary grind of -300 µm, scheelite was virtually 100% liberated. The QEMSCAN results for molybdenite were inconclusive due to the low content in the samples combined with the platy nature and high aspect ratio of the molybdenite grains.

Heavy Liquid Separation (HLS) testing on a coarse crush (-3/8 inch) of the two samples showed mixed results, indicating that a pre-concentration step using dense media separation (DMS) on crushed plant feed should be evaluated on its economic as well as metallurgical benefits in the next stage testing.

Grindability testing on the two composites confirmed the medium-hard to hard classification of these samples at an average bond work index of 17 kWh/t.

Molybdenite floated well, with a rougher recovery of greater than 90% of the Mo in the feed. Overall sulphide removal was 87%. Scheelite rougher flotation recovered 85% of the overall tungsten in the main feed or equating close to 90% stage-recovery in the tungsten rougher flotation feed.

With the availability of freshly drilled core samples in late 2011 the direction and all metallurgical testing efforts shifted to the feasibility level metallurgical test program at that time.

The scope of work for the feasibility level metallurgical test program included ore characterization, comminution, heavy liquid separation, gravity, flotation process development, optimization testwork, process confirmation testing in support of definitive metallurgy, and tailings water treatment testwork.

Analytical methods used at SGS for this program included ICP, WRA (whole rock analysis), and XRF as and where appropriate for the subject samples.

### 13.2.1 Sample Selection and Metallurgical Composites

Northcliff carried out a summer drilling program in 2011 to collect a metallurgical sample for a Feasibility Study planned for 2012. Drill hole locations for the metallurgical samples were selected to represent major rock types and areas that would be mined according to the mine plan at the time. Approximately 30 t of full drill core from 15 PQ and HQ drill holes, with a small segment previously removed for interval assays, were shipped to SGS Lakefield in October 2011. Upon receipt, the material was inventoried in the presence of Northcliff representatives and divided into 10 Metallurgical Composites, according to Rock Type and Mining Year as listed in Table 13.1 below. A small portion of the as-received drill core did not fall into one of these 10 categories and was set aside.

Table 13.1 As Received Composite Samples and Weights					
Met Domains	Yr 1-2 (kg)	Yr 3-5 (kg)	Yr 6-10 (kg)	Yrs >10 (kg)	Total (kg)
Intrusive (IN)	676	761	264	5,987	7,688
Metavolcanic (MV)	1,442	3,993	5,433	6,350	17,218
Metasedimentary (MS)	-	1,751	1,005	-	2,756
Total	2,118	6,505	6,702	12,337	27,662

Over the course of the program, various additional test composites were generated by combining the products generated through master sample preparation as summarized in Table 13.2 below. This included ~3 t of Master Composite material for a Concentrate Production Plant (CPP), ~ 3 t of material for HPGR testing, and 2 lots of material for Pilot Plant (PP) testing, as Years 1-10 Master Composite and a Years >10 Composite.

Table 13.2 Sample Preparation and Composites of Crushed Drill Core Materials			
Test Comp ID	Prep Date	Purpose	Composite Mass (kg)
Y 1-10 MC#2	Feb-12	CPP Feed	3,050
HPGR Comp	Apr-12	HPGR Testing	3,250
Y 1-10 Comp	Jun-12	PP Feed (Years 1-10)	8,005
Y 10 Comp	Jun-12	PP Feed (Years >10)	13,185

### 13.2.2 Head Characterization

#### Chemical Analysis

The eight main test composites were submitted for a detailed suite of assays and select chemical analysis are shown in Table 13.3. The head assays indicate no major deleterious elements that could readily impact product recoveries or impurities in a significant way. Note that the Rock Type (or Domain) composites were comprised of the individual yearly constituents on a weighted basis according to Table 13.1 above. Similarly, the Yearly composites were comprised of the Rock Type composites on a weighted basis.

Table 13.3 Select Chemical Analysis of Test Composites								
Element	DOM-MS Comp	Y1-10 MC	DOM-I Comp	DOM-MV Comp	Y1-2 MC	Y3-5 MC	Y6-10 MC	Y>10 PP-Feed
WO <sub>3</sub> %	0.081	0.11	0.12	0.084	0.14	0.080	0.050	0.090
Mo %	0.016	0.037	0.043	0.039	0.015	0.026	0.019	0.027
S %					0.32	0.42	0.45	0.42
F %	0.19	0.20	0.21	0.21				
SiO <sub>2</sub> %	65.0	64.1	53.6	65.8	61.9	62.6	65.0	59.0
Al <sub>2</sub> O <sub>3</sub> %	14.4	14.4	18.1	14.2	14.5	14.6	14.2	15.8
Fe <sub>2</sub> O <sub>3</sub> %	7.10	6.88	8.92	6.41	7.13	7.05	6.44	7.46
MgO %	2.78	2.65	4.87	2.37	2.71	2.79	2.34	3.39
CaO %	2.86	3.13	7.01	2.89	3.71	3.40	2.83	4.85
Na <sub>2</sub> O %	2.52	2.58	2.66	2.75	2.73	2.49	2.58	2.57
K <sub>2</sub> O %	3.00	3.06	2.39	3.12	2.86	3.12	3.20	2.81
TiO <sub>2</sub> %	0.99	0.97	1.35	0.89	1.03	1.01	0.88	1.06
P <sub>2</sub> O <sub>5</sub> %	0.18	0.17	0.21	0.16	0.16	0.17	0.17	0.19
MnO %	0.14	0.15	0.23	0.14	0.18	0.15	0.14	0.18
Cr <sub>2</sub> O <sub>3</sub> %	0.02	0.02	0.02	0.02	0.02	0.02	0.03	0.04
V <sub>2</sub> O <sub>5</sub> %	0.02	0.02	0.03	0.02	0.03	0.03	0.02	0.03
LOI %	1.90	1.52	1.91	1.53	2.01	1.92	1.91	2.15
Sum %	100.9	99.7	101.4	100.3	98.9	99.4	99.8	99.6

Table 13.4 below shows a comparison of the main direct head assays as provided in Table 13.3 above to the average reconstituted head assays from testing.

Table 13.4 Reconstituted Head Assay Comparison									
Composite	WO <sub>3</sub> , %			Mo, %			S, %		
	Direct	Recon.	Rel St Dev	Direct	Recon.	Rel St Dev	Direct	Recon.	Rel St Dev
Y1-10 MC	0.11	0.097	11.9	0.037	0.025	12.9		0.43	7.8
DOM-MS Comp	0.081	0.098	7.2	0.016	0.021	3.8		0.42	3.8
DOM-I-Comp	0.12	0.10	7.0	0.043	0.022	11.1		0.37	3.5
DOM-MV Comp	0.084	0.082	13.2	0.039	0.026	8.0		0.45	7.6
Y12 MC	0.14	0.11	1.1	0.015	0.012	0.6	0.32	0.32	6.2
Y35 MC	0.080	0.11	5.9	0.026	0.023	1.3	0.42	0.39	6.2
Y610 MC	0.050	0.074	1.6	0.019	0.026	18.0	0.45	0.50	9.5
Y >10	0.090	0.083	15.4	0.027	0.026	17.6	0.42	0.41	9.5

### Mineralogical Characterization

The Master Composite (Y 1-10 MC) and each of the 3 Domain Composites were submitted for mineralogy examination with a third party, Process Mineralogical Consulting Ltd (PMCL), Maple Ridge, BC. PMCL prepared the samples to a grind size of approximately 80% passing 100 µm and divided the ground sample into +75, -75+38 and -38 µm fractions. Each of the fractionated composites was analyzed for modal distribution and liberation characteristics. The following are the major observations:

- All of the samples have similar modal distributions with quartz, plagioclase, biotite, and K-feldspar dominating the mineral assemblage with 88-90% of the overall mineral content.
- Approximately 87-92% of the tungsten occurs as scheelite with the remainder as wolframite. Molybdenum occurs exclusively as molybdenite.
- Scheelite liberation generally improves from approximately 85% free / liberated (>80% of the mineral by surface area) to 90% in the finer fractions.
- Molybdenite liberation is inconsistent across the fractions, but is generally in the range of 80-90% for all samples.

### **13.2.3 Comminution Testing**

#### Work Index Testing

Bond Work index testing was performed by SGS Lakefield on major test composites representing various ore types and predicted yearly composites. Testing included Crusher Work Indices (CWI), Rod mill Work Indices (RMI), Ball mill Work Indices (BMI), and abrasion Indices (AI).

Results indicate that the ore is moderately hard to hard in response to these tests. The Bond low-energy impact test results on drill core indicated hardness ranges from medium to hard. The Bond abrasion index

test results indicate moderately abrasive to very abrasive material. The test results are shown in Table 13.5 below.

Table 13.5 Bond Work Index Test Summary				
Sample Name	Work Indices (kWh/t)			A I g
	CWI	RWI	BWI	
DOM-1 Comp	-	17.2	17.5	-
DOM-MS Comp	-	18.3	17.4	-
DOM-MV Comp	-	17.6	16.7	-
Y-1-10MC	-	17.8	17.1	-
Y-1-10MC - PP		-	16.9	-
WIC-FTQ	19.5	-	-	-
WIC-IDQ	15.3	-	-	-
WIC-MCT	17.6	-	-	-
WIC-IGB	15.8	-	-	-
WIC-WKB	15.6	-	-	-
WIC-MTF	14.7	-	-	-
WIC-FTA	13.1	-	-	-
IN-Y12	-	-	-	0.724
IN-Y35	-	-	-	0.438
IN-Y610	-	-	-	0.430
MS-Y35	-	-	-	0.412
MS-Y610	-	-	-	0.371
MV-Y10	-	-	-	0.554
MV-Y12	-	-	-	0.544
MV-Y35	-	-	-	0.543
MV-Y610	-	-	-	0.505

#### High Pressure Grinding Roll (HPGR) Testing

Comminution testing for the High Pressure Grinding Rolls (HPGR) on a 3,250 kg sample of Yr 1-10 Master Composite was performed by Koeppern Machinery at their pilot test facility at the University of British Columbia in Vancouver, B.C.

Approximately 3.25 t of sample was tested through a high pressure grinding roll (HPGR) crusher employing 0.75 m dia. and 0.22 m width rollers. Test results of the single pass and closed circuit tests indicated that:

- The Sisson samples were amenable to HPGR crushing,
- Substantial size reduction was achieved,
- Variations in feed size had a well-defined effect on HPGR sizing and process parameters, and

- Changes in feed moisture did not have a significant effect on HPGR performance.

Information generated through test work was sufficient to confirm the employment of HPGR crushing in the Sisson comminution circuit. And further, the information collected allowed equipment sizing and preparation of specifications for the intended duty at feasibility study level.

#### **13.2.4 Heavy Liquid Testwork**

Heavy liquid separation testing was conducted to gauge the amenability of the Sisson samples to pre-concentration prior to subsequent recovery unit operations. The objective was to concentrate a substantial portion of the molybdenum and tungsten in as small a portion of the feed sample mass as technically and economically feasible while rejecting the low grade waste to tailings without further processing. The testwork was conducted on the Master Composite and each of the three Domain Composites using feed samples crushed to -1/2" and screened at 14 mesh to remove fines in advance of HLS testing.

The results showed that only 20-40% of the mass is rejected at acceptable recoveries of tungsten and molybdenum. In addition, it is noted that the separation is very sensitive to subtle changes in specific gravity. On the basis of these observations, and of the results of negatively concluded trade-off analysis on its economic benefits, dense media separation was rejected as a potential pre-concentration or recovery technique from the Sisson process flowsheet development testing.

#### **13.2.5 Flowsheet Development Testing**

Following a review and analysis of results of all major testwork conducted to date, including the phase testing mentioned above, the process flowsheet development test program was embarked on as summarized below.

##### Molybdenum and Bulk Sulphide Circuit Development

###### Initial Batch Flotation Tests

A series of 18 batch tests were conducted on the Y1-10 MC composite to confirm the conditions previously developed for the Moly and Sulphide Roughers. Major test conditions included 100% -300 µm grind, 8 minutes of moly rougher flotation using fuel oil and 6 minutes of sulphide flotation using mainly a xanthate collector. These tests confirmed the moly recovery to the rougher concentrate at approximately 88%.

###### Moly Rougher Concentrate and Tungsten Circuit Feed Production Mini Pilot Plant (CPP)

A small concentrate production plant (CPP) was conducted on approximately 3 t of material representing the first 10 years of mine life material (Y1-10 MC). The objectives were to confirm the moly and bulk sulphide conditions developed in the laboratory and more importantly, to generate a large moly rougher concentrate sample for moly cleaner circuit development and bulk sulphide flotation (BSF) tailings for tungsten circuit development.

Moly rougher concentrate and BSF rougher concentrate were collected in drums, while tungsten circuit feed (BSF tailings) was collected in 20 liter pails interchanged at a pre-determined frequency. The CPP



demonstrated that approximately 90% of the moly deported to the moly rougher concentrate, while 95% of the tungsten deported to the tungsten circuit feed.

#### Moly Cleaner Flotation Tests

Moly rougher concentrate collected from the CPP was divided into test charges for laboratory flotation testing. Moly cleaner circuit flowsheet development investigated the role and location of regrind, and the use of various sulphide depressants and moly collectors. A total of 14 batch tests were completed. The following conditions showed the most potential: P80 regrind to 45  $\mu\text{m}$ , 6 cleaning stages, regrind on Mo 1<sup>st</sup> Cleaner Concentrate, iron sulphide depressants, and a generic moly collector.

#### Tungsten Circuit Development

##### Batch Flotation Testing

Approximately 60 batch tests were completed to determine optimum conditions for the tungsten circuit. All tests were conducted on Master Composite material. The first 18 tests were completed on BSF tailings immediately following sulphide flotation testing of whole ore feed, while the final ~40 tests were completed on CPP tailings. Various modifiers, depressants, and collectors were trialed, building on the general conditions determined through previous testwork. Several key observations were made over the course of testing including the following.

- Dosage of sodium silicate was critical in determining mass pulls;
- Tungsten recovery and selectivity improved at a finer grind P<sub>100</sub> of 212  $\mu\text{m}$  versus a P<sub>100</sub> of 300  $\mu\text{m}$ .

##### Gravity Testwork

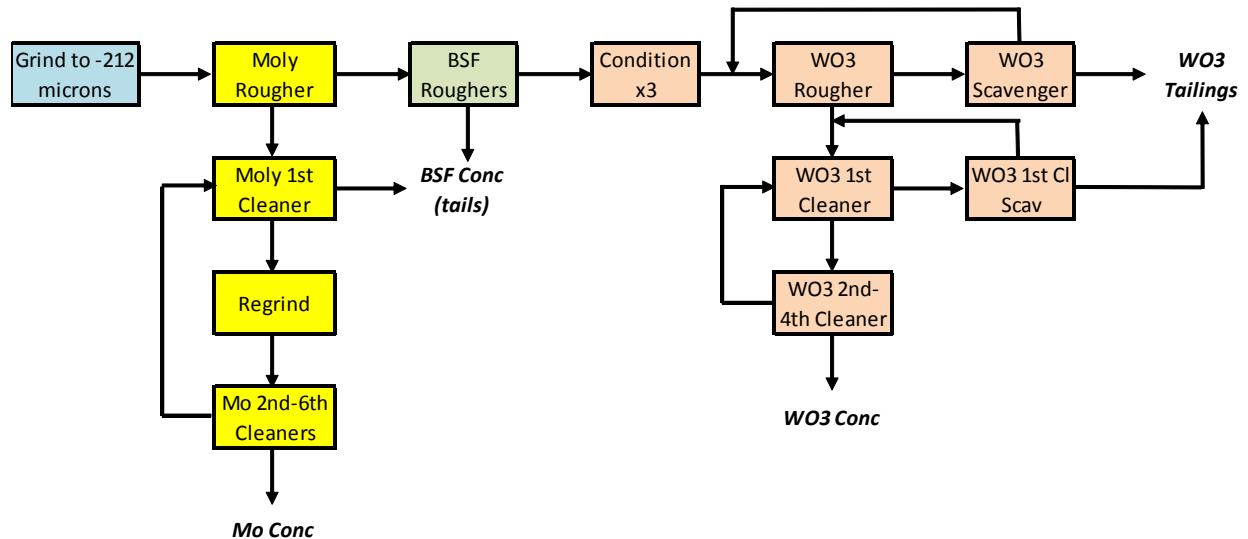
A potential employment of gravity separation within or ahead of the tungsten recovery circuit as a pre-concentration step was examined on whole-ore feed from the Master Composite. Preliminary testing was completed using a lab scale centrifugal gravity concentrator and a shaking table on material ground to - 300  $\mu\text{m}$  and divided into 3 fractions. Results were similar with both pre-concentration methods used and showed that lower than targeted tungsten and molybdenum were concentrated in approximately 50% of the mass. As with the HLS testing discussed previously, gravity recovery was dismissed for whole ore processing as a pre-concentration technique due to excessive losses particularly for molybdenum.

Gravity separation on tungsten circuit feed was evaluated in conjunction with flotation. The strategy was to apply spiral concentration on un-sized tailings, with upgrading on the spiral concentrate using a shaking table and further upgrading the table concentrate using a high-intensity magnetic separator. It was demonstrated that about 40% of the available tungsten could be recovered into a concentrate assaying in the range of 45% WO<sub>3</sub>. Two locked cycle tests on the tailings from this process were completed. In both cases, flotation achieved expected recoveries with flotation concentrate grade in the lower than acceptable ranges.

Ultimately, gravity separation was rejected from the final flowsheet as it did not yield matching or superior performance to a flotation-only process and due to its perceived complexity to implement in a large scale operation as an addition to a flotation circuit.

Final Developed Process Flowsheet

A block diagram of the final process flowsheet as developed and optimized through the batch testing described in the previous sections is illustrated in Figure 13.1 below.



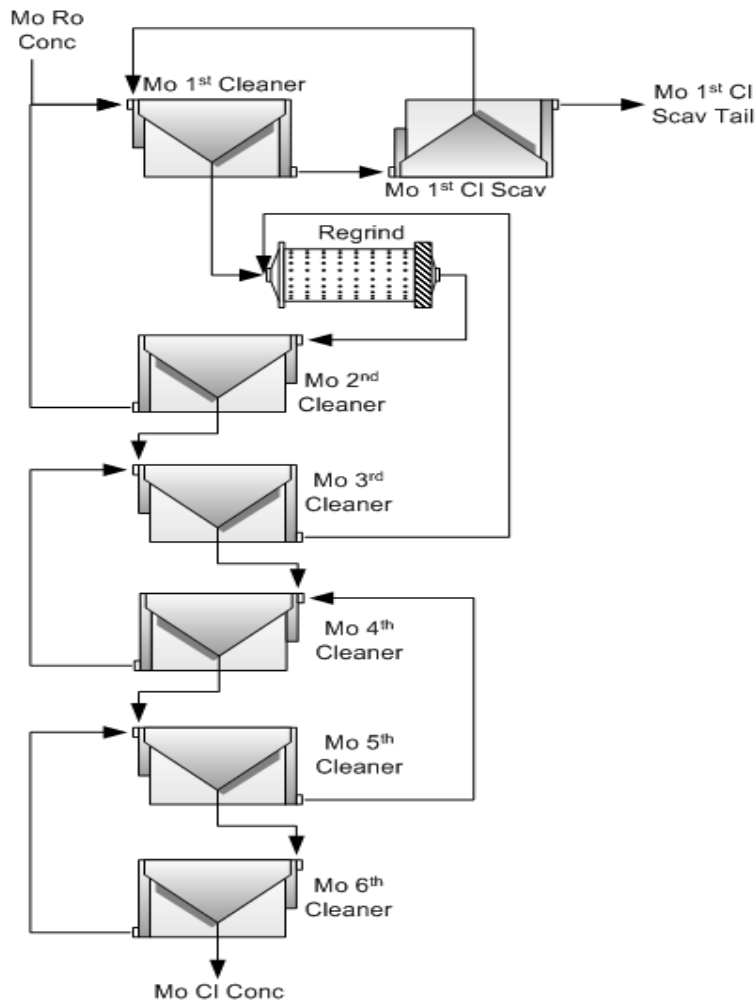
**Figure 13.1: Process Flowsheet**

Ore ground to -212  $\mu\text{m}$  ( $P_{80}$  of approximately 120  $\mu\text{m}$ ), is processed through moly flotation. The Mo rougher concentrate is cleaned once, regrind, and cleaned five more times. A moly 1<sup>st</sup> cleaner tail is rejected and combined with BSF rougher concentrate as tailings. BSF tailings is conditioned and subjected to tungsten rougher-scavenger flotation. The tungsten rougher concentrate is cleaned four times. Tungsten rougher plus 1<sup>st</sup> cleaner scavenger tailings are combined as tailings.

**13.2.6 Locked Cycle Flotation Testing**

Moly Locked Cycle Testing

Three locked cycle tests were completed on Mo Rougher Concentrate samples generated from the pilot plant runs. Best results were obtained using the previously discussed moly regrind and cleaner flotation conditions. These tests resulted in a preferred flowsheet which is based on 6 stages of cleaning with counter current recycling of cleaner tailings and a regrind step after the first cleaner as presented in Figure 13.2 below.



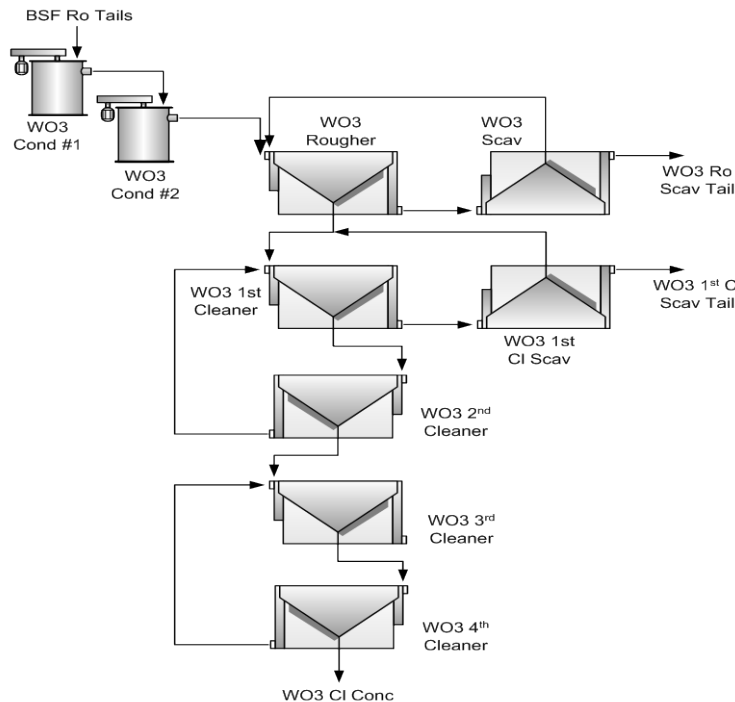
**Figure 13.2: Moly Locked Cycle**

Metallurgical projections are tabulated in Table 13.6 below with recoveries expressed both on the basis of the feed to the cleaner test as well as on original ore feed. By averaging over M-LCT1 and M-LCT3 (M-LCT-2 used an abbreviated version of the preferred flowsheet), an overall moly cleaner concentrate recovery of 83% may be projected at a grade of 48% Mo. Note that marketable molybdenum concentrates are typically better than 50% Mo. For the purposes of the feasibility study, the molybdenum concentrate has been projected at 51% Mo grade with a lower overall recovery of 82%. Based on available data, the projected concentrate grade and recovery should be achievable in a large continuous circuit.

Table 13.6 Summary of Mo Cleaner LCT Results						
Test No.	Comp. ID	Product	Mass	Grade, %	Mo Distribution, %	
			%	Mo	Test	O'All
M-LCT1	CPP	<b>6<sup>TH</sup> Clnr Conc</b>	1.32	<b>48.9</b>	<b>95.0</b>	<b>84.7</b>
		1st. Clnr Scav Tail	98.7	0.034	5.0	4.4
		Head (calculated)	100.0	0.68	100.0	89.1
M-LCT2	PP	<b>6<sup>TH</sup> Clnr Conc</b>	1.9	<b>51.6</b>	<b>84.8</b>	<b>75.6</b>
		2nd Clnr Scav Tail	7.1	0.86	5.2	4.6
		1st. Clnr Scav Tail	91.0	0.13	10.0	8.9
		Head (calculated)	100.0	1.17	100.0	89.1
M-LCT3	PP	<b>6<sup>TH</sup> Clnr Conc</b>	2.29	<b>46.9</b>	<b>91.8</b>	<b>81.8</b>
		1st. Clnr Scav Tail	97.7	0.098	8.21	7.32
		Head (calculated)	100.0	1.17	100.0	89.1

Tungsten Locked Cycle Testing

A total of 12 locked cycle tests were completed on the tungsten circuit. The first 2 tests were conducted at a coarse (-300 μm) grind with high silicate dosage. The next two tests involved a combination of gravity and flotation. The final 8 tests all examined the optimized flowsheet described in Tungsten Circuit Development section above with intermediate products re-circulated according to Figure 13.3 below.



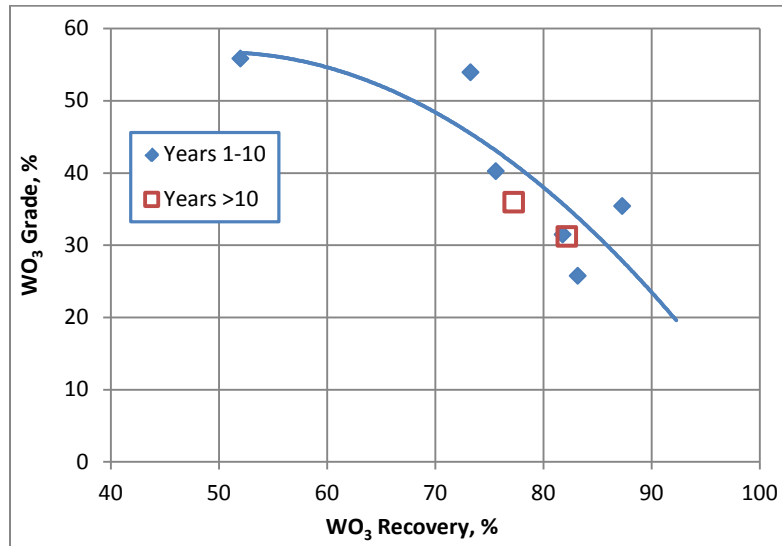
**Figure 13.3: Tungsten Locked Cycle**

These tests all employed the finer grind and optimized dosages of reagents. Minor adjustments to depressant, collector, flotation residence times, and pull rates were made over the course of the final 8 confirmatory tests.

The feed for the locked cycle tests came from several sources, including Y1-10 and Y>10 materials. Two of the tests (LCT-11 and LCT-12) processed whole ore feed to evaluate the entire circuit on freshly generated samples. Results from the eight confirmatory locked cycle tests are given in Table 13.7 below with tungsten recoveries expressed both in terms of feed to the LCT (BSF Tails) and whole ore feed.

Table 13.7 Tungsten Locked Cycle Flotation Tests										
Comp ID	Test ID	Product	Mass %	Grade, %			% Distribution (of BSF Tails)			O'All Dist, % WO <sub>3</sub>
				WO <sub>3</sub>	SiO <sub>2</sub>	S	WO <sub>3</sub>	SiO <sub>2</sub>	S	
Year 1-10	LCT-5	<b>W Cleaner Conc</b>	<b>0.2</b>	<b>35.4</b>	<b>3.88</b>		<b>87.3</b>	<b>0.0</b>		<b>82.7</b>
		W 1st Clnr Scav Tail	1.5	0.19	44.2		2.9	1.0		2.8
		W Scav Tail	98.3	0.01	65.1		9.8	99.0		9.3
		W Circuit Feed	100.0	0.10	64.6		100.0	100.0		94.8
	LCT-6	<b>W Cleaner Conc</b>	<b>0.2</b>	<b>53.9</b>	<b>2.88</b>		<b>73.3</b>	<b>0.0</b>		<b>69.4</b>
		W 1st Clnr Scav Tail	1.1	0.92	38.8		8.7	0.7		8.2
		W Scav Tail	98.7	0.02	66.4		18.0	99.3		17.1
		W Circuit Feed	100.0	0.12	65.9		100.0	100.0		94.8
	LCT-7	<b>W Cleaner Conc</b>	<b>0.4</b>	<b>25.7</b>	<b>2.45</b>	<b>0.86</b>	<b>83.2</b>	<b>0.0</b>	<b>5.0</b>	<b>78.8</b>
		W 1st Clnr Scav Tail	0.9	0.54	41.5	0.57	4.2	0.6	8.1	4.0
		W Scav Tail	98.8	0.01	64.6	0.05	12.6	99.4	86.9	12.0
		W Circuit Feed	100.0	0.11	64.2	0.06	100.0	100.0	100.0	94.8
	LCT-9	<b>W Cleaner Conc</b>	<b>0.1</b>	<b>55.8</b>	<b>3.39</b>	<b>4.49</b>	<b>52.0</b>	<b>0.0</b>	<b>6.1</b>	<b>49.3</b>
		W 1st Clnr Scav Tail	0.3	0.49	53.6	0.41	1.6	0.3	2.0	1.5
		W Scav Tail	99.6	0.05	64.7	0.06	46.4	99.7	91.9	44.0
		W Circuit Feed	100.0	0.10	64.6	0.07	100.0	100.0	100.0	94.8
LCT-11	<b>W Cleaner Conc</b>	<b>0.3</b>	<b>31.5</b>	<b>3.14</b>	<b>0.30</b>	<b>81.8</b>	<b>0.0</b>	<b>1.7</b>	<b>79.3</b>	
	W 1st Clnr Scav Tail	0.8	0.41	44.4	0.13	3.1	0.6	2.2	3.0	
	W Scav Tail	98.9	0.02	65.5	0.05	15.1	99.4	96.1	14.6	
	W Circuit Feed	100.0	0.11	65.2	0.05	100.0	100.0	100.0	97.0	
LCT-12	<b>W Cleaner Conc</b>	<b>0.2</b>	<b>40.2</b>	<b>2.85</b>	<b>0.27</b>	<b>75.6</b>	<b>0.0</b>	<b>0.9</b>	<b>72.1</b>	
	W 1st Clnr Scav Tail	0.7	0.56	47.8	0.12	4.1	0.5	1.4	3.9	
	W Scav Tail	99.2	0.02	65.3	0.05	20.3	99.5	97.7	19.4	
	W Circuit Feed	100.0	0.094	63.6	0.43	100.0	100.0	100.0	95.4	
Year 10+	LCT-8	<b>W Cleaner Conc</b>	<b>0.4</b>	<b>31.2</b>	<b>2.06</b>	<b>0.32</b>	<b>82.2</b>	<b>0.0</b>	<b>1.6</b>	<b>77.9</b>
		W 1st Clnr Scav Tail	0.9	0.81	36.0	0.26	5.2	0.5	3.2	4.9
		W Scav Tail	98.8	0.02	59.8	0.07	12.7	99.5	95.2	12.0
		W Circuit Feed	100.0	0.14	59.4	0.07	100.0	100.0	100.0	94.8
	LCT-10	<b>W Cleaner Conc</b>	<b>0.2</b>	<b>35.9</b>	<b>1.48</b>	<b>0.15</b>	<b>77.3</b>	<b>0.0</b>	<b>0.7</b>	<b>73.2</b>
		W 1st Clnr Scav Tail	0.7	0.65	38.8	0.25	4.2	0.4	3.4	4.0
W Scav Tail		99.1	0.02	60.9	0.05	18.5	99.5	96.0	17.5	
	W Circuit Feed	100.0	0.11	60.6	0.05	100.0	100.0	100.0	94.8	

Concentrate  $WO_3$  grade-recovery relationship is shown in Figure 13.4 below. A second order polynomial trend-line was fit through the six Years 1-10 LCT results. This relationship was used as a basis for predicting tungsten metallurgy.



**Figure 13.4: Tungsten LCT Grade Recovery Relationship**

To maximize  $WO_3$  recovery, 30% concentrate grade was targeted. The APT plant was designed accordingly to handle concentrates in this grade range and is discussed later in this report. A 30% concentrate translates to a  $WO_3$  recovery of 85% in terms of tungsten circuit feed and 82% overall recovery for Years 1-10 material grading 0.097%  $WO_3$ .

**13.2.7 Variability Flotation Testing**

Each of the 8 composites tabulated in Tables 13.2 and 13.3 were submitted to variability rougher flotation tests in duplicate. The conditions applied were similar in all tests and consistent with the optimized conditions previously established. The results indicate a general trend of increasing molybdenum and tungsten recoveries versus increasing feed grade. Tungsten feed grades varied from 0.075% to 0.11%  $WO_3$  and 0.012% to 0.029% Mo. There were some inconsistent results, independent of feed grade and lithology, and further testing is recommended to determine these variables.

**13.2.8 Pilot Plant Testing**

Overview and Objectives

A pilot plant was set up and operated to meet project objectives of producing a bulk tungsten concentrate sample for the subsequent APT conversion tests and to develop feasibility level engineering data as well as definitive predictions for the tungsten circuit.

Upon review of the ore feed grades coupled with available remaining ore, a decision was made to pilot only the rougher portions of the flowsheet and upgrade rougher concentrates in laboratory locked cycle tests. Two lots of ore were prepared for the pilot plant; the plan was to use the  $Y >10$  material as a



commissioning and optimization composite, and then to use the Y1-10 Master Composite material for the objectives mentioned above.

The plant did not perform as expected. The majority of the attention throughout pilot operations was focused on improving tungsten circuit froth stability. It was recognized mid-way through the operations that the low throughput rate of the pilot plant feed was likely the root cause in generating flotation conditions that were too sensitive to achieve circuit stability in the pilot plant cells. Although the plant had periods of operation when metallurgy was as expected, this performance could not be maintained on a consistent basis. Required froth mass pull rates were too small to achieve at reasonable consistency, and increased pull rates would require larger feed rate (and thus larger metallurgical sample). For this reason, the operation of the pilot plant was ceased. Metallurgical predictions of recovery in the concentrator are therefore based on locked cycle tests described above. Further pilot plant testing will be considered at a later stage of the project development.

#### Tungsten Upgrading

The tungsten rougher concentrates generated over the course of the pilot plant were upgraded by various batch flotation tests and gravity separation tests as required. Three batches of tungsten concentrate, totaling approximately 5 kg and typical of cleaner flotation concentrate grades and quality from the locked cycle tests were prepared and provided for APT testing.

### **13.3 Metallurgical Testing – Tungsten Leach, Purification and APT Production**

During 2012 and in support of the Sisson Feasibility Study, a hydrometallurgical test program was conducted on samples from the Sisson tungsten-molybdenum deposit in New Brunswick, Canada. The test program was conducted by SGS Lakefield Research Ltd, in Lakefield, Ontario for Northcliff Resources Ltd., Vancouver, BC. The results of this program were provided in a report titled “An Investigation into Tungsten Pressure Leaching from a Tungsten Scheelite Concentrate Sample from the Sisson APT Project prepared for Northcliff Resources, January 14, 2013”.

The test program included pressure leach tests on tungsten concentrates, alkali recovery from leach solutions, and purification of the resulting sodium tungstate solution. Analytical methods used at SGS for this program included ICP, WRA (whole rock analysis), and XRF as and where appropriate.

#### **13.3.1 Test Objectives and Samples**

Main objectives of the hydrometallurgical test program included the following:

- Determine alkali leach characteristics of the Sisson tungsten concentrate samples in the 30% WO<sub>3</sub> to mid-40% WO<sub>3</sub> range,
- Develop purification methods to remove impurities from the leached solutions, and
- Conversion of purified sodium tungstate solution to ammonium paratungstate (APT).

Test objectives further included the use of conventional and proven process methods as practiced in the industry. The results of the test program were used in the design and confirmation of the APT process flowsheet for the Sisson Project.

Test samples for this program were prepared from pilot plant runs and batch upgrading tests that used Sisson master composite samples representing years 1-10 and 10 plus as discussed previously. Three batches of tungsten concentrates, typical of cleaner flotation concentrate grades and quality, were provided for the APT test program. Two smaller batches were provided in the range of 40% WO<sub>3</sub> and one larger batch was generated in the range of 30% WO<sub>3</sub>.

### **13.3.2 Test Results**

Initially, two leach tests were performed on tungsten concentrate composite samples generated by the Sisson mineral processing test program. The results of the first two tests indicated greater than 95% tungsten recovery from the concentrates. Based on these results, a series of 18 tests were designed to determine the parameters required to achieve the highest extraction of tungsten and to determine the impurities in the pregnant leach solutions which would have to be removed in the subsequent processing steps. These experiments were conducted in batch autoclaves and tested process variables that included feed WO<sub>3</sub> grade and size distribution, leach slurry density, leach residence time and other conventional parameters for pressure leaching. Particle size range tested is 100% passing 93 µm to 158 µm.

Most of these tests resulted in tungsten extraction efficiencies ranging from 97% to 98.9%. An extraction efficiency of greater than 98% was achievable at predicted and economically acceptable process conditions such as alkali addition, residence time and feed particle size distribution.

Furthermore, the resulting pregnant leach solutions (PLS) showed that the majority of the impurities such as Al, Fe, P & Si in the tungsten concentrate (leach feed sample) reported to the gangue, allowing efficient and relatively economic impurity removal from the pregnant leach solutions.

Subsequent testing involved the recovery of alkali from the filtered and concentrated sodium tungstate leach solutions, dissolution, and preparation of the same for impurity removal. The results from these tests indicated high degree of recovery and low impurities.

A series of tests was developed for the purification of the resultant sodium tungstate solution by addition of specific reagents and liquid solid separation of the precipitates to remove major impurities such as Al, As, Mo, P and Si. This series of tests has resulted in greater than 99.5% unit operation recovery.

## **13.4 Mineral Processing and Metallurgical Recovery Predictions**

### **13.4.1 Mineral Processing Recovery Predictions**

Using the metallurgical information presented in the previous sections, an overall mass balance is presented in Table 13.8 below for the Year 1-10 Master Composite.

Table 13.8 Overall Metallurgical Balance for Y 1-10 Master Composite							
Stream	Mass %	Grade			% Distribution		
		Mo, %	S, %	WO <sub>3</sub> , %	Mo	S	WO <sub>3</sub>
Master Composite Feed	100.0	0.025	0.43	0.097	100.0	100.0	100.0
Mo Concentrate	0.044	47.9	34.0	0.070	83.2	3.5	0.0
BSF Concentrate	6.1	0.05	6.00	0.070	12.2	85.4	4.4
WO <sub>3</sub> Concentrate	0.27	0.11	0.30	30.0	1.2	0.2	82.1
WO <sub>3</sub> Tailings	93.6	0.001	0.05	0.014	3.4	11.0	13.5

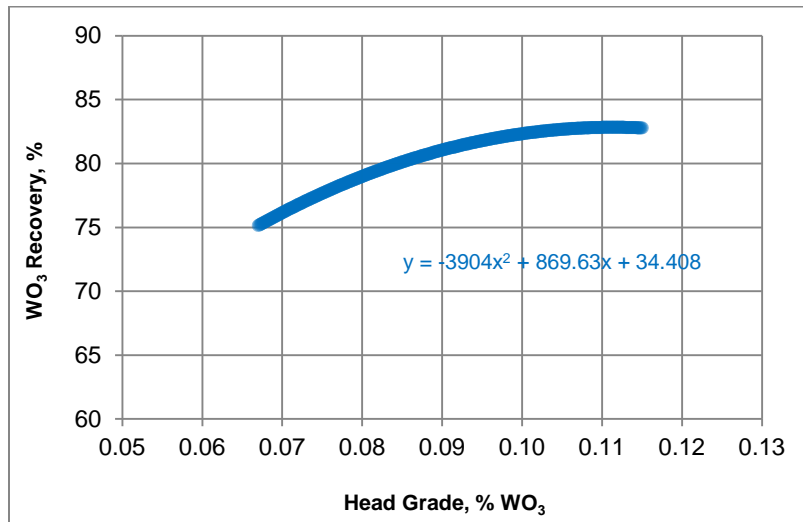
The variability test data was used to develop a preliminary relationship between moly recovery and feed grade. Only the Yearly composites were used to develop the relationship; the Domain composites were excluded since individual lithologies would never be individually mined or processed.

Based on the review and analysis of available test data, it was assumed that there was no measurable relationship between Moly recovery and feed grade. A constant Mo recovery of 82.1% as was robustly determined for the Master Composite as summarized in Table 13.6 above, should be assumed over the feed grade range tested. Further testing is recommended to refine this model over a wider feed grade range as in the mine production plan.

The variability test data, which were based on batch rougher flotation tests, and the locked cycle test results were used to develop a relationship between tungsten feed grade and recovery. Again, only the yearly composite test data was used for the predictions.

The approach used in developing the tungsten recovery predictions was similar to that used for moly and involved the fitting of a second-order polynomial curve intersecting the Master Composite locked cycle test results summarized in Table 13.7 and Figure 13.4 above to the batch variability flotation test recovery data at a constant rougher concentrate upgrade factor for each feed grade.

The equation and the second-order polynomial plot provided in Figure 13.5 below shows the tungsten overall recovery as a function of feed grade. Further testing is required to refine this model at a wider feed grade range using locked cycle test results directly without the need to use batch variability rougher flotation test data.



**Figure 13.5: Tungsten Grade Recovery Relationship**

**13.4.2 Metallurgical (APT Process) Recovery Predictions**

Tungsten leach tests indicated that an extraction efficiency of greater than 98% was achievable using conventional leach conditions as practiced in the industry. Solution purification results have shown greater than 99.5% recovery into purified solution. And the subsequent ammonium tungstate conversion and crystallization tests that are in progress are also expected to result in tungsten efficiencies exceeding 99.5%, allowing an overall APT process recovery prediction of greater than 96.75%.

## **14.0 Mineral Resource Estimate**

### **14.1 Summary**

Northcliff prepared a Mineral Resource estimate for the Sisson Project in early 2012. This estimate was audited by RPA and is described in the June 29, 2012 Technical Report by RPA. The effective date of the estimate was February 29, 2012. Mineral Resources were estimated using a block model constrained with wireframe models of the principal geological domains. Values for bulk density,  $WO_3$ , and Mo were interpolated into the blocks using Inverse Distance Squared ( $ID^2$ ) weighting. A Net Smelter Return (NSR) value for each block was also created which provided a basis for applying a cut-off to the block model.

For the purpose of this report, RPA was requested to revise the NSR calculation to reflect the processing parameters derived during the Feasibility Study. The details of this calculation are provided in the section of this report entitled NSR and Cut-Off Grade.

This most recently revised estimate is considered to be current to December 31, 2012 and is summarized in Table 14.1.

Table 14.1 Mineral Resource Estimate to December 31, 2012							
NSR Cut-Off (\$/t)	Tonnage (Mt)	WO <sub>3</sub> (%)	Mo (%)	WO <sub>3</sub> (M mtu)	Mo (M lb)	WO <sub>3</sub> Eq (%)	Avg NSR (\$/t)
<b>Measured</b>							
15	77.1	0.087	0.026	6.67	44.8	0.113	32.58
11	96.1	0.077	0.024	7.36	51.9	0.102	28.68
<b>9</b>	<b>108</b>	<b>0.072</b>	<b>0.023</b>	<b>7.70</b>	<b>55.3</b>	<b>0.096</b>	<b>26.67</b>
7	121	0.067	0.022	8.03	58.4	0.090	24.66
<b>Indicated</b>							
15	189	0.079	0.022	15.0	91.7	0.101	28.94
11	245	0.069	0.021	17.0	112	0.091	25.26
<b>9</b>	<b>279</b>	<b>0.065</b>	<b>0.020</b>	<b>18.0</b>	<b>122</b>	<b>0.086</b>	<b>23.42</b>
7	317	0.060	0.019	19.0	131	0.080	21.56
<b>Measured + Indicated</b>							
15	266	0.081	0.023	21.6	136	0.105	30.00
11	342	0.071	0.022	24.4	164	0.094	26.22
<b>9</b>	<b>387</b>	<b>0.067</b>	<b>0.021</b>	<b>25.7</b>	<b>178</b>	<b>0.089</b>	<b>24.33</b>
7	438	0.062	0.020	27.0	189	0.083	22.42
<b>Inferred</b>							
15	87.9	0.070	0.024	6.19	47.0	0.097	26.60
11	144	0.056	0.022	8.14	69.8	0.082	21.21
<b>9</b>	<b>187</b>	<b>0.050</b>	<b>0.020</b>	<b>9.41</b>	<b>82.6</b>	<b>0.074</b>	<b>18.63</b>
7	241	0.045	0.018	10.8	94.1	0.066	16.25
Notes:							
1. CIM definitions were followed for Mineral Resources.							
2. Mineral Resources are estimated at a net smelter return (NSR) cut-off grade of \$US9.00/t.							
3. Mineral Resources are estimated using a long-term metal prices of US\$350 per mtu WO <sub>3</sub> and \$US15/lb Mo, and a US\$/C\$ exchange rate of 0.9:1.							
4. Metallurgical recoveries for the NSR calculation were 82% for Mo and averaged 77% for WO <sub>3</sub> over the life of mine. WO <sub>3</sub> recovery is a function of mill head grade.							
5. Numbers may not add due to rounding.							

The Sisson Project is in an advanced stage of development but to date, permits for development of a mine operation has not yet been acquired. RPA notes that there is a risk that environmental, legal, land tenure, socio-economic, or permitting issues could prevent development of the Sisson Project. However, to the extent delineated in this report, RPA is not aware of any such constraint.

## 14.2 Previous Resource Estimate

### 14.2.1 Comparison with Previous Estimate

The block NSR values for the Mineral Resource estimate were modified from the February 2012 estimate to be consistent with the parameters used for the Feasibility Study. The principal changes to the NSR calculation are as follows:

- Tungsten price changed from US\$300/mtu WO<sub>3</sub> to US\$350/mtu WO<sub>3</sub>
- Molybdenum recovery changed to 82%
- Tungsten recovery changed to a variable amount based on grade
- Parameters for concentrate treatment charges, freight, and payable metals adjusted to reflect mill process in the Feasibility Study.

Table 14.2 compares the most recent Mineral Resource estimate (December 31, 2012) with the previous one (February 29, 2012). RPA notes that the tonnage and metal contents of all categories are largely unchanged. There was a minor increase in the tonnage of the Inferred Mineral Resources, but this was partially offset by reductions in grade. The net result was that metal contents did not change significantly.

Table 14.2 Comparison of Current and Previous Estimates						
February 29, 2012						
Category (%WO <sub>3</sub> Eq)	Cut-Off (NSR)	Tonnage (Mt)	WO <sub>3</sub> (%)	Mo (%)	WO <sub>3</sub> (M mtu)	Mo (M lb)
Measured	\$9.00	107	0.072	0.023	7.70	55.3
Indicated	\$9.00	276	0.065	0.020	18.0	122
Inferred	\$9.00	178	0.051	0.020	9.41	82.6
December 31, 2012						
Category (%WO <sub>3</sub> Eq)	Cut-Off (NSR)	Tonnage (Mt)	WO <sub>3</sub> (%)	Mo (%)	WO <sub>3</sub> (M mtu)	Mo (M lb)
Measured	\$9.00	108	0.072	0.024	7.7	56.6
Indicated	\$9.00	279	0.065	0.020	17.9	121.7
Inferred	\$9.00	187	0.050	0.021	9.08	82.4
Percent Difference						
Category (%WO <sub>3</sub> Eq)	-	Tonnage (Mt)	WO <sub>3</sub> (%)	Mo (%)	WO <sub>3</sub> (M mtu)	Mo (M lb)
Measured	-	0.9%	0.0%	4.3%	0.0%	2.4%
Indicated	-	1.1%	0.0%	0.0%	-0.6%	-0.2%
Inferred	-	5.1%	-2.0%	5.0%	-3.5%	-0.2%



## 14.3 Mineral Resource Estimation Methodology

### 14.3.1 Database – General Description

The Mineral Resource estimate for the Sisson Project is based primarily on information from surface drilling, supplemented in part by historical surface mapping and geophysics to assist in the interpretations. As stated above, the database provided to RPA contained collar records for 304 holes. All of these are diamond drill holes. Most of the holes were collared at a dip shallower than  $-75^{\circ}$ . Hole lengths vary widely but are typically in the range between 100 m and 350 m.

Drilling on Sisson covers an approximate area of 2,300 m (north-south) by 1,500 m (east-west), with the density of drilling decreasing toward the north and northwest. Results from 259 drill holes within the block model boundaries were used for the resource estimate. Further detail on drilling can be found in Section 10.0, Drilling.

The Mineral Resource estimate was completed using Maptek Vulcan (Vulcan) software using a conventional approach including 3D solid modeling and block modeling.

### 14.3.2 Assays

The assay database provided to RPA contains 31,357 assay intervals, and of these 29,897 have non-zero values for  $WO_3$  and 31,336 have non-zero values for molybdenum. Most sampled intervals are 1.5 m in length. Brief statistical summaries of the grade domain assays are provided in Table 14.3 and Table 14.4.

Table 14.3 Mo Sample Statistics by Estimation Domain			
Statistic	All Domains	Ellipse Zone	Zone III
Samples	31,336	4,930	26,406
Minimum	0	0	0
Maximum	19.8	2.73	19.8
Mean	0.020	0.019	0.021
Standard Deviation	0.131	0.072	0.140
CV	6.456	3.767	6.791
Variance	0.017	0.005	0.020

Table 14.4 WO <sub>3</sub> Sample Statistics by Estimation Domain						
Statistic	All Domains	Ellipse HG	Ellipse LG	Z3 HG East	Z3 HG West	Z3 LG
Samples	31,357	2,858	2,072	5,730	11,064	9,633
Minimum	0	0	0	0.001	0	0
Maximum	4.439	2.383	0.676	3.493	3.354	4.439
Mean	0.062	0.072	0.033	0.09	0.082	0.027
Standard Deviation	0.127	0.113	0.048	0.161	0.148	0.072
CV	2.039	1.574	1.467	1.797	1.802	2.72
Variance	0.016	0.013	0.002	0.026	0.022	0.005

### 14.3.3 Geological and Structural Models

Drill data was partitioned into 13 separate lithology wireframes corresponding to the eight rock types and their respective sub-domains, discussed in Section 7.0. Descriptive statistics were generated based on these 13 lithology wireframes and the results suggest that lithology does not have strong control over the mineralization distribution. As a result, the lithology model was not used for the purpose of this estimation.

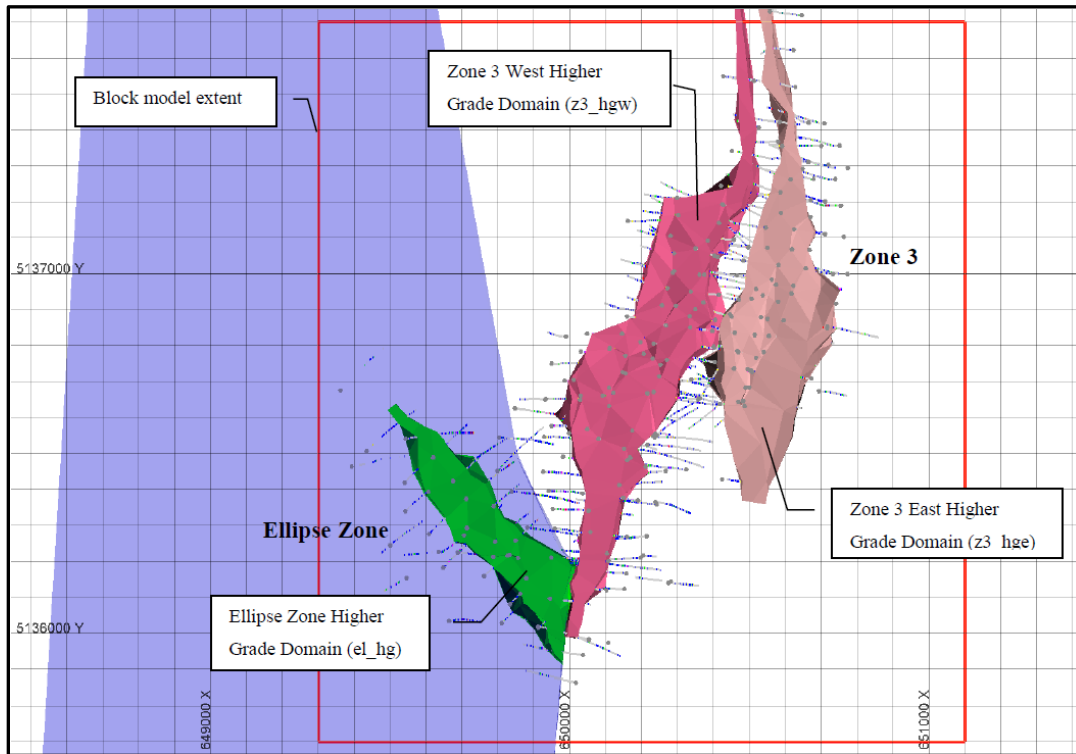
After visual data inspection and statistical analysis, the grade domain approach was chosen for partitioning of WO<sub>3</sub> data. Minimum grade parameters for higher grade WO<sub>3</sub> domains were not rigidly fixed but effectively defined zones with acceptable continuity that grade above 0.035% WO<sub>3</sub> over ten or more meters. There were two higher grade zones modeled for Zone III and a single high grade zone modeled for the Ellipse Zone. Figure 14.1 is a plan map showing the position of the domains at surface.

After a visual inspection of molybdenum data it was decided to not use the grade domaining technique in this case. Molybdenum data was partitioned into Zone III and Ellipse domains only.

### 14.3.4 Assay Capping (Cutting)

RPA notes that the sample grade distributions for WO<sub>3</sub> and molybdenum are positively skewed, in some cases resembling log-normal distributions. For skewed distributions of this type, the highest grade samples can have an inordinately large effect on the average grades, which can result in biased grade interpolations. In order to reduce the influence of these high samples, a cap is applied prior to compositing. For the 2012 estimate, a series of log-normal probability plots and histograms for tungsten and molybdenum were prepared from data within the interpreted zones to examine the distribution of the assay data. Interpretation of the log-normal probability plots and histograms has yielded capping levels of 1.1% WO<sub>3</sub> and 0.65% Mo for both the Ellipse and Zone III domains.

In total, 66 WO<sub>3</sub> and 68 molybdenum assay intervals were capped. These intervals represent approximately 0.2% of the total number of assays. The net impact of the capping was to reduce the average WO<sub>3</sub> and molybdenum assay grades by a negligible amount. Samples were capped prior to compositing.



**Figure 14.1: Ellipse & Zone III WO<sub>3</sub> High Grade Domains within Lower Grade Ellipse & Zone III Domains**

### 14.3.5 Composites

For all Sisson domains, assay intervals were composited on the basis of their respective grade domains. Samples were composited in downhole intervals of four meters, starting at the wireframe pierce-point for each zone, continuing to the point at which the hole exited the zone. Inevitably, the final composite in each zone will be shorter than the fixed composite length unless the zone intercept is an exact multiple of the selected length.

Since the short composites were relatively few in number, their impact was considered to be insignificant, and they were left in the database. The four meter composite length was deemed most suitable, based on the prospective equipment size for potential future mining. Grade domain solids were used to flag composites, based on their centroids (see Table 14.5).

Table 14.5 Grade Domain Composite Flags		
Zone/Domain	Variable	Value
Zone III HG East WO <sub>3</sub>	Bound	z3_hge
Zone III HG West WO <sub>3</sub>	Bound	z3_hgw
Zone III LG WO <sub>3</sub>	Bound	Z3_lg
Ellipse HG Mo	Bound	el_hg
Ellipse LG Mo	Bound	el_lg

In RPA's opinion, insofar as the most common sample lengths are 1.5 m and three meters, a more appropriate composite length would have been three meters. This would have reduced the number of composites that straddle the assay intervals. This is not considered to be a serious concern.

A summary of composite statistics is provided in Table 14.6 and Table 14.7.

<b>Table 14.6</b>			
<b>Mo 4 m Composite Statistics by Estimation Domain</b>			
<b>Statistic</b>	<b>All Domains</b>	<b>Ellipse Zone</b>	<b>Zone III</b>
Samples	13349	2062	11287
Minimum	0	0	0
Maximum	0.62	0.491	0.62
Mean	0.018	0.019	0.018
Standard deviation	0.035	0.036	0.035
CV	1.952	1.894	1.963
Variance	0.001	0.001	0.001

<b>Table 14.7</b>						
<b>WO<sub>3</sub> 4 m Composite Statistics by Estimation Domain</b>						
<b>Statistic</b>	<b>All Domains</b>	<b>Ellipse HG</b>	<b>Ellipse LG</b>	<b>Z3 HG East</b>	<b>Z3 HG West</b>	<b>Z3 LG</b>
Samples	13356	1274	788	2427	4468	4399
Minimum	0	0.001	0.001	0.001	0	0
Maximum	0.986	0.49	0.262	0.916	0.986	0.564
Mean	0.059	0.07	0.035	0.086	0.079	0.025
Standard Deviation	0.074	0.062	0.032	0.091	0.085	0.035
CV	1.248	0.887	0.932	1.065	1.074	1.375
Variance	0.005	0.004	0.001	0.008	0.007	0.001

### 14.3.6 Block Model and Grade Estimation Procedure

#### Dimensions and Coding

A block model of sufficient dimension to encompass the known mineralized system was generated using a block size of 10 m x 10 m x 10 m. Variables were then created to contain estimated WO<sub>3</sub>, Mo, and WO<sub>3</sub>Eq, as well as specific gravity (SG), metal revenue, average distance of estimation samples, material (air/overburden/rock) and resource classification.

### Variogram Models

Variograms were developed for  $WO_3$  in each of the grade domains and for molybdenum in each of the two zones (Zone III and Ellipse). For all metals a two-structure, spherical model was fitted to the experimental variograms. This analysis was carried out by Northcliff using Snowden Supervisor software.

The drill spacing throughout the deposit is approximately 50 m along strike in the north-south direction by 100 m along sections in the east-west direction. This drill spacing generally yielded the best set of variograms using lag distances between 50 m and 90 m.

### Grade Interpolation

Grade estimation was performed using inverse distance estimator with a power of two. Search region geometries were based on variography analysis results (a factor of 1.5 was applied to the ranges). For  $WO_3$  grade estimation, higher grade domain solids were employed as hard boundaries (i.e., only samples from this domain were used to estimate blocks for this domain). Lower grade domains were estimated using soft boundaries – samples from both lower grade domains were used to estimate blocks in either of these domains. Two estimation passes were performed, a minimum of three and maximum of 12 samples from a minimum of two drill holes were required before a block would be estimated in the first pass. No minimum drill hole limit was applied in the second pass. For molybdenum grade estimation, blocks in both zones were estimated using all composites (soft boundaries).

Other estimation parameters are summarized in Table 14.8.

Table 14.8 Inverse Distance Estimation Parameters										
Estimator	Zone	Estimation ID	Sample Selection	Block Selection	Major Axis	Semi-Major Axis	Minor Axis	Bearing	Plunge	Dip
WO <sub>3</sub> (ID <sup>2</sup> )	Zone III HG West	z3w1_1, z3w1_2	Bound=z3_hgw	wo3_zone eqs "z3_hgw" and yworld le 5137300	165	123	60	20	20	-90
	Zone III HG West (North of 5137300)	z3w1n1, z3w1n2	Bound=z3_hgw	wo3_zone eqs "z3_hgw" and yworld gt 5137300	165	123	60	350	20	-90
	Zone III HG East	z3w1_1, z3w1_2	Bound=z3_hge	wo3_zone eqs "z3_hge" and yworld le 5137300	180	144	48	20	-30	-90
	Zone III HG East (North of 5137300)	z3w1n1, z3w1n2	Bound=z3_hge	wo3_zone eqs "z3_hge" and yworld gt 5137300	180	144	48	350	-30	-90
	Zone III LG	z3w0_1, z3w0_2	Bound="z3_lg" or "el_lg"	zone eqs "z3" and wo3pct_id2 lt 0	273	255	180	20	0	-90
	Ellipse Zone HG	elw1_1, elw1_2	Bound=el_hg	zone eqs "el_hg"	225	156	138	140	0	-90
	Ellipse Zone LG	elw0_1, elw0_2	Bound="z3_lg" or "el_lg"	zone eqs "el" and wo3pct_id2 lt 0	234	225	420	140	0	-90
Mo (ID <sup>2</sup> )	Zone III	mo_z32	MoPCT > 0	zone eqs "z3"	500	280	300	35	0	-90
	Ellipse Zone	mo_el2	MoPCT > 0	zone eqs "el"	460	180	70	140	0	-90
SG	Zone III + Ellipse Zone	sg	SG > 0	material eq 1	260	300	100	15.8	39.8	-83.5

### 14.3.7 NSR and Cut-Off Grade

Both molybdenum and  $WO_3$  are considered to be of economic interest for the Sisson Project. Northcliff has derived and applied an NSR cut-off value to include contributions from both metals. The NSR was based on metal prices, plant recovery factors, downstream treatment, insurance, transportation losses, and cost of sales parameters. The principal economic and metallurgical assumptions used in the NSR calculation are listed below:

- $WO_3$  price = US\$350/mtu
- Mo price = US\$15/lb
- Exchange rate = US\$0.90 per C\$1.00
- Plant recovery for  $WO_3$  variable (see equation below)
- Plant recovery for Mo = 82%
- APT plant recovery = 97%
- $WO_3$  conc grade = 30%
- Mo conc grade = 50%
- Payable Mo = 99%
- Transportation losses (Mo only) = 1.5%

Metallurgical test work conducted by Northcliff determined that there is a relationship between plant recovery for tungsten and head grade. Block tungsten recovery was estimated using a regression line formula based on the results of the metallurgical testing. This formula is shown below:

$$WO_3 \text{ recovery (\%)} = ((-3,904.0 \times WO_3\%^2) + (869.6 \times WO_3\%) + 34.4) \times APT\_rec$$

Where:  $WO_3\%$  = estimated block grade  
 $APT\_rec$  = APT plant recovery = 97%

The calculated recovery was limited to a theoretical maximum of 82% for blocks with grades greater than 0.0968%  $WO_3$ .

RPA reviewed the inputs to the NSR calculation, and considers them to be reasonable.

The recoverable NSR values per unit contributed by each metal were determined. These values were multiplied by the estimated block grades to obtain an NSR value for the blocks. These unit values for the in situ metals were determined to be:

$$NSP\_WO_3 = C\$17.46 / \text{lb } WO_3$$

$$NSP\_Mo = C\$14.50 / \text{lb Mo}$$

The block NSR equation was as follows:

$$NSR = (WO_3\% \times NSP\_WO_3 \times REC\_WO_3 \times 2204.6 \text{ lb/t} / 100\%) + (Mo\% \times NSP\_Mo \times REC\_Mo \times 2204.6 \text{ lb/t} / 100\%)$$



Where:  $WO_3\%$  = estimated block grade

$Mo\%$  = estimated block grade

$NSP_{WO_3}$  = C\$17.46 / lb  $WO_3$

$NSP_{Mo}$  = C\$14.50 / lb Mo

$REC_{Mo}$  = metallurgical recovery for Mo

$REC_{WO_3}$  = metallurgical recovery for  $WO_3$

### Equivalence Calculation

Tungsten equivalence was calculated for each block as a metric for comparison with other estimates. The  $WO_3Eq$  values were calculated for each block using a formula derived from the NSR calculation parameters. These parameters are described in the section of this report entitled NSR and Cut-Off Grade. The block NSR value was then divided by the NSR multiplier for  $WO_3$  to obtain the  $WO_3Eq$ . The calculation is provided below:

$$WO_3Eq = WO_3\% + ((Mo\% \times NSP_{Mo} \times REC_{Mo}) / (NSP_{WO_3} \times REC_{WO_3}))$$

### Bulk Density

In previous estimates, density values were calculated by Geodex using a water immersion method and the data were compiled and averaged by Mercator on the basis of sampled lithology. For the block model, all 555 SG samples from all lithologies were grouped into four major categories for the purpose of density assignment.

For the current estimate, Northcliff elected to create an interpolated SG model for density and tonnage calculations. The 2011 drill program has increased the number of density determinations within the database to a total of 1,743 samples. Using interpolated densities in this model is expected to provide better local estimates than the previous simple average of density values within grouped lithologies.

The procedures of the water immersion method are as follows:

- Whole core samples which are typical of the surrounding rock and free of visible moisture were selected. They ranged from seven centimeters to 20 cm in length, and averaged 15 cm.
- The sample was weighed in air on a digital scale and the mass in air ( $Ma$ ) recorded to the nearest 0.1 g in an Excel spreadsheet.
- The sample was then suspended in water below the scale and the mass in water ( $Mw$ ) was entered into the same sheet.
- Calculation of the bulk density was done by the following formula:

$$BD = Ma / (Ma - Mw).$$

Table 14.9 is a summary of the bulk density data.

Table 14.9 Bulk Density Summary					
Year	Number	BD Min	BD Max	Mean	Median
2006	205	2.61	3.51	2.79	2.77
2007	172	2.59	3.02	2.76	2.76
2008	40	2.61	2.92	2.82	2.84
2009	60	2.59	2.94	2.76	2.76
2010	224	2.59	3.17	2.76	2.76
2011	1,042	2.56	3.37	2.77	2.77
<b>Total</b>	<b>1,743</b>	<b>2.56</b>	<b>3.51</b>	<b>2.77</b>	<b>2.77</b>

An average deposit density of 2.77 t/m<sup>3</sup> was assigned to blocks where SG variable was not populated by the estimation routine.

#### 14.4 Limiting Pit Shell

In order to comply with the CIM definitions of “reasonable prospects for economic extraction”, Northcliff prepared a preliminary Lerchs-Grossman pit shell using the estimated costs and parameters shown in Table 14.10. Only those blocks contained within the preliminary pit shell are reported as Mineral Resources (Table 14.1) at the NSR reporting cut-off grade of US\$9/t.

Table 14.10 Limiting Pit Shell Parameters	
Item	Amount
WO <sub>3</sub> Price	US\$13.61/lb
Mo Price	US\$15.00/lb
Mining Cost	US\$2.25/t
Milling + G&A Cost	US\$8.00/t
Pit Slope	45°
Mo Recovery	74%
WO <sub>3</sub> Recovery	80%

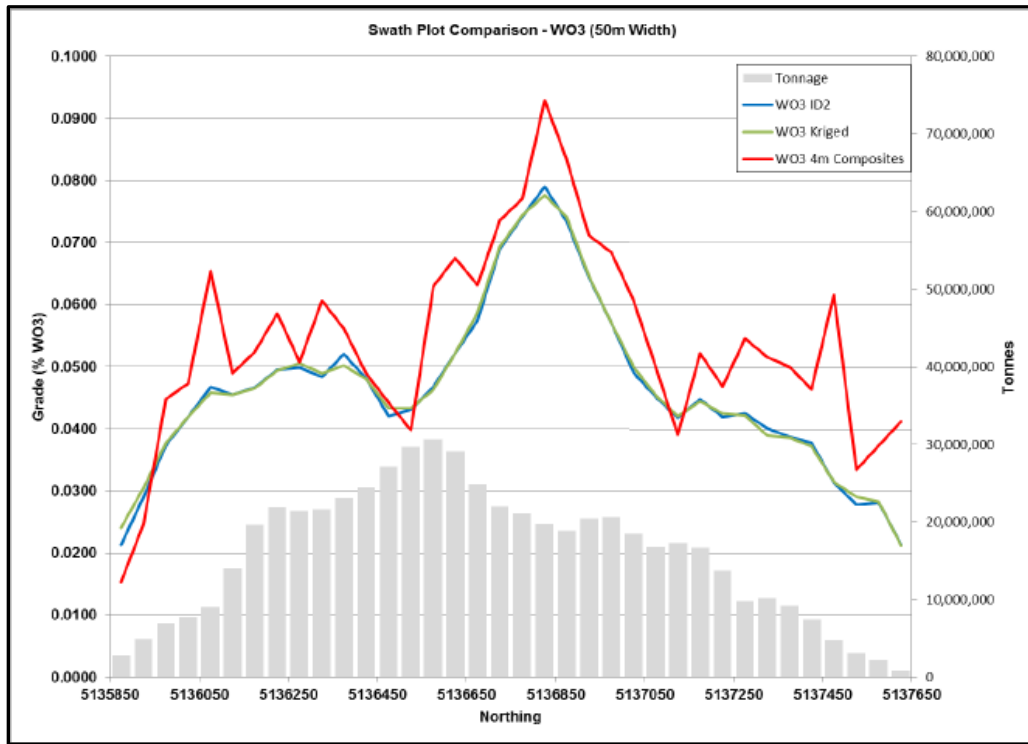
The block model variable resource\_2012 was set to 1 if a block is within the pit shell and of inferred or better category.

#### 14.5 Model Validation by Northcliff

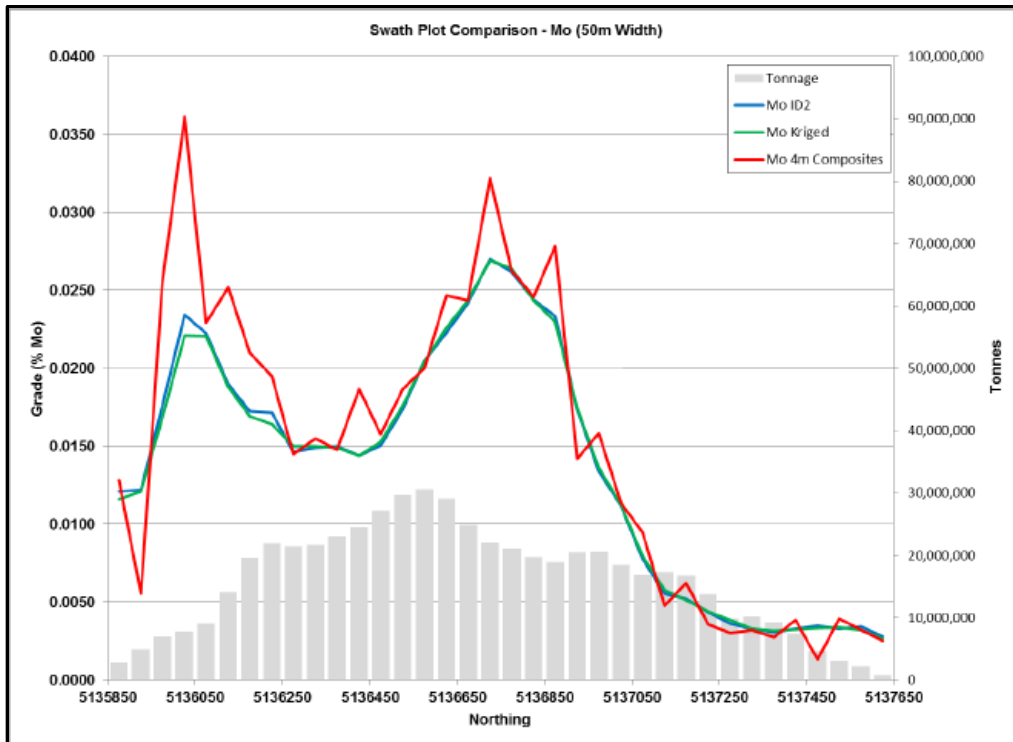
A visual comparison completed by Northcliff on a section by section basis showed acceptable degree of consistency between the block model and drill hole assay grade distribution.

Swath plots for WO<sub>3</sub> and molybdenum were generated to assess the model for global bias by comparing ID<sup>2</sup> values with Ordinary Kriging estimates and four meter run length composites. These plots were

compiled on 50 m vertical East-West panels through the deposit. Only blocks and composites that are within the Indicated category outline were used for swath plot generation. In RPA's opinion, the plots show a good comparison between the three methods, particularly in the main portions of the deposit. At the north end of the deposit, some deviation between  $WO_3$  estimated grades and assay composites was noted. This deviation between estimated and composite grades may be due to the lower number of samples available to populate blocks at the edges of the model. The differences in grade between models and raw data are shown in Figure 14.2 and 14.3.



**Figure 14.2: Swath Plot Comparison for WO<sub>3</sub>**



**Figure 14.3: Swath Plot Comparison for Mo**

Swath plots for SG were generated to assess the model for global bias by comparing ID<sup>2</sup> values with drill hole SG measurements on 100 m vertical South-North panels through the deposit. Results show a good comparison between the two data sets (Figure 14.4).



**Figure 14.4: Swath Plot Comparison for SG**

## 14.6 RPA Validation

The following is a list of the checks performed on the drill hole database and resource models by RPA:

- Checked for duplicate drill hole collar locations and hole numbers.
- Checked collar locations for zero/extreme values.
- Checked assays in database for missing intervals, long intervals, extreme high values, blank/zero values, reasonable minimum/maximum values.
- Ran validity report to check for out-of-range values, missing intervals, overlapping intervals, etc.
- Checked for overlapping wireframes to assess potential double counting of resource volumes.
- Checked mineralized domain/wireframe extensions beyond last holes to see if they are reasonable and consistent.
- Compared basic statistics of assays within wireframes with basic statistics of composites within wireframes for both uncut and cut values.
- Checked the capping of extreme values and effect on coefficient of variation.
- Checked for reasonable compositing intervals.
- Checked that composite intervals start and stop at wireframe boundaries.
- Checked that assigned composite rock type coding is consistent with intersected wireframe coding.
- Checked if block model size and orientation is appropriate for drilling density, mineralization, and mining method.
- Checked search volume radii and orientations against available variography.
- Checked estimation parameters against available variography.
- Visually checked block resource classification coding for isolated blocks.

- Visually compared block grades to drill hole composite values on section and plan views.
- Visually checked for grade banding, smearing of high grades, plumes of high grades, etc., on sections and plans.

While performing the above checks, RPA noted the following:

- Even after cutting, over 20% of the total molybdenum metal content resides within the top percentile of the data. This indicates that there is still a risk of overestimation of global molybdenum in the block model. This is not expected to be severe but warrants further review as the Sisson Project advances.
- The risk of overestimation of the molybdenum metal content may also be increased somewhat by the fact that the interpolation is effectively unconstrained. It is noted, however, that this risk is greatest on the periphery of model, in areas that were often not included as Mineral Resources.
- There are a couple of areas in the model where high SG estimates have been allowed to fill blocks in an unusual manner. This occurs, again, at the edges of the model, presumably where there are few data points and not enough constraining data. Some of these blocks, however, are within the pit limit. The net impact on the global resource estimate is not anticipated to be overly severe but it is likely that some block tonnages will be overestimated. RPA notes that, in poorly drilled portions of the deposit, it may be more appropriate to use average SGs rather than trying to interpolate them. The impact of making this change should be evaluated.
- There also appear to be differences in the statistics for the various zones, which suggest that it might be more prudent to develop independent top cuts on a zone by zone basis rather than a single top cut for all.

In RPA's opinion the assumptions, parameters, and methodology used in the Mineral Resource estimate are appropriate for the style of mineralization and proposed mining method. The resources were further constrained by an 80 m limit to the external boundary of the model. This limit was used because it represents two-thirds of the semi-major axis range of the tungsten variogram.

## 14.7 Classification

As previously stated in the section of this report entitled Limiting Pit Shell, only those blocks contained within the preliminary pit shell are reported as Mineral Resources (Table 14-1) at the NSR cut-off grade of \$9/t.

Estimated blocks were classified by geostatistical means using Vulcan scripts. The principal criterion for assignment of resource classification was the average distance to the nearest three drill holes. The classification was assigned using the following criteria:

1. Blocks with an average distance to the nearest three holes of 45 m or less were classified as Measured (class = 1).
2. Blocks with an average distance to the nearest three holes between 45 m and 80 m were nominally classified as Indicated (class = 2).

3. Blocks with an average distance to the nearest three holes between 80 m and 125 m were nominally assigned to the Inferred category (class = 3).

For the Indicated and Inferred classes, three dimensional category solids were created from sectional interpretations of category blocks. This allowed local smoothing of category boundaries and elimination of isolated classification artefacts. These solids were then used for repopulation of the block model variable class. A script was also created to make sure grade values were erased where no class variable had been assigned.



## **15.0 Mineral Reserve Estimate**

The mineral reserve estimate was based on several series of pit designs, beginning with conceptual Lerchs-Grossman (LG) pit optimization and culminating with detailed phased pit designs.

Three individual sets of LG pits were generated, each testing the sensitivity of a particular design parameter. Series 1 tested pit slope sensitivity, Series 2 tested processing cost sensitivity, and Series 3 tested the sensitivity to a range of design parameters including costs and recoveries. In all cases, the LG sensitivities to these design parameters were minimal.

### **15.1 Assumptions and Methods**

The pit optimization was carried out using MineSight® mine planning software for open pit mining. All mine planning work was carried out on a 3D block model imported by MMTS from a .csv file format received from Northcliff Resources on February 13, 2012 and audited by RPA. The MMTS 3D block model is equivalent to the resource model outlined in Section 14.0.

Conceptual LG pit optimizations were carried out based on the best available economic, geotechnical, and metallurgical parameters at the time, with ongoing refinement of these parameters occurring concurrently with the mine design.

### 15.1.1 Economic, Geotechnical, and Metallurgical Basis for Pit Design

The economic, geotechnical and metallurgical basis used for the pit design is summarized in Table 15.1.

Table 15.1 Base Case Economic, Geotechnical, and Metallurgical Design Parameters			
Design Parameter	Values	Units	Description
LG (overall) Pit Slope Angle, East Side	43	degrees	Azimuth 10-190, Estimated based on 2 ramps w/ IRA of 48 degrees
LG (overall) Pit Slope Angle, West Side	45	degrees	Azimuth 190-10, Estimated based on 3 ramps w/ IRA of 52 degrees
Waste/Ore Mining Cost	\$2.01	C\$/t mined	Based on latest LOM average available at the time
Processing Cost	\$6.88	C\$/t	Processing + G&A + TSF
Recovery (WO <sub>3</sub> )	80.00	%	Assumed flat recovery
Recovery (Mo)	74.00	%	Assumed flat recovery
Base Case NSP Tungsten	\$11.96	C\$/lb WO <sub>3</sub>	NSP = Price - Offsites
Tungsten Price	\$300.00	US\$/mtu	APT price of WO <sub>3</sub> in concentrate
WO <sub>3</sub> Price in common units	\$15.12	C\$/lb	Unit Conversion
Total Offsite WO <sub>3</sub> Costs	\$3.16	C\$/lb	As per Smelter Schedule V3
Base Case NSP Molybdenum	\$15.37	C\$/lb Mo	NSP = Price - Offsites
Molybdenum Price	\$15.00	US\$/lb	Mo in concentrate
Mo Price in common units	\$16.67	C\$/lb	Unit Conversion
Total Offsite Mo Costs	\$1.30	C\$/lb	As per Smelter Schedule V3
Exchange Rate	0.9	US\$/C\$	Northcliff estimate
Factor	22.05	(lb/t)/100	Converts the grade item in % of a tonne into lb.
Default SG	2.77	t/m <sup>3</sup>	SG typically read from 3D block model

#### Economic Basis

The NSP is defined, in basic terms, as metal price after offsite costs with conversions to a common unit such as C\$/lb. Metal market prices were provided by Northcliff based on a marketing report prepared by Roskill Consulting Group Ltd., U.K. (refer to Section 19.0 – Market Studies and Contracts), with an assumed long-term exchange rate of 0.9 US\$/C\$.

Offsite costs include items such as transportation, losses, insurance, treatment charges, and smelter costs. This is summarized in Table 15.1. Further details can be found in 'Smelter Schedule V3'.

### Geotechnical Basis

LG pit designs are conceptual and as such do not include safety berms or haul road ramps, both of which affect the overall pit slope angle. Berms can be accounted for by using inter-ramp angles provided by KP. In order to account for ramps at a conceptual level, an assumed number of ramps are allocated to each side of the pit (based on previous scoping level detailed pit designs); two are on the east and three on the west. When combined with the inter-ramp angle, this gives a rough approximation of the overall slope angle.

### Metallurgical Basis

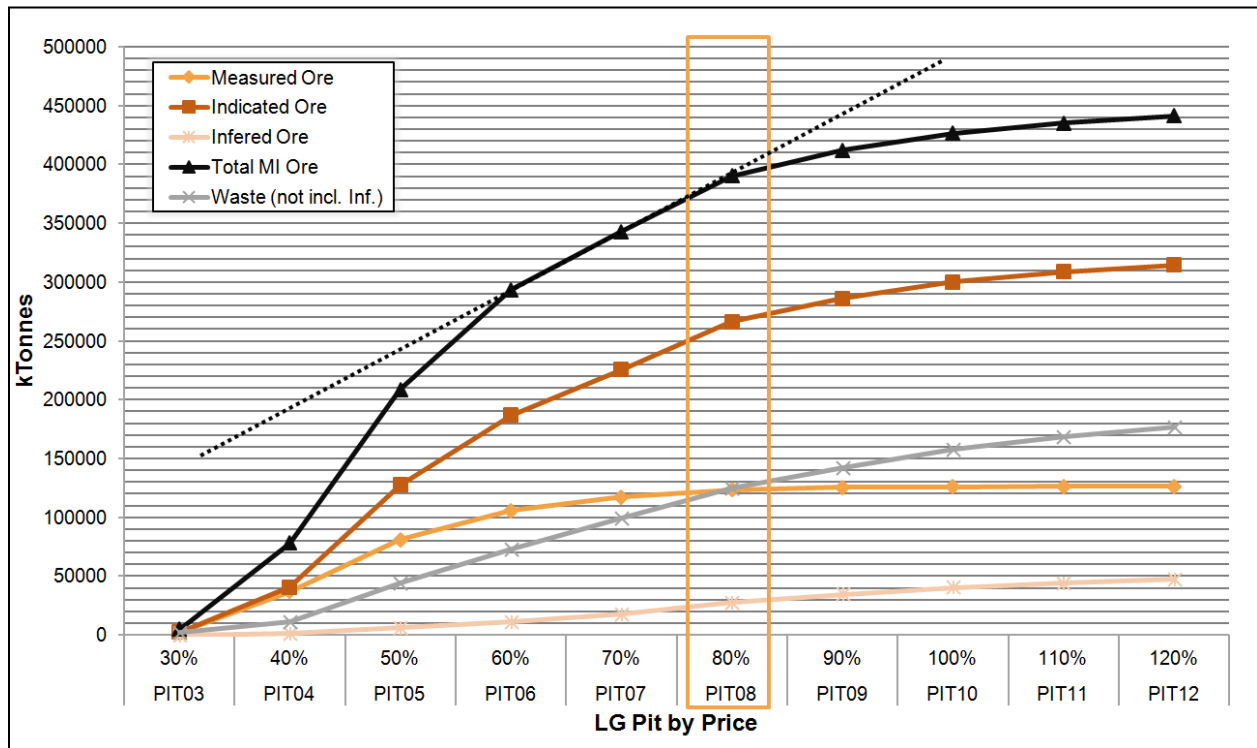
Early estimates of process recoveries assumed a flat 80% recovery for  $WO_3$  and 74% for Mo. However, with more metallurgical test work, it was determined that  $WO_3$  recovery is dependent (non-linear) on relative  $WO_3$  grade. Therefore, the  $WO_3$  recovery formula is built into the 3D block model with a maximum recovery of 82%. Also, subsequent testing improved Mo recoveries to 82% overall (see Table 15.3).

#### **15.1.2 Lerchs-Grossman (LG) Pit Optimization**

Unsmoothed pit limits have been developed using a MineSight® LG algorithm. The prices, recoveries and operating costs are used to estimate the value of each block in the model. Only blocks which contain measured or indicated resources are considered as revenue generating. Blocks containing inferred mineralized material are treated as barren and only accrue mining costs. A series of nested pit limits were developed using varying revenue factors between the base price for each metal and 10% incremental change in price. The nested pit shells are used to determine ultimate pit shell limits as well as guide phase design for the detailed pit phase designs.

The LG analysis uses a base case NSP (100% case) and varies this price by 10% down to the 30% case and up to the 120% case. This generates a series of consecutively nested LG pit shells, using revenues to demonstrate the pit resource sensitivity. Since the cut-off grades are not changed when pit resources are calculated, this does not represent a price sensitivity analysis.

In order to determine the economic pit limit, an estimate of the reserves contained within each of the pit shells is plotted in Figure 15.1. A net-zero revenue cut-off grade (CoG) is used to determine waste and ore tonnages. However, the Sisson pit displays similar trend lines when plotted with higher CoGs. The economic pit limit is selected based on the incremental increase from one LG pit shell to the next. A steeper sloping line between incremental pit shells represents a more favorable incremental increase, and as the slope flattens the incremental economic margin (revenues minus costs) declines. Therefore, by identifying inflection points, where an upwards trending line suddenly flattens, a favorable pit shell is selected since including material from pit shells to the right of the selected inflection point will add only marginal positive cash flow to the project on a non-discounted basis. Time value discounting for long term projects, and cost and price sensitivities, will also favor selecting an ultimate pit left of an inflection point. Typically, by choosing the last pit shell prior to a downwards deviation from the trend line, a favorable balance of increased ore tonnage versus increased barren rock tonnage can be identified. In Figure 15.1, this is demonstrated to be Pit08 (the 80% NSP case) and it is therefore adopted as the economic pit limit.



**Figure 15.1: Economic Pit Limit Selection (Inflection Point Graph)**

**15.1.3 LG Pit Resource Estimate**

The resources contained within the economic (LG) pit limit are expressed in terms of NSR CoG and WO<sub>3</sub>Eq CoG, both of which are calculated and stored in the 3D block model as presented in Equation 15.1.

Equation 15.1 - NSR Calculation:

$$NSR = (WO3\% * NSPWO3 * RECOVERY_{WO3} * 22.046) + (MO\% * NSPMO * RECOVERY_{MO} * 22.046)$$

All calculations for LG pit resource generation are applicable only at the time the resource estimate was carried out.

The CoG selected for LG pit resource calculations in Table 15.2 is the NSR grade at which a given block in the 3D block model breaks even with the sum of re-handle, processing, and general and administrative costs. This represents a block that could be stockpiled and reclaimed at the end of the mine life without a loss in net revenue.

Table 15.2 Resources for the Economic Pit Limit (assuming PIT08 as optimal pit)							
Category	Cut-off Grade	Ore Above CoG	Average Grade Above CoG			Contained Metal above CoG	
	NSR (\$/t)	ROM kt	NSR	WO <sub>3</sub> %	Mo%	Tungsten (M mtu)	Molybdenum (M lb)
Measured	7.13	124,010	19.71	0.067	0.022	8.3	60.2
Indicated	7.13	266,648	18.57	0.064	0.020	17.1	117.7
Total	7.13	390,658	18.93	0.065	0.021	25.5	177.9

In order to accurately estimate resources, a 5% allowance for mining loss and 10% allowance for mining dilution was included where diluted grades were set as the average grades below the break even CoG, 4.55 NSR, 0.016 WO<sub>3</sub>%, and 0.0047 Mo%. These numbers were further refined as data became available and are detailed in Section 15.2.

## 15.2 Summary of Reserves from Detailed Pit Design

The open pit reserve estimate for the Sisson Project were based on the most current design parameters available at the time the estimate was completed. This included several updates to the project, including but not limited to updated marketing studies and metallurgical testing. It is also important to note that the offsite costs for Tungsten were reduced due to the decision to include an APT Processing Plant at the Sisson Project site. The associated costs therefore become part of the processing operating costs, as seen in Table 15.3.

The reserves, contained within the detailed ultimate pit, are expressed in terms of NSR CoG, where NSR is calculated and stored in the 3D block model as outlined in Equation 15.1, found in Section 15.1.3.

Table 15.3 Design Parameters for Reserve Estimate			
Design Parameter	Parameter	Units	Description
Processing Cost	\$7.11	C\$/t	Updated as per TSmith Jan 10, 2013
APT Cost	\$1.53	C\$/t	Updated as per TSmith Jan 28, 2013
TSF Cost	\$0.19	C\$/t	Updated as per TSmith Jan 10, 2013, from KP Cost Model (not incl. sustaining capital)
Incremental Break Even CoG	\$8.83	C\$/t	=\$PC+\$APT+\$TSF
Recovery (WO <sub>3</sub> )	Variable*	%	*read from Block Model item RECWO MAX(82% or -3,904WO <sub>3</sub> % <sup>2</sup> +869.6WO <sub>3</sub> % <sup>2</sup> +34.4) x97% APT RECOVERY
Recovery (Mo)	82.00*	%	*read from Block Model item RECMO
Base Case NSP Tungsten	\$17.46	C\$/lb WO <sub>3</sub>	NSP = Price - Offsites
Tungsten Price	\$350.00	US\$/mtu	APT price of WO <sub>3</sub> in concentrate
WO <sub>3</sub> Price in common units	\$17.64	C\$/lb	Unit Conversion
Total Offsite WO <sub>3</sub> Costs	\$0.18	C\$/lb	As per Smelter Schedule V5.3
Base Case NSP Molybdenum	\$14.49	C\$/lb Mo	NSP = Price - Offsites
Molybdenum Price	\$15.00	US\$/lb	Mo in concentrate
Mo Price in common units	\$16.67	C\$/lb	Unit Conversion
Total Offsite Mo Costs	\$2.18	C\$/lb	As per Smelter Schedule V5.3
Exchange Rate	0.9	US\$/C\$	Northcliff estimate
Factor	22.05	(lb/t)/100	Converts the grade item in % of a tonne into lb
Default SG	2.77	t/m <sup>3</sup>	SG typically read from 3D block model

The CoG for the detailed ultimate reserve calculations in Table 15.4 is the incremental break even CoG, the sum of Processing, APT and TSF costs.

Table 15.4 Reserves for the Series 2 Detailed Ultimate Pit Design Economic Pit Limit							
Category	Cut-off Grade	Ore Above CoG	Average Grade Above CoG			Contained Metal Above CoG	
	NSR (\$/t)	ROM kt	NSR	WO <sub>3</sub> %	Mo%	Tungsten (M mtu)	Molybdenum (M lb)
Proven	8.83	105,415	25.48	0.069	0.023	7.3	53.1
Probable	8.83	228,948	23.54	0.065	0.020	14.9	101.7
Total	8.83	334,363	24.15	0.066	0.021	22.2	154.8

See Table 15.5 for mining loss and dilution applied in this table.

In order to accurately calculate reserves for the detailed ultimate pit design, a loss and dilution study was completed as part of the Feasibility Study but it is not included in the Technical Report. The study uses the

NSR value in the 3D block model blocks to define the ore/waste boundaries and an estimation of dilution is accounted for at these boundaries. This “contact dilution” is a result of mining on the perimeters of the ore/waste zones. For a given ore block, the magnitude of the loss and dilution increases as the number of waste contact edges increases.

The mining reserves used for scheduling were estimated from grades in the 3D block model within the detailed pit designs with the appropriate mining loss and dilution applied as described above. The mining recoveries and dilution convert the in-situ resource material tonnages into a ROM ore.

This method of contact loss and dilution modeling is in addition to any whole block dilution that may be included in the 3D block model. Contact dilution needs to be accounted for because of the lens type nature of the mineralization, where there are significant boundaries between ore and barren rock, rather than just bulk mining whole block dilution. There are three main parts to contact loss and dilution:

- Dilution of barren rock into ore where separate ore and barren rock blasts are not possible
- Loss of ore into barren rock where separate ore and barren rock blasts are not possible, and
- General mining losses and dilution due to handling (haul back in truck boxes, stockpile floor losses, etc.).

For this study, a factor for mining loss and dilution based on the waste contact edges has been estimated for each pit phase. The estimated overall mining loss and dilution values are shown in Table 15.5. This loss and dilution is added to the whole block grade which already includes internal dilution.

<b>Table 15.5 Loss &amp; Dilution by Phase</b>				
<b>Phase</b>	<b>Handling Loss</b>	<b>Contact Block Loss</b>	<b>Total Loss</b>	<b>Dilution</b>
Phase1	1.24%	3.87%	5.11%	3.87%
Phase2	1.24%	4.18%	5.42%	4.18%
Phase3	1.24%	5.97%	7.21%	5.97%
Phase4	1.24%	7.17%	8.41%	7.17%
Phase5	1.24%	5.77%	7.01%	5.77%
Phase6	1.24%	8.13%	9.37%	8.13%

For the detailed pit designs reserves calculation, an NSR dilution grade of \$10.539/t is used. This is based on the assumption that the envelope of waste surrounding each ore block appears to be in the \$9.29-\$11.85 NSR range. The dilution grade is the average grade of material that dilutes the in-situ tonnes into ROM ore.

### **15.3 Conclusion and Comments**

The mineral reserve estimate accurately represents the in-pit resources for the economic LG pit limit and detailed ultimate pit limit reserves. However, they do not take into account any cut-off grade optimization



used for mine production scheduling. These activities (discussed in Section 16.2) will defer lower grade material into a stockpile throughout the mine life.

There is a 14.6% drop in measured and indicated (M&I) ore tonnes above CoG between the conceptual LG pit limit resources (Table 15.2) and the detailed ultimate pit limit reserves (Table 15.4). This is due to the fact that conceptual work was carried out ahead of the selection of final input parameters which necessitated re-running reserves at the detailed design level. These reserves were verified against the conceptual designs and are within acceptable tolerance. As such, the conceptual designs have not been updated.

The reserves estimate summarized in Table 15.4 was also verified with the mine production schedule and any differences were deemed to be insignificant (less than 1%). This is because the production schedule is based on a CoG higher than the incremental breakeven cutoff grade.

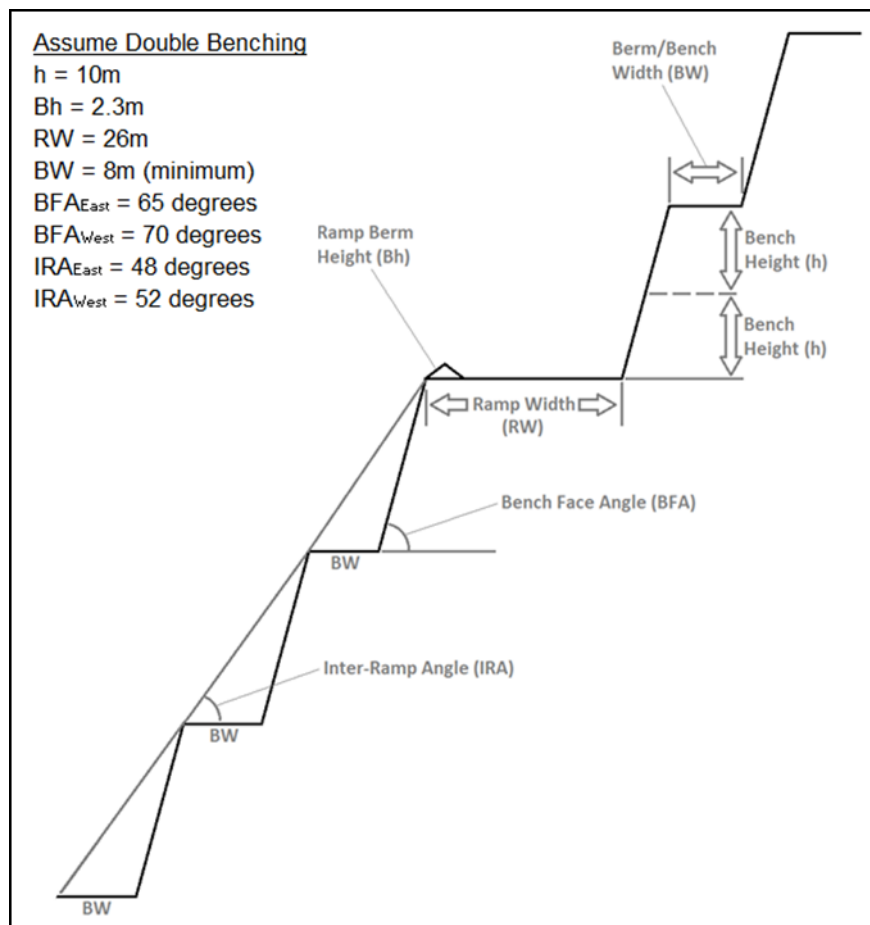
**16.0 Mining Methods**

**16.1 Mining Operations**

Mining operations for the Sisson Project are typical of open pit, truck-and-shovel mining methods for year round operations in Canadian climates.

**16.1.1 Geotechnical Considerations**

Geotechnical parameters used in the pit optimization process are provided by KP and are summarized in Figure 16.1. Geotechnical parameters are divided into the East and West Domains, where the East/West Domain boundary runs at approximately +10 degrees azimuth from North-South.



**Figure 16.1: Cross Sectional Schematic of Pit Wall with Geotechnical Design Parameters**

**16.1.2 Mining Datum**

The Sisson Project mine design work is based on UTM zone19 NAD83 coordinates. Effort has been made to ensure that all disciplines have used the same topography data.

### 16.1.3 Production Rate Consideration

A number of factors are considered when establishing an appropriate mining and processing rate. Typical key factors include:

- Resource Size: Typically, a planned mine life is set at 12.5 to 20 years; beyond this, time-value discounting shows an insignificant contribution to the NPV of the project.
- Capital Payback: Capital investment is typically targeted at projects with a payback period of 3 to 5 years.
- Operational Constraints: Power, water, or supplies and services for support of operations can limit production. For the Sisson Project, there are no significant operational constraints.
- Site Delivery Constraints: Physical size and weight of equipment and shipping limits can determine the maximum size of units that can be delivered to site. For the Sisson Project, site delivery is typically not a constraining factor.
- Project Financial Performance (economies of scale, market share, etc.).

In earlier scoping studies, a 20,000 t/d mill throughput case was examined. However, higher production rates generally pay back fixed capital earlier and provide a higher rate of return on capital, which improves project NPV. As such, the throughput for the feasibility study was increased to 30,000 t/d to provide better project economics. This sets the open pit mine life at 27 years for the existing reserves.

### 16.1.4 Design Standards

#### Mine Haul Roads

Mine haul roads were designed to provide safe and efficient haulage throughout the site and have restricted access for all non-mine equipment and vehicles. In the absence of definitive New Brunswick regulation, the Sisson mine haul roads follow the British Columbia Mines Regulations' minimum width specifications. The British Columbia Mines Regulation is conservative as compared to other jurisdictions.

An allowance for ditches is included within the travel width. Ditches are not added to the in-pit highwall roads; there is adequate water drainage at the edge of the road between the crowned surface and lateral embankments, such as highwalls or lateral impact berms.

Based on a 137 t truck, the haul road design basis is as follows:

- Largest vehicle overall width: 6.3 m
- Double lane highwall haul road allowance: 25.6 m
- Double lane external haul road allowance: 32.3 m
- Single lane highwall haul road allowance: 19.3 m
- Single lane external haul road allowance: 25.9 m

#### Minimum Mining Width

A minimum mining width between pit phases and at pit bottoms is reserved to maintain a suitable mining platform for efficient mining operations. This width is established based on equipment size and operating characteristics. For this study, pit bottoms generally conform to a minimum mining width 30 m, which

provides sufficient room for two-sided truck loading. Between phases, the minimum mining width is increased to 100 m wherever possible to allow for increased operating efficiency.

#### Access Considerations

As stated above, haul road widths are dictated by the configuration of the road (one-way versus two-way) and the size of the equipment. Two-way roads are used for primary long-term haul routes, since they enable safe by-passing of trucks and allow for higher haul productivity. One-way roads are less expensive to build and maintain, and are an appropriate option for low-volume traffic flow or for shorter-term operations. For this study, the use of one-way haul roads is limited to the bottom two or three benches of some pit phases.

Road grades are designed at a maximum grade of 8% for downhill loaded hauls. Uphill loaded hauls are typically designed at a maximum grade of 8%. However, some phases may include short steeper sections, up to 10%, in order to facilitate access in areas with tight pit geometry. Switchbacks are designed flat, with ramps entering and exiting at design grade. In practice, however, grades are transitioned such that visibility and haul speeds are optimized going around the switchback. Where possible, switchbacks are located such that they tie into future phase access development.

#### Bench Height

The Sisson pit designs are based on the digging reach of the mine shovels (10 m operating bench) with double benching between high wall berms; therefore, there are berms every 20 m vertical.

#### Berm Width

Safety berm widths are a minimum of 8 m and may be larger in areas where pit geotechnical parameters govern. Where haul roads intersect designed safety berms, the haul road width is counted towards the safety berm width for the purpose of calculating the overall pit slope angle.

### **16.1.5 Detailed Pit Design**

#### Conceptual Phase Design

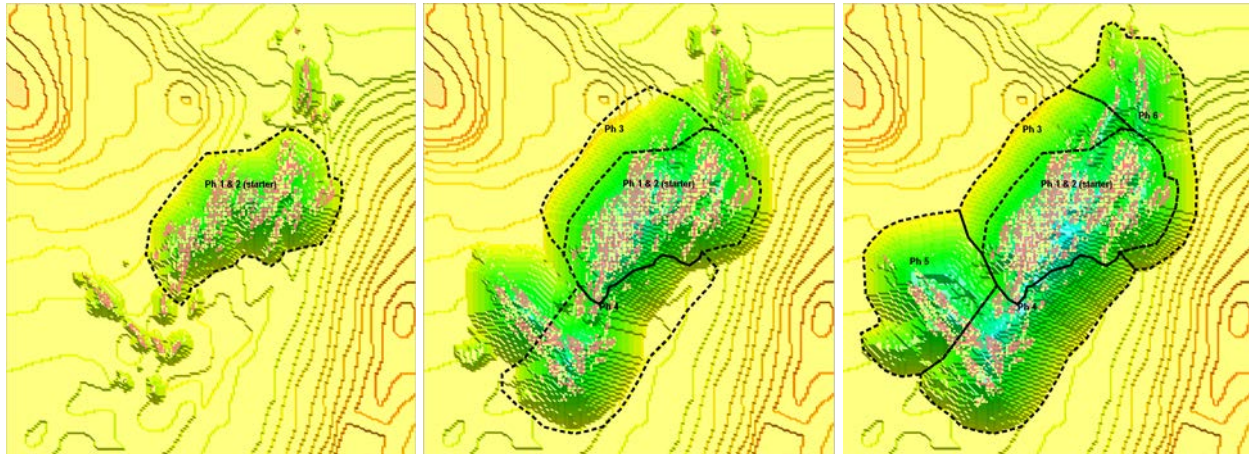
The Sisson detailed pit design is a 6 phase design approach. The LG pits discussed in Section 15.0 were used to evaluate shells for determining the economic pit limit and the optimal pushbacks or phases before commencing detailed design work. Details considered were the addition of roads and bench access, removal of impractical mining areas with a width less than the minimum working width, and ensuring the pit slopes meet the detailed geotechnical recommendations.

Smaller pit shells within the ultimate pit limit were used to design phases from higher to lower margins. This provided the opportunity to maximize revenues and minimize mining costs at the start of mining operations and thereby shortens the Sisson Project capital payback as well as even out Life of Mine stripping requirements. Conceptual phase designs are presented in Figure 16.2.

The lower LG price case pits were used as guides to place the initial starter pit with the following constraints. The starter pit must:

- Be large enough to accommodate the multiple unit mining operations of drilling, blasting, loading, and hauling;
- Have bench sizes large enough so the number of benches mined per year is reasonable (sinking rate); and
- Be wide enough so the shovels can load the trucks efficiently.

At Sisson, the LG shell that indicates a ‘starter phase’ was split into 2 starter pits. This allowed the first pit to mine down into better grades more quickly.



Series1 Pit04: Phase 1 & 2

Series1 Pit06: Phase 3 & 4

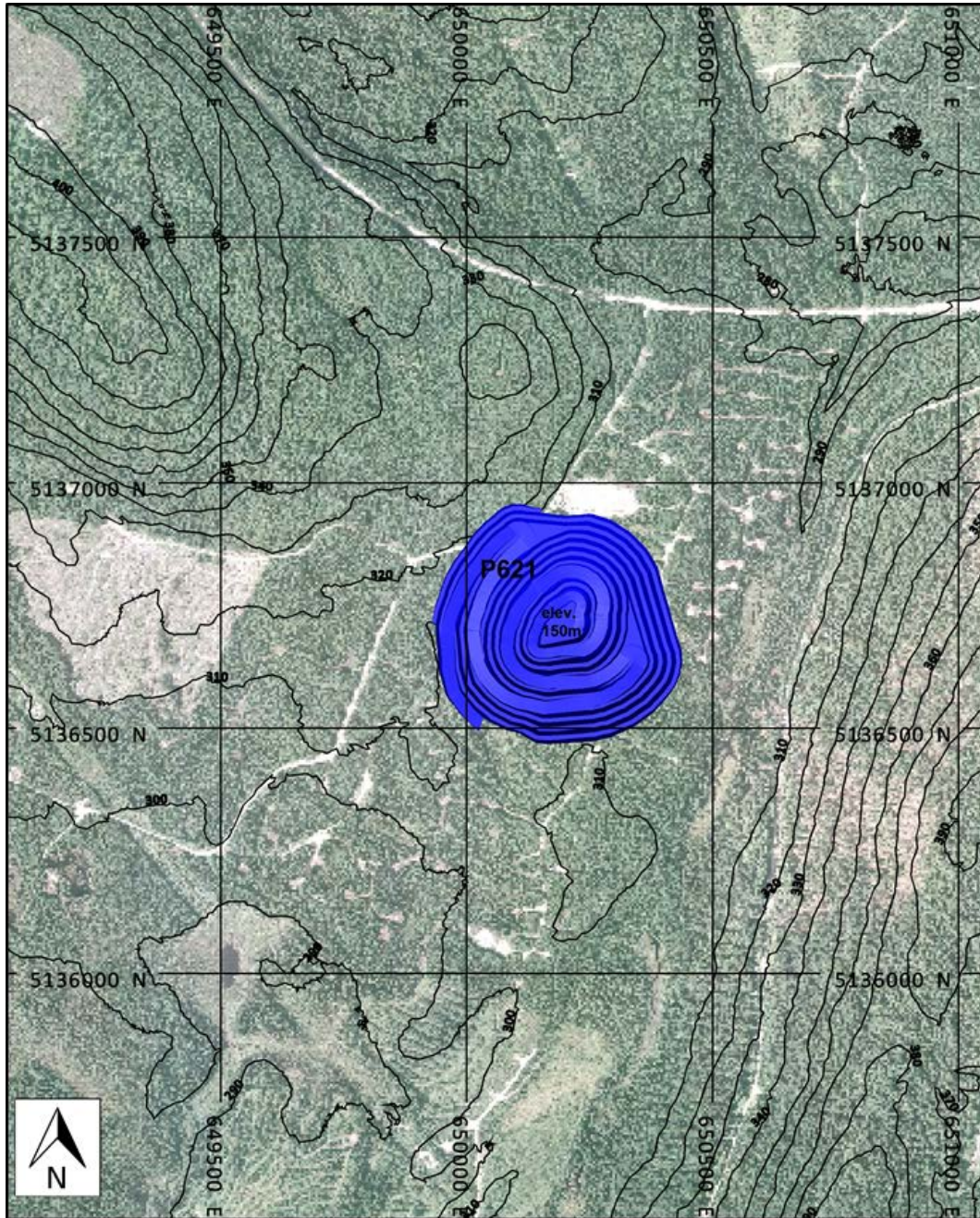
Series1 Pit08: Phase 5 & 6

**Figure 16.2: Conceptual Phase Design**



Detailed Pit Design, Phase 1 (P621)

Phase 1 of the detailed pit design, illustrated in Figure 16.3, is a relatively small starter pit. This phase mines from surface (approximately 320 masl) down to 150 masl. Two switchbacks in the ramp design are planned to tie into future phases. Ramps to the bottom two benches are single lane only.

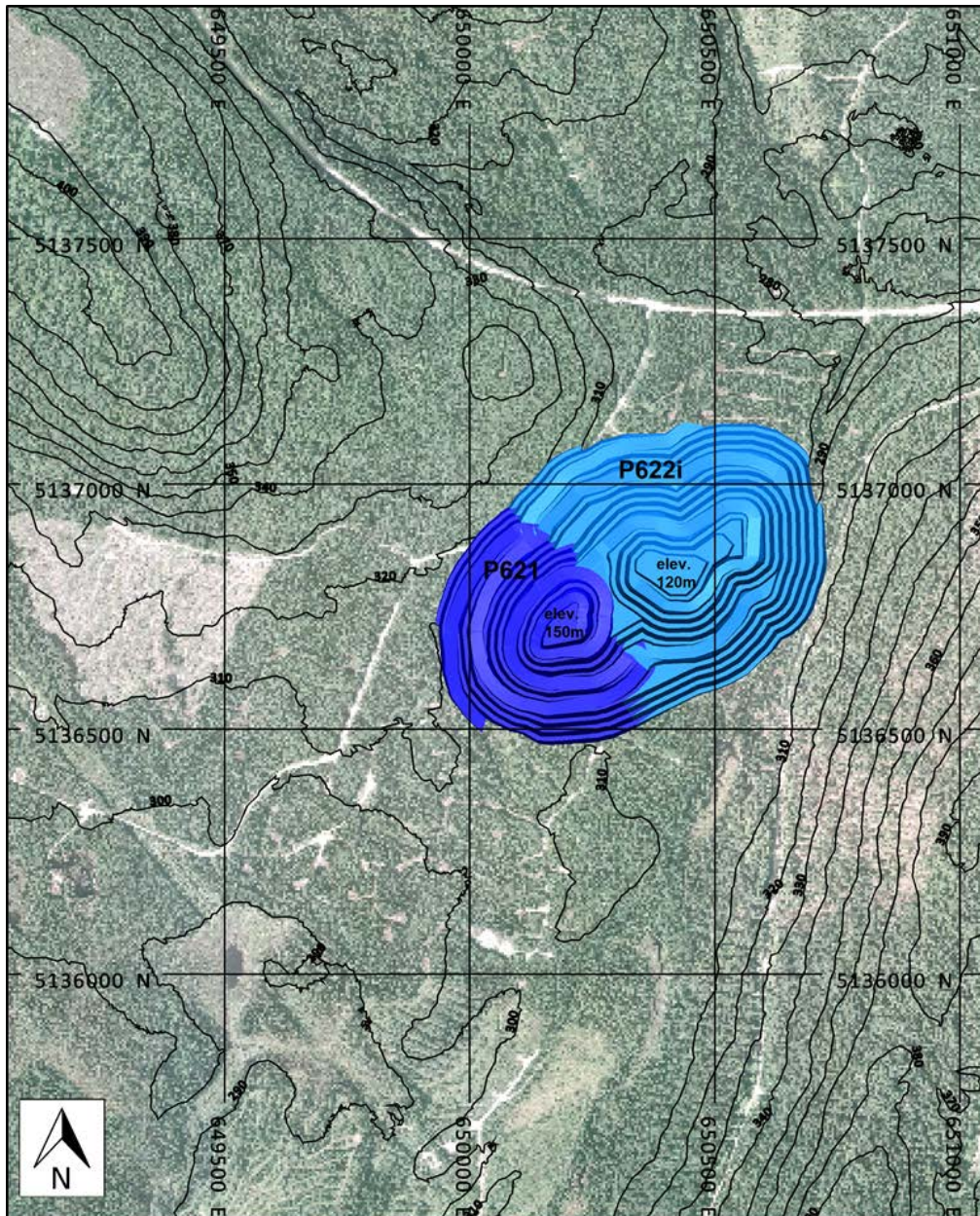


**Figure 16.3: Detailed Pit Design, Phase 1 (P621)**



Detailed Pit Design, Phase 2 (P622)

Phase 2 of the detailed pit design, illustrated in Figure 16.4, is a slightly larger starter pit which acts as a pushback of over 500 m to the northeast of Phase 1. This phase mines from surface (approximately 320 masl) down to 120 masl. Phase 2 also attempts to provide higher grade ore to the mill while keeping the strip ratio reasonably low. Ramps are designed to tie into Phase 1 at several elevations. Ramps to the bottom three benches are single lane only, with the ramp to the final bench (not shown) mined out upon retreat.

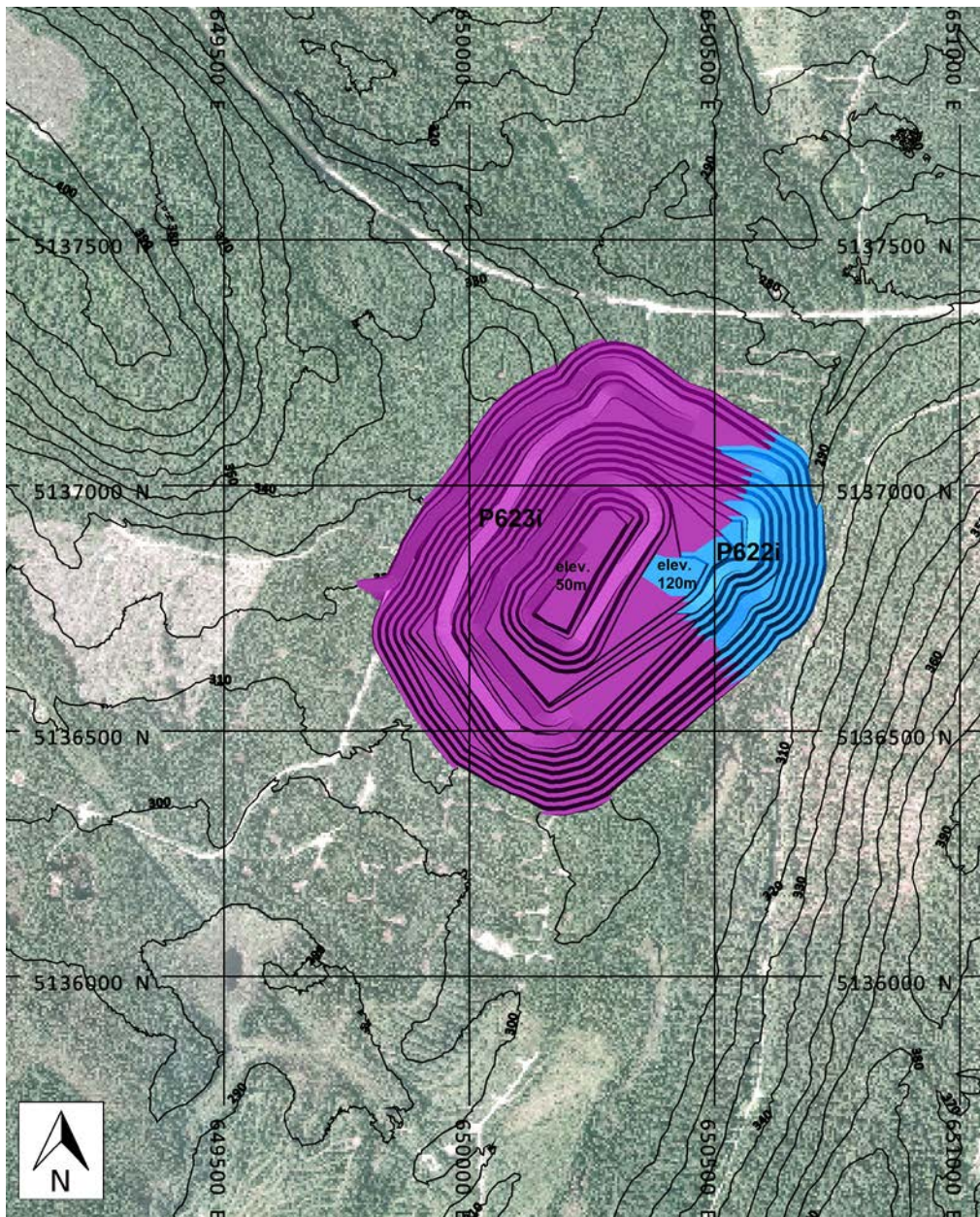


**Figure 16.4: Detailed Pit Design, Phase 2 (P622i)**



Detailed Pit Design, Phase 3 (P623)

Phase 3, illustrated in Figure 16.5, mines out most of Phase 2 and all of Phase 1 and establishes the pit exit for the remainder of the life of mine. This phase mines from surface (approximately 340 masl) down to 50 masl, and mines to the ultimate pit limit on the northwest side. The ramps are designed to allow future phases to tie into the permanent ramps on the northwest high wall. Ramps to the bottom two benches are single lane only.

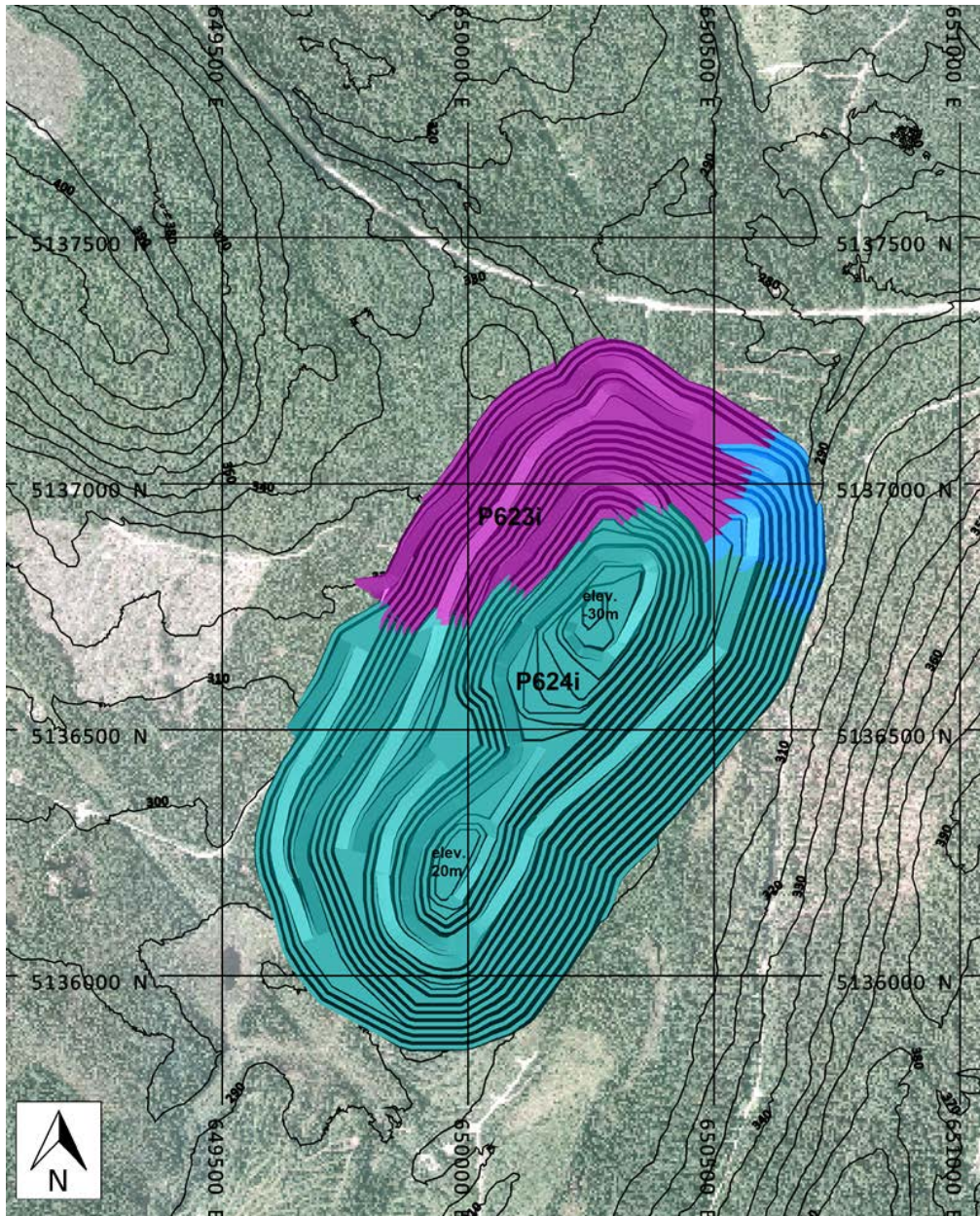


**Figure 16.5: Detailed Pit Design, Phase 3 (P623i)**



Detailed Pit Design, Phase 4 (P624)

Phase 4 is the largest phase, illustrated in Figure 16.6, and requires significant pre-stripping. This phase mines from surface (approximately 310 masl) down to two separate pit bottoms, 20 masl on the south end and -30 masl on the north end (ultimate pit bottom). This phase mines to the ultimate pit limit on the south and southeast side. An unused ramp is cut into the southeast highwall to ensure access to the pit bottom during later mining phases. Phase 4 also has a separate pit exit for mining activities on the upper benches.

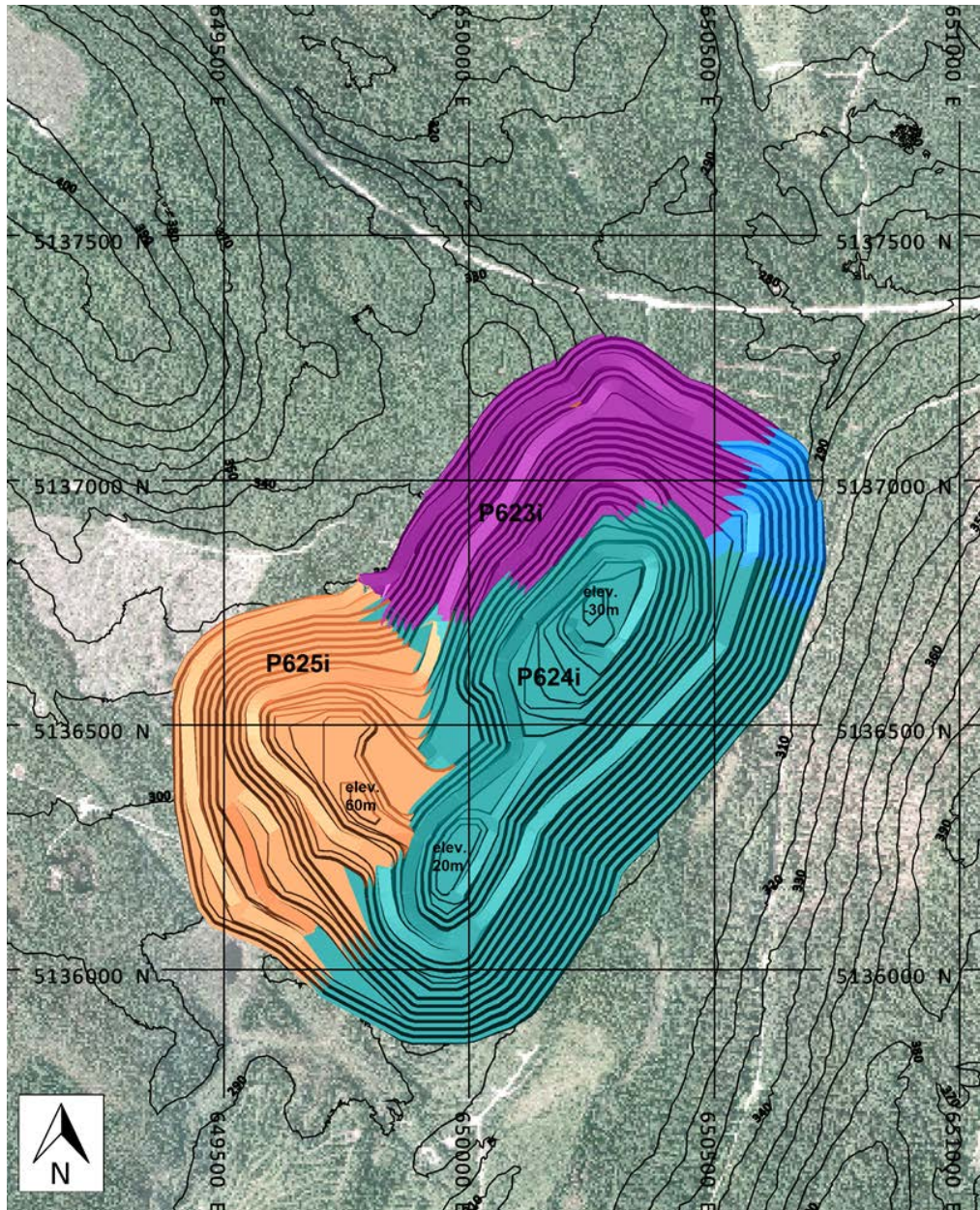


**Figure 16.6: Detailed Pit Design, Phase4 (P624i)**



Detailed Pit Design, Phase 5 (P625)

Phase 5, illustrated in Figure 16.7, is a push back to the west edge of the ultimate pit limit which mines from surface (approximately 320 masl) down to 60 masl. The upper half of Phase 5 is hauled independently out of the pit, merging with the existing ramps only at the pit exit. The remainder of this phase relies heavily on tie-ins to existing ramps from Phase 3 and 4. The bottom of Phase 5 is hauled out into Phase 4 on a short double lane ramp. This phase includes some sections of steeper ramps (up to 10% grade); however, they only see loaded uphill hauls and unloaded downhill hauls.

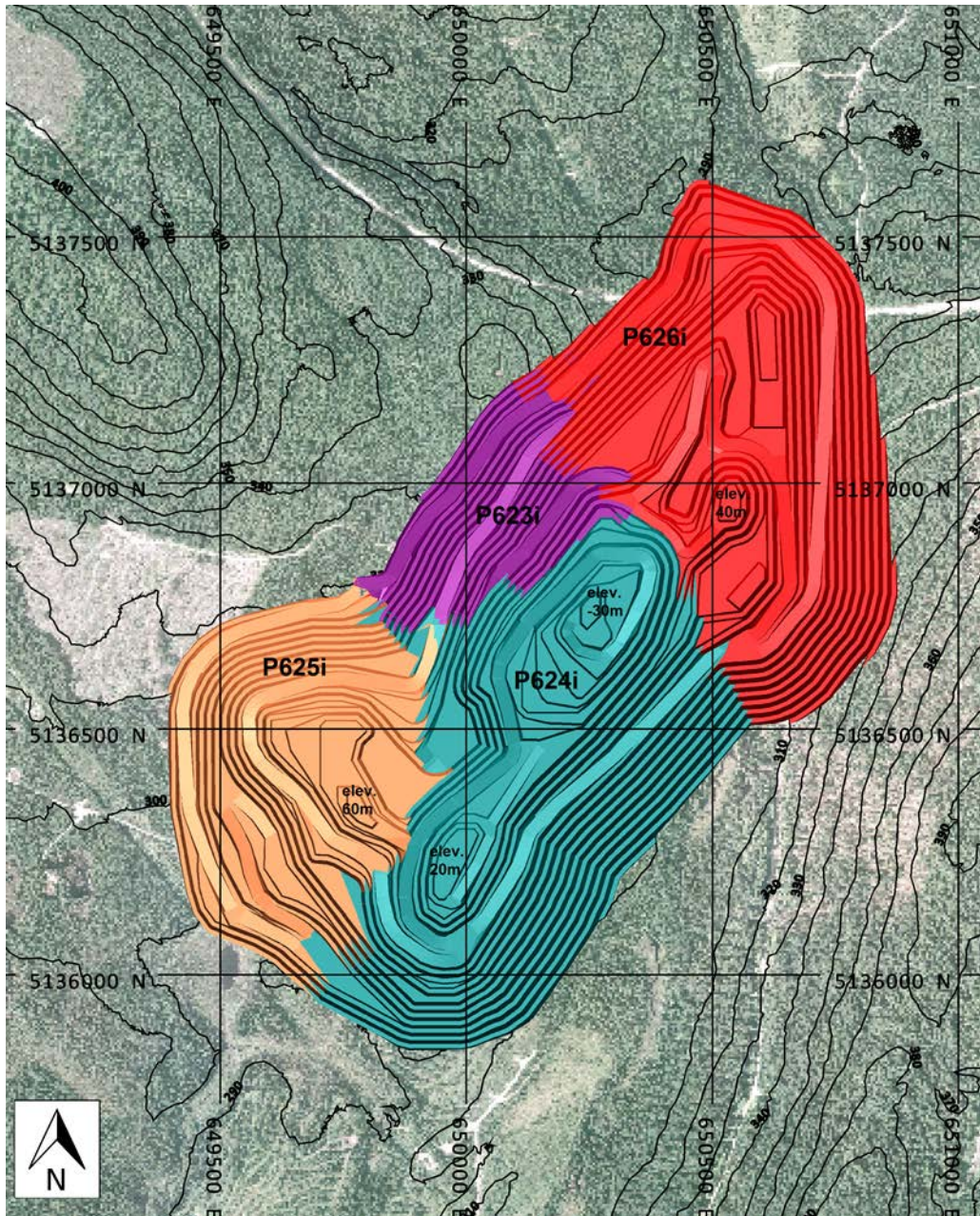


**Figure 16.7: Detailed Pit Design, Phase 5 (P625i)**



Detailed Pit Design, Phase 6 (P626), Ultimate Pit Limit

Phase 6 is the final push back to the ultimate pit limit, illustrated in Figure 16.8. On the north end it is a substantially large pushback. However, it narrows down on the east side to a minimum 100 m mining width. The phase mines from surface (approximately 330 masl) down to 40 masl. Early pre-stripping is hauled out independent of other phases, after which material is hauled out though Phase 3. The switchback at 150 masl ties into the unused highwall ramp from Phase 4 and allows activity elsewhere in the pit to continue throughout mining of Phase 6. Ramps to the final two benches are single lane only.



**Figure 16.8: Detailed Pit Design, Phase 6 (P626i)**

### Detailed Pit Design, Reserves by Phase

The reserves, as summarized by the detailed pit design phases, are shown in Table 16.1. It is important to note that these reserves are generated prior to any scheduling of phases and as such use a non-optimized NSR CoG, of \$8.83/t (the break-even NSR). In reality, scheduled reserves are less than those reported in Table 16.1 due to CoG optimization and scheduling, which use a variable CoG by period with a minimum NSR CoG of \$9.29/t. Using a higher CoG in effect will create smaller selective mining areas on the operating pit benches and increase the impact of loss and dilution. The loss and dilution study recognizes this fact and as such loss and dilution is modeled using the higher more conservative values from an estimated scheduled average NSR CoG of \$11.85/t.

Table 16.1 Detailed Pit Design, Reserves by Phase						
Phase	Ore kt	NSR (\$/t)	WO <sub>3</sub> %	Mo%	Barren kt	S/R*
Phase1	CoG: 8.83		Loss: 5.11%		Dilution: 3.87%	
Proven	12310	30.96	0.0856	0.0230	-	-
Probable	14288	27.12	0.0765	0.0193	-	-
Total	26598	28.90	0.0807	0.0210	9243	0.35
Phase2	CoG: 8.83		Loss: 5.42%		Dilution: 4.18%	
Proven	20585	30.77	0.0807	0.0280	-	-
Probable	16664	29.66	0.0819	0.0223	-	-
Total	37249	30.27	0.0812	0.0255	12856	0.35
Phase3	CoG: 8.83		Loss: 7.21%		Dilution: 5.97%	
Proven	19595	26.67	0.0720	0.0235	-	-
Probable	35343	25.86	0.0735	0.0184	-	-
Total	54938	26.15	0.0730	0.0202	45198	0.82
Phase4	CoG: 8.83		Loss: 8.41%		Dilution: 7.17%	
Proven	37291	21.43	0.0559	0.0240	-	-
Probable	98161	21.46	0.0552	0.0248	-	-
Total	135452	21.45	0.0554	0.0246	82046	0.61
Phase5	CoG: 8.83		Loss: 7.01%		Dilution: 5.77%	
Proven	5861	18.27	0.0546	0.0139	-	-
Probable	28551	21.44	0.0616	0.0167	-	-
Total	34412	20.90	0.0604	0.0162	25867	0.75
Phase6	CoG: 8.83		Loss: 9.37%		Dilution: 8.13%	
Proven	9773	24.86	0.0767	0.0111	-	-
Probable	35941	24.35	0.0748	0.0113	-	-
Total	45714	24.46	0.0752	0.0113	59074	1.29
Total	CoG: 8.83		Loss: 6.73%		Dilution: 5.52%	
Proven	105415	25.48	0.0691	0.0228	-	-
Probable	228948	23.54	0.0652	0.0202	-	-
<b>Total</b>	<b>334363</b>	<b>24.15</b>	<b>0.0664</b>	<b>0.0210</b>	<b>234284</b>	<b>0.70</b>

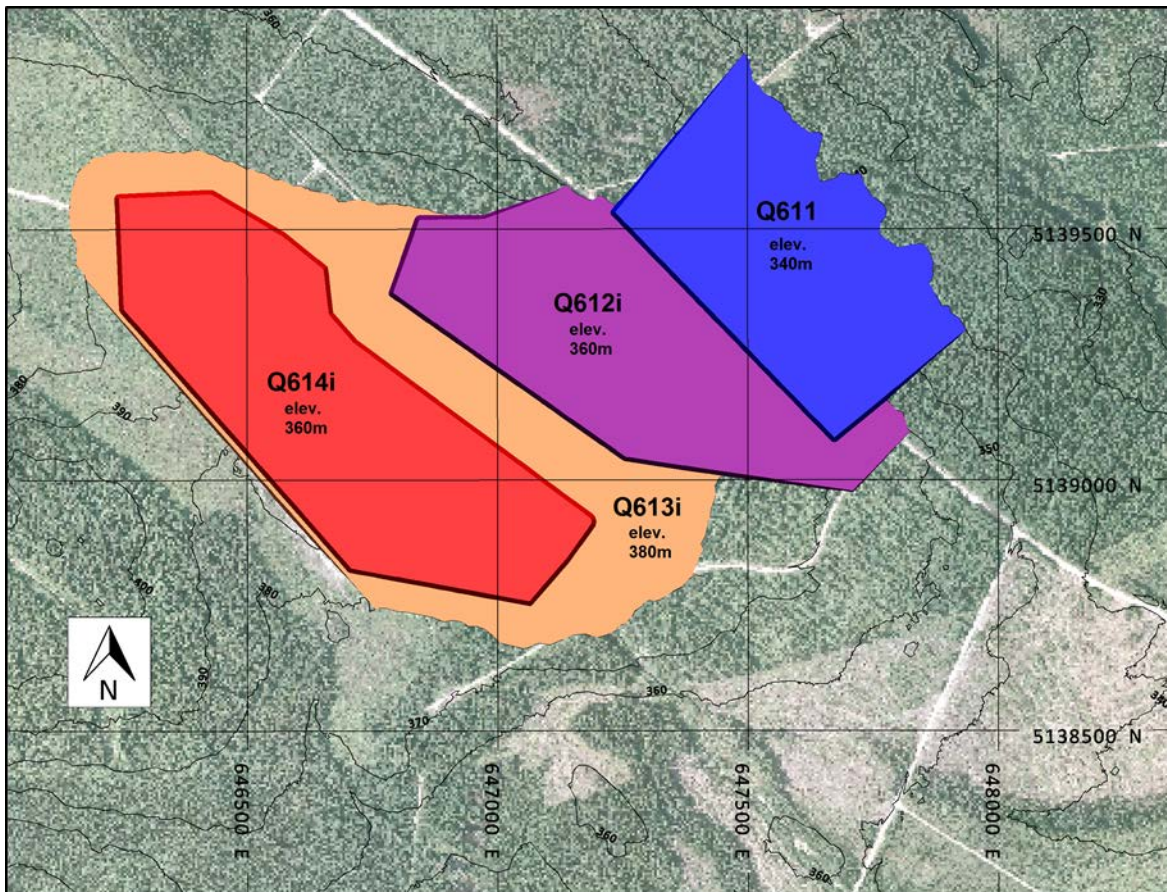
\*Strip ratio (Barren/Ore) comparative purposes only, does not represent optimized scheduled S/R as it does not use an optimized CoG.

### 16.1.6 Detailed Quarry Design

Due to the presence of potentially acid generating barren rock within the open pit, the tailings embankment cannot be constructed using material from the open pit. As such, material for tailings



embankment construction will be sourced from a separate, non-potentially acid generating quarry site. The quarry will be located within the footprint of the TSF to minimize surface disturbance and is therefore designed as a tiered, 4 phase design to accommodate the tailings filling schedule. The design also allows for the future operations to direct any surface run-off into the TSF and not impact any other drainage areas. Detailed quarry designs use the same design parameters as the open pit. Phases 1-4 (Q611, Q612, Q613 and Q614) of the detailed quarry design are presented in Figure 16.9



**Figure 16.9: Detailed Quarry Design, Phases 1-4**

Phase 1, Q611 on Figure 16.9, contains approximately 8.2 Mt of material and is intended to provide early quarried rock for embankment construction, mining from surface (approximately 370 masl) down to 340 masl. While more volume could be made available by mining to a lower elevation, this would reduce the window of available time to mine Phase 1 due to the rapid advancement of the tailings elevation during the early years of the mine life. Phase 1 will be completely filled by tailings by the time the mine life is complete.

Phase 2, Q612 on Figure 16.9, mines approximately 18.3 Mt from surface (approximately 400 masl) down to 340 masl. This provides an additional 20 m of elevation for quarry operations to remain ahead of the tailings elevation, which is now advancing at a more linear rate. There is still, however, a fixed window of time where Phase 2 must be mined, as tailings will eventually partially fill this phase.



Phase 3, Q611 on Figure 16.9, mines the hill top down from surface (approximately 410 masl) to 380 masl. There are approximately 25.4 Mt of material in this phase. With the final elevation at 380, this phase will remain above the tailings after mine closure. The footprint for this phase was designed to reduce its impact on natural drainage as much as possible.

Phase 4, Q611 on Figure 16.9, is the final phase of the quarry; it contains approximately 19.3 Mt of material, though not all of this material may be required. It is a 20 m sink cut at the bottom of Phase 3 (380 masl) that provides the remaining balance of required quarried rock fill for the tailings embankment. Mining of Phase 4 will occur when tailings have already exceeded 360 masl, as such, a minimum set back of 100 m between Phase 4 and Phase 2 is observed to reduce the risk of seepage into Phase 4. At mine closure, a channel will be cut in this 100 m setback to facilitate flooding of Phase 4 of the quarry.

#### **16.1.7 Barren Rock Storage Areas (BRSA)**

This study designates any material mined from the open pit that is not ore as Barren Rock. Barren rock can still possess tungsten and/or molybdenum mineralization; however, not in quantities significant enough to designate it as ore. If mineralization of the barren rock is reasonably significant it is designated as Mid-Grade and stored in the Mid-Grade Stockpile also located within the TSF footprint and water catchment. As the TSF fills with tailings over the course of the mine life, the BRSA will be encapsulated within the tailings.

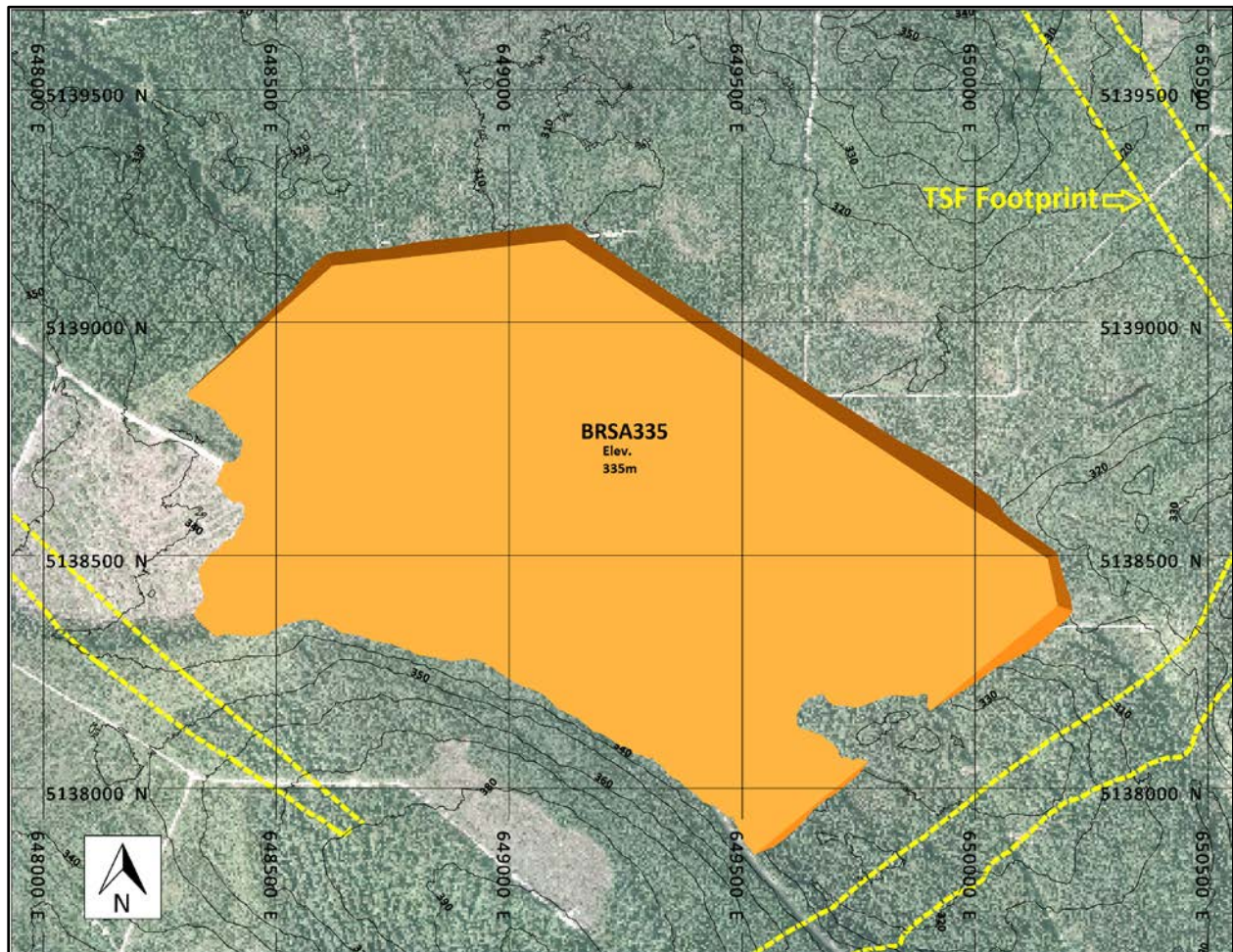
All BRSA are designed with a natural angle of repose of 37°. A 30% swell factor is applied to in situ volumes to calculate the volumes that need to be placed.

##### Construction Methods

Barren rock is placed using primarily bottom-up construction methods. Bottom-up placement involves the truck placing the material in lifts and constructing the BRSA to final limits from the bottom working upwards. Some top-down placement, involving trucks dumping the material from the top bench crest down to the platform or topography below will occur on the outer edges of subsequent lifts as well as during back filling operations.

##### BRSA 335 Construction Detail

The first lift of the BRSA, illustrated in Figure 16.10, is at an elevation of 335 masl and has a capacity of approximately 59 Mt. As it is built on the topographic surface, the thickness varies considerably. The maximum thickness is approximately 35 m. Construction begins on the Southern corner of the lift and expands outwards.

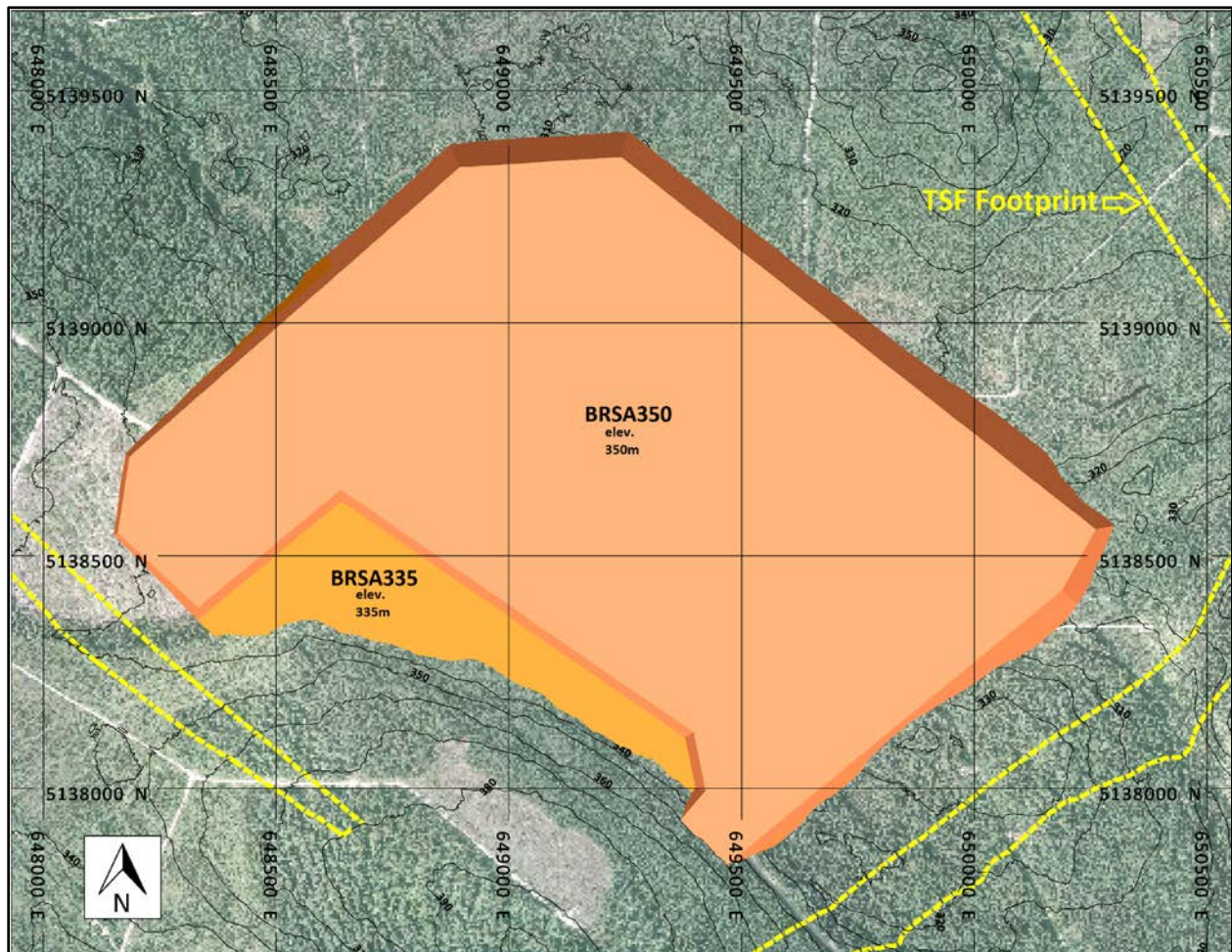


**Figure 16.10: Barren Rock Storage Area 335**



BRSA350 Construction Detail

The second lift of the BRSA, illustrated in Figure 16.11, is at an elevation of 350 masl and has a capacity of 67 Mt. As tailings have begun to encroach on the 335 m elevation it is necessary to add another lift on top of the previous BRSA335 lift. The majority of this lift is 15 m thick, except on the Northeast edge where the toe spills out approximately 180 m beyond the toe of the previous lift. When free-dumping off the Northeast side, the lift height can reach a maximum of approximately 50 m; however, a portion of this height is actually comprised of dumping into the tailings pond. On the Southwest side, a pocket is left open to accommodate the mid-grade stockpile. Construction begins on the southern corner of the lift and expands outwards.

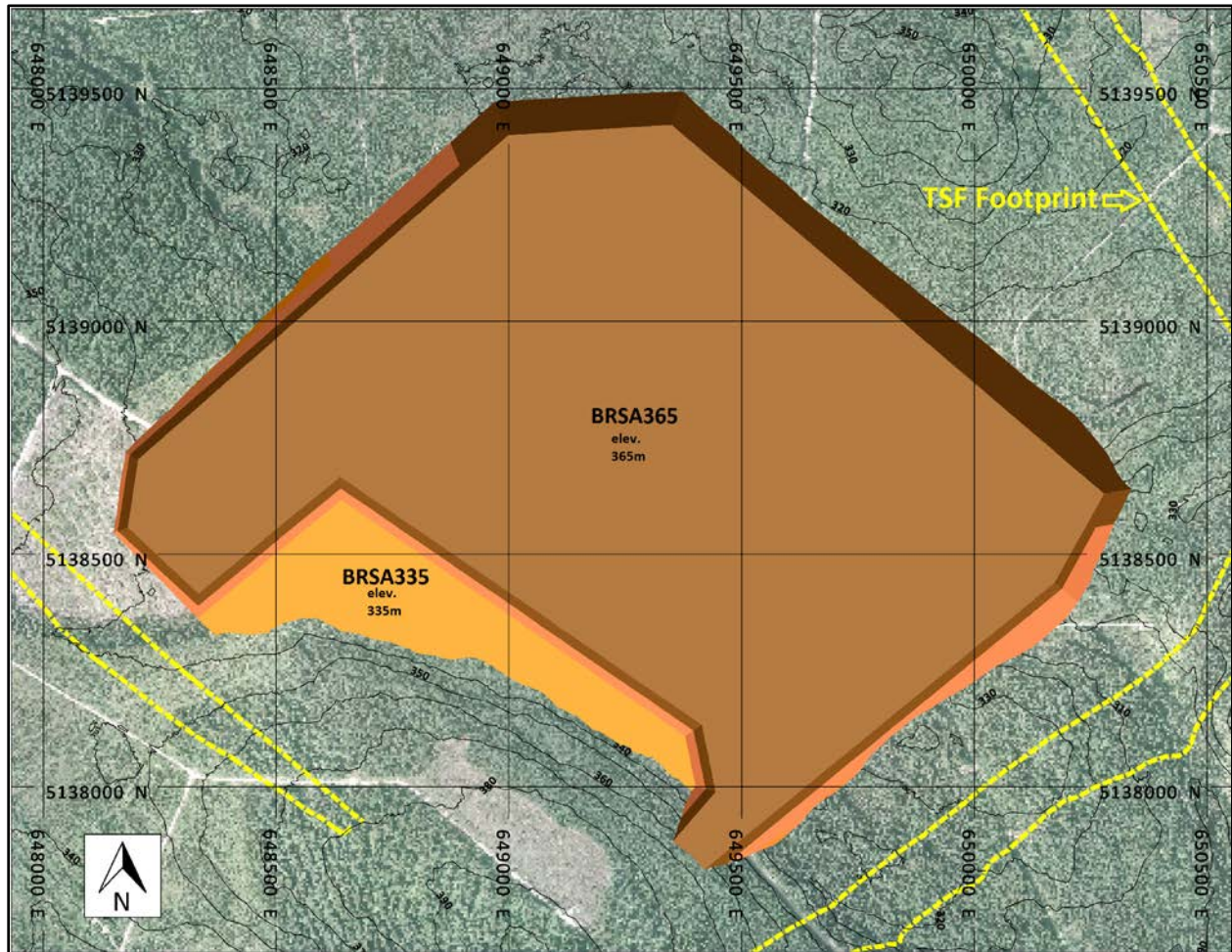


**Figure 16.11: Barren Rock Storage Area 350**

BRSA 365 Construction Detail

The third and final lift of the BRSA, illustrated in Figure 16.12, is at an elevation of 365 masl and has a capacity of 66 Mt. With the final tailings elevation estimated to be 372 masl, this puts the BRSA submerged a minimum of 7 m below the tailings surface at closure. The typical thickness of this final lift is 15 m, except on the Northeast edge where the toe spills out approximately 110 m beyond the toe of the

previous lift. When free-dumping off the northeast side, the lift height can reach a maximum of approximately 65 m. By the time this lift is required, tailings will have encroached upon the previous BRSA350 lift and therefore a portion of this height is actually comprised of dumping into the tailings pond. On the Southwest side, the final lift is toed into the previous lift to continue to accommodate the mid-grade stockpile. Construction begins on the Southern corner of the lift and expands outwards.



**Figure 16.12: Barren Rock Storage Area 365**

Mid-Grade (MG) Stockpile

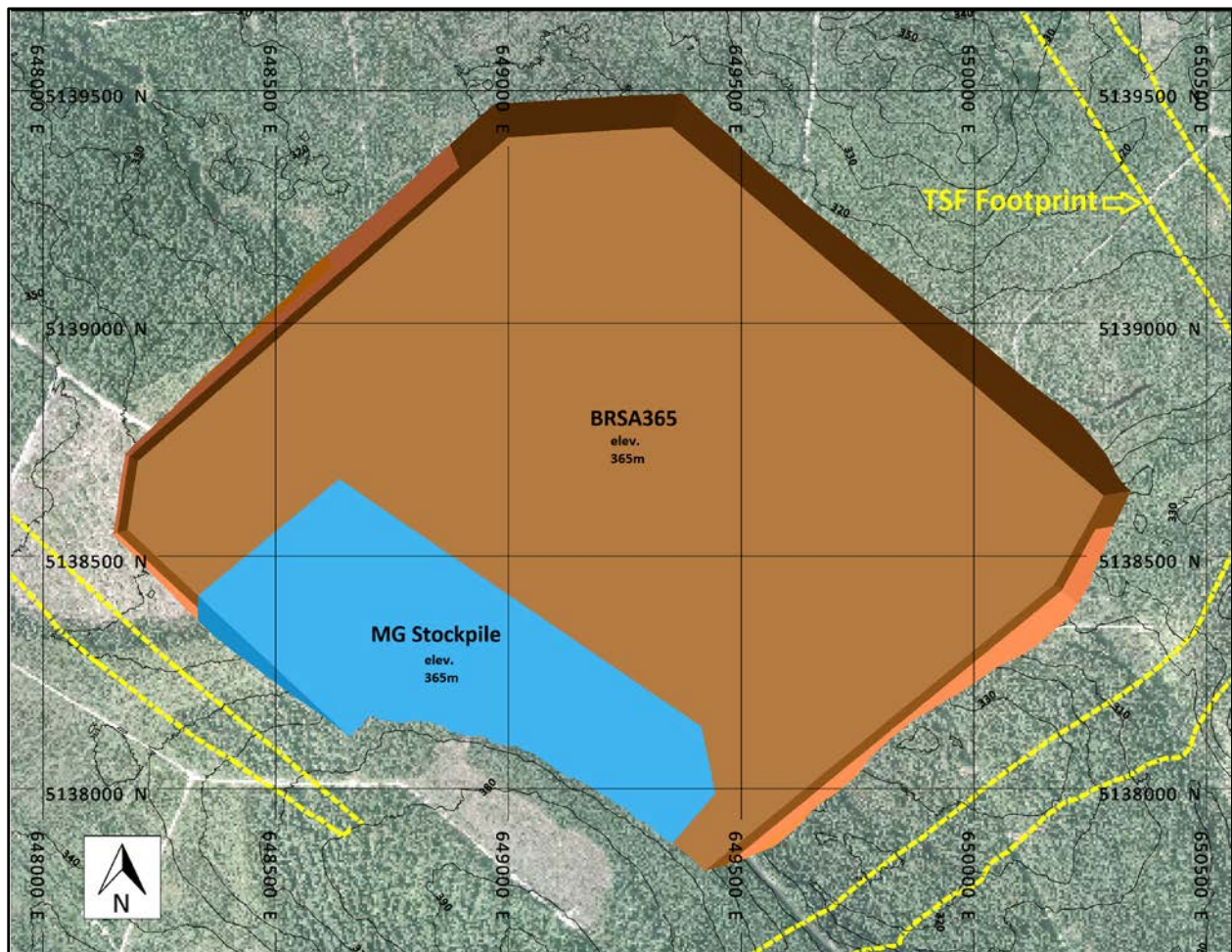
The MG stockpile, illustrated in Figure 16.13, is constructed in a free-dumping, top-down method, expanding outwards from the 365 m contour line. The bottom of the MG stockpile sits on top of the first BRSA lift at 335 masl. This makes the typical thickness of the MG stockpile approximately 30 m. As the MG stockpile expands throughout the mine life, it toes into the 37° (angle of repose) slopes of the BRSA ensuring that no MG stockpile material is buried beneath BRSA lifts, even after the BRSA is complete. Constructed in this manner, there is capacity for almost 17 Mt of mid-grade material.

In theory this would allow all MG stockpile material to be reclaimed, even after tailings overtops the 335 m elevation, if appropriate measures were taken to ensure that tailings seepage did not infiltrate the MG



stockpile (i.e. liners, dykes, drains, etc.). This is a potentially costly endeavor and as such no such measures to prevent infiltration have been taken. Therefore, the current mine plan does not reclaim any of the MG stockpile at any point during or after the mine life.

Once tailings overtops the BRSA, the midgrade material in the pit is no longer segregated and stockpiled, the material instead reports to backfilling in the pit along with all other barren rock.



**Figure 16.13: Barren Rock Storage Area Final, Including Mid-grade Stockpile**

Backfilling

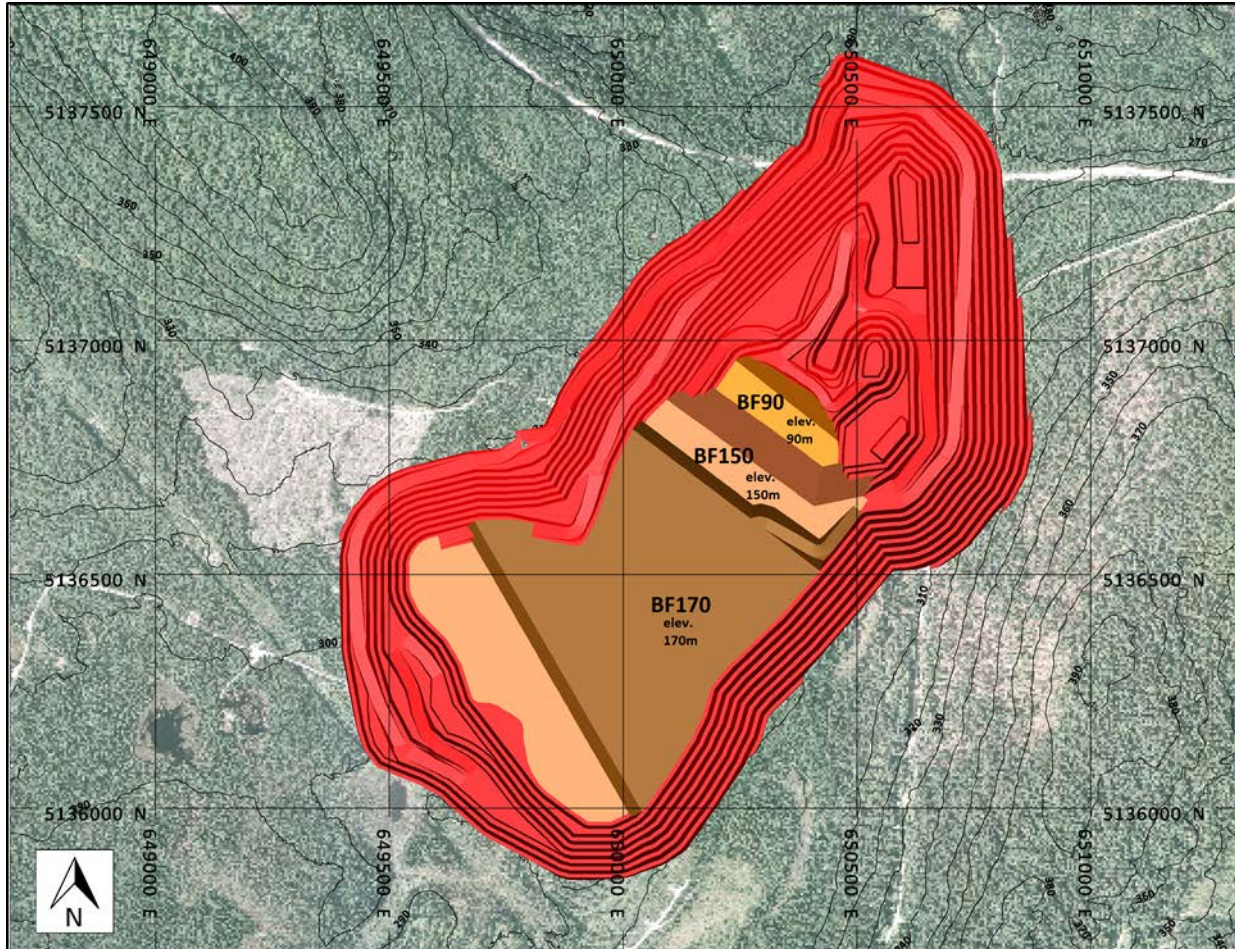
Backfilling barren rock into the mined out pit begins in year 21, when there is a suitable mined out pit bottom, and continues through LOM. It should be noted that when backfilling begins, production is from lower benches and incremental economics designates the midgrade material to the backfill rather than hauling it out of the pit. Backfilling occurs in three stages, as illustrated in Figure 16.14.

The first stage of backfill has capacity for 24.3 Mt of barren rock and mid-grade material. This stage is at an elevation of 90 m with a maximum fill height of 120 m. The second stage of backfilling has capacity for 42.4 Mt of material, and fills the entire South half of the pit up to the 150 m elevation.



The final stage of backfill is a 20 m lift, with a final elevation of 170 masl and capacity for 11.6 Mt.

In all stages, the back fill is constructed in such a way that any active working areas below the toe of an active fill slope is protected from dangers posed by rolling rocks or debris.



**Figure 16.14: Final Backfill**

### 16.1.8 Mine Plan, Open Pit and Quarry

The description of the mine plan, including both open pit and quarry operations, is detailed in Section 16.2.5, 'Detailed Mine Plan'.

### 16.1.9 Open Pit Mine Operations

Open pit mine operations for the Sisson Project are typical for Canada and employ accepted mining equipment and techniques. There is considerable operating and technical expertise, services, and support in Canada, as well as access to a readily available labor force in the province of New Brunswick.



## General Organization

Mine operations are organized into three departments: mine maintenance, direct mining, and technical services. Other areas of the organization are dealt with elsewhere in the report. The mine department is estimated to employ 141 people initially and increase to 173 people at the peak of mine operations.

The mine maintenance department reports to the General Manager through the Maintenance Manager. Under the supervision of the General Maintenance Foreman, mine maintenance accounts for supervision, planning, and implementation of all maintenance activities pertaining to the mine fleet or direct mining infrastructure, whether in the maintenance shops or in active mining areas.

The direct mining and technical services departments report to the General Manager through the Mine Manager. Direct mining, under the supervision of the General Operations Foreman, accounts for supervision, training, and implementation of all drilling, blasting, loading, hauling, and pit maintenance activities in the mine. It also accounts for any other areas where mine fleet activity is present, such as the construction of haul roads or BRSAs. Technical services, under the supervision of the Chief Engineer, account for all technical support from mine planning, geology, surveying and mine engineering personnel.

In this study, direct mining and mine maintenance were planned as an Owner operated fleet with the equipment ownership and labor being directly under operations. It may be possible to contract out some of the direct mining activities under typical mine stripping contracts and maintenance and repair contracts as per other operating mines. The viability and cost effectiveness of contracting can be determined in future detailed planning and commercial negotiations. The exception for this study involves blasting where, due to the specialty expertise required, the supply and onsite manufacturing of blasting materials, blasting crew and their immediate supervisor are assumed to be contracted out. All infrastructure required for the blasting supply contractor will be purchased by the Owner. The mine will employ a drilling and blasting foreman that will act as a liaison between the contractor and the mine, and a drilling and blasting engineer to design and manage the blasting operation.

## Mine Maintenance

Mine maintenance accounts for the supervision and planning of the mine maintenance activities. Mine maintenance activities will be directed under the Maintenance General Foreman who will assume overall responsibility for mine maintenance and will report to the Maintenance Manager, who will in turn report to the General Manager. Maintenance planners will coordinate planned maintenance schedules. The daily maintenance shift coordination will be carried out by a maintenance foreman during day shift and by the mechanical foreman during night shift. Mine maintenance is also responsible for maintaining the warehouses and shops.

The mine maintenance department will perform breakdown and field maintenance and repairs, regular preventive maintenance, component change-outs, in-field fuel and lube servicing, and tire change-outs. Major component rebuilds will be done by specialty shops off-site.

## Direct Mining

Direct mining accounts for the drilling, blasting, loading, hauling, and pit maintenance activities in the mine.

In situ rock will require drilling and blasting to create suitable fragmentation for efficient loading and hauling of both ore and barren rock. Ore limits will be defined in the blasted muck pile through blast hole, assays and grade control technicians. Support personnel and equipment will be required to maintain the mining area, ensuring the operation runs safely and efficiently. General descriptions of the direct mining unit operations are included in this section but specific equipment descriptions and specifications are presented later in Section 16.1.11, 'Mine Equipment'.

### Drilling

Areas will be prepared on the bench floor for blast patterns in the in situ rock. The spacing and burden between blast holes will be varied as required to meet the specified powder factor for the various rock types. The drill operators will be responsible for blast hole sampling for the ore control system (OCS). If the future operation decided to install automatic samplers, the drill will be responsible for bagging and tagging the drill cutting from the sampler for shipment to the assay lab. If manual sampling is done, the driller will be responsible for taking the samples from the drill cuttings, and bagging and tagging it.

Controlled blasting techniques will be used for high wall rows, pioneering drilling during pre-production, and development of initial upper benches. Where required, dozers will be used to establish initial drilling benches for the upper portions of each phase.

### Blasting

#### *Powder Factor*

An appropriate powder factor will be used to provide adequate fragmentation and digging conditions for the shovels, with a targeted top-size between 1400-1500 mm for barren rock and quarried rock. The powder factors selected will be dependent on rock types and will therefore vary. In barren rock, the powder factor will range from 0.14-0.18 kg/t, while in quarried rock it will range from 0.17-0.19 kg/t.

Powder factors for ore are higher as additional explosive effort is used to increase the fragmentation to meet the ideal size distribution specified for mill feed by the process modelers. The mill feed size distribution is listed in Table 16.2. To achieve this size distribution powder factors for mill feed range from 0.18-0.23 kg/t, depending on the rock type.

<b>Table 16.2 Target Size Distribution for Mill Feed</b>	
	<b>mm</b>
Top-size	1100
F80	730
F50	360
F20	80

More detailed information is available in the drill and blast studies found in the Feasibility Study.

### *Explosives*

A contract explosives supplier will provide the blasting materials and technology for the mine, as well as manufacture bulk explosives on site. The nature of the business relationship between the explosives supplier and the mining operator will determine who is responsible for obtaining the various manufacture, storage and transportation permits, as well as any necessary licenses for blasting operations. This will be established during commercial negotiations. For this study, the explosives contractor delivers the prescribed explosives to the blast holes and supplies all blasting accessories. Blasting accessories will be stored in magazines.

Specifications for blasting plant and explosives storage magazines and the locations of these facilities must adhere to the Explosives Act of Canada regulations as published by the Explosives Regulatory Division of Natural Resources Canada. The location of the blasting plant and the explosives magazines are determined by the table of distances that govern the manufacturing and storage of explosives and blasting agents. The contractor will be responsible for proper placement of magazines and facilities.

Different contractors have various explosives products and specifications. The chosen contractor will be responsible for providing all MSDS and product fact sheets as applicable. For this study, all contract explosives providers recommended 100% emulsion products.

### *Explosives Loading*

Loading of the explosives will be done with bulk explosives loading trucks provided by the explosives supplier. The trucks should be equipped with GPS guidance or otherwise tied into the in-pit data network, and should be able to receive automatic loading instructions for each hole from the engineering office.

Explosives loading will be carried out by the contractor's crews with their immediate supervisor; however, they must report to the owner's Drilling and Blasting Foreman who will work alongside the Drilling and Blasting Engineer to ensure that explosives loading is carried out according to the mine's specific needs.

The blast holes will be stemmed to avoid fly-rock and excessive air blasts. Crushed rock will be provided for stemming material and will be dumped adjacent to the blast pattern. A small front-end loader, owned and operated by the mine, will be used for blast hole stemming. Any crushed rock required for blast hole stemming will be provided by the onsite rock crusher specified for mine roads and quarrying operations.

### *Blasting Operations*

The blasting crew will be provided by the contractor and will coordinate the drilling and blasting activities to ensure a minimum of two weeks of broken material inventory is maintained for each shovel. During winter months, the drilled holes may need to be covered due to the snow. Also, the blast patterns will not be staked; therefore, the blasting activities will also need to be tied into the in-pit data network. The blasters will require hand-held electronic location devices to identify the holes for the pattern tie-in. The pattern size may be limited by the rate of snowfall in some months. A detonation system will be provided by the contractor and will consist of an initiation device, detonating cord, surface delay connectors, and boosters.

Blasting activities are day shift activities only and can only occur during the five days each week that the mine's Drilling and Blasting Foreman is present.

### Loading

Ore will be defined in the blasted muck pile as determined by the dig lines generated by the OCS. The fleet management systems will provide on-board dig maps and assist in optimizing deployment and utilization of the loading units into ore and barren rock headings, to meet the short term production plan. The fleet management systems are able to communicate directly with the mine geology model and the OCS dig limits, therefore accurately define ore or barren rock zones for the shovel operator.

Bench widths are designed to ensure operating room is suitable for efficient double-sided loading of trucks at the shovels. This study does not reduce shovel productivities in areas where double-sided loading is not possible (such as the upper benches of the pit phases where the end of the bench meets topography), as it is assumed that ancillary equipment will be deployed in non-productive operating areas, to prepare the digging areas for higher shovel productivity. This can entail dozing small benches down slope to the next bench, trap dozing, and other dozing activities.

After mill start-up, there will be a requirement each year to mine a given quantity of quarried rock. The intention is to campaign mine the required quarry tonnes for one or two months each year. During these months, it is intended to relocate a single shovel and matching truck fleet to the quarry area for the required length of time. When not operating in the quarry, this shovel can be utilized elsewhere for support or production activities. If it is not required elsewhere it may at times sit idle in the quarry.

Any significant move of a primary loading unit will be carried out with the use of a rental low-boy truck and trailer in order to reduce wear and tear on the shovel, reduce down time, and to minimize the cost of electric cable moves.

### Hauling

Haulage of ore, barren rock, and quarried rock will be handled by off-highway 136-t haul trucks. The trucks will be outfitted with Fleet Management systems. See Section 16.1.11, 'Loading' for additional information.

The study is based on a detailed haulage network database, estimated and designed using MineSight® haulage software. Haulage profiles were created from the centroid of every bench of every phase of the open pit to each unique destination location (such as a BRSA lift, the primary crusher, MG stockpile, etc.). These haul profiles were input into the schedule optimizer, which is set to maximize project NPV by using the lowest cost haul to a feasible destination. Due to the wide variety of potential destination locations for quarried rock along the circumference of the TSF embankment, no haul profiles were built in MS Haulage for the quarry. Instead, conservative default average cycle times were input for all quarry activity. The payload, loading time, and haul cycle then determine the truck productivity.

### Pit Maintenance

Pit maintenance services include haul road maintenance, mine dewatering, transporting operating supplies, relocating equipment, snow removal, and pit floor clean-up. Mine pit de-watering is presented in Section 18.12.3.

Haul road maintenance is necessary for low haulage costs; dozer, grader, and water truck hours are allocated and adjusted to maintain the haul road network throughout the LOM production schedule.

Regular application and grading of crushed rock to haul road surfaces improves truck travel speeds, reduces mechanical fatigue to the haul trucks, and enhances tire life, which is a major mine operating cost. Crushed rock requirements for road maintenance are incorporated into the crushing and screening operation located at the quarry.

Additional ancillary equipment is included in this section such as maintenance vehicles, crew and supervisor pickup trucks, cranes and other support equipment.

### General Mine Expense (GME)

The GME portion of direct mining accounts for the supervision, safety, environmental, and training for the direct mining activities, typically pertaining to salaried direct mining staff. Direct mining supervision will extend down to the Shift Foreman level.

The Operations General Foreman will assume responsibility for overall supervision for the mining operation and will be responsible for open pit supervision and equipment coordination. He reports to the Mine Manager, who in turn reports to the General Manager. A mine Shift Foreman is required on each 12 hour shift, with overall responsibility for the shift. A day shift only, (five days a week) Drilling and Blasting Foreman will provide supervision for drilling and blasting during day shifts. When not present, no blasting activities may occur; however, drilling can be supervised by the Shift Foreman.

Initial training and equipment operation will be provided by experienced operators. As performance reaches adequate levels, the number of trainers can be decreased to a sustaining level, with a single Training Foreman providing supervision for continuous day shift only shift trainers.

Direct mining GME also includes annual allowances for general equipment rental, licensing and maintenance of mine design and fleet management software.

### Technical Services

Technical Services accounts for the technical support from mine engineering, planning, geology and surveying functions. The majority of Technical Services activities fall under GME.

The Chief Mine Engineer will direct the Technical Services department and will report to the Mine Manager, who in turn will report to the General Manager. The Senior Mining Engineer will coordinate mining engineering, drilling and blasting engineering, mine planning, and surveying. The Senior Geologist will be responsible for local step out and infill drill programs for onsite exploration activities and updating the long range mine ore body models. The geology department will also provide grade control support to

mine operations, and will manage and execute the blast hole sampling and blast hole kriging of the short range blast hole models for operations planning and ore grade definition.

A separate Project Engineer will assume split responsibilities for all mine geotechnical issues including pit slope stability, monitoring, and hydro-geological studies, as well as TSF engineering. The Project Engineer will also have oversight for the whole property for any geotechnical monitoring and assessment programs being carried out by safety personnel or third party consultants, or any other unspecified projects on the property.

Technical Services GME also includes engineering consulting on an ongoing basis for specialty items such as geotechnical, operations support, environmental and geo-hydrology expertise and third-party reviews. In addition, in-fill exploration drilling is included as an allowance in the GME in the year prior to phase push-backs.

### 16.1.10 Mine Closure and Reclamation

At mine closure, the BRSA inside the TSF footprint will already be submerged beneath a minimum of seven meters of tailings and therefore no reclamation efforts will be required. The remaining barren rock will be back filled into the pit and the pit flooded to ensure no barren rock is left exposed after closure. In order to facilitate flooding of the open pit, ahead of natural precipitation and run-off, a spill-way channel will be cut into the TSF embankment to allow additional volume to be shifted from the TSF to the open pit.

The majority of the quarry will also be submerged beneath the TSF by closure; the exception is the Phase 4 sink cut. At closure, a channel will be cut to allow Phase 4 to flood with TSF supernatant.

All disturbed areas that remain un-submerged will be reclaimed and re-seeded as required. Closure and reclamation plan for the Sisson Project is discussed further in Section 20.6.

### 16.1.11 Mine Equipment

The mining equipment descriptions and specifications in this section provide general information of the size, dimension, capacity, etc. of the selected equipment. These specifications are not intended to target equipment from any specific manufacturer or vendor.

The complete mining fleet is summarized in Table 16.3 showing the initial Y-1 requirement, and the max LOM requirement.

Table 16.3 Mobile Fleet			
Mine Mobile Fleet	Task / Description	Initial Qty	LOM Max Qty
<b>Drilling</b>			
Drill - Diesel Hydraulic - 165 mm	Primary Drill	1	3
<b>Blasting</b>			
Blasthole Loader - 75 kW	Blast hole stemmer	1	1
<b>Loading</b>			



<b>Table 16.3 Mobile Fleet</b>			
<b>Major:</b>			
ELEC Hydraulic Shovel - 16.5 m <sup>3</sup>	Loading Ore & Barren	1	3
<b>Support:</b>			
Dozer – 433 kW	Shovel support, pit ramps and roads	1	1
Wheel Dozer – 372 kW	Pit clean up	1	1
<b>Hauling</b>			
<b>Major:</b>			
Haul Truck – 136t	Hauling Ore/Barren	3	14
<b>Support:</b>			
Water Truck – 4000 gal	Road maintenance	1	2
Water Truck – 20000 gal	Road maintenance	0	1
Dozer – 306 kW	Barren rock facility maintenance	1	2
Grader – 221 kW	Road Grading, maintenance	1	2
FEL -Tire Manipulator – 274 kW	Multi-Tool, tire changes, cable reeler	1	1
<b>PIT MAINTENANCE</b>			
Dozer – 306 kW	Pit Support	0	1
Excavator – 301 kW	Utility Excavator	1	1
Mobile Screening Plant	Road and Stemming Crush	1	1
Light Plant – 20 kW	Lighting plant	2	4
Forklift – 10t	Forklift	1	1
Forklift – 30t	Forklift	1	1
Fuel/Lube Truck - 4000 liters	Mobile Fuelling	1	1
Jaw Crusher	Road and Stemming Crush	1	1
FEL – 274 kW - loader	Crushing-Screening rehandle and loading	1	1
Crew Van - 15 Passenger	Crew Transport	2	2
Warehouse Truck – 1t	Warehouse truck	1	1
Crew Cab Pickup	Crew Cabs, Supervisor trucks	4	8
Service Truck – 1t	maintenance + overhauls	1	2
Welding Truck - 1t	Welding Truck	1	2
Picker Truck - 10t	Picker Truck	0	1
Dozer – 306 kW	Quarry / TSF Dozer	0	1

## Drilling

Primary production drilling at the Sisson Project utilizes diesel hydraulic rotary drills. These drills are outfitted with 165 mm drill bits for use in ore, barren rock, and quarry applications. Drills are outfitted with high precision drill positioning or GPS systems for efficient and accurate positioning, and superior data collection from each drill unit and drill hole.

The production drills operate in all highwall rows, final walls, and other controlled blasting areas. Drill requirements, including the highwall drilling, require 1 drill in Y-1, and 2 drills in Y1. As production increases, a third drill is added in Y6.

Detailed drill studies were requested from the drill manufacturers for the drilling equipment they recommended. However, the manufacturers declined. Instead, rock characteristics of density and compressive strength were provided and manufacturers provided estimates of drill penetration rates. These estimates are compiled by MMTS for this study and a median range penetration rate of 30 m/h is used.

Infill drilling has not been detailed in this study. However, an allowance is added in the mine GME area for Y-1, Y1 and Y2, and the year prior to new phase pushbacks for infill drilling.

## Blasting

Blasting at the Sisson property will be performed by contractors at a mine owned explosives storage and handling facility. The contractor will operate a proprietary MMU (Mobile Manufacturing Unit), which will deliver blasting materials to the blast pattern and mix them into explosives at the drill hole. This unit will be leased by the mine.

The mine will own and operate a small front-end loader for blast hole stemming. This loader will load drill cuttings, crushed rock, or gravel into the hole to stem it and more effectively direct the energy of the blast.

One 110 kW FEL (front end loader) is sufficient for the LOM; it will be purchased in Y-1.

## Loading

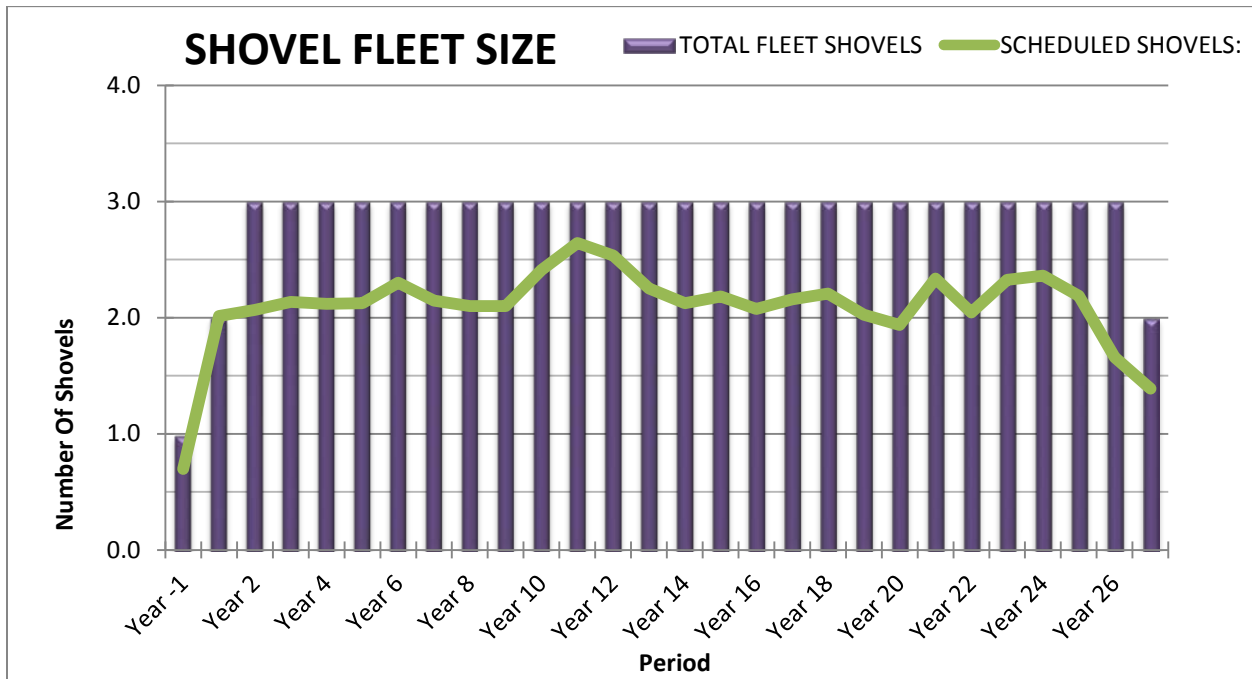
### Major Equipment

Production loading will be performed by 16.5 m<sup>3</sup> electric hydraulic shovel units. Electric hydraulic units have been chosen due to the proximity of electric infrastructure and the efficiency of electric drive units versus diesel units. Electric drive units achieve reduced operating costs and maintenance down-time compared to the diesel units. The capital cost difference between the configurations is not significant.

Shovels are selected based on a feed rate and barren rock volume per day or per year. Using 30,000 tpd mill feed and 30,000 tpd barren rock, the 16.5 m<sup>3</sup> class of hydraulic shovel paired with the 136-t haul truck is the most cost effective combination. Using this match, and with the addition of the quarry and machine availability, 3 shovel units are specified, with one shovel being underutilized.

In addition, the 136-t truck is the largest payload to efficiently match the shovel production rate with 4 pass loading.

Shovel requirements in Y-1 require only one shovel purchase. An additional shovel is required in Y1 to accommodate the quarry volumes. This shovel is recommended to be configured as a backhoe. A third production shovel is added in Y2. The shovel fleet schedule is presented in Figure 16.15.



**Figure 16.15: Shovel Fleet Size**

Shovels are equipped with high precision positioning and GPS systems that communicate directly with the OCS by way of an onboard display terminal. This means that an operator is provided the ore/barren rock dig limit from the OCS.

Support Equipment

Shovel support equipment includes the loading equipment listed in Table 16.4.

Table 16.4 Loading Support Equipment		
Mine Mobile Fleet	Initial Qty	LOM Max Qty
<b>Loading Support:</b>		
Dozer – 433 kW – Pit clean-up and shovel support	1	1
Wheel Dozer – 372 kW – Pit clean-up and shovel support	1	1

A 433 kW dozer will be stationed in the pit. This dozer is larger than others on site and is included for heavy ripping and in-pit ramp and road cuts.

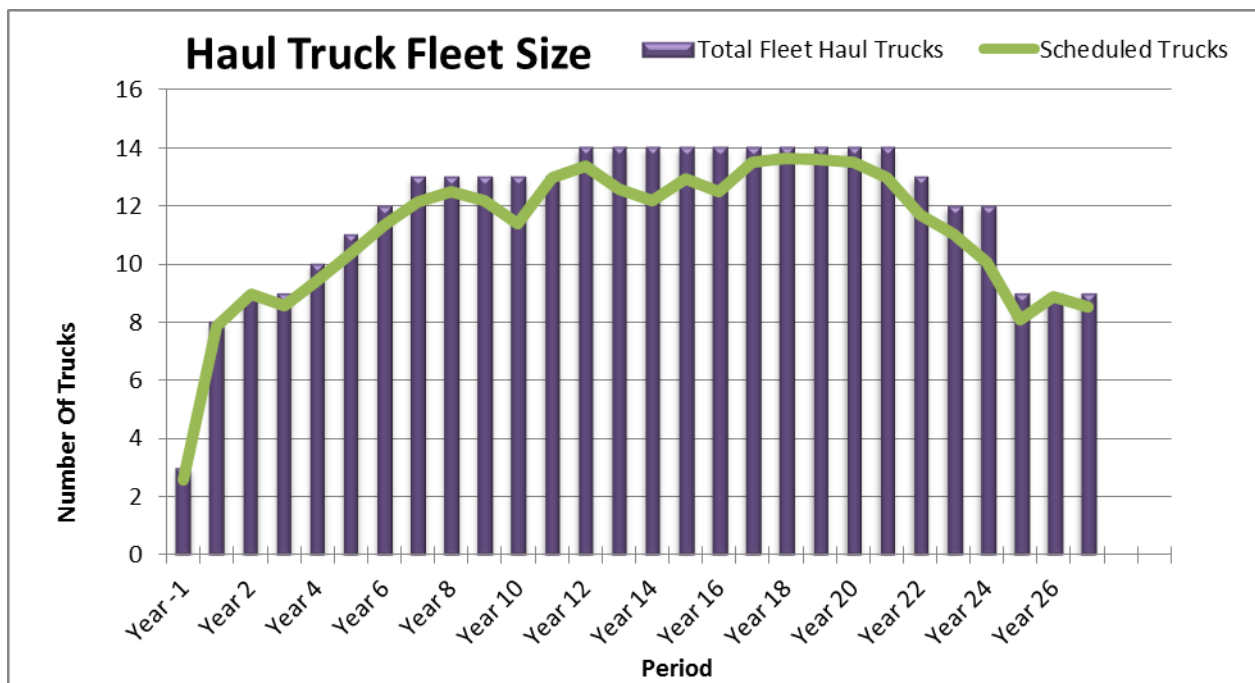
A 372 kW wheel dozer is included for cleaning up spilled rock at the shovel face. The wheel dozer is highly mobile and so is assumed to support both shovels operating in the pit. In addition, the in-pit dozer, or the pit maintenance or utility dozer may be available to support pit floor clean-up if the shovels are working far apart.

These large support pieces are fitted with vehicle health monitoring systems. High precision navigation is not specified for this equipment.

Hauling

Major Equipment

The hauler selected to match the 16.5 m<sup>3</sup> shovels is the 136-t payload class diesel haul truck. As described in Section 16.1.11, 'Loading' the 136-t trucks have been determined from a previous truck/shovel matching study. The size of the haul fleet is determined by the production schedule and required truck operating hours to meet the scheduled tonnage over the haul road network for each operating period. The schedule optimizer evaluates the cycle time and destination requirements for each period and accumulates the required hours. These are combined with the appropriate availability and utilization factors to determine the required number of trucks. The resulting truck fleet size is illustrated in Figure 16.16.



**Figure 16.16: Haul Truck Fleet Size**

Where possible, optimization of the haulage fleet was undertaken in this study to even out the truck fleet requirements. The LOM maximum haul fleet is 14 units.

All haul trucks are fitted with fleet management systems. These are state-of-the-art data centers that report on all facets of machine health. These include machine operating temperatures, vibration, fuel

consumption, etc. The trucks are not equipped with a dispatching or positioning system as the haul fleet is not large enough for this to be of significant advantage.

### Support Equipment

The haul support fleet (see Table 16.5) maintains roads and assists maintenance of the trucks (i.e. tire manipulator). In addition, the dozers for BRSA maintenance are included in the haul support fleet.

<b>Table 16.5 Haul Support Fleet</b>		
<b>Mine Mobile Fleet</b>	<b>Initial Qty</b>	<b>LOM Max Qty</b>
<b>Haul Support:</b>		
Water Truck – 4000 gal - Road maintenance	1	2
Water Truck – 20000 gal – Road Maintenance	0	1
Dozer – 306 kW – Barren Rock Facility maintenance	1	2
Grader – 221 kW - Road maintenance	1	2
FEL – 274 kW - Tire Manipulator and tire moves	1	1

### Pit Maintenance and Ancillary Equipment

A detailed rationale of pit maintenance and ancillary equipment selection is included in the Feasibility Study. Table 16.6 summarizes the pit maintenance and ancillary equipment list:

<b>Table 16.6 Pit Maintenance and Ancillary Equipment</b>			
<b>Pit Maintenance</b>		<b>Initial Qty</b>	<b>LOM Max Qty</b>
Dozer – 306 kW	Pit Support	0	1
Excavator – 301 kW	Utility Excavator	1	1
Mobile Screening Plant	Road and Stemming Crush	1	1
Light Plant – 20 kW	Lighting plant	2	4
Forklift – 10 t	Forklift	1	1
Forklift – 30 t	Forklift	1	1
Fuel/Lube Truck - 4000 litres	Mobile Fuelling	1	1
Jaw Crusher	Road and Stemming Crush	1	1
FEL – 274 kW	Crushing-Screening re-handle and loading	1	1
Crew Van - 15 Passenger	Crew Transport	2	2
Warehouse Truck – 1t	Warehouse truck	1	1
Crew Cab Pickup	Crew Cabs, Supervisor trucks	4	8
Service Truck – 1t	maintenance + overhauls	1	2
Welding Truck - 1t	Welding Truck	1	2
Picker Truck - 10 t	Picker Truck	0	1
Dozer – 306 kW	Quarry Dozer, TSF Dozer	1	0



## 16.2 Open Pit Production Schedule

The Sisson mine production schedule was developed with MineSight® Strategic Planner (MS-SP), a comprehensive long-range scheduling tool for open pit mines. It is typically used to produce a LOM schedule that will maximize the NPV of a project, subject to user specified conditions and constraints. Annual production requirements, mine operating considerations, product prices, recoveries, destination capacities, equipment performance, and operating costs are used to determine the optimal production schedule. This mine plan provides mill feed at a rate of 30,000 tpd or 10,500 kt/a, with a derating commissioning allowance in the first quarter at 50% of capacity. Averaged over the year, this results in 9,220 kt of mill feed in the first year.

The overall mine production was scheduled by pit phase and bench on an annual basis. In the mine schedule, "Time 0" refers to the mill start date. The activities in the pre-production periods are mainly related to construction of the facilities and the TSF dams. With the mining phase as described earlier, the first pit phase provides continuous mill feed after start-up with minimal pre-stripping in the last half of Year -1. Full mill feed production capacity is expected in Year 2. The production schedule specifies:

- Pre-production: Year -2, -1
- Pre-stripping in second half of Year -1
- LOM operations: Year 1 onward.

### 16.2.1 Mine Load and Haul Fleet Selection

The mine load and haul fleet were selected prior to production scheduling. Productivities of the selected equipment are derived from truck/shovel matching studies, and include detailed truck haul cycle estimates, for multiple pit-to-destination combinations.

### 16.2.2 Schedule Criteria

The production schedule setup includes a large number scheduling parameters, and can be modified to a high level of detail. As such, only key scheduling parameters used in MS-SP are defined here.

Truck and shovel criteria are a key component in the calculation of equipment hours in MS-SP. An 80% equipment efficiency factor was applied to both trucks and shovels, based on the combination of 83% operating efficiency and 96% utilization efficiency. Truck and shovel availability assumptions for MS-SP are shown in Table 16.7.

Table 16.7 Shovel and Truck Availabilities Used in MS-SP										
Haul Trucks										
Lifetime Operating Hours (khours)	0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	100+
Availability	89%	88%	87%	86%	85%	84%	83%	82%	81%	80%
Electric Hydraulic Shovel										
Lifetime Operating Hours (khours)	0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	100+
Availability	89%	88%	87%	86%	85%	84%	83%	82%	81%	80%

Loading time is used with other design parameters to calculate the required equipment hours. Design parameters are summarized in Table 16.8.

Table 16.8 Equipment Hour Design Parameters	
Criteria	Value
Loading Time	3.74 min./load
Cycle Time per Pass	36 sec.
# of Passes	5
Spot & Wait Time per Load	10 sec
Job Efficiency Factor	84%
Spot & Dump Time per Load	0.5 min
Delay Time	1.0 min
Max Speed on Haul Roads	50 km/h
Max Speed on Active Bench	20 km/h
Max Speed w/in 200m of Active Dump Face	20 km/h
Rolling Resistance In-pit	5%
Rolling Resistance On Ramp or Haul Road	3%
Rolling Resistance on Dump	8%

In the production scheduling, mining precedence is required to specify the mining order of the pit phases based on relative location of the phases. In addition to pit precedence, MS-SP tracks the haul cycle time and resultant variable unit cost from each pit and bench to the primary crusher, stockpiles, or designated barren rock dumps to determine appropriate costs for optimization.

The primary program objective in each period is to maximize the NPV. The MS-SP NPV calculation is guided by estimated operating and capital costs, process recoveries, and metal prices.

The MS-SP schedule assumes 360 mine operating days scheduled per year and a 21.5 hour operating day.

### 16.2.3 CoG Optimization

In order to optimize the project NPV, grade bins were specified (based on NSR block values); the MS-SP optimizer developed a CoG strategy to increase the project NPV by stockpiling lower grade material where possible and effectively increased the revenue per tonne milled early in the schedule. The mid-grade stockpiled material could be sent to the mill at the end of the production schedule; however, this study assumed no mid-grade stockpile reclaim at this point in time. The grade bins were calculated as shown in Table 16.9 to group the ore for mill feed material and mid-grade stockpiling in a manner that allows the schedule optimization software to optimize head grades. It is assumed that during operations, blast hole assays will be used for ore CoGs and for a CoG strategy.

Table 16.9 Stockpile Grade Cut-offs			
Grade Bin	NSR CoG Range		Destination
Barren Rock	0	8.69*	BRSA
Sub Grade	8.69*	9.29	BRSA
Bin01 (Low Grade)	9.29	11.85	BRSA or Mill
Bin02 (Mid-Grade)	11.85	12.00	Mill or MGSP
Bin03 (Mid-Grade)	12.00	13.00	Mill or MGSP
Bin04 (Mid-Grade)	13.00	14.00	Mill or MGSP
Bin05 (Mid-Grade)	14.00	15.00	Mill or MGSP
Bin06 (Mid-Grade)	15.00	16.00	Mill or MGSP
Bin07 (Mid-Grade)	16.00	17.00	Mill or MGSP
Bin08 (Mid-Grade)	17.00	18.00	Mill or MGSP
Bin09 (High Grade)	18.00+		Mill
*NSR8.69 is the break-even CoG calculated in Q4 2012 and does not include 2013 updates to cost, see Section 15.3.			

A series of scheduling iterations were run to determine the optimized CoG strategy. Results from the scheduler indicate that by using a variable CoG each year the schedule could be further optimized. The final CoGs selected for each period are detailed in Table 16.10. However, as a result of using variable CoGs, it became necessary to send Bin01 (Low Grade) material to the mill in years 6, 10, and 21 in order to meet mill feed targets.

Table 16.10 Variable CoG by Year				
Year	Mill Feed CoG	NSR	Stockpile CoG	NSR
1	Bin09 (High Grade)	18.00+	Bin02 (Mid-Grade)	11.85
2	Bin09 (High Grade)	18.00+	Bin02 (Mid-Grade)	11.85
3	Bin09 (High Grade)	18.00+	Bin02 (Mid-Grade)	11.85
4	Bin06 (Mid-Grade)	15.00	Bin02 (Mid-Grade)	11.85
5	Bin05 (Mid-Grade)	14.00	Bin02 (Mid-Grade)	11.85
6	Bin01 (Low Grade)	9.29	No Stockpile - All Ore to Mill	
7	Bin05 (Mid-Grade)	14.00	Bin02 (Mid-Grade)	11.85
8	Bin06 (Mid-Grade)	15.00	Bin02 (Mid-Grade)	11.85
9	Bin03 (Mid-Grade)	12.00	Bin02 (Mid-Grade)	11.85
10	Bin01 (Low Grade)	9.29	No Stockpile - All Ore to Mill	
11	Bin03 (Mid-Grade)	12.00	Bin02 (Mid-Grade)	11.85
12	Bin03 (Mid-Grade)	12.00	Bin02 (Mid-Grade)	11.85
13	Bin02 (Mid-Grade)	11.85	No Stockpile - All Mid-Grade to Mill	
14	Bin03 (Mid-Grade)	12.00	Bin02 (Mid-Grade)	11.85
15	Bin04 (Mid-Grade)	13.00	Bin02 (Mid-Grade)	11.85
16	Bin04 (Mid-Grade)	13.00	Bin02 (Mid-Grade)	11.85
17	Bin04 (Mid-Grade)	13.00	Bin02 (Mid-Grade)	11.85
18	Bin03 (Mid-Grade)	12.00	Bin02 (Mid-Grade)	11.85
19	Bin03 (Mid-Grade)	12.00	Bin02 (Mid-Grade)	11.85
20	Bin02 (Mid-Grade)	11.85	No Stockpile - All Mid-Grade to Mill	
21	Bin01 (Low Grade)	9.29	No Stockpile - BRSA submerged	
22	Bin02 (Mid-Grade)	11.85	No Stockpile - BRSA submerged	
23	Bin02 (Mid-Grade)	11.85	No Stockpile - BRSA submerged	
24	Bin02 (Mid-Grade)	11.85	No Stockpile - BRSA submerged	
25	Bin02 (Mid-Grade)	11.85	No Stockpile - BRSA submerged	
26	Bin02 (Mid-Grade)	11.85	No Stockpile - BRSA submerged	
27	Bin02 (Mid-Grade)	11.85	No Stockpile - BRSA submerged	

#### 16.2.4 Schedule Results

Scheduling results are tabulated annually and summed for the LOM. These results include:

- Tonnes and grade (NSR, WO<sub>3</sub>EQ, WO<sub>3</sub>, Mo) mined by year and bench, broken down by material type;
- Shovel and truck requirements by period in number of units and number of operating hours (summarized previously in Figure 16.15 and 16.16, respectively); and

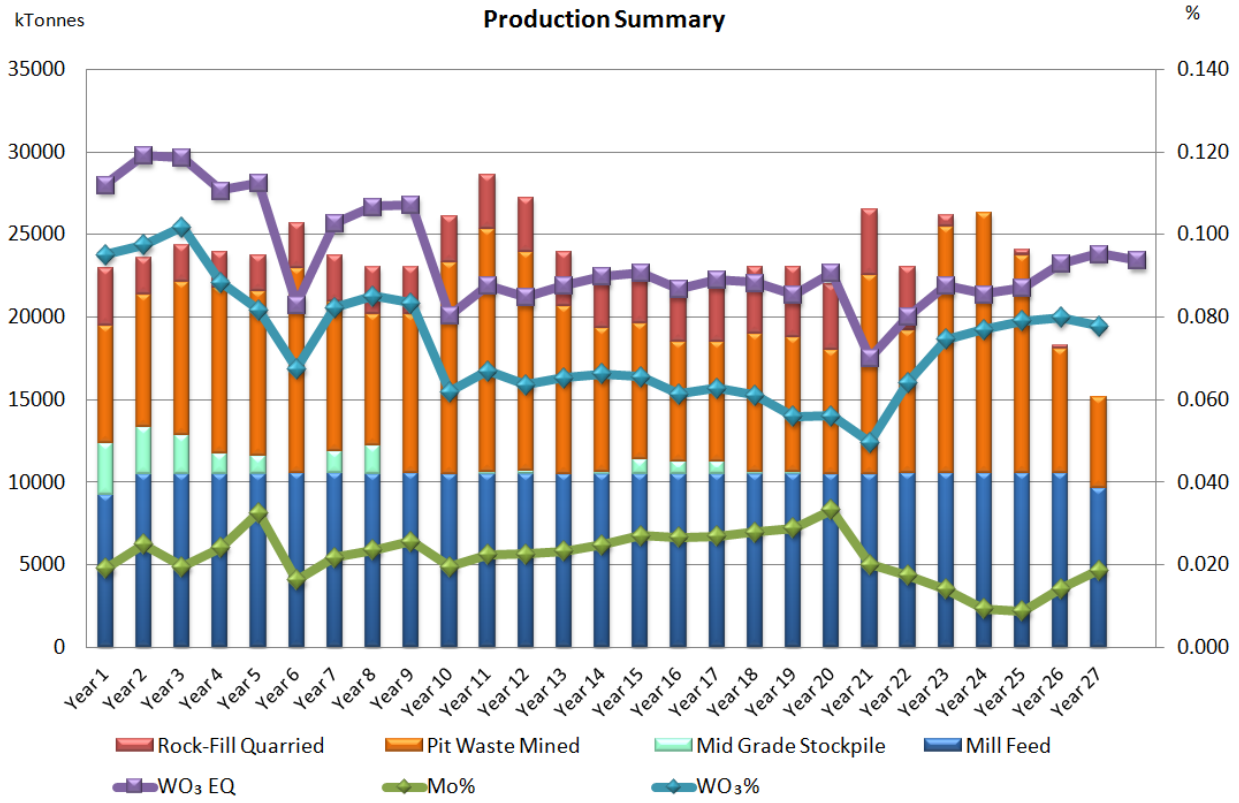
- Tonnes transported by period to different destinations (i.e. TSF embankment, mill, MG stockpile, and BRSA).

The general schedule of mining by pit phase, year and tonnes, is summarized in Table 16.11. The annual mine production forecast is summarized in Table 16.12 and presented graphically on Figure 16.17.

Table 16.11 Mining Schedule by Phase, Year, and Total Tonnes							
	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5	Phase 6	Total
Year	kt	kt	kt	kt	kt	kt	kt
0	2648	-	-	-	-	-	2648
1	19505	-	-	-	-	-	19505
2	12398	8959	-	-	-	-	21358
3	911	17329	3896	-	-	-	22136
4	379	13072	8296	-	-	-	21747
5	-	9148	12410	-	-	-	21557
6	-	828	22102	-	-	-	22929
7	-	770	20188	-	-	-	20957
8	-	-	17196	3002	-	-	20198
9	-	-	11923	8275	-	-	20198
10	-	-	2096	21214	-	-	23310
11	-	-	1574	23787	-	-	25360
12	-	-	457	23458	-	-	23915
13	-	-	-	20646	-	-	20646
14	-	-	-	19301	-	-	19301
15	-	-	-	19635	-	-	19635
16	-	-	-	18523	-	-	18523
17	-	-	-	17293	1203	-	18496
18	-	-	-	14244	4438	314	18996
19	-	-	-	12553	3649	2538	18741
20	-	-	-	12963	-	5053	18016
21	-	-	-	2603	19960	-	22563
22	-	-	-	-	18027	1143	19170
23	-	-	-	-	8297	17175	25472
24	-	-	-	-	4703	21628	26332
25	-	-	-	-	-	23740	23740
26	-	-	-	-	-	18067	18067
27	-	-	-	-	-	15130	15130
LOM	35840	50105	100137	217498	60278	104789	568647

Table 16.12 Sisson Production Schedule Summary (2.3B R6)				
Mine Production:				
	Year	1-5	1-10	LOM
<b>TOTAL Ore Mined</b>	ktonnes	61,872	117,583	298,357
Average Ore Mined per Year	ktonnes	12,374	11,758	11,050
NSR	\$/tonne	30.536	28.752	25.947
WO <sub>3</sub> EQ	%	0.105	0.100	0.092
WO <sub>3</sub> %	%	0.084	0.079	0.071
Mo%	%	0.023	0.022	0.022
<b>Total Mill Feed</b>	ktonnes	51,220	103,780	281,489
Average Mill Feed per Year	ktonnes	10,244	10,378	10,426
NSR	\$/tonne	33.865	30.686	26.670
WO <sub>3</sub> EQ	%	0.115	0.105	0.094
WO <sub>3</sub> %	%	0.093	0.084	0.073
Mo%	%	0.024	0.023	0.022
<b>Total Mid Grade Stockpiled</b>	ktonnes	10,652	13,803	16,868
Average Mid Grade Stockpiled per Year	ktonnes	2,130	1,380	625
NSR	\$/tonne	14.524	14.215	13.869
WO <sub>3</sub> EQ	%	0.058	0.057	0.056
WO <sub>3</sub> %	%	0.041	0.039	0.038
Mo%	%	0.016	0.017	0.017
<b>TOTAL Ore Milled</b>	ktonnes	51,220	103,780	281,489
Average Ore Milled per Year	ktonnes	10,244	10,378	10,426
<b>TOTAL Rock-fill Quarried</b>	ktonnes	12,220	26,190	71,208
Average Rock-fill Quarried per Year	ktonnes	2,444	2,619	2,637
<b>TOTAL Barren Rock Mined</b>	ktonnes	44,430	96,312	270,290
Average Barren Rock Mined per Year	ktonnes	8,886	9,631	10,011
Strip Ratio (Barren Rock/Ore Mined)		0.72	0.82	0.91
Effective Strip Ratio ((BR+stk)/MF)		1.08	1.06	1.02
<b>Total Material Mined</b>	ktonnes	118,523	240,085	639,855
Average Material Mined	ktonnes	23,705	24,009	23,698



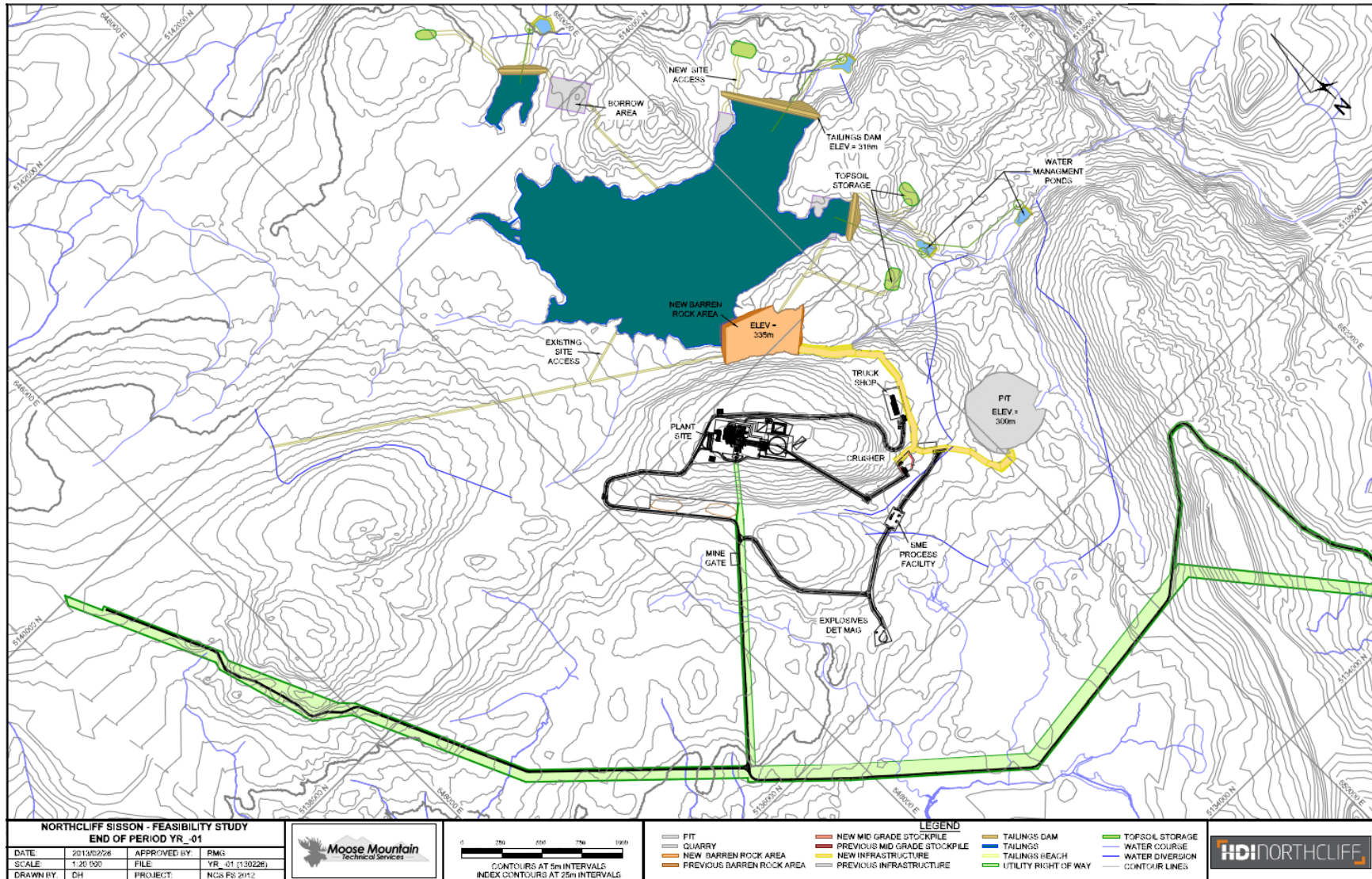


**Figure 16.17: Summary of Production Schedule (2.3B R6)**

**16.2.5 Detailed Mine Plan**

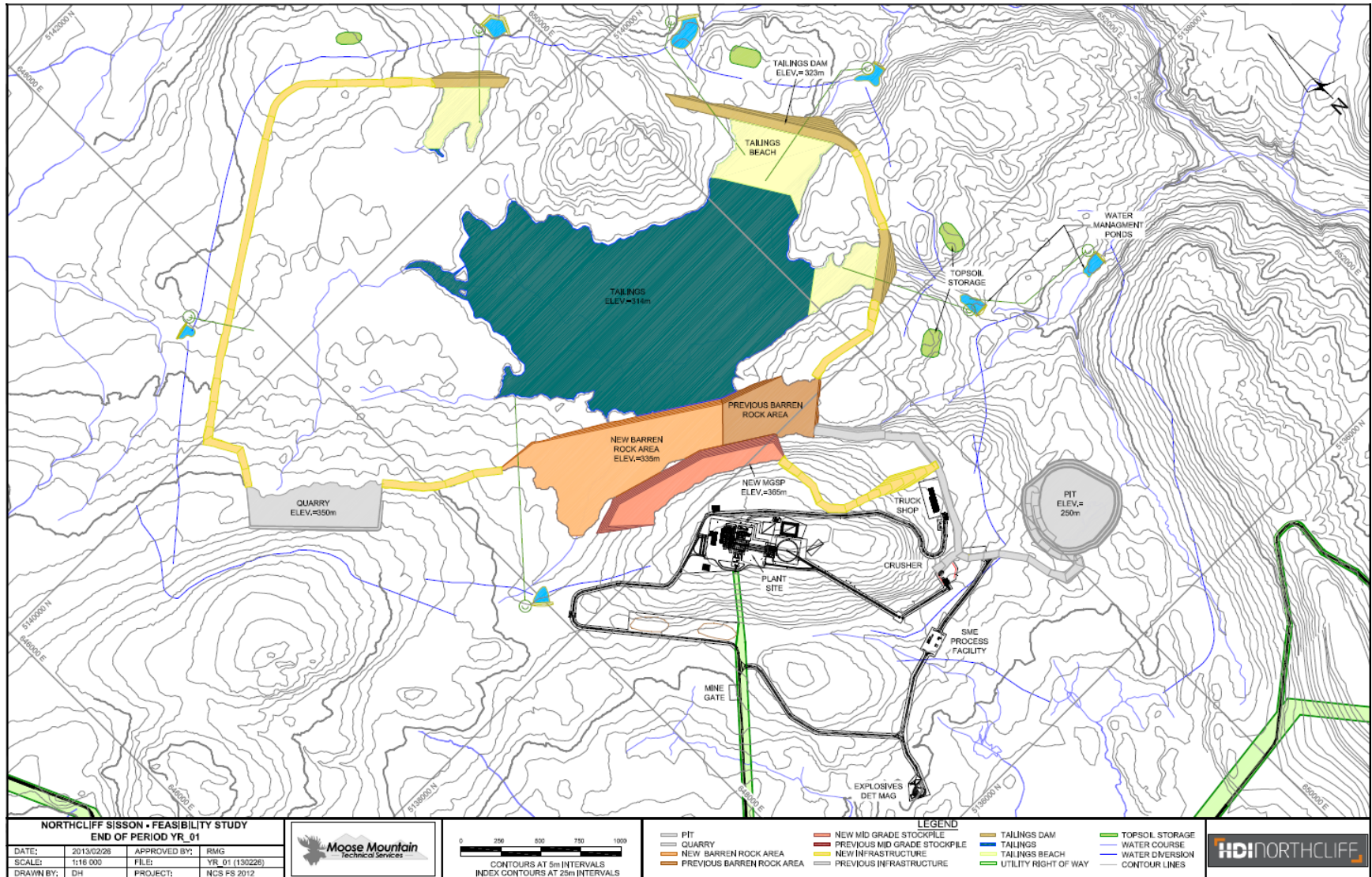
The description of the detailed mine plan, for both the open pit and quarry operation, is based on the production schedule. A set of End of Period (EoP) maps were generated from the details of the production schedule. These are snapshots of what the Sisson Project site is forecast to look like at the end of the year listed, and are used as a schematic to illustrate the detailed mine plan. EoP maps are generated for two years of pre-production (-02 & -01) and production years 1 through 10, 15, 20, 25, 27, where year 27 represents the Life of Mine (LOM). Each of these periods is discussed in detail in the Feasibility Study. However, only EoP maps for the following years are included in this report:

- Pre-production Year -01 (At mill start-up)
- Production Year 01
- Production Year 05
- Production Year 10
- Production Year 20
- Production Year 27 (LOM)



**Figure 16.18: EoP Map, Pre-production - Year -01 (Mill Start-up)**





**Figure 16.19: EoP Map, Production Year 01**

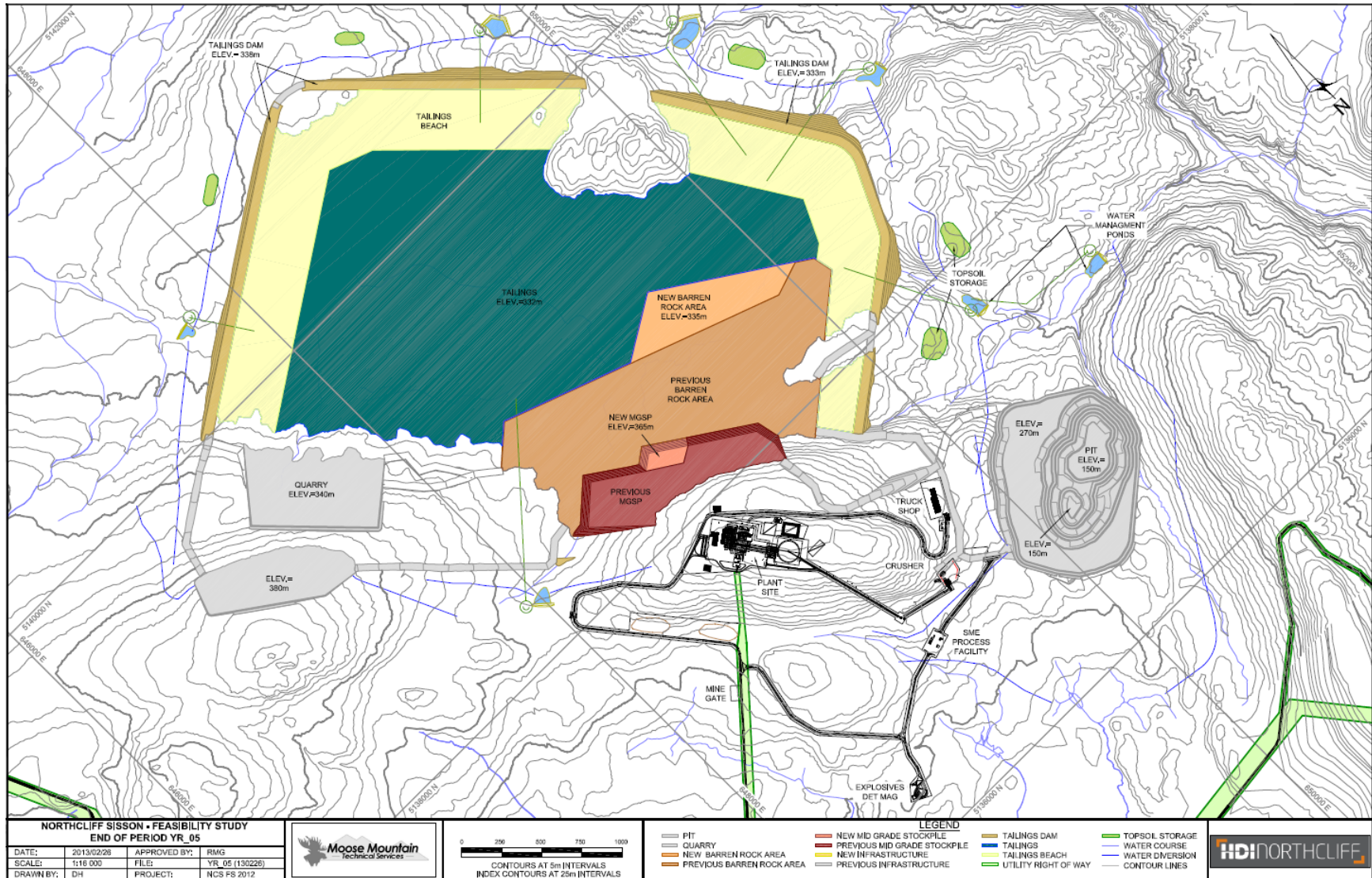
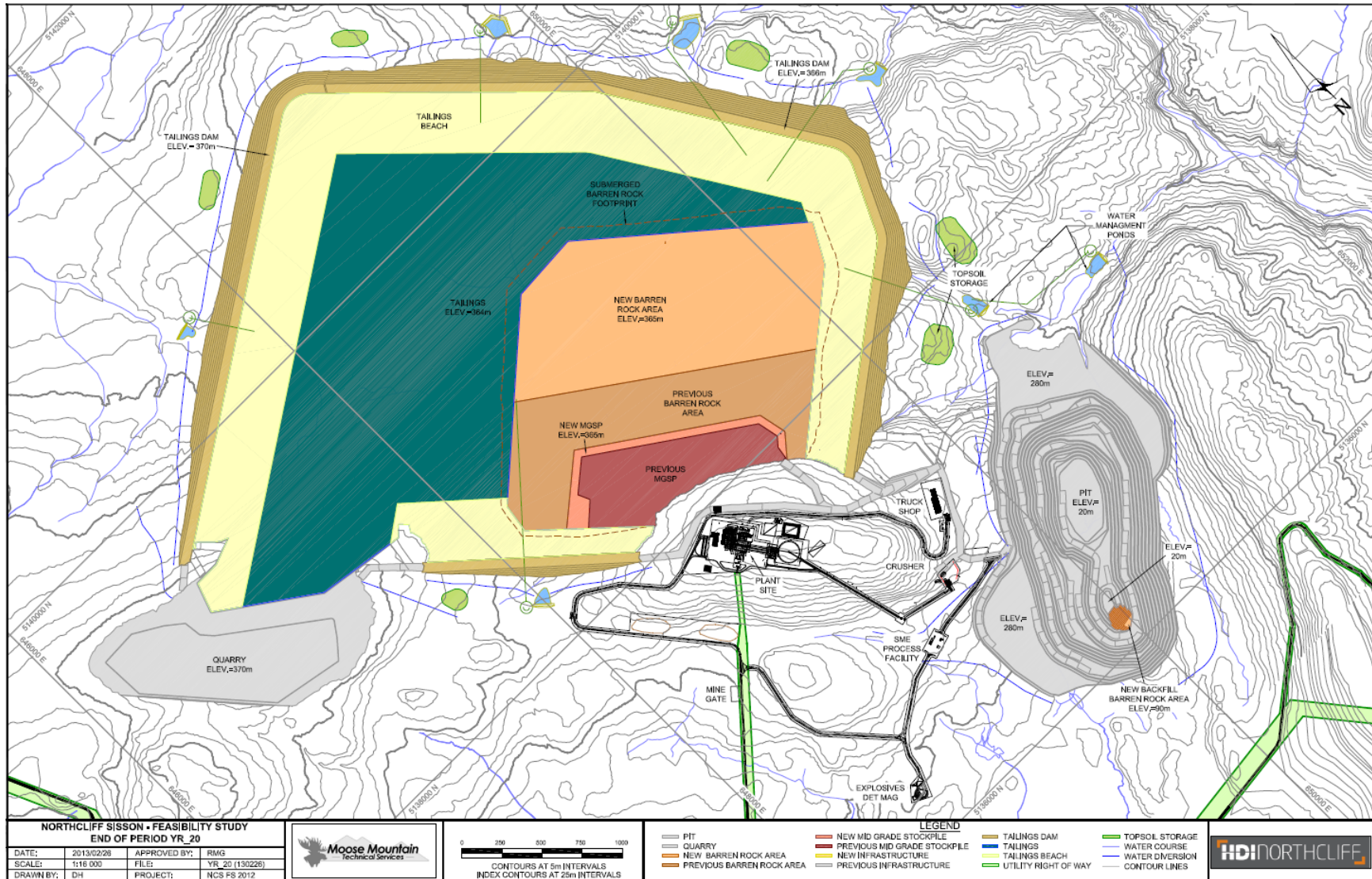


Figure 16.20: EoP Map, Production Year 05







**Figure 16.22: EoP Map, Production Year 20**



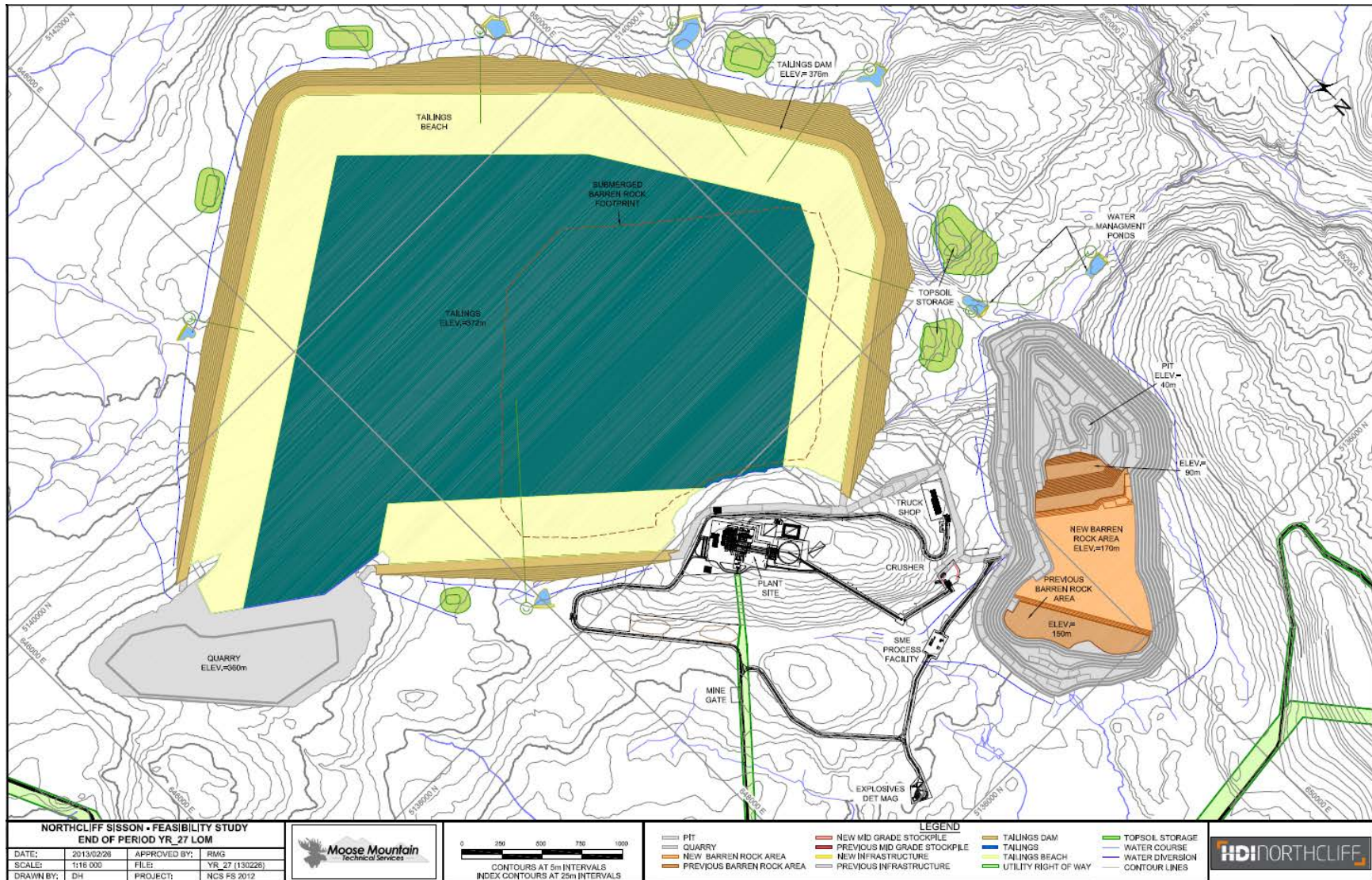


Figure 16.23: EoP Map, Production Year 27 (LOM)

## 17.0 Recovery Methods

The principal economic minerals of the Sisson deposit are scheelite ( $\text{CaWO}_4$ ) and molybdenite ( $\text{MoS}_2$ ) and in accordance with the above described test results, the Sisson concentrator process flowsheet is based on the recovery of these two minerals into their respective concentrators.

The run of mine ore will be processed through an onsite concentrator that will produce a molybdenum flotation concentrate and a tungsten flotation concentrate. The molybdenum concentrate will be shipped offsite to a purchaser, most likely for roasting into molybdenum tri-oxide ( $\text{MoO}_3$ ), while the tungsten concentrate will be processed onsite to high purity ammonium paratungstate (APT) product.

The concentrator process flowsheet development for Sisson was based on the results of the 2012 Feasibility Study metallurgical test program as described in Section 13 above. Locked cycle flotation tests which were optimized from extensive batch flotation test program formed the basis of concentrator process flowsheet development. Of the total of 12 locked cycle tests conducted, the final eight tests examined the optimized flowsheet where intermediate products were re-circulated according to the developed locked cycle test flowsheet.

The Sisson APT plant process flowsheet design was based on industry proven metallurgical and chemical processes confirmed by testing as described in Section 13 above as well as substantial in-house expertise in APT production and related technologies.

Process description and plant design characteristics of the concentrator and APT plants are provided in the following sub sections.

### 17.1 Concentrator Process Flowsheet Design

Based upon the results of the 2012 Feasibility Study metallurgical test program, the process flowsheet for the Sisson Concentrator includes the following major processing steps:

- Three stage crushing
- Single stage, dual-line grinding and classification
- Molybdenum rougher-scavenger and bulk sulphide flotation
- Molybdenum regrind and four-stage cleaner flotation
- Molybdenum concentrate dewatering and packaging
- Tungsten rougher-scavenger flotation
- Tungsten three-stage cleaner flotation
- Reagent preparation and utilities

A general, simplified process flowsheet for the concentrator is provided in Figure 17.1 below.

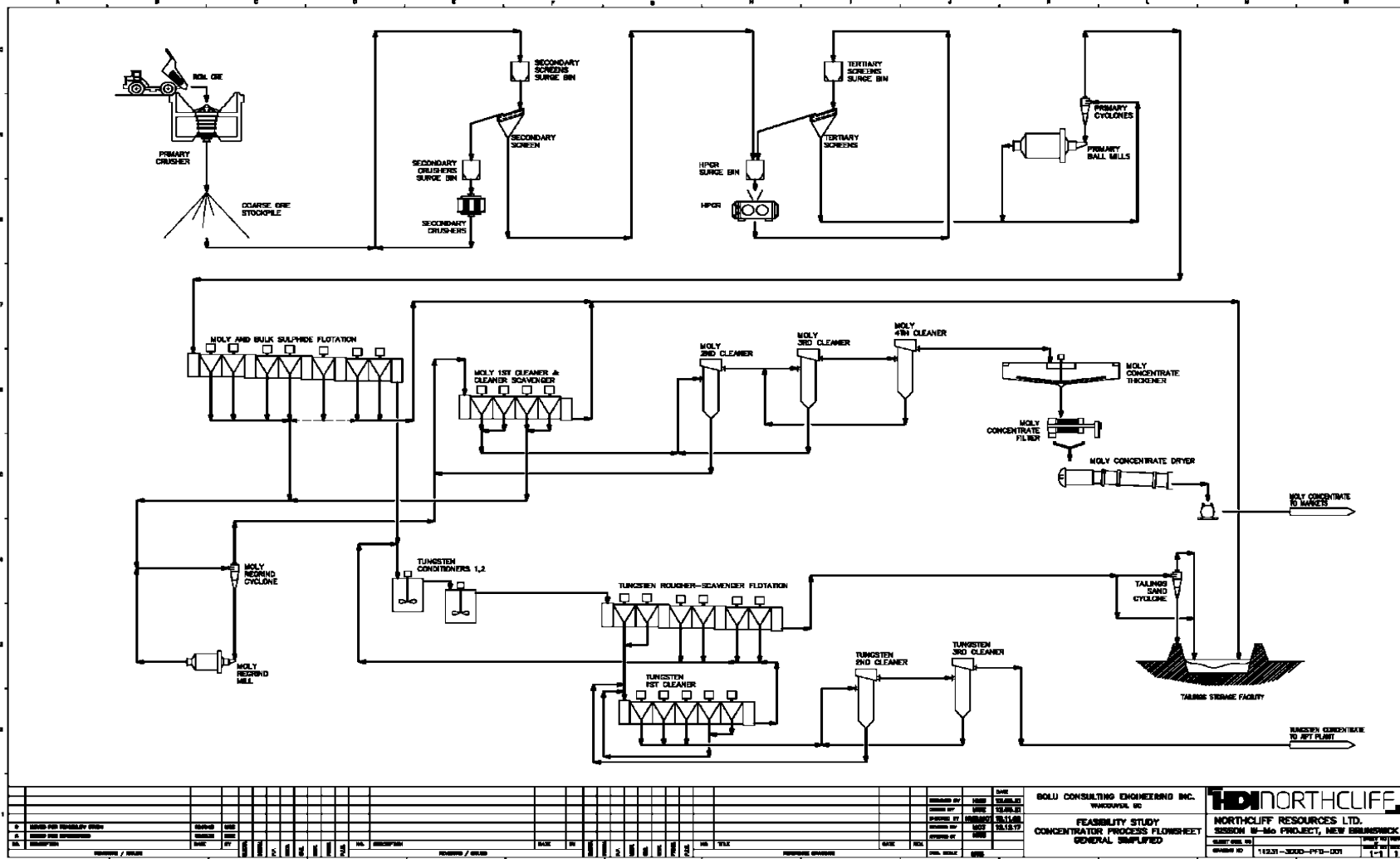


Figure 17.1: Concentrator Simple Flowsheet

The concentrator process was designed to treat 10.5 M t/a of run-of-mine (ROM) feed using conventional comminution and flotation techniques. A three-stage crushing and screening circuit followed by two parallel closed circuit ball mills will be utilized to produce a suitable feed for flotation as determined by metallurgical test work.

A molybdenite rougher concentrate is then floated, reground and cleaned in four stages. The final molybdenite concentrate is thickened, filtered, dried and bagged for markets.

The molybdenite tailings stream enters an adjoining Bulk Sulfide Flotation (BSF) circuit. The BSF concentrate will contain pyrite and other sulfide minerals which are removed to mitigate their interference in the downstream scheelite flotation process. Furthermore, the BSF concentrate forms the potentially ARD generating molybdenum tailings stream and is sent to the tailings storage facility (TSF) for sub-aqueous disposal to prevent oxidation.

The BSF tailings stream is then conditioned in two stages with depressants and collectors for tungsten flotation. The conditioned pulp enters the tungsten rougher circuit followed by an adjoining scavenger flotation circuit. The rougher concentrate is cleaned three times, thickened, filtered, dried and then refined to APT. The scavenger concentrate is recycled to the scheelite conditioners while the tailings, containing low levels of sulphides, are disposed to the TSF as tungsten tailings stream from the plant.

## **17.2 Concentrator Plant Design and Description**

The Sisson concentrator was designed to process 10.5 Mt of ROM feed materials per year, and operate 365 days per year at an average operating availability of 92%. Daily average operating throughput rate is 28,767 t and design operating rate will be 31,269 t or 1,303 t/h

### **17.2.1 Primary Crushing**

ROM ore with a maximum size of approximately 1,000 mm will be end dumped into a primary crushed feed hopper with a live capacity of 171 t total. ROM ore will gravitate to a 54 inch by 75 inch or equivalent gyratory crusher equipped with a 454 kW motor. Any oversize rocks will be handled with a dedicated rock breaker. The crushed product at 80% passing 150mm will fall onto an apron feeder equipped with a variable speed drive. The apron feeder in turn discharges onto a conveyor equipped with a weightometer and a self-cleaning magnet. Two subsequent conveyors will transport the material to an open coarse ore stock pile with a nominal 30,000 t live capacity.

In a reclaim tunnel below the stockpile, three apron feeders will reclaim primary crushed ore from the stockpile and discharge onto the secondary crusher screen surge bin conveyor. The primary crusher and reclaim areas are equipped with dust collectors. Crushing plant design is based on 75% operating availability processing 1,598 t/h of ROM material to provide an average daily throughput of 28,767 t.

### **17.2.2 Secondary Crushing – Standard Crusher**

The primary crusher discharge material is conveyed to the secondary crusher screens' surge bin. The bin feeds two 3.0 m wide by 7.1 m long double deck screening units. The screen undersize, minus 45 mm, reports to a conveyor supplying the tertiary crusher screen surge bin, while the oversize is conveyed to the standard crusher surge bin with a 382 t live capacity. The conveyor is equipped with a weightometer,



metal detector, and tramp metal bypass chute. In the event that the metal detector is activated, a diverter will bypass the surge bin. The surge bin supplies ore to one of two standard head cone crushers (one operating, one standby) equipped with 750 kW motor each. The discharge of the standard crusher, at 80% passing 40mm joins the feed to the secondary screening plant.

### 17.2.3 Tertiary Crushing – HPGR

The secondary screen undersize, the finished secondary crushing product, is conveyed to the tertiary crusher surge bin where it is joined by the tertiary screen oversize stream. The bin will have 336 t live capacity and will feed a high pressure grinding roll (HPGR) crusher. The HPGR crusher will be equipped with two 2.2 m diameter by 1.55 m long rolls and 2 x 2,200 kW motors. The discharge of the HPGR is conveyed to the tertiary screens surge bin.

The tertiary screens surge bin feeds two double deck screens operating in parallel. The screens are each 3.6 m wide by 8.5 m long and equipped with a 37 kW motor and will use wet screening for efficient removal of fines. The screen undersize is a nominal minus 6 mm or 80% passing 4,200  $\mu\text{m}$ . The undersize from each screen gravity flows to primary cyclone feed pump box in open launders. The screen oversize material is conveyed to the HPGR surge bin where it joins the secondary screens' finished product. The conveyor is equipped with a weightometer, metal detector, and a self-cleaning magnet. In the event that the metal detector is activated, a diverter will bypass the surge bin momentarily via a tramp metal bypass chute.

### 17.2.4 Grinding and Classification

The grinding circuit is comprised of two 6.7 m dia. x 11.6 m effective grinding length (EGL), dual pinion ball mills operating in parallel. Each ball mill will be equipped with 2 x 5,500 kW variable speed motors. Ball mill bond work index, based on testwork, is estimated at 17.4 kWh/t for design purposes. Each ball mill circuit will further employ a primary cyclone feed pump box, cyclone feed pumps (1 operating, 1 standby) and a cyclone cluster of 5+1+1, 660 mm diameter cyclones (5 operating, one standby, one blank). Each mill is equipped with a magnet on the discharge end trunnion to collect magnetics and a discharge trommel screen for trash including wood and plastics. A single pipe line will receive the cyclone overflow streams from the two cyclone clusters ahead of flotation for sampling purposes.

The undersize product from each tertiary screen will feed a primary cyclones feed pump box where it is joined by discharge of the ball mill of the corresponding grinding line. The combined product is pumped to the cyclone pack for classification. The cyclone underflow product at a nominal 65-70% solids density gravitates to the ball mill feed chute while the cyclone overflow product, at nominal 40% solids density and 80% passing 110  $\mu\text{m}$ , proceeds to cyclones overflow standpipe for furtherance to flotation as finished product of the grinding circuit. Process water is added to the cyclone feed pump box and mill feed inlet as required for density adjustment. Fuel oil, the primary collector for the downstream molybdenite flotation is added to each mill feed inlet. The flotation feed will be sampled for particle size analysis and metallurgical accounting.

### 17.2.5 Molybdenum and Bulk Sulphide Rougher Flotation

The molybdenum and bulk sulphide flotation circuit is comprised of seven 250 m<sup>3</sup> tank cells in series of which the first four cells will float a molybdenite rougher concentrate and the remaining three a bulk sulphide concentrate. Valving on the fifth cell launder piping will allow flexibility for the froth to join moly concentrate or BSF concentrate depending on Mo and S grades at the time. Each tank cell will be equipped with 224 kW motor and use low pressure air supplied by flotation air blowers. The molybdenite rougher concentrate will be sent to a regrind circuit for further liberation and upgraded in four stages. The bulk sulphide concentrate stream will join the moly cleaner scavenger tailings and will be discharged to a sulphide tailings trench in the plant for disposal to TSF through its dedicated pipeline. The BSF tailings stream will proceed to tungsten flotation circuit.

Reagent additions include a sulphide collector PAX (potassium amyl xanthate) and a glycol frother to aggressively float the remaining sulphides. The BSF concentrate and BSF tailings streams are sampled for on-stream analysis. The BSF tailings are pumped to the tungsten conditioning tanks.

### 17.2.6 Molybdenum Cleaner Flotation

Molybdenum cleaner circuit consists of a regrinding and a four-stage cleaner flotation circuit which is designed to operate in counter current configuration.

The rougher molybdenite concentrate flows to a cyclone feed pump which pumps the combined regrind mill discharge and rougher concentrate to a regrind cyclone pack of four 250 mm dia. cyclones (three operating one standby). Regrinding is accomplished in a 2.9 m diameter by 4 m EGL ball mill operating in closed circuit with the cyclone pack. The ball mill motor power is estimated at 350 kW for a target finished product P80 size of 45 µm.

The cyclone underflow discharges to the regrind mill feed inlet accompanied with an iron sulphide depressant sodium cyanide. The regrind circuit finished product, cyclone overflow, flows by gravity to a bank of four 10 m<sup>3</sup> first cleaner flotation tank cells for upgrading. Reagent addition to the tank cells includes a second depressant sodium sulphide and pine oil. A cleaner concentrate is collected from the first two cells and a cleaner scavenger concentrate from the remaining two cells. The cleaner scavenger concentrate is returned to the molybdenite regrind circuit and the cleaner scavenger tailings are pumped to the sulphide tailings trench in the plant and piped to the TSF for disposal.

The first cleaner concentrate is further upgraded in the subsequent 2<sup>nd</sup>, 3<sup>rd</sup>, and 4<sup>th</sup> cleaner flotation stages which will employ a column cell each; 1.2 m dia. x 8 m high, 0.9 m dia. x 6 m high, and 0.9 m dia. x 6 m high, respectively. The final cleaner concentrate assaying +51% Mo is thickened, filtered, dried and packaged before shipment.

### 17.2.7 Tungsten Rougher-Scavenger Flotation

The tungsten flotation is accomplished by conventional techniques involving conditioning, rougher and scavenger flotation and three stages of cleaning to produce a final tungsten concentrate. The final cleaner concentrate is thickened, filtered, dried and then refined in the APT plant also as discussed in this report.



A series of two agitated conditioning tanks will sequentially adjust the pH of the slurry, and progressively condition the feed with dispersants, gangue depressants, collectors and frothers. These will include sodium hydroxide, sodium carbonate, sodium silicate, quebracho and fatty acids. The first conditioning tank at 192 m<sup>3</sup> live capacity will provide 5 minutes of conditioning time while the second conditioner at 269 m<sup>3</sup> live volume will provide 7 minutes conditioning time. The overflow from the second conditioner will report to the rougher flotation bank.

Six - 250 m<sup>3</sup> tank cells equipped with 224 kW motors will be used in the tungsten rougher-scavenger flotation circuit to recover the tungsten. Tank cells will use low pressure forced air supplied by flotation air blowers. The first two cells will float a rougher concentrate which will be sent to cleaning. The remaining four cells will produce a scavenger concentrate which is pumped back to the second conditioner. Supplementary collector and frother are added to the scavenger cells.

### **17.2.8 Tungsten Cleaner Flotation**

The rougher concentrate is cleaned in three stages. The first stage consists of five 2.8 m<sup>3</sup> tank cells each equipped with 5.6 kW motors. The first 2 cells produce Cleaner 1 concentrate and the remaining three cells produce a cleaner scavenger concentrate which is recycled to the head end of the cleaner circuit. Low pressure forced air to the flotation cells will be provided by air blowers. Supplementary frothers and depressants will be added as needed to the first stage of cleaning.

The Cleaner 1 concentrate is cleaned two more times using two 1.22 m dia. X 8.0 m high column cells in series. The final concentrate of approximately 30% WO<sub>3</sub> is thickened, filtered, and dried. The dried final concentrate will then be refined in the onsite APT plant.

### **17.2.9 Quality Control**

Quality control will be achieved in the mill with the use of an on-stream analyzer and particle size analyzer.

The on-stream analyzer (OSA) sampling stations will be established at six locations in the mill to automatically collect samples and provide timely analysis for mill operating crews. Shift samples will be collected from the automatic samples for laboratory analyses and the metallurgical balance. The six locations are as follows:

- Primary cyclone overflow – head sample
- Moly Cleaner Concentrate #4
- Bulk Sulphide Concentrate
- Bulk Sulphide Tails Sample
- Tungsten Cleaner Concentrate #3
- Tungsten Scavenger Tails

A particle size analyzer will receive a sample from the cyclone overflow and determine the particle size analysis on a set time frequency to the operations crew.

### 17.2.10 Reclaim Water Clarification

Sisson concentrator will utilize recycled water from the TSF. Reclaim water from the TSF, containing low levels of unsettled fine suspended solids, will first be sent to the reclaim water clarification plant for lime treatment. The clarification plant major equipment will include 2 – 220 m<sup>3</sup> lime conditioning tanks, a 61m diameter clarifier, and lime and flocculant preparation/mixing systems. After clarification and pH adjustment with carbon dioxide, the settled solids will be separated, and the clarified water containing acceptable levels of suspended solids for process water make up purposes will be recycled back to the concentrator process water tank for use in the plant. Settled solids from the water clarification plant will be sent back to a dedicated area of the TSF to prevent buildup of hard-to-settle fine solid particles. The water clarification plant is designed to process approximately 2,635 m<sup>3</sup>/h of recycle water to meet all process make-up water requirements of the process facilities.

### 17.2.11 Tailings Disposal

Flotation plant tailings will consist of non-PAG and PAG generating streams. Tungsten flotation circuit tailings are expected to be non-PAG and will constitute approximately 95% of total tailings mass. The non-PAG classification of the tungsten tailings is on account of their sulphur content being below 0.1%. The molybdenum and bulk sulphide circuit tailings are expected to be PAG. The two tailings streams will be pumped to the TSF separately, to allow the sub-aqueous deposition of the molybdenum tailings.

Process water will be reclaimed from the tailings impoundment by pumps located on a floating barge to the reclaim water clarification plant.

Further details on tailings pumping, disposal, and reclaim water pumping are provided in Section 18.13, Tailings Storage Facility.

## 17.3 APT Plant Process Flowsheet Design

The Sisson APT plant process flowsheet design was based on proven metallurgical and chemical processes confirmed by testing conducted at the laboratories of SGS Lakefield, an independent testing facility in Ontario, Canada. And furthermore, substantial metallurgical expertise on APT production and the related technologies available in-house has been used in the testing, process flowsheet development and design of the APT plant described herein. The process as designed is a series of continuous and batch operations with storage hold points and based on alkali pressure leach technology.

The process flowsheet for Sisson APT plant will include the following major processing steps:

- Feed Preparation
- Digestion and Residue Filtration
- Alkali Recovery and Solution Purification
- Conversion to Ammonium Tungstate
- Ammonium Paratungstate (APT) Crystallization
- APT drying and packaging
- Reagent preparation and utilities

Tungsten concentrates will first be reground and dewatered in the feed preparation circuit in order to allow a uniform feed ahead of digestion. Tungsten in the concentrates will be digested using an alkali leach system and the sodium tungstate solution will be filtered from the undigested leach residue. The residue will be disposed to the TSF while the sodium tungstate solution will be processed thru an alkali recovery and purification process. Common impurities including As, Mo, P, and Si (arsenic, molybdenum, phosphorous, and silicon) will be removed and stored for disposal at an offsite facility. The resulting sodium tungstate solution will be converted to ammonium tungstate and subsequently to ammonium paratungstate (APT) crystals.

The aqueous solution effluent from the ammonium tungstate conversion will be disposed to a lined containment pond within the TSF after pH adjustment. The dried and screened APT will be packaged for markets. Vapors from the crystallizer and other areas and processes in the plant will be sent to their respective scrubbers and stripping systems for reclaim and re-use before discharge to atmosphere. The main reagents used in the process are sodium hydroxide (NaOH), sulphuric acid (H<sub>2</sub>SO<sub>4</sub>), anhydrous ammonia, (NH<sub>3</sub>), lime, and organic exchange media.

A simplified process flow diagram (Figure 17.2) shows the main process steps as designed.



## 17.4 APT Plant Design and Description

The APT plant was designed to process Sisson tungsten concentrates at a maximum feed rate of 29 kt/a containing 881 k mtu  $WO_3$  per year at 92% operating availability. On average, and based on Feasibility Study LOM mine plan, the APT plant will process 19 kt of concentrates per year containing 581 k mtu of  $WO_3$  per year to produce 555 k mtu per year of  $WO_3$  contained in high quality APT product. The APT plant recovery is predicted at 96.75%. Potential exists for additional tungsten concentrate processing through the APT plant when production from the Sisson concentrator is less than APT plant capacity stated above.

### 17.4.1 Feed Preparation

Tungsten concentrate slurry from the Sisson concentrator plant will be pumped to the APT plant feed preparation circuit which will employ a wet grinding mill to facilitate size reduction and further exposure of tungsten mineral grains. The target particle size is less than 74  $\mu m$  95% passing. The mill will operate in closed circuit with a hydrocyclone and the finished product, cyclone overflow, is fed to a thickener for dewatering and density adjustment prior to filtering. The thickener overflow will return to concentrator process water tank as make up water while the filter cake discharge from the filter press is fed to a continuous dryer to further reduce moisture to less than 0.5%. The dryer discharge is screened on a sifter to ensure a minimum particle size necessary for efficient downstream digestion of the concentrate.

Oversize from the sifter is returned to the grinding mill for further grinding. The ground and dried concentrate is stored in a hopper for feed to the digesters. An emergency storage bin with a surge capacity equivalent to five days of dried concentrate production is provided between the feed preparation and the digestion circuit for unscheduled maintenance in the downstream circuits. This surge capacity may be extended by bagging the concentrates for external storage.

### 17.4.2 Digestion and Residue Filtration

The digestion section consists of digesters, dilution tanks, filter presses, residue processing equipment, and storage tanks.

The digesters are designed to extract tungsten in the concentrate by digesting with alkali solution for a period of time which will allow maximum recovery of the tungsten at optimum conditions as concluded by the test results. This process is based on methods traditionally and widely used in the global tungsten industry and confirmed by testing at SGS Lakefield laboratories on Sisson tungsten concentrates. After digestion the digested slurry is transferred to filtration of the gangue from the sodium tungsten solution.

Filtering is designed to separate the sodium tungstate solution from the undigested gangue and wash the filter cake with water for maximum tungsten recovery.

Filter cake, the undigested residue, is hauled to the tailings storage facility for disposal.

### 17.4.3 Alkali Recovery and Solution Purification

The sodium tungstate solution is next processed through a purification process where undesirable impurities are removed from the solution.

The first step is an alkali recovery step where the products are alkali and concentrated sodium tungstate solution. This is accomplished by removing excess water and then separating the two streams. The alkali is reused in the digestion step and the recovered water is used where required in the process.

The sodium tungstate solution is further processed to remove impurities such as Al, As, Mo, P, and Si. This is accomplished by pH adjustment of the treated solution to various levels followed by filtering and absorbing the impurities from the solution.

#### **17.4.4 Conversion to Ammonium Tungstate**

The conversion of sodium tungstate to ammonium tungstate is accomplished by loading the  $WO_4$  anion on to an organic cation in a pH controlled process. The resulting aqueous solution is pumped to the tailings storage facility for disposal. The loaded organic is then washed and stripped with an ammonium hydroxide solution to form ammonium tungstate solution and a free radical organic. The free radical organic is regenerated for reuse in the process and the ammonium tungstate is sent to the next process.

#### **17.4.5 Crystallization of Ammonium Tungstate to Ammonium Paratungstate**

The APT is crystallized in a continuous evaporator crystallizer. The ammonium tungstate is pumped from storage to the APT crystallizer. The crystals as formed are continuously removed from the crystallizer and separated from the mother liquor by use of a filter. Mother liquor is returned to the crystallizer. The crystals are washed and sent to a dryer.

Vapors from the crystallizer and other related sources in the plant are sent to a scrubber and stripping system for reclaim and re-use in the generation of ammonium tungstate.

#### **17.4.6 Drying and Packaging**

The APT crystals are dried in a dryer and sent to a screening system before packaging and shipment to markets.

### **17.5 Plant Process Control Philosophy and System Description**

The Process Control System (PCS) for the Concentrator Plant shall be controlled by Distributed Control System (DCS) microprocessor-based with components capable of being installed in separate locations and will incorporate APT Plant wide digital process control communications. Data communications between controllers shall be accomplished by redundant data communication networks utilizing fiber optic cabling. The control system will handle all process plant digital controls including motor control, interlocks, switches and all analog process control loops, process indicators and analog control devices.

All concentrator data collection and plant operation will be operated from a single concentrator centralized control room located on the top floor between Flotation and Grinding Area with operator ability to view both areas from control room. Primary Crusher Area, located away from the Concentrator, will be operated from primary crusher dedicated control room with operator ability to view primary crusher and control primary crushing discharge and conveyor handling to coarse ore stockpile area.



All data collection and APT plant operation will be from a single centralized control room located in a central location in the APT building near the digesters. The PCS level of automation shall provide control room operators with the ability to perform all monitoring, direct control, regulatory, advanced control functions, supervisory control functions and data acquisition from any operator stations located in concentrator and APT plant areas. Any process equipment can be operated, started or stopped locally or remotely from the control room.

The level of automation for APT plant will provide adequate automation to allow all critical process equipment to be operated and monitored from a central control room. The intent is that the operators will be on the plant floor, so process gauges and local indicators will be provided to allow non-critical processes to be monitored and adjusted locally.

The Concentrator PCS architecture is based on process controllers located in concentrator electrical rooms. In addition, primary crusher may have its own dedicated package controller and the secondary crushers, HPGR and Ball Mills as well as crystallizers will have a dedicated programmable logic controller (PLC) to handle specific drive and control functions.

The DCS and its communication network shall be able to facilitate future expansion by adding new nodes and inputs/outputs (I/O) modules without any modification to the existing control system or the communication network. A total of two (2) or three (3) operator stations shall be provided for each of the Concentrator and APT control rooms. A combined engineering work station and data historian will be located in the Concentrator and APT control rooms or electrical rooms to allow modifications to be made to the PCS and to collect and store process data for trending and analysis purposes. All process information and status will be displayed in graphic form on the operator stations. Data will be presented in the conventional overview, group, and point hierarchical display screen.

Field Operator Stations (FOS) shall be provided to interface to the DCS in remote location for the following Concentrator plant areas; Primary Grinding, Moly and Bulk Sulphide Rougher Flotation, Moly Re grind and Cleaner Flotation, Tungsten Rougher and Cleaner Flotation. Field Operator Stations (FOS) considered to be deployed in the field for the following APT plant areas; Digestion, Solution Preparation/Purification and spare.

The APT PCS should communicate with Vendor supplied packages through Vendor PLC in a transparent manner such that the operational status of motors and instruments that are controlled by the Vendor packages are also monitored from APT DCS network.

Not all Vendor system shall require full communications with the APT PCS. Non -process control units with proprietary controls, such as air compressors, may only provide running and error status.

The PCS will use power supplies configured in a redundant format so that the failure of one power supply will not shut down the entire system. In addition, the PCS will have a dedicated uninterruptible power supplies (UPS) with batteries backup for the processors, communications, modules, and operator stations, so that these systems will remain operational for specified time following a power outage.

## 17.6 Plant Services and Utilities

The requirements for plant services and utilities for the Concentrator and APT plants are combined and rationalized for maximum operating flexibility and cost efficiency. These services and utilities will include the following:

- Analytical and metallurgical laboratory support services,
- Air supply and distribution,
- Water supply and distribution, and
- Steam supply system.

Building services such as fire protection system and power supply are covered in Section 18.0, Project Infrastructure.

A centralized laboratory located near the concentrator in a separate building will provide analytical and metallurgical testing needs of the overall process facilities. The lab will be manned by a chief metallurgical engineer, a chief chemist, three metallurgists, 6 analytical personnel and 2 buckers for a total of 13 positions. The lab will utilize XRF, ICP, AA, and other modern analytical equipment as well as sample preparation and metallurgical testing equipment such as flotation, particle size analyses, gravity test equipment and other.

Three stationary compressors, two operating and one standby, each 2,000 Nm<sup>3</sup>/h at 1,034 kPa discharge pressure will supply high pressure air to the following areas and circuits: column flotation cells (after pressure adjustment), grinding mills, instrument air, concentrate filters, APT plant filters, dust collection and suppression systems, and plant air distribution for both the concentrator and the APT plants.

Low pressure air will be supplied by three air blowers; two in operation and one in standby mode, each sized at 15,660 m<sup>3</sup>/h providing 64 kPa pressure air. All induced air flotation cells, other than column cells, will be served by the low pressure air system.

The plant water systems will consist of process water, filtered process water, fresh water, potable water, soft water, de-ionized water and recycled raw water. Process water and filtered process water make up the vast majority of the site requirements. This water is recycled / reused from the TSF.

The process water system is primarily made up of tailings reclaim water and with lower quantities of thickener overflow waters. The mine water, if found suitable, will be considered for recycle through the process water system as make up water. The process water will be stored in and distributed from a 1,045 m<sup>3</sup> tank which will provide 30 minutes of supply at average consumption rates. The process water system will supply water to the secondary and tertiary screening plant, grinding circuit, flotation circuits, hose stations, and filtered process water system.

The filtered process water system is comprised entirely of process water and is used in applications where high turbidity and particulate matter are not tolerated. It will be used for the following applications: fire water, crusher lube system, mill lube system, OSA, reagent mixing, gland seal water, and spray water in select and suitable areas. The filtering process is achieved in two stages with strainers followed by cartridge filters. The filtered water will be stored in and distributed from a 500 m<sup>3</sup> tank. The upper

portion of the tank will be allocated to filtered water system use while the lower portion will hold a dedicated amount of water for fire protection.

The upper portion of the tank for filtered water storage will be 280 m<sup>3</sup> capacity which will provide 2 hours supply at average consumption rates. The lower portion of the tank will hold 220 m<sup>3</sup> of water for firefighting purposes at all times. The filtered water tank will be located outside the grinding building along with the process water and fresh water tanks.

An estimated 5 to 10 groundwater wells will be developed to meet the fresh water requirements for the project. Fresh water will be stored in a 205 m<sup>3</sup> tank located at the process area and will provide 6 hours of supply at nominal consumption rates. The fresh water system will be used to supply a potable water system, APT plant, select reagent mixing, and for dust suppression.

The potable water system is comprised of a hypochlorite treatment plant and a 48 m<sup>3</sup> distribution tank and pump. The water will be used for potable water throughout the process plant, administration building, eye wash stations and showers, and dust suppression in select areas.

Fresh water will also supply a fire water tank located in the truck shop area. This tank will hold approximately 110 m<sup>3</sup> of water for fire firefighting purpose at the truck shop. Potable water will be delivered by truck to potable water tanks at the truck shop and other remote, site facilities.

De-ionized and soft water systems will be generated on site using fresh water supply. Both water systems will mainly serve the APT plant facility which will have its internal recycled water system.

Process plant utilities will include fuel oil which will be used for mobile equipment, concentrate dryers and for the APT plant. A fuel oil tank located at the tank farm will be used to store and distribute fuel oil as required in a self-contained area which will be equipped with a sump pump for spill recovery.

A diesel-powered emergency power generator will be provided near the process plants to provide an alternate power supply for lighting, critical process loads and other process sensitive areas during scheduled or non-scheduled power outages.

## **17.7 Reagents Preparation, Storage and Operating Consumables**

Reagents and chemicals for the process plants will be used in flotation, dewatering, reclaim water clarification and APT conversion circuits. The reagents described in this paragraph will be delivered in bulk or by specific container and stored onsite in separate and secure and designated areas near or attached to process plant buildings. Covered and open storage areas for all reagents will be self-contained and equipped with spill recovery sump pumps as required. Minimum storage capacities for reagents and chemicals will vary from 5 days to over 30 days depending on form of delivery and specific consumption rates.

Reagents will be mixed with filtered process water where necessary and pumped to day-tanks from where they will be dosed to their appropriate circuits using metering pumps or slurry pumps as in lime distribution. Some select reagents such as flocculants will use fresh water for mixing.

The reagents for the process plant will be fuel oil, pine oil and Methyl Isobutyl Carbinol (MIBC). A conventional frother will be used in molybdenum floatation to promote and collect molybdenite mineral. Small amounts of sodium cyanide and sodium sulphide will be used in molybdenum cleaner floatation as depressing and controlling agents. Potassium Amyl Xanthate (PAX) and MIBC will be used in the bulk sulphide floatation circuit to promote selective floatation of the sulphide minerals. Sodium hydroxide, sodium silicate, sodium carbonate, quebracho, fatty acid, and frother will be used in tungsten floatation to enhance selective floatation of the scheelite mineral.

Lime will be used in water clarification and the APT process. APT reagents will further include sodium hydroxide, ammonium hydroxide, sulphuric acid, sodium hydrosulphide, liquid nitrogen, magnesium chloride, ammonia and amine. These will be delivered into their respective storage tanks for distribution as needed. Flocculant will be used to aid in the settling of solids in concentrate thickening and water clarification applications. Liquid carbon dioxide will also be used in water clarification process.

Fuel oil, pine oil, MIBC, fatty acid, and tungsten floatation frother will be shipped in tanker trucks and stored in Enviro-safe tanks where they will be transferred, as required, into day tanks for dosing. PAX, sodium cyanide, sodium sulphide, quebracho and flocculant will be shipped to site in dry solid flakes or pellet form in bags or drums. These will be stored in the reagent storage area next to the reagent preparation building. After mixing, the solution will be pumped to their respective day tanks for distribution to their appropriate addition points. Bulk reagents such as sodium hydroxide, sodium carbonate, and lime will be shipped to site in tanker trucks and pneumatically unloaded into their dedicated on site storage bins before mixing to required solution strength and distribution as required. Sodium silicate will be delivered in liquid form to its storage tank where it will be pumped to the mix tank for mixing and subsequent delivery to addition points.

Comminution media such as grinding balls, grinding mill and crusher liners are the major consumables for the process plants. Steel balls for primary grinding will be shipped by trucks and dumped into ball bins located in the grinding building. Smaller regrind steel balls will be shipped in drums and stored on site near the warehouse for use as needed. Mill and crusher liners will also be stored by the warehouse until required for use in their respective areas.

## 17.8 Process Design Criteria

Major process design criteria for the Sisson Concentrator and the APT plants has been developed based on the Feasibility Study 2012 test program and is summarized in Table 17.1 and Table 17.2 below.

Table 17.1 Concentrator Major Process Design Criteria		
Description	Units	Value
<b>Production</b>		
Operating Days/year	(d)	365
Pr. Crushing Plant Operating Availability	%	75
Concentrator Operating Availability	%	92
Tonnes Milled/Calendar Day, Average	(t/d)	28,767

<b>Table 17.1 Concentrator Major Process Design Criteria</b>		
Tonnes Milled/Calendar Day, Design	(t/d)	31,269
Tonnes Milled/Year	(kt/a)	10,500
<b>Ore Characteristics</b>		
Ore Specific Gravity	SG	2.8
Ore Bulk Density	(t/m <sup>3</sup> )	1.68
Ore Moisture Content, Average	(%)	2.5
<b>Work Indices (metric)</b>		
Impact, average	(kWh/t)	15.9
Impact, design	(kWh/t)	17.6
Rod Mill, average	(kWh/t)	17.7
Rod Mill, design	(kWh/t)	18.3
HPGR (E <sub>SPnet</sub> )	(kWh/t)	1.62-2.15
Ball Mill average	(kWh/t)	17.2
Ball Mill, design	(kWh/t)	17.4
<b>Concentrate ROM Feed Grades</b>		
Years 1-5	(%) WO <sub>3</sub>	0.093
Years 1-10	(%) WO <sub>3</sub>	0.084
LOM; Years 1-27	(%) WO <sub>3</sub>	0.073
Years 1-5	(%) Mo	0.024
Years 1-10	(%) Mo	0.023
LOM; Years 1-27	(%) Mo	0.022
<b>Concentrator Metal Recovery Predictions</b>		
Years 1-5	(%) WO <sub>3</sub>	81.5
Years 1-10	(%) WO <sub>3</sub>	80.0
LOM; Years 1-27	(%) WO <sub>3</sub>	77.1
Years 1-5	(%) Mo	82.0
Years 1-10	(%) Mo	82.0
LOM; Years 1-27	(%) Mo	82.0
<b>Concentrate Production and Grades</b>		
Concentrate Grade, LOM; Years 1-27	(%) WO <sub>3</sub>	30.0
Concentrate Grade, LOM; Years 1-27	(%) Mo	51.0
Concentrate Production, LOM; Years 1-27, Average	WO <sub>3</sub> mtu/a	585,771
Maximum Concentrate Production, Design	WO <sub>3</sub> mtu/a	881,363
Molybdenum Production, LOM; Years 1-27	k lb/a of Mo	4,144

<b>Table 17.2</b> <b>APT Plant Major Process Design Criteria</b>		
<b>Description</b>	<b>Units</b>	<b>Value</b>
<b>Production</b>		
Operating Days/year	(d)	365
APT Plant Operating Availability	%	92
APT Plant Throughput Capacity, Average	WO <sub>3</sub> mtu/a	585,771
APT Plant Throughput Capacity, Design	WO <sub>3</sub> mtu/a	881,363
Daily Concentrate Processed, Average	t	58.1
Daily Concentrate Processed, Design	t	87.5
Tungsten Recovery into APT Product	%	96.7
<b>Plant Feed Characteristics</b>		
Feed WO <sub>3</sub> Grades to be Processed; Min/Max	%WO <sub>3</sub>	30/45+
Typical Feed Analysis	% As	0.003
	% Mo	0.16
	% P	6
	% Si	10
Concentrate Specific Gravity	SG	4.0 to 4.6+
<b>Typical APT Product Analysis</b>		
	% WO <sub>3</sub>	89.5+
	Mo, ppm	6
	Si, ppm	4
	Al, ppm	2
	P, ppm	5



## 18.0 Project Infrastructure

General site facilities planned for the Sisson Project are:

- Site roads and lay-down areas
- Waste and overburden stockpiles
- Settlement ponds for open pit drainage water
- Explosives bulk plant and magazine
- Primary crusher, coarse ore storage and conveyor system
- Process plant with Concentrator and APT buildings
- Tailings storage facility (TSF)
- Power distribution, and communication system
- Truck shop, warehouse and vehicle shelter
- Administration building
- Laboratory building
- Water storage and distribution
- Sewage treatment and garbage handling
- Fire protection
- Fuel storage and distribution
- Reclaim Water Clarification Facility

Site infrastructure and ancillary buildings are presented in the ensuing sections.

### 18.1 Site Access Roads

As mentioned in Section 5.1, Forestry Service Roads (FSR) off paved highways, provide access to most areas of the property. One FSR, Fire Road, runs through the Sisson Project site and will be relocated for about 11 km around the southwest side of the site; a 3 km site access road will be established from it to the main process site area. FSRs north to Highway 107 and south to Highway 105 will be renovated, as needed, to accommodate the increased traffic associated with project construction and operation.

### 18.2 Site Roads

Offshoots from the main access road will connect to the primary crusher, the site mixed explosive (SME) facility, and mine pit. Ancillary roads from the site process bench connect to the truck shop and fuel storage facility. Site and access roads are depicted on Figure 18.1.

Mine haul roads are designed by MMTS and are presented as part of the pit design and mining production schedule in Section 16.0.

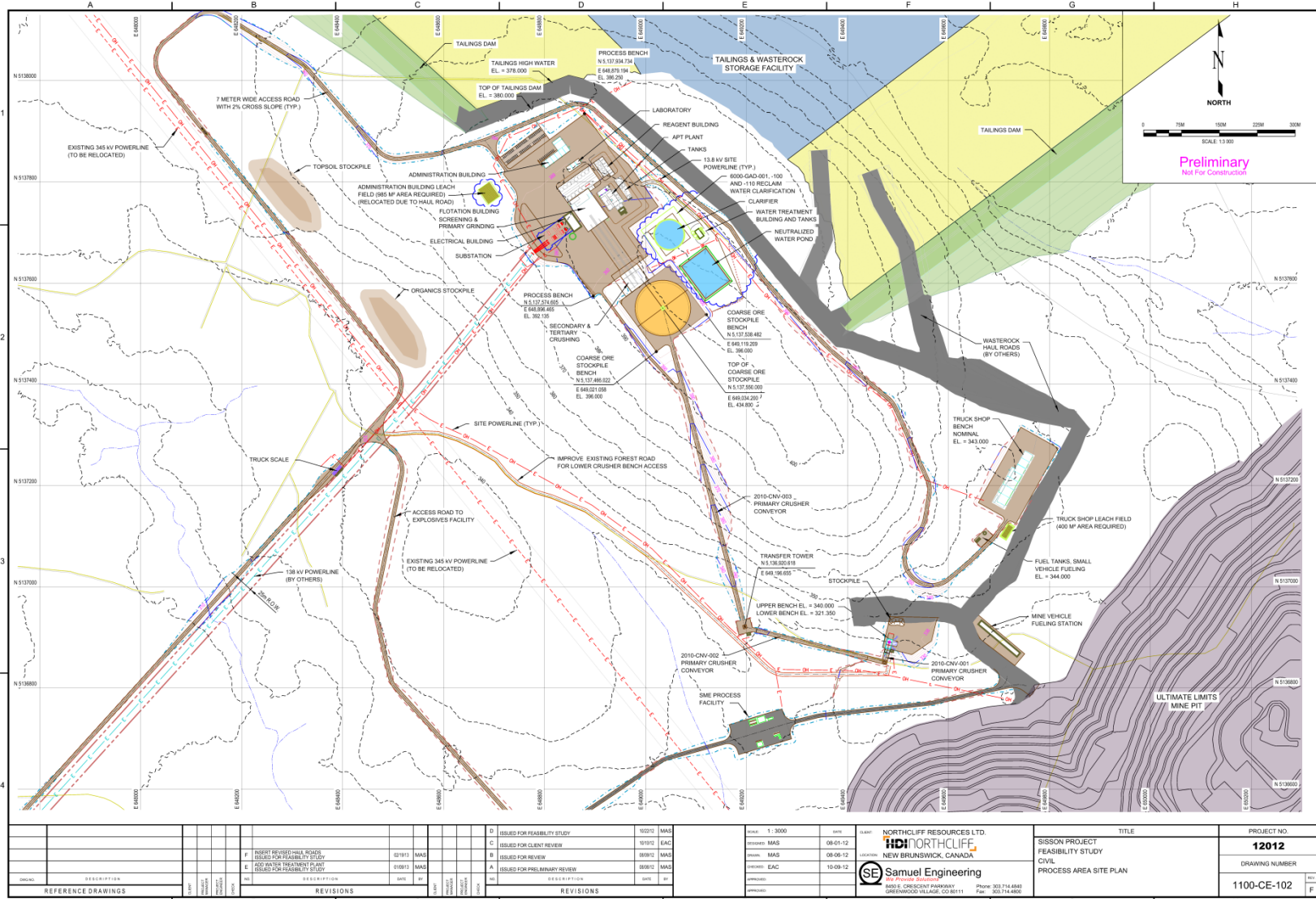


Figure 18.1: Project Area Site Plan

### 18.3 Power Supply

A 9 km section of an existing 345 kV transmission line (number 3011), which runs within the property boundary, will be re-routed at a minimum of 500 meters away from the mine pit. This line is part of the bulk transmission grid which is not intended to supply loads directly to industrial substations; it therefore cannot be used to provide power to the Sisson Project.

A new 42 km, 138 kV transmission line from the New Brunswick Power Transmission Corp. (NB Power) Keswick switching terminal will supply power to the Sisson Project substation. This new line will be constructed by NB power alongside the existing 345 kV line. A wood pole H-frame design will be used due to greater span lengths, proven reliability and the ability to withstand extreme weather events. Infrastructure at the Keswick terminal will be upgraded to accommodate the extension. NB Power will own the line and the Keswick switchgear but Northcliff will own the mine site terminal station. Power costs are estimated at \$0.066 per kWh in the Feasibility Study.

New Brunswick Power Transmission Corporation (September 21, 2012) conducted a design study to determine the cost to re-route the existing 345 kV line, to perform the required upgrade to the Keswick terminal and, to install the new 138 kV line up to the plant substation. The study also addressed cost for delivered power, right-of-way permitting, and environmental impact.

The 138 kV line will be terminated at a utility meter supplied by NB Power. The meter will be installed within a fenced substation located close to the site's main electrical room and concentrator building. The substation will include the main 138 kV disconnect switch, two 138 kV – 13.8 kV, 25/33.4/41.7 MVA transformers, and a 13.8 kV bus with distribution switchgear; the facility will operate on both transformers.

Power will be distributed to the plant facilities at 13.8 kV. Distribution will be routed via duct banks to facilities adjacent to the main substation (i.e. secondary crushing, grinding, flotation, and APT plant) while the power supply to remote locations, such as primary crushing, reclaim water system, quarry, truck shop, mine pit, and SME facility will be routed via overhead lines. Electrical equipment will be located inside electrical equipment rooms built onsite, and in pre-fabricated power distribution centers.

An electrical load analysis was developed based on data from the Sisson Project's Concentrator and APT plant equipment lists. The analysis utilized motor efficiencies and power factors typical of North American manufacturers published data. All motors, except those for the ball mills, assumed a maximum demand of 80% of the connected load. Average demand was set at 90% of the maximum demand load. Two standby generators (800 kW and 350 kW) will provide stand-by power to critical equipment during utility main power failure.

Availability factors used in the annual energy consumption calculation varied for different connected load types. Lighting loads are based on building floor space and an estimated load of 33 watts per square meter. A summary of the electrical load analysis is presented in Table 18.1.

<b>Table 18.1 Electrical Load Analysis Summary</b>	
Connected hp	69,295
Connected kW	51,694
Max. kW Demand	47,565
Avg. kW Demand	42,804
Avg. kVAR Demand	25,510
Power Factor	0.88*
<b>Area kWh/y:</b>	
Concentrator	266,996,725.31
APT Plant	9,309,451.73
Standby Generators	1,480,841.86
Mine Pit & Quarry	21,052,879.12
SME Facility	584,826.82
Weigh Scales Area	584,826.82
Water Clarification	1,449,864
Tailings Storage facility	6,049,932.63
Truck Shop	2,924,134.11
<b>Total kWh/y</b>	<b>310,433,482.16</b>
*The plant utilizes two ball mills with large synchronous motors so, with motor excitation, the power factor will exceed 0.90.	

#### 18.4 Crushing and Process Buildings

Secondary and tertiary crushing consists of two cone crushers and a high pressure grinding roll (HPGR), respectively. These will be housed in a single crusher building with a total area of approximately 1,120 m<sup>2</sup>.

Two ball mills make up the grinding circuit; they will be housed in a separate mill building with an area of approximately 3,395 m<sup>2</sup>. The concentrator building measuring approximately 3,416 m<sup>2</sup> will consist of large molybdenum and tungsten bulk flotation and scavenger cells, and reagent preparation and storage area. This building will also house the mine main control room as well as all concentrator operating personnel offices and a maintenance shop. A reagent storage shed measuring about 250 m<sup>2</sup> will be erected outside the reagent preparation and storage area of the concentrator building.

The APT building will be a two story building with a floor plan of approximately 25 m x 42.5 m. This building will house APT processing equipment, an electrical room, APT control room, lab, and a small personnel office.

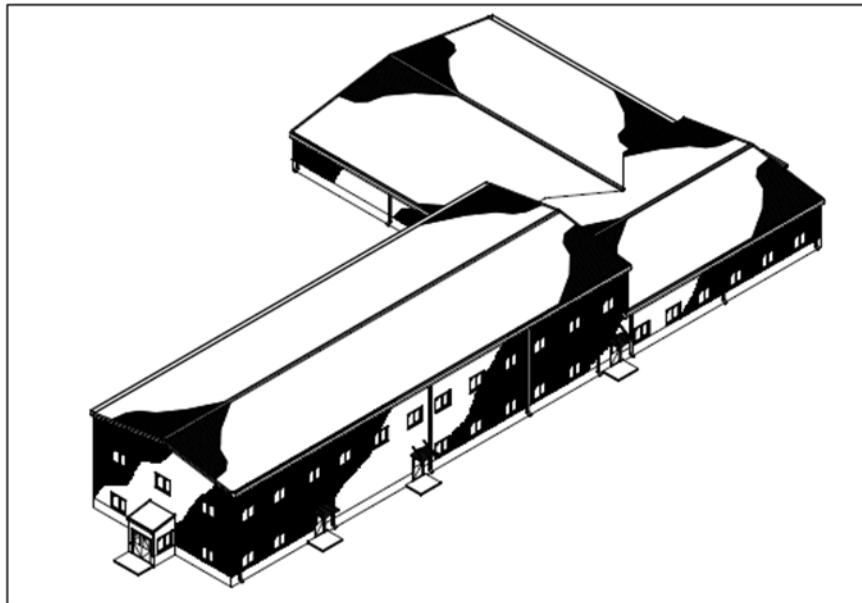
## 18.5 Site Ancillary Facilities

Site ancillary facilities will include an administration building, a laboratory building, truck shop and warehouse, fuel storage, site mixed explosive (SME) plant, and explosives and detonator magazines.

### Administration Building

The administration building will be a steel-framed, prefabricated, slab-on-grade building. The building footprint is L-shaped with a two-story segment measuring approximately 14 m by 40 m, and a single story segment measuring approximately 20 m by 34 m (see Figure 18.2).

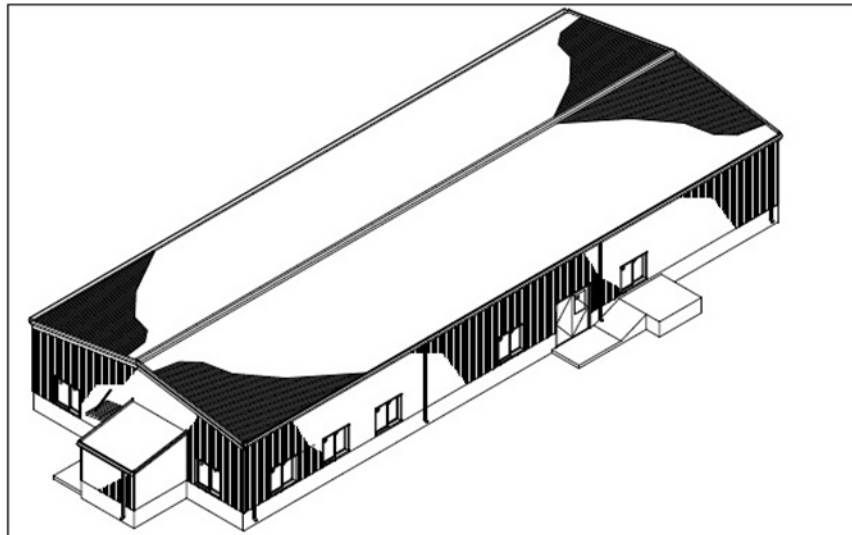
The administrating building will house space for site management, administration, mine management, engineering offices, conference rooms, archiving, building mechanical services, and washrooms. Dry change, and medical and safety offices will also be located in this facility. The building will be located north of the process plant.



**Figure 18.2: Administration Building**

### Laboratory Building

The laboratory building will be a single-story, steel-frame, prefabricated, slab-on-grade building measuring approximately 12 m by 30 m (see Figure 18.3). This building will house an analytical lab, metallurgical lab, sample preparation area, small office area, break room, and a washroom. The building will be located north of the process plant, adjacent to the administration building.



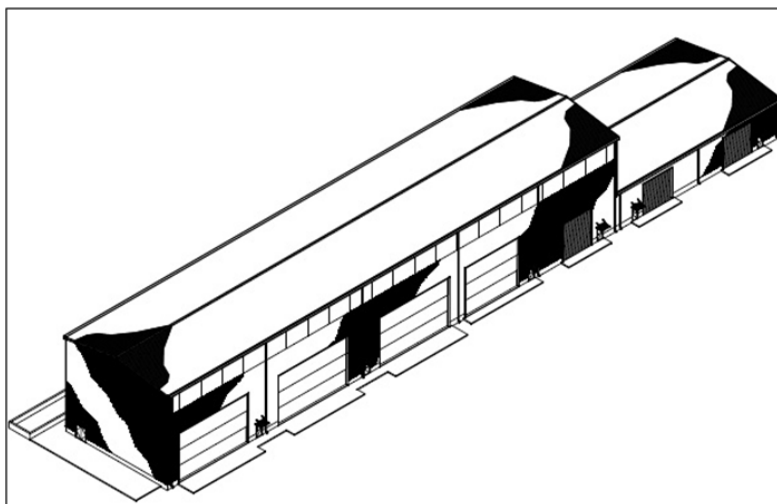
**Figure 18.3: Laboratory Building**

Truck Shop and Warehouse

The truck shop and warehouse building will be a single story, steel-framed, prefabricated, slab-on-grade building measuring approximately 25 m by 114 m (see Figure 18.4).

The building will house fleet repair facilities, wash bays, workshops, machine shop, a small office area, washrooms, and warehouse space for both mining and process facilities equipment. The building will be located approximately 800 m southeast of the process facility, close to the mine and mine haul roads.

A fuel storage depot and dispenser terminals will be located close to the truck shop. A storage shelter for a fire truck and mine rescue truck will be located adjacent to the truck shop.



**Figure 18.4: Truck Shop and Warehouse**



### Site Mixed Explosives (SME) Plant

A site mixed explosive (SME) plant and, explosives and detonator magazines will be located some distance west of the mine pit. The SME facility will store bulk ingredients required for producing the emulsion explosive used in the blast hole. It will also house all required pumps and tanks, truck wash bay and, blasting personnel offices and change rooms. The bulk ingredients are not explosive and at no time will explosive product leave the Sisson Project site. Explosives prepared at the SME would be transported directly to the open pit for placement in blastholes.

## **18.6 Communications**

The site telecommunication and high speed internet access needs will be served by a fiber-optic line that will be installed from the town of Keswick to the mine site. The fiber optic line will be strung on the same poles as the 138 kV power line. Radio systems will provide site and other local mobile communication to vehicles and personnel. Each major worksite will have a computer network that will support e-mail, internet, file access, project applications, and data backup services.

## **18.7 Sewage Treatment and Garbage Disposal**

Sewage treatment for the process plant area, administration building, and laboratory will be by leach-bed system. Leach fields will be sized based on the personnel requirements at the ancillary facilities. The main leach field, approximately 985 m<sup>2</sup>, will be located to the west of the main process bench and will include a berm for containment in the event of a failure. The truck shop and primary crusher leach field (approximately 400 m<sup>2</sup>) will be located southeast of the truck shop bench.

No landfill will exist at site; rubbish will be hauled offsite for disposal at municipal landfills, recycling yards, and approved construction and demolition sites. Certain areas around the plant site will be designated for dumpsters.

## **18.8 Fire Protection**

Filtered, process water will supply a combined filtered water and fire water tank. Firewater will be pumped from the fire water tank to the Concentrator and APT Plant firewater distribution system. The pumping system will consist of an electric pump, a jockey pump and, a diesel pump for use in the event of a power failure.

The filtered water discharge connection is at an elevation above the bottom of the water tank to ensure the remaining volume will be available for firewater purposes. Distribution will consist of a buried ring main around major facility buildings with hydrants and stand pipes connected to indoor hose stations. Allowances have been made for portable cart-type and handheld extinguishers for localized protection.

In addition to the hydrants and indoor stations, the APT building will employ a mist (fog) fire protection system at its solvent extraction area.

A separate firefighting system will be employed at the truck shop. It will consist of pumping and distribution system, hydrants, and stand pipes connected to hose stations inside the truck shop and will use fresh water from area wells.

## 18.9 Security and Fencing

Although no wildlife or security fencing is planned to encompass the entire plant site, security fencing will be installed around the substation and explosive storage areas.

There will be a remote operated security gate on the plant access road, just prior to the turn-off to the SME plant. This gate will be monitored 24 hours per day, 356 days per year by cameras and operated as needed from the administration building. Also, a vehicle barrier will be installed on the SME plant access road.

Weigh scales, used by delivery trucks, will be positioned on the site access road just prior to the security gate. The scales will be remotely monitored by cameras and scale readings will be accessible from the plant control room. Site delivery personnel and visitors will check in at the administration building.

The ore stockpile area and main process benches will be large enough to accommodate laydown areas during construction, no security is planned for these locations.

## 18.10 Logistics and Transportation

Sisson's operations workforce will primarily travel to site from neighboring communities to assembly points where the FSRs meet paved highways, and then by company-provided buses to site; some personal transportation to site is anticipated. Six light pickup trucks and one van will be assigned to key management, G&A, and operations personnel for transportation to and from site, and around site.

No onsite housing is required; construction labor will reside in the surrounding communities. Based on the assumption that about 70% of the labor force will require travel accommodations during the construction phase of the Sisson Project, a contractor busing allowance for the cost of busses, drivers, fuel, etc., has been made in the capital cost estimate. Bussing will be arranged and managed by each individual contractor.

Deliveries of equipment, materials and supplies to the Sisson Project site will be by truck. Products (molybdenum concentrate in 2-t bags and APT in drums) will be trucked from site, likely to a rail siding at Napadogan for onward shipment by rail. Overseas shipments will be handled through existing ports at Saint John or Belledune.

## 18.11 Offsite Offices and Storage

An offsite office for administrative and management personnel, whose direct presence at the mine and processing site is not required, will be employed at Fredericton. Approximately 8 people will work out of this office.

An offsite storage facility will be located at a nearby rail siding at Napadogan. This storage facility will include a warehouse and laydown area.

## **18.12 Mine Waste Management and Water Control**

### **18.12.1 Mine Waste**

Waste from mining operations will include tailings from the mill process and barren rock from open pit mining. Barren rock will be progressively encapsulated by tailings and submerged within the TSF to effectively mitigate the potential onset of acid generation. Barren rock will be stored in the TSF for the first 21 years of the mine life and in the open pit thereafter. Tailings will cover the barren rock piles after approximately Year 21 of the Sisson Project life. Details on the TSF and the barren rock storage areas are presented in Sections 18.13 and 16.1.7, respectively.

### **18.12.2 Water Management**

The general water management strategy is to divert non-contact surface water back to natural drainages using diversion channels to the extent possible, and to collect all contact water either in the TSF or in water management ponds for recycle back to the TSF. Contact water that exceeds the Sisson Project water demand will be treated as necessary to meet water quality objectives and released. Additional detail on the water management plan is presented in Section 18.13.5.

### **18.12.3 Open Pit De-watering**

Direct precipitation and groundwater infiltration will be pumped from the open pit during mining operations. Sumps will be installed in the low points within the open pit from which water will be pumped to a water management pond located adjacent to the pit rim. The pumps and pipelines will be sized to remove the inflow volume resulting from the 1 in 10-year design flood event within 10 days.

## **18.13 Tailings Storage Facility (TSF)**

The TSF is designed for secure and permanent storage of approximately 282 Mt of tailings, 17 Mt of mid-grade ore and 270 Mt of barren rock from the proposed mining operations over a 27 year mine life. All potentially acid-generating materials will be stored sub-aqueously within the TSF. General arrangements of the TSF are presented on the end of period layouts shown on Figures 16.18 through 16.23.

### **18.13.1 TSF Alternatives Assessment**

A Waste and Water Management Alternatives Assessment was completed to compare various TSF site locations, tailings technologies, and TSF embankment construction materials; the results of the assessment are summarized below.

#### Site Location Alternatives

Four potential TSF sites in the Sisson Project area were identified during the scoping stage. The current site was selected as the preferred location on the basis of environmental, logistical and economic factors. This site is located closest to the open pit and results in a minimized overall project footprint. Three TSF configurations were further assessed at the preferred site, from which the current project configuration was chosen.

### Tailings Technology Alternatives

A trade-off study was completed to evaluate the following tailings technologies:

- Un-thickened slurry tailings
- Paste tailings, and
- Filtered dry stack tailings.

The resulting recommendation was that an un-thickened tailings system, operating at approximately 35% solids content by weight, be used as the basis for further project development. This conclusion was based on several factors including the local climate, site water balance, overall system complexity, and ease of operation.

### Embankment Construction Material Alternatives

The TSF embankment design that was presented in the 2011 Preliminary Engineering Study assumed the use of barren rock from the open pit as a construction material. Geochemical evaluation of the barren rock in early 2012 indicated that some of the barren rock may be potentially acid-generating and would not be suitable embankment fill material. The mitigation strategy is to place and submerge all barren rock within the TSF and use quarried rockfill (characterized as non-potentially acid generating) for embankment construction. A trade-off study was undertaken in 2012 to compare cyclone sand and quarried rockfill as construction material alternatives. The results of the study indicated that quarried rockfill was the preferred embankment fill material option based on economic, environmental, ease of construction, and closure and reclamation considerations.

## **18.13.2 Tailings Storage Facility Design**

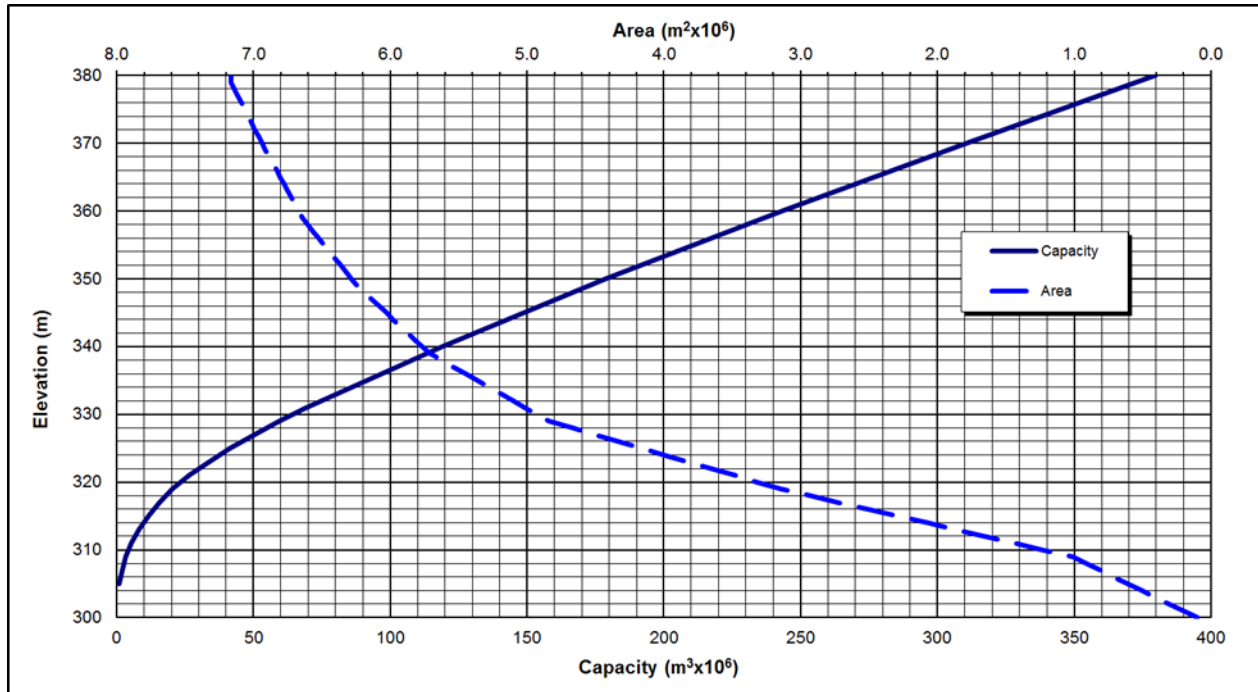
### General

A single TSF, confined by a perimeter embankment on the north, east, and south sides, and a saddle embankment on the west side, will be constructed to store all tailings and barren rock produced over the mine life.

The primary aspects of the TSF design include:

- Zoned embankments constructed of earthfill and rock
- Upslope TSF diversion channels
- Access roads and haul roads for embankment construction
- Seepage and embankment runoff collection ditches and ponds
- Tailings transport and deposition system
- Reclaim water system
- Tailings beaches
- Supernatant water pond, and
- Barren rock and mid-grade ore storage.

The depth-area-capacity relationship for the TSF design is shown on Figure 18.5. The TSF embankments are designed for staged expansions as the elevation of the stored tailings and ponded water increases over the mine life. TSF layouts at the various stages are shown on Figures 16.18 through 16.23.



**Figure 18.5: TSF Depth-Area-Capacity Relationship**

Embankments

The embankments will be constructed in stages as zoned earthfill and rockfill structures. Stage 1 includes the initial starter embankment that will be constructed prior to mill start-up; Stages 2 through 4 will be constructed throughout the mine life to meet tailings and water storage requirements. The final embankment will have an elevation of 376 masl.

Starter Embankments (Stage 1)

Three starter embankments will be constructed at the low points in the TSF impoundment using select overburden from local borrow sources near the embankment sites. The embankments will have a geosynthetic liner on the upstream face to elevation 318 masl to provide containment for a start-up water pond and the first year of tailings deposition. The liner will be anchored into a trench keyed into lower permeability bedrock along the upstream toe of the embankment.

Ongoing Embankment Raises (Stages 2 to 4)

The TSF embankments will be progressively raised by the modified centerline construction method using compacted tailings sand along with quarried rockfill. Transition and filter zones will also be incorporated to ensure compatibility and internal stability of the embankment fill materials. The low permeability zone of compacted tailings will be constructed on the upstream side of the exposed tailings beaches using dozer

compaction in hydraulic sand cells. The tailings zone will also have a relatively low permeability, and will mitigate seepage migration through the embankment.

#### Access

Temporary roads will be constructed within the TSF impoundment to provide access to the TSF starter embankments, borrow sources, and the initial water management ponds. Access will be provided by upgrading existing Forestry Service Roads (FSR) with new extensions built to FSR standards. The construction access roads will eventually be flooded by the TSF.

Permanent access to the TSF and water management ponds will be provided by the active haul roads built by the mine fleet. The crest of the embankments has been sized to allow for 2-way haul truck traffic with additional width for safety berms and pipelines. The location of access roads will change throughout the mine life to suit the demands of the mining operation and TSF construction.

#### Surface Water Diversion Channels

Diversion structures will be constructed upstream of the TSF to limit the inflow of non-contact surface run-off where possible. These diversion channels will consist of trapezoidal ditches or collection berms to divert clean runoff flow around the TSF.

#### Tailings Distribution

Tailings slurry from the tungsten circuit in the mill will be distributed around the TSF in pipelines and discharged from a series of offtakes located along the embankment crest.

The coarse fraction of the tailings is expected to settle rapidly and will accumulate closer to the discharge points forming a gentle beach with a slope of about one percent. Finer tailings particles will travel further and settle at a flatter slope adjacent to and beneath the supernatant pond. The beaches will be developed with the intent of maximizing the storage capacity and to control the location of the supernatant pond. Selective tailings deposition will be used to maintain tailings beaches and keep the supernatant pond a suitable distance from the embankments. Successful management of tailings deposition and beach development will reduce seepage through the embankments.

A separate tailings line will run from the mill directly into the TSF pond for subaqueous discharge of the molybdenum tailings, which represent approximately 5% of the total tailings produced over the mine life.

#### Barren Rock and Mid-Grade Ore Stockpile

The TSF is sized to store the tailings, water, barren rock, and mid-grade ore produced over the life of the Sisson Project. The barren rock and mid-grade ore will be placed in the TSF by the mine trucks; the active lift will remain above the TSF supernatant pond to provide a safe working platform. The barren rock will be located a sufficient distance from the embankment to ensure that the pile is completely encapsulated by deposited tailings solids.



Seepage and Contact Water Management

Seepage from the TSF will be largely controlled by the tailings beach and the upstream compacted tailings zone; seepage that is intercepted in the embankment will be routed to the water management ponds through embankment drainage systems. Surface water runoff from the embankment faces or other impacted areas in the vicinity of the TSF will also be collected in the water management ponds located at topographic low points along the downstream toe of the embankments. Water collected in the water management ponds will be continuously monitored and pumped back into the TSF depending on water quality. Groundwater monitoring wells will be installed around the TSF to monitor seepage and water quality. If necessary, water interception wells would be developed where required to recover groundwater seepage and to pump it back to the water management ponds.

**18.13.3 TSF Construction Methodology**

The construction of the TSF is divided into the four main stages as shown in Table 18.2.

<b>Table 18.2 TSF Staging</b>			
<b>TSF STAGE</b>	<b>EMBANKMENT CREST ELEVATION</b>	<b>END YEAR</b>	<b>PRIMARY CONSTRUCTION BY</b>
1	318 masl	-2	Contractor
2	338 masl	7	Mine Fleet
3	362 masl	19	Mine Fleet
4	376 masl	27	Mine Fleet

The TSF starter embankments will be constructed by a contractor and ongoing embankment raises will be built by the mine fleet. Construction of the TSF has been divided into three phases:

1. Site Establishment
2. Starter Embankment Construction
3. Ongoing Embankment Construction

Site Establishment

Site establishment will consist of the activities required prior to beginning construction of the starter embankments:

- Clearing timber from the construction and laydown areas
- Upgrading existing FSRs to an access road sufficient for the contractor’s equipment
- Establishing any temporary camps, maintenance shops, or other infrastructure that the contractor may require
- Preparing suitable laydown areas for equipment and cleared timber
- Construction of temporary by-pass channels, or cofferdams (depending on contractor strategy)
- Best management practices for silt and sediment control (e.g. sediment control ponds, silt fences, straw bales)

### Starter Embankment Construction (Stage 1)

The Stage 1 starter embankments will be constructed by a contractor two years before mill start-up. The Stage 1 elevation was selected to provide sufficient capacity to store water for mill start-up and the first year of tailings. The major construction activities will include:

- Clearing and grubbing of the starter embankment footprint
- Excavation and re-compaction of overburden material for the Stage 1 Embankment footprints
- Installation and operation of construction dewatering equipment (where required)
- Overburden and topsoil stockpile development
- Development of local borrow sources
- Cofferdam construction upstream of the embankments and installation and operation of dewatering systems (if required)
- Construction of the Stage 1 Embankments
- Installation of the HDPE liner and placement of ice protection layer
- Removal of dewatering equipment
- Installation of tailings and reclaim pipework
- Construction of water management ponds and pumping systems

### Ongoing Embankment Construction (Stage 2 onwards)

Ongoing construction will include staged embankment raises and the installation of additional tailings and reclaim pipelines. Embankment raises will be completed using rockfill from the quarry located at the northwest corner of the TSF. The mine fleet will deliver quarried rock to the embankments, including processed filter and transition zone materials. A contractor will likely be used to spread and compact the filter and transition zones as they may be too narrow for the mine equipment to operate on efficiently. The major Stage 2 construction activities are:

- Continued clearing of the impoundment, as required
- Continued grubbing, stripping, and excavation of unsuitable overburden beneath the expanded embankment footprints
- Modified centerline embankment raises using:
  - Hydraulic placement and dozer compaction of deposited tailings in sand cells in the upstream zone of the embankment
  - Placement of select overburden and processed filter and transition materials by a contractor
  - Quarried rockfill delivered by the mine fleet for the downstream shell zone
- Installation of additional tailings pipelines to reach the full extent of the embankments
- Construction of a rockfill causeway in the TSF using barren rock to allow repositioning of the reclaim barge.

### Scheduling Considerations

The primary scheduling constraints are due to seasonal variations and the resulting changes in temperature, snow pack, precipitation, and stream flow. The major scheduling constraints are summarized below:

### Environmental

- Water quality downstream of the embankments (silt and sediment control from impacted areas).

### Precipitation and Temperature

- Access restrictions due to snowfall in winter months
- Lower construction productivities due to cold weather (e.g. ice hazards on access roads, material cohesion)
- Cold weather limitations on placing and compacting cohesive materials and welding HDPE liner
- Snow clearing
- Spring freshet, storm events, and flood flows.

## **18.13.4 TSF Seepage and Stability**

### Seepage Analysis

Steady state seepage analyses were completed to estimate the phreatic surface for stability analyses and to estimate seepage flows for design of the seepage collection system. Seepage rates were estimated for each unique section of the embankments at various stages in the project life, including start-up, operations, and closure. These analyses were carried out to support the design of seepage management features in the TSF.

### Stability Analysis

Embankment stability analyses were carried out for both static and seismic conditions under the following cases:

- Static conditions during operations and post-closure
- Earthquake loading from the Operating Basis Earthquake (OBE) and the Maximum Design Earthquake (MDE), and
- Post-earthquake conditions using residual (post-liquefaction) tailings strengths.

The results of the stability analyses satisfy the minimum requirements for factor of safety (FOS) and indicate that the proposed design is acceptable to maintain both short term (operational) and long term (post-closure) stability. The seismic analyses indicate that any embankment deformations during earthquake loading from the OBE or MDE would be minor and would not have a significant impact on embankment freeboard or result in any loss of embankment integrity. The results also indicate that the embankments are not dependent on tailings strength to maintain overall stability and integrity, as tailings strength would be minimal following a seismic event.

## **18.13.5 TSF Water Management Plan**

The following sections outline the water management plan for the TSF from pre-production (Year -2) through to the end of operations (Year 27).

### Construction Water Management

Construction water management at the TSF will commence approximately two years prior to mill start-up and coincide with initial construction of the facility. This phase is characterized by clearing, grubbing and stripping, and the development of access roads and haul roads. Water management during this phase will consist of establishing collection ponds, cofferdams, pumping systems, runoff collection ditches, and diversion channels. Some of the temporary works such as cofferdams and by-pass diversion channels will be removed once the initial starter embankments have been constructed. Sediment collection ponds and collection ditches will remain in place throughout the life of the Sisson Project.

### Operational Water Management

The operational water management plan for the TSF includes:

- All un-diverted runoff from within the TSF catchment will report to the TSF.
- Process water contained in the tungsten and molybdenum tailings will be discharged into the TSF with the tailings slurry at an average rate of approximately 2,022 m<sup>3</sup>/h at full production.
- Tailings supernatant water will be reclaimed, treated and pumped back to the mill to the extent possible to meet the average process water requirement of approximately 2,003 m<sup>3</sup>/h at full production.
- Water will be discharged from the TSF to a treatment plant when the facility is operating in a water surplus condition, likely starting mid-way through the mine life under average climatic conditions, to maintain an acceptable TSF operating pond volume. The reclaim barge has been sized to accommodate this additional flow rate.
- Tailings will be selectively deposited from the crest of the embankments to develop tailings beaches which function as an extensive low permeability zone to mitigate seepage through the embankments. The operational supernatant pond volume will be managed to reduce the potential for dust generation and to ensure that sufficient storage exists for operational flexibility and storm inflow storage.
- Diversion channels upstream of the TSF will divert non-contact water away from the TSF to the extent possible. This water may be collected to control sediment before discharge.
- Water management ponds at low points around the TSF perimeter will collect seepage and runoff from the TSF embankments. This water will be pumped back to the TSF unless the water quality is suitable for discharge.
- Groundwater monitoring wells will be located below the water management ponds. Water interception wells will be developed as necessary if excessive seepage is detected to supplement the water management ponds.
- Water from the open pit will be pumped to a collection pond near the pit rim and discharged or pumped to the water treatment plant depending on water quality. Any surplus water must meet applicable water quality criteria prior to being discharged to the environment.

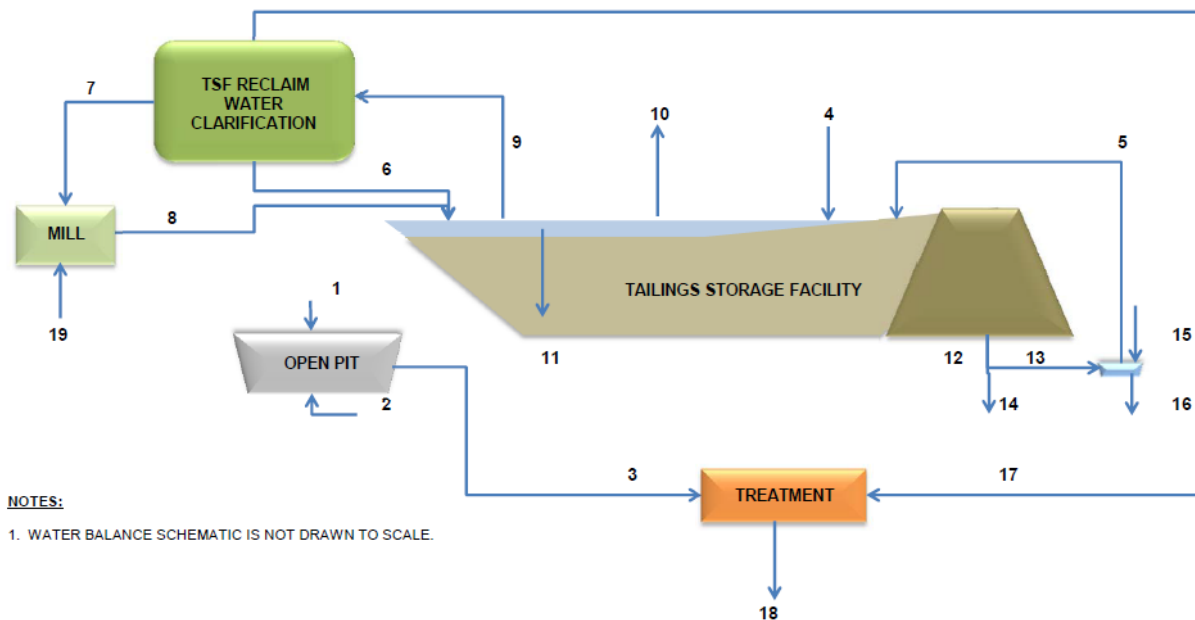
### Water Balance

A stochastic analysis was carried out on the base case monthly operational mine site water balance using the GoldSim<sup>®</sup> software package. The intent of the modeling was to estimate the magnitude and extent of any water surplus and/or deficit conditions in the TSF based on a large range of possible climatic

conditions. The modeling timeline included one pre-production year (Year -1), and 27 years of mine operation. The model is shown schematically on Figure 18.6 and incorporates the following major project components:

- TSF
- Mill
- Open pit
- Tailings Reclaim Clarification Plant, and
- Water Treatment Facility

The assumptions and parameters used in the model are discussed in the following section.



NO.	DESCRIPTION
1	Open Pit direct precipitation and catchment runoff
2	Open Pit groundwater inflows
3	Open Pit dewatering direct to treatment
4	TSF catchment and beach runoff; direct precipitation on pond and exposed
5	TSF seepage pond recycle
6	Water from Clarification Plant to TSF pond
7	Water from Clarification Plant to Mill
8	Water in slurry to TSF
9	TSF reclaim water to Clarification Plant
10	TSF pond evaporation
11	Water trapped in tailings and waste rock void spaces
12	TSF emankment seepage - total
13	TSF embankment seepage - captured by seepage collection system/pond
14	TSF embankment seepage - lost (unrecoverable)
15	TSF seepage pond embankment and catchment runoff
16	TSF seepage pond - water lost to environment
17	Excess reclaim water to treatment
18	Treated water discharged to environment
19	Freshwater make-up to Mill

**Figure 18.6: Mine Site Water Balance Schematic**

## Model Assumptions

### Average Climatic Conditions

The base case monthly operational water balance model was developed using average estimated values for runoff and precipitation. The mean annual unit runoff for undisturbed basins in the Sisson Project area was estimated to be approximately 827 mm. The mean annual precipitation used in the model was 1350 mm, with 75% of the annual precipitation falling as rain and the remainder as snow. The annual average potential evapotranspiration for the Sisson Project site was estimated to be 500 mm.

### Stochastic Inputs

The variability of climatic conditions was addressed using a stochastic version of the water balance model that included Monte Carlo type simulation techniques. The monthly climate parameters were modeled as probability distributions rather than simply as mean values.

### Water Balance Results

The water balance model results were used to estimate the likelihood of having a surplus or deficit of water in the TSF. The TSF pond is predicted to be in a surplus condition for the entire operating life of the mine, indicating that the system (including the TSF and contributing catchments) is able to supply more than enough water to meet the mill process water requirements, even under dry conditions.

## **18.13.6 Tailings Management Systems**

### General

Tailings will flow from the mill to the TSF by gravity in HDPE pipelines during the initial years of operation. Pumping will be required in approximately Year 13 as the tailings elevation rises. The tailings distribution system includes:

- Full Capacity Tungsten Tailings Line
- Half Capacity Tungsten Tailings Dump Line
- Molybdenum Tailings Line
- Clarification Plant Underflow Discharge Line, and
- Neutralization Pond Precipitate Discharge Line

### Full Capacity Tungsten Tailings Line

The full capacity tungsten tailings pipeline is designed for the nominal mill throughput of 30,000 tpd plus 20% surge capacity. Tailings will be discharged from the mill in two 34" pipelines running in opposite directions along the TSF embankment crest. Each pipeline will cover approximately half of the total embankment crest length and is sized to handle the full mill throughput to allow for increased operational flexibility and reduced pumping requirements.

Tailings will be discharged from a series of spigots along the TSF embankment crest. The discharge spigots will consist of flanged tees, approximately 20 m of HDPE discharge piping, and a knife gate



valve. Moveable sections of pipeline with discharge spigots will be incorporated into each pipeline to facilitate beach development through strategic deposition of the tailings.

#### Half Capacity Tungsten Tailings Dump Line

A minimum tailings flow velocity is required to prevent pipeline blockage due to settlement of solids. If the plant is running at a reduced throughput, for example during scheduled maintenance or unexpected downtime, the full capacity tailings lines may be too large to prevent the tailings from settling. A smaller diameter 24" half capacity dump line is included as a design provision to allow for continued plant operation at reduced throughputs. The half capacity dump line will run north along the embankment crest from the mill directly into the TSF and will discharge at a single point in the supernatant pond.

#### Molybdenum Tailings Line

A 10" diameter molybdenum tailings line will run parallel to the half capacity dump line to the TSF supernatant pond. The molybdenum tailings will be discharged sub-aqueously, beneath the supernatant pond, to limit the potential for the tailings to be exposed to oxygen. The discharge point will be placed an adequate distance away from the embankment and from the water reclaim barge intake.

#### Clarification Plant Underflow Discharge Line

Underflow from the clarifier will be pumped to the TSF in an 8" diameter HDPE pipeline. Discharge will occur on top of the barren rock pile or directly into the supernatant pond.

#### Neutralization Pond Precipitate Discharge Line

Neutralization pond precipitate will be pumped to the TSF in an 8" diameter HDPE pipeline. Discharge will occur on top of the barren rock pile or directly into the supernatant pond.

### **18.13.7 Water Reclaim System**

The water reclaim system will pump water from the TSF supernatant pond to the process water head tank at the mill for reuse in the process. The pumping requirements and power consumption will decrease over the Sisson Project life as the elevation of the supernatant pond increases. The design flow for the reclaim pipeline is sufficient to meet the mill demand, with additional capacity for discharge to a water treatment plant, if required. Additional surge capacity is also included. The reclaim water demand for the mill is approximately 2,003 m<sup>3</sup>/h based on a nominal 30,000 tpd milling rate and a tailings slurry at 35% solids content by mass. The pipeline and barge have been sized to allow for an additional 250 m<sup>3</sup>/h of TSF pond water discharge and 30% surge capacity.

The water reclaim system will consist of a floating pump barge and a single pipeline running to the mill head tank. The barge will be repositioned throughout the Sisson Project life to ensure adequate pump submergence and to minimize the length of the pipeline. Access to the barge will be provided by a causeway constructed using barren rock. The pump barge design includes provisions for winter operation, including a prefabricated building enclosure and a de-icing system consisting of mechanical agitators.

### 18.13.8 Instrumentation and Monitoring

#### Instrumentation

Geotechnical instrumentation comprising vibrating wire piezometers, slope inclinometers and movement monuments will be installed along instrumentation planes within the TSF embankments and foundations during initial construction and throughout the life of the Sisson Project. The instrumentation will be monitored during the construction and operation of the TSF to assess embankment performance and to identify any conditions different to those assumed during design. Amendments to the ongoing designs and/or remediation work will be implemented to respond to changing conditions, should the need arise.

Groundwater monitoring wells will be installed at suitable locations downstream of the embankments.

Flow and level monitoring instrumentation will be installed in the embankment foundation drainage system and seepage collection ponds.

#### Monitoring and Inspections

Instrumentation will be monitored routinely during construction and operations. Measurements will be taken and analyzed regularly during construction to monitor the response of the foundations and embankments to fill loading.

The frequency of monitoring for the piezometers and inclinometers may be decreased to bi-monthly readings once the effects of initial construction have dissipated. Surface movement monuments will be surveyed at least quarterly during operations. Water quality monitoring of the seepage through the embankment and foundation will be monitored monthly during operations.

In addition to the routine inspections carried out by mine personnel on a shift/daily/weekly and monthly basis, the facility will be audited regularly by a suitably qualified professional engineer to ensure it is operating in a safe and efficient manner. Such audits will be conducted annually. A dam safety review will be conducted every five years by a qualified Geotechnical Engineer.

### 18.13.9 Tailings Characteristics

Tailings produced from the mill process will be discharged to the TSF at an average slurry solids content of 35% by weight, at a throughput of 30,000 tpd, or approximately 10.5 Mt/a. Geochemical characteristics are provided in Section 20.2.

Laboratory testing of representative tailings samples was completed during the feasibility study. Tailings samples were obtained from the Sisson Pilot Test Plant and tested by KP at its laboratory in Colorado, USA. Samples for both the tungsten tailings and molybdenum tailings were tested. Bench scale tests were carried out to determine the physical properties of the tailings, including classification tests, slurry settling tests, air drying tests, and consolidations and permeability tests.

The average settled dry densities of the tungsten and molybdenum tailings were estimated to be approximately 1.47 t/m<sup>3</sup> and 1.0 t/m<sup>3</sup>, respectively. Actual field densities will depend on the rate of rise,

degree of consolidation, overall depth of deposited tailings, and tailings segregation during deposition into the impoundment.

#### 18.13.10 Geotechnical – TSF Foundation Conditions

Site investigations were conducted in the summer and fall of 2011 to evaluate the geotechnical and hydrogeological conditions at the TSF and other proposed mine site infrastructure. The 2011 site investigation included:

- 604 m of drilling in 17 geotechnical drill holes using auger and coring techniques for the TSF and plant site areas
- Detailed geotechnical logging to characterize the overburden and rock mass quality
- In-situ hydraulic conductivity testing to evaluate permeability of the rock mass
- 65 test pit excavations completed in the TSF, plant site and open pit areas
- Installation of 11 hydrogeological wells (5 shallow and 6 deep) to monitor groundwater levels, test hydraulic conductivity, and for groundwater sampling, and
- Laboratory test work on soil samples recovered from the drill holes and test pits.

The geotechnical conditions of the overburden and bedrock were assessed based on the geological and geotechnical information collected. The site is characterized by the following:

- **Overburden** - Typically includes a sand and gravel unit overlying a till unit. The till includes sand, gravel, and silt with varying amounts of cobbles and boulders. The overburden thickness is between 1.5 m and 17 m and is generally thicker in the valley bottoms and becomes thinner on the hill slopes. The overburden typically comprises the following units:
  - Topsoil comprised of organic materials and a variety of roots and leaves with varying quantities of silt and sand. The topsoil ranged from 0 to 1.4 m thick.
  - A sand and gravel unit between 0 to 7 m thick, and/or
  - A till unit ranging from 1.5 to 9.5 m thick. The till was found to comprise silt, sand, gravel and cobbles.
- **Granite Bedrock** – Typically black and white, fine to medium grained crystallized granite with the following properties:
  - Classified as ‘GOOD’ rock using Rock Mass Rating classification (RMR) with values ranging from 29 to 90 and a mean of 66,
  - Rock Quality Designation (RQD) values range from 0 to 100% with a mean value of 78%. The lower values are from the samples recovered from broken zones, and
  - In-situ hydraulic permeability testing indicates that the rock mass has a hydraulic conductivity ranging from  $10^{-6}$  to  $10^{-3}$  centimeters per second (cm/s).

#### 18.13.11 Hazard Classification

A dam classification was carried out to enable appropriate design earthquake and flood events to be determined for the TSF. The selection of the design flood and earthquake events is based on the

classification criteria provided by the Canadian Dam Associations (CDA) Dam Safety Guidelines (2007). These guidelines are the recommended standard design practice for major impoundments, water management facilities and dams.

### **18.13.12 Hydrometeorology**

Details on climate, climate data collection, hydrometric monitoring and, climate analyses to predict long-term regional climate patterns at the Sisson Project site are presented in Section 5.2.

#### Design Storm Events and Inflow Design Flood

Selection of an appropriate Inflow Design Flood (IDF) was required to carry out a safety assessment of the TSF and to estimate flood storage requirements. The size of the IDF increases with increasing consequences of failure. Based on the classification assigned to the TSF, an appropriate IDF is a probabilistically derived event with a return period of two-thirds between the 1-in-1,000-year flood and the PMF. However, for this study, the deterministically derived 24-hour PMF has been conservatively selected as the IDF for design of the TSF. The TSF is designed with sufficient capacity and freeboard to store the Probable Maximum Flood (PMF) at all times during operations. The storm storage volume required during operations is approximately 4.8 Mm<sup>3</sup>, corresponding to an equivalent runoff depth of 583 mm.

### **18.13.13 Seismicity**

A preliminary assessment of the regional seismicity has been carried out to enable selection of appropriate design earthquake events and ground motions. A summary of the assessment and results is presented below.

#### Seismicity Assessment

Eastern Canada is located in a stable continental region within the North American tectonic plate, and has a relatively low rate of seismic activity. However, moderate to large earthquakes have occurred in the region and project planning has been done with the view that this can happen in the future.

The Sisson Project is located within the Northern Appalachians seismic zone, where the level of historical seismic activity is low. The largest earthquake instrumentally recorded in New Brunswick was a Magnitude 5.7 event in 1982, located in the north-central Miramichi Highlands. Seismicity is greater along the St. Lawrence River, to the north and northwest of the Sisson Project site. This includes the Charlevoix seismic zone, which is the most seismically active region of eastern Canada. There is potential for large earthquakes of up to about Magnitude 7.5 along the fault zones associated with the St. Lawrence River. However, these events would be located over 200 km from the Sisson Project site, and therefore the amplitude of ground motions experienced at the site would be low due to attenuation over a large distance.

Review of historical earthquake records and regional tectonics indicates that the Sisson Project site is situated in a region of low seismicity. A probabilistic seismic hazard analysis has been carried out using historical earthquake data and the regional tectonics to identify potential seismic sources and to estimate the maximum earthquake magnitude for each seismic source. The corresponding median maximum acceleration is 0.07 g for a return period of 500 years.

## Design Earthquake

Consistent with the current design philosophy for geotechnical structures such as dams, two levels of design earthquake have been considered: the OBE for normal operations, and the MDE for extreme conditions (ICOLD 1995). Values of maximum ground acceleration and design earthquake magnitude have been determined for both the OBE and MDE.

The CDA Dam Safety Guidelines recommend that the mean maximum acceleration value should be used for dam design. This is likely to be similar or slightly higher (by about 20%) than the median value provided by Natural Resources Canada (NRCan). Consequently, estimated mean maximum acceleration values have been adopted for the design earthquake events used in seismic stability analyses.

The OBE has been taken as the 1-in-500-year return period event for the design of the TSF. The maximum mean average acceleration is estimated to be 0.07g for the 1-in-500-year earthquake. A design earthquake magnitude of 7.0 has been selected for the OBE, based on a review of regional tectonics and historical seismicity. The TSF would be expected to function in a normal manner after the OBE.

An appropriate MDE for embankment design has been selected based on the dam hazard classification and the criteria for design earthquakes provided by the CDA Dam Safety Guidelines (2007). The CDA guidelines require that the dam contemplated hazard be designed for a probabilistically derived event (known as the Earthquake Design Ground Motion) having an annual exceedance probability (AEP) of 1-in-5,000. Consequently, the MDE selected for the TSF is the 1-in-5,000-year earthquake. The mean average maximum acceleration is estimated to be 0.37 g for the 1-in-5,000-year earthquake. A design earthquake magnitude of 7.0 has been selected for the MDE based on a review of regional tectonics and historical seismicity. Limited deformation of the tailings embankment is acceptable under seismic loading from the MDE, provided that the overall stability and integrity of the TSF is maintained and that there is no release of stored tailings or water (ICOLD 1995).

### **18.14 Water Supply**

#### **18.14.1 Mill Process Water Supply**

The feasibility study water balance indicates that the Sisson Project will operate in a surplus condition over the 27-year mine life. Prior to mill start-up, water will be impounded in the TSF for two freshet periods in order to collect an adequate volume of water for mill start-up. Water for processing will be pumped from the TSF supernatant pond to a head tank located at the mill via a floating reclaim pump barge and pipeline at a nominal flow rate of 2635 m<sup>3</sup>/h. The recirculation and reuse of water for mill processing represents a significant saving in fresh water use.

#### **18.14.2 Mill Fresh Water Supply**

A fresh water supply pipeline from groundwater wells will supply the mill process fresh water requirements at a flow rate of approximately 21 m<sup>3</sup>/h. It should be noted that the fresh water requirements are small as compared to the mill process water requirements that will be recycled from the TSF supernatant pond, as summarized above.

**18.14.3 Potable Water Supply**

Potable water for use in sanitary systems will be supplied by the mill fresh water supply wells. Drinking water will use the mill fresh water supply system with treatment, or will be delivered to site.

**18.15 Reclaim Water Clarification Facility**

The reclaim water clarification facility will be a single-story, engineered, concrete building measuring approximately 10 m by 18 m. The building will contain flocculant and lime systems with mixing and dosing equipment, storage and mixing tanks, and associated piping, pumps and electrical components. Barge-mounted pumps located in the tailings storage facility pond area will feed the plant. Treated effluent will flow, by gravity, to a neutralized water pond and from there fed by gravity to the process water tank located at the process bench. The treatment plant will be located on the southeast side of the process bench.



## 19.0 Market Studies and Contracts

### 19.1 Markets

Northcliff engaged Roskill Consulting Group Ltd., London, U.K. (Roskill) to conduct a comprehensive study dated November 2, 2012 and titled “Tungsten and Molybdenum Market Study and Price Forecast” for the Sisson Project. Roskill is an independent metals market research company with extensive experience working with metals producers and refiners worldwide. The following analysis is a summary of this independent marketing report.

#### 19.1.1 Tungsten

##### Tungsten Demand

Current global tungsten consumption is estimated to be 88,400 t W. In a base-case demand scenario, demand for tungsten is forecast to reach 112,750 t W by 2017 and 148,500 t W by 2025 (Table 19.1). In a more optimistic scenario, where growth is closer to that seen in the mid-2000s, tungsten demand could reach 178,900 t W by 2025. However, the global economy remains fragile following the 2009 downturn, especially because of continuing debt problems in Europe, and this could weigh on future tungsten demand growth; in a pessimistic scenario, demand would reach 123,000 t W by 2025, more consistent with the longer-term trend in consumption.

Table 19.1 World: Forecast Tungsten Consumption, 2011-2025 (t W)			
Year	Pessimistic	Base-case	Optimistic
2011	88,400	88,400	88,400
2017	105,000	112,750	121,100
AAGR '12 - '17(%)	3.5	5.0	6.5
2025	123,000	148,500	178,900
AAGR '17 - '25(%)	2.0	3.5	5.0
2011	88,400	88,400	88,400

Source: Roskill forecast

##### Tungsten Supply

From 2012, primary production of tungsten is likely to grow relatively strongly as a number of tungsten projects are at later stages of feasibility, construction and/or financing. However, not all of the potential new primary tungsten supply will be commissioned in the short-term and may be delayed until market demand increases.

##### Tungsten Historical Prices

In the 1990s, the tungsten market was characterized by the cheap availability of tungsten from China, with prices below US\$100/mtu for APT. The tungsten market witnessed a sharp growth in consumption in the

mid-2000s, combined with limited availability of material from China because of production and export quotas and a lack of new supply coming on stream outside of China. This caused the market to tip into a large deficit and APT prices increased more than three-fold from US\$80/mtu to over US\$250/mtu. Prices then stabilized, as sales of stockpiled material and illegal/unlicensed output in China increased. Prices declined by around US\$50/mtu in 2009 as the global economic downturn significantly reduced demand for products using tungsten in their manufacture, but this was not as severe a fall as for some other metals.

In 2010, APT prices recovered to 2008 levels and then accelerated through 2011 reaching over US\$460/mtu by mid-year. Prices then stabilized, but from early 2012 have fallen fractionally to around US\$360/mtu cost, insurance and freight (CIF) Europe.

### Tungsten Market Outlook

Illegal mining and smuggling, and sales of stockpiled material, have overhung the tungsten market in recent years and tended to act as a brake on price rises; this was particularly true between 2005 and 2008. However, most of the material contained in stockpiles has now been sold and illegal mining and smuggling is now much more closely controlled in China. Trends in future tungsten prices should therefore correlate more closely to the underlying supply/demand fundamentals, and perhaps explain why prices have increased so sharply in recent years.

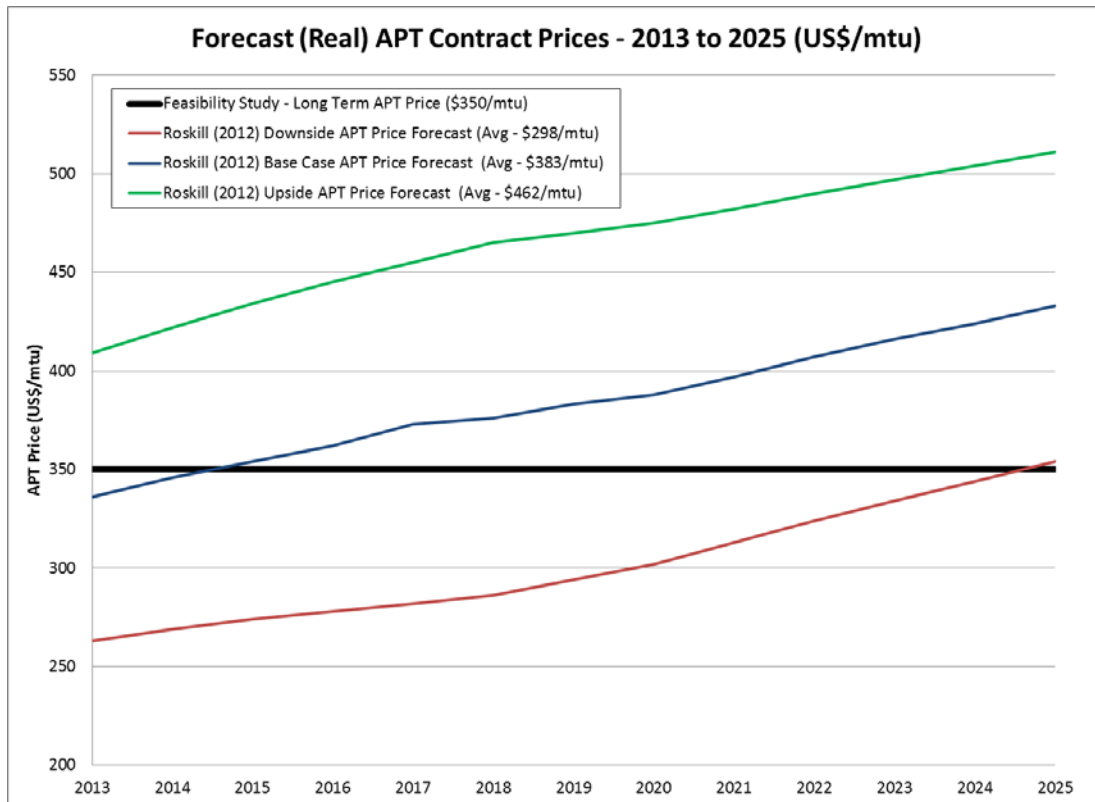
Using the combined supply and demand forecasts, Roskill has forecast the future tungsten market supply/demand balance to 2025. The base-case and pessimistic scenarios show that should supply continue to grow as projected, demand will be covered at least until 2025 and the market will remain in balance. In an optimistic demand scenario, supply may not be sufficient to meet demand in 2013/14, while new projects are commissioned, and again from 2020. In reality, in the longer-term, more new projects may emerge to meet demand in the optimistic scenario.

Long-term, tungsten prices are expected to increase irrespective of the supply/demand balance. Tungsten production costs have been rising globally and Chinese producers are now more sensitive to tungsten prices than in the past due to rising energy, labor, equipment and reagent costs. Based on marginal costs of US\$250/mtu in 2012 and assuming cost inflation rises by an above-inflation 5%/py, APT prices would have to be at least above US\$370/mtu in 2020 and US\$470/mtu by 2025 in nominal terms. If prices fell below these levels for any extended period of time then tungsten production would be removed from the market until prices reflected the true cost of production.

In 2011, average values of exports of APT freight on board (FOB) China were US\$366/mtu and in the first six months of 2012 the average was US\$376/mtu. Very few of the significant new tungsten projects delivered any substantial tonnages of tungsten in 2012, so the market will be relying on existing producers (principally China). Given that APT prices are currently on a short-term downward trajectory, the average APT price for 2012 was estimated at US\$370/mtu.

Prices are expected to ease further in 2013 as new tungsten production capacity is expected to enter the market. Roskill forecasts an average price of US\$345/mtu in 2013; this will probably mark the low point for prices in the short-term. From 2014, prices are anticipated to rise by almost 4% per year and reach an average of US\$450/mtu in 2020 (US\$388/mtu in real terms). By 2025, average annual prices for APT could be US\$550/mtu (US\$433/mtu in real terms).

Based on its analysis, Roskill projects APT metal prices for the period 2013-2025 as shown in Figure 19.1.



**Figure 19.1: Forecast (Real) APT Contract Prices – 2013 to 2025**

For the Sisson Project Feasibility Study economic analysis purposes, SE is using a flat APT price over the mine life of \$350/mtu. This base case price was derived from the average price for 2014 (\$346/mtu) and 2015 (\$354/mtu) which is the earliest time construction of the Sisson Project could commence.

Tungsten Offsite Charges

Prices quoted in the Roskill report are based on contracted APT price and it is assumed that Northcliff will be responsible for transportation of APT to the buyer. A global tungsten container freight charge is conservatively estimated to be \$125/t and other offsite costs (losses, insurance, representation, etc.) are estimated to be 0.15% of the value of the shipment. These terms have been used as the average realization charges over the life of the mine for APT in the economic model, equivalent to \$0.10/t milled.

**19.1.2 Molybdenum**

Molybdenum Outlook

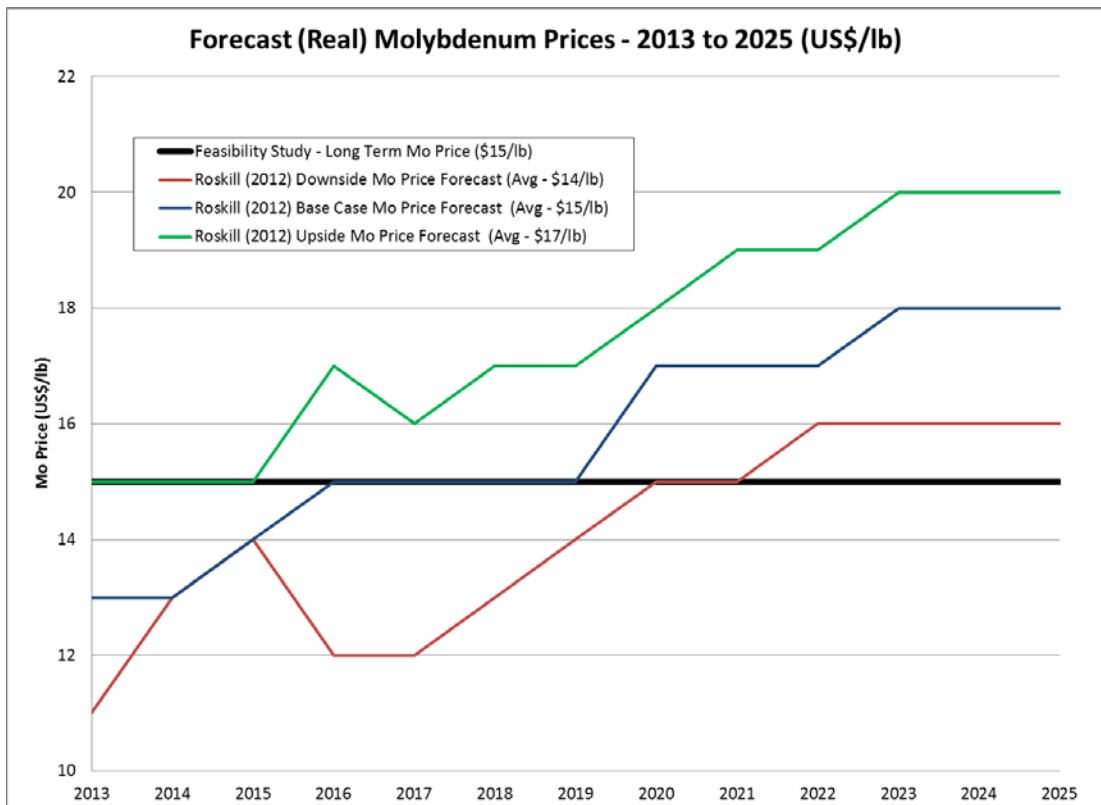
With the requirement for high grade steel alloys continuing to rise in a number of industries, demand for molybdenum appears set to increase steadily, particularly in the industrializing and emerging economies of Asia and South America. Growth is forecast at 5% per year, somewhat higher than forecast global

GDP growth, resulting in molybdenum demand almost doubling from 225kt Mo in 2011 to about 435kt Mo in 2025.

Existing molybdenum mine capacity is probably adequate to satisfy global demand until 2017/2018. Mine production will be surplus to demand until 2015; thereafter, the market will be broadly in balance and may show a small deficit by the early 2020s. However, in the longer-term, new/expansion projects will probably emerge to balance demand.

The combination of stable and adequate supply and steadily rising demand should be reflected in equally stable prices, albeit at lower levels than witnessed over the last five years. Prices will rise slowly irrespective of market balance, largely in response to production costs. The price of technical molybdenum oxide is forecast to reach US\$20/lb Mo in 2025 (US\$18/lb Mo in real terms) and average US\$17/lb Mo (US\$15/lb Mo in real terms). This presupposes that concentrate roasting capacity remains adequate for mine output, which appears likely. Price spikes could occur if China were again to become a major importer of molybdenum concentrates because of world prices being lower than domestic production costs.

Based on its analysis, Roskill projects molybdenum metal prices for the period 2013-2025 as shown in Figure 19.2.



**Figure 19.2: Forecast (Real) Molybdenum Prices – 2013 to 2025**

The Sisson Project Feasibility Study economic analysis utilizes average long-term prices of \$15/lb molybdenum based on Roskill’s average base case price to 2025.

### Molybdenum Offsite Charges

Prices for unroasted molybdenum concentrate vary depending on the grade and impurities present. Typical toll treatment charges for unroasted molybdenum concentrates are around US\$ 1.25/lb, but with penalty elements, Roskill notes that this could rise to above US\$ 2.5/lb.

The Sisson Project is expected to produce an average molybdenum concentrate grade of at least 50% which is considered a minimum concentrate grade before any significant discounts would arise. At this grade, and given the expected quality of the molybdenum concentrate, molybdenum treatment and refining charges (TCs/RCs), plus other offsite charges (including freight, insurance, discounts due to grade and impurities, etc.) are estimated to be 12% of the Mo value in concentrate, or \$1.80/lb. These terms have been used as the average realization charges over the life of the mine for molybdenum in the economic model, equivalent to \$0.79/t milled.

#### **19.1.3 Net Smelter Return**

Considering the molybdenum and tungsten deductions, the average net smelter return (NSR) value for the Sisson life of mine is estimated to be \$26.24/t milled. The average NSR value over the first five years of production is estimated to be \$30.75/t milled.

#### **19.2 Contracts**

To the authors knowledge, at this time Northcliff has not entered into any contracts or arrangements that apply to mining, concentration, smelting, refining, transportation, handling, sales and, hedging and forward sales that apply specifically to the Sisson Project.

## 20.0 Environmental Studies, Permitting and Social or Community Impact

### 20.1 Environmental Studies

In 2008, Geodex initiated water quality, climate, and hydrology studies for the Sisson Project, and early biophysical field programs were also initiated. Northcliff has continued these programs and, in 2011, initiated additional studies of air quality, acoustics, surface and groundwater resources, environmental geochemistry, terrestrial and aquatic habitats, fish and wildlife, wetlands, land and resource uses, heritage resources, socio-economics, and traditional Aboriginal land uses. The purpose of these studies is to provide information for use in project planning and design and in preparing the Environmental Impact Assessment (EIA) Report. While the EIA requires a comprehensive study of baseline conditions and assessment of project effects, particular attention is being paid to aquatic resources to understand potential project effects on fish and fish habitat and to develop the habitat compensation plan required to obtain project authorization under the federal Fisheries Act. As well, comprehensive water quality monitoring and modeling provides the basis for project design to meet what are expected to be stringent water quality discharge standards and receiving environment quality criteria, and for effective long-term monitoring to meet regulatory requirements. There are no known environmental issues that are not being addressed through project planning and design that could materially affect implementing the Sisson Project.

### 20.2 Waste and Water Management

Waste materials are defined here as wastes produced by mining and ore processing (waste rock/barren rock and tailings) but also include mid-grade ore, pit walls exposed by mining and borrow materials used for construction purposes (quarry rock). These materials have been characterized for metal leaching/acid rock drainage (ML/ARD) potential by a number of analytical laboratory and field techniques including:

- Geological assessment for ML/ARD potential;
- Acid-base accounting (ABA) tests;
- Element and mineralogy composition analyses;
- Laboratory water leaching tests (humidity cells and water extraction tests); and
- Field leaching tests.

Based on results obtained to date (SRK 2012a; SRK 2012b), waste at Sisson is generally characterized as follows:

- **Waste/Barren Rock:** Potentially acid generating (PAG) with metal leaching potential; a delay to onset of ARD of several decades is expected. It should be noted that only 70% of waste rock was actually identified as PAG but that segregation during mining did not appear practical and therefore all waste rock was classified as PAG.
- **Pit Walls:** PAG with ML potential; a delay to onset of ARD of several decades is expected.
- **Mid-Grade Ore:** PAG with metal leaching potential; a delay to onset of ARD of one decade is expected.
- **Quarry material:** non-PAG, low ML potential.
- **Tungsten tailings:** non-PAG, low ML potential



- **Molybdenum tailings:** PAG, ML potential; onset to ARD is expected to be rapid (less than one year).

Waste management strategies have been developed to effectively mitigate the potential for long term ML/ARD impact on water quality from the Sisson Project. The Tailings Storage Facility (TSF) will handle all tailings and waste/barren rock, including the subaqueous disposal of PAG materials. Mine-contact water will be collected and delivered to the TSF for use and re-use in the process plant. During operations, any surplus water from the TSF and open pit will be discharged to the environment, treated if necessary to meet permit conditions to be set by New Brunswick and the requirements of the federal Metal Mining Effluent Regulations. Similarly, post-closure, any necessary treatment of surplus water discharged to the environment will be continued until water quality meets discharge standards. Comprehensive monitoring of discharged water as well as the receiving environment will be carried out in accordance with regulatory requirements and to assist in project decision-making throughout operations.

### **20.3 Environmental Assessment**

Mining projects in New Brunswick must undergo EIA review and approval by both the federal and provincial governments. Guidelines for the provincial EIA were issued in March 2009. In April 2011, the Canadian Environmental Assessment Agency accepted the Sisson Project Description to launch the federal EIA process. The Sisson Project is undergoing a harmonized federal/provincial EIA review. Draft Terms of Reference (TOR) for the EIA were defined by a joint federal/provincial Technical Review Committee (TRC), and were released for review and comment in August 2011. The TOR were finalized in April 2012, and Northcliff is completing an EIA Report to meet the TOR requirements. Completion of the Draft EIA Report, ready for submission to the TRC, is scheduled for 2013. The Draft EIA Report will then be reviewed by the TRC, finalized with Northcliff, and released for review and comment by the public, First Nations and stakeholder groups. Subsequent provincial and federal EIA approval processes differ somewhat, but are expected to result in EIA decisions in 2014.

### **20.4 Project Permitting**

As noted above, the Canada and New Brunswick project review and permitting process has started with the preparation of an EIA Report. A comprehensive permitting plan is being developed so that the individual permits and authorizations required for the Sisson Project will be obtained in a timely fashion to allow the start of construction shortly after the EIA decisions are made. An approved reclamation and closure plan and a financial security held by the province for the associated costs are required before approval to construct the Sisson Project is granted.

### **20.5 Social or Community Requirements**

Since early 2011, Northcliff has conducted numerous discussions, presentations, workshops and open houses with interested individuals, local communities, First Nations, and stakeholder groups to disseminate Sisson Project information and to learn about and help address specific concerns. In addition, information sharing is facilitated through the Sisson Project website; a community office in the town of Stanley, near the Sisson Project site; and newsletters and project updates by mail and email. The information gained from these consultation activities has helped shape the scope of the EIA and the factors to be considered in the environmental effects analyses. These activities will continue throughout project permitting and into

construction and operations. Northcliff funded an Aboriginal Traditional Use Study carried out by First Nations communities, and is consulting First Nations about the EIA and potential project benefits. A study of the economic benefits of the Sisson Project has been completed along with the Feasibility Study.

## **20.6 Closure and Reclamation**

Following the completion of mining, decommissioning, reclamation and closure of the Sisson Project will be undertaken to establish self-sustaining physical, chemical and biological stability of the site, and to meet desired end land uses, all as required under provincial and federal legislation and regulations. All facilities, buildings and other infrastructures will be removed and the sites reclaimed except for those that will be used for ongoing care and maintenance of the site. The water management system will be reconfigured as needed to ensure the long-term stability of the site. The TSF embankments and beaches will be capped and re-vegetated, and a spillway will direct runoff to the open pit. Once the pit fills, water will be treated if necessary to meet regulatory requirements and discharged to Sisson Brook.

A financial security is required by the province to ensure acceptable closure of the Sisson Project. The amount of the required security will grow over the life of the Sisson Project to an estimated value of \$50 million at closure in Year 27 in today's dollars. The estimated closure costs are based on a conceptual level closure and reclamation plan developed for the Feasibility Study. The closure concept is based on a number of assumptions pertaining to water treatment requirements, soil replacement and revegetation details, and desired land end-uses.

## 21.0 Capital and Operating Costs

### 21.1 Capital Costs

The capital cost is based on the Feasibility Study and addresses the engineering, procurement, construction and start-up of the Sisson Project, which consists of an open-pit tungsten-molybdenum mine, a concentrator capable of processing 30,000 tpd, an ammonium paratungstate (APT) plant and, associated ancillary facilities.

Northcliff engaged SE and other consultants to provide estimates of cost for portions of the project that fall within their specialized scope of work. SE has received input from the Consultants and compiled an estimate that represents the total project.

The breakdown of responsibility for the Sisson Project capital cost is as noted below:

Mine & Mining	Moose Mountain Technical Services
Haul Roads	Moose Mountain Technical Services
Process Facilities	Bolu Consulting Engineering & Samuel Engineering
Ancillary Facilities	Samuel Engineering
Utilities	Bolu Consulting Engineering & Samuel Engineering
Power line	New Brunswick Power Transmission Corp.
Access Roads	Samuel Engineering
Plant Site Roads	Samuel Engineering
Environmental Facilities (including TSF)	Knight Piésold Ltd.
Water Treatment	Bolu Consulting Engineering & Knight Piésold Ltd.
Indirect Cost	All
Owner's Cost	Northcliff & Samuel Engineering
Contingency	All

The estimate is based on the following data:

- Mine capital expenditure model
- Metallurgical and mass balance
- Process flow diagrams
- Process design criteria
- Site/plot plans
- General arrangement drawings
- Concentrator mechanical equipment list (FS Issue Rev 4, dated January 2, 2013)
- APT plant mechanical equipment list (FS Issue Rev 4a dated December 21, 2012)
- Electrical equipment list
- Budgetary equipment quotations from vendors
- Budgetary pre-engineered building quotations from vendors

- Earthwork, concrete, steel, platework and electrical MTOs
- In-house historical data and database information

The capital cost estimate was developed to a level that was sufficient to assess/evaluate the Sisson Project concept and the project overall viability. After inclusion of the recommended contingency, the capital cost estimate is considered to have an accuracy of minus 5% plus 15% with a base date of third-quarter 2012. The total estimated cost to design, procure, construct and start-up the facilities described in this report is \$579 million in Canadian currency. Tables 21.1 and 21.2 summarize the capital cost.

Table 21.1 Capital Cost Summary (000s)						
Description	Mine (incl. SME Facility)	Concentrator & APT Plant	TSF & Environmental	Infrastructure (Anc. Facilities, Power Line, Access Roads)	Owner's Cost	Total
<b>Direct Cost</b>						
Earthwork	\$ 8,275	\$ 6,254	\$ 14,070	\$ 3,541		\$ 32,140
Buildings	\$ 1,210	\$ 34,817		\$ 6,943		\$ 42,970
Concrete	\$ -	\$ 16,466		\$ 3,720		\$ 20,185
Steel	\$ -	\$ 21,171		\$ 205		\$ 21,375
Equipment	\$ 24,547	\$ 126,445	\$ 11,246	\$ 926		\$ 163,163
Piping	\$ -	\$ 18,364		\$ 221		\$ 18,586
Electrical	\$ -	\$ 14,467		\$ 14,171		\$ 28,638
Instrumentation	\$ -	\$ 9,965		\$ 405		\$ 10,371
<b>Subtotal Direct Cost</b>	<b>\$ 34,032</b>	<b>\$ 247,948</b>	<b>\$ 25,316</b>	<b>\$ 30,133</b>	<b>\$ -</b>	<b>\$ 337,428</b>
Contractor Indirects	\$ 1,246	\$ 52,075	\$ 2,025	\$ 4,866		\$ 60,212
Contracted Indirects	\$ -	\$ 42,605	\$ 759	\$ 4,069		\$ 47,433
Spares	\$ 683	\$ 1,489				\$ 2,172
Initial Fills	\$ -	\$ 5,012				\$ 5,012
Owner's Cost	\$ -	\$ -			\$ 35,985	\$ 35,985
<b>Subtotal Indirect Cost</b>	<b>\$ 1,929</b>	<b>\$ 101,181</b>	<b>\$ 2,785</b>	<b>\$ 8,934</b>	<b>\$ 35,985</b>	<b>\$ 150,814</b>
Freight	\$ 729	\$ 14,197		\$ 787		\$ 15,713
Taxes (Excluded)	\$ -	\$ -				\$ -
Import Duties	\$ -	\$ 1,904				\$ 1,904
Contingency	\$ 2,425	\$ 54,478	\$ 3,406	\$ 6,285	\$ 6,388	\$ 72,982
<b>Total Cost</b>	<b>\$ 39,116</b>	<b>\$ 419,708</b>	<b>\$ 31,506</b>	<b>\$ 46,139</b>	<b>\$ 42,372</b>	<b>\$ 578,841</b>

<b>Table 21.2 Capital Cost Summary</b>	
<b>Cost Area</b>	<b>\$M</b>
Mine	34.1
Concentrator & APT Plant	247.9
Site Infrastructure & Ancillary	55.4
Owner's Costs & Indirects	168.4
Contingency (approximately 15%)	73.0
<b>Total</b>	<b>578.8</b>

An exchange rate of \$1US = 1 Canadian dollar has been used for the capital cost estimate given the prevailing spot exchange rate at the date of this report and the shorter term outlook for the exchange rate for the period when the capital costs are expected to be incurred. As a result, the estimate can therefore be expressed in either third-quarter 2012 Canadian or United States dollars.

The estimate includes a contingency allowance of approximately 15% or \$73 million. Contingency is an allowance to cover unforeseeable costs that may arise during the project execution, but which cannot be explicitly defined or described at the time of the estimate due to lack of information. It is assumed that contingency will be spent; however, it does not cover scope changes or project exclusions.

Sustaining capital represents capital expenses for additional construction cost and equipment purchases that will be necessary during the operational years of the project, and are not included in the regular annual operating costs. Examples include replacement of mining equipment and vehicles, additional raises of the tailings dam after the initial construction, and additional mine haul roads. Life of mine sustaining capital is estimated to be \$196 million.

No provision has been included in the capital cost to offset future escalation.

### **21.1.1 Exclusions**

Items not included in the capital estimate are:

- Sunk costs and costs prior to the start of basic engineering phase
- Allowance for special incentives (schedule, safety, etc.)
- Escalation beyond Q3 2012
- Working capital
- Sustaining capital
- Interest and financing cost
- Taxes
- Reclamation Bonding
- Force majeure occurrences, such as risks due to political upheaval, government policy changes, labor disputes, permitting delays, etc.

## 21.1.2 Estimating Methodology

### Direct Cost

The capital cost estimate has been built up by cost centers as defined by the Sisson Project's Work Breakdown Structure (WBS) and by prime commodity accounts which include mining, earthwork, concrete, structural steel, process building, mechanical equipment (including platework), piping, electrical, instrumentation and tailings and environmental facilities. The estimate includes the pre-production period (initial capital).

Costs are based on the assumption that equipment and materials will be purchased on a competitive basis and installation contracts will be awarded in defined packages for lump sum or unit rate contracts.

### Contracted Indirect Costs

Common distributable and contracted indirect costs apply to multiple parties (suppliers, contractors, service providers, etc.) across multiple areas of the Sisson Project. These costs have been calculated using percentages based on contractor input or historical data of similar type projects.

### Contractor Indirects

Contractor indirects include, but are not limited to, overhead staff and support facilities; bonding; insurance; construction permits; contract administration; schedule management; management of subcontractors; onsite busing; surveying; mobilization and demobilization; construction equipment and small tools; supervision; safety; temporary power, toilets and communication; warehousing; cleanup and waste removal; construction vehicles, fuel and maintenance.

Process facilities contractor indirect costs are included at an overall rate of 48% of the direct field labor cost. All bulk materials are assumed to be purchased by the construction contractors except for structural steel and mechanical platework, which will be purchased by the EPCM services contractor. A markup of 8% on contractor furnished materials has been used to cover contractor overhead and profit.

Mining indirect costs have been provided by MMTS and include costs for mining fleet erection, equipment first fills, and the hiring and training of senior managers, technical staff, foreman, and senior operators.

Environmental facilities contractor indirects have been provided by KP. They are estimated to be 8% of the direct costs, based on past experience with similar work and recent estimates from contractors.

The cost for construction equipment has been calculated using contractor supplied rental rates and applying those rates over the construction period of equipment use.

Construction labor will reside in the surrounding communities so no onsite housing will be required. However, a contractor bussing allowance has been allocated.



### Spare Parts and Initial Fills

An allowance has been made for spare parts required for start-up and commissioning of the Sisson Project. Where available, vendor supplied quotations for suggested spare parts have been used. Where unavailable, a percentage of the equipment value has been assigned.

One-year spares have been calculated but are included in the working capital portion of the financial model; insurance spares are not included.

Initial fills (or first fills) include grinding media for the mills, flocculant and other reagents, water treatment chemicals and lubricants. Costs for all reagents and grinding media, plus fuel oil for the APT plant have been provided by Bolu, and include freight.

### EPCM Services

EPCM services for basic and detailed engineering design, procurement, and construction management of the processing and ancillary facilities have been included at 14% of the direct cost. Environmental facilities engineering costs have been provided by KP at 3% of the direct costs. These percentages are based on past experience with similar work.

Mining engineering and mass earthwork engineering will be provided by the Owner and are therefore not included in the EPCM cost.

### Pre-Operational Testing

Pre-operational testing costs, including both labor and expenses, are based on estimated manpower needed for engineering supervision, craft labor support, and vendor representative support. This cost is for pre-operations checkout only; the Owner operations staff will perform the actual start-up and commissioning of the plant so these costs are not included as contracted indirects.

An allowance has been made for utility power requirements during pre-commissioning and the preparation of facilities operations and maintenance manuals, procedures, and training materials.

### Owner's Costs

Owner's costs have been developed jointly by Northcliff and SE, and include:

- Preproduction Employment for G&A, Mine and Mill Personnel
- Owner's Project Management Team
- Light Vehicles for Owner's Team
- Mobile Equipment
- Communications Infrastructure and Expense
- Medical, Security & Safety Infrastructure
- Security/Medical Services
- Environmental Monitoring
- Land Acquisition, ROW, Easements, Etc.

- Community Development
- Project Insurances
- Legal, Permits & Fees

Mobile equipment for the plant and off-site facilities, road maintenance, etc. are included in Owner's cost. Corporate services (including travel and expenses) are excluded.

### Freight

Transportation costs have been included for delivery of equipment and materials to the jobsite. In general, it has been assumed that most equipment and bulk materials will be purchased in North America and can be trucked to site.

To the extent possible, freight has been applied to equipment packages based on information provided in quotations received from vendors. In cases where freight charges were not provided by vendors or equipment and materials were not quoted, ex-works locations or assumptions were made with regard to sourcing so that percentages could be applied for the various freight cost components.

### Contingency

Contingency has been included in the capital cost in recognition of the degree of detail on which the estimate is based. Contingency percentages have been assessed by considering possible ranges of cost uncertainties for various elements of the estimate. Each element is assigned a percentage rate based on the best judgment of the project team.

No external risk factors, such as escalation, currency fluctuations, weather, force majeure, etc., have been addressed in the contingency analysis. A project reserve provision should be made by the Owner to cover elements of risk outside of the defined variables.

### General Risk Factors

General factors that could have an impact on the Sisson Project could include, but are not limited to:

- Commodity price volatility;
- Global inflation;
- Shrunken industry suppliers;
- Shortage of skilled labor;
- Currency exchange rate swings;
- Project cancellations;
- Limited access to capital; and
- Escalation of local materials and labor

Given potential market volatility, cost saving opportunities may be available by remaining flexible on the Sisson Project schedule and execution plan.

Foreign exchange rate volatility could have an adverse impact on the value of labor and materials (including freight, duties and taxes). However, no additional funds have been allocated in the estimate to offset any currency fluctuations. Other potential risks could include government policy changes, labor disputes, permitting delays, weather delays or any other force majeure occurrences.

No allowance has been made in the capital cost for any of the potential risk items discussed.

Costs provided herein are dependent on the various underlying assumptions, inclusions and exclusions used in developing them. Actual cost may differ and could be significantly affected by factors such as changes in the external environment, the manner in which the Sisson Project is implemented, and other unknown factors that could impact the estimate's basis. Accuracy ranges are only projections based on cost estimating methods and are not a guarantee of actual project costs.

## 21.2 Operating Costs

The operating costs estimate includes the cost of mining, processing, waste management, and G&A services. The costs were estimated using information that was developed by the integrated feasibility study project team including MMTS, Bolu, KP and Northcliff.

The operating costs, which vary on an annual basis, were built up using organizational charts, estimated consumption rates for consumables, and unit prices for the commodities that are delivered to the site. The life-of-mine average operating cost for the Sisson Project is summarized in Table 21.3.

Table 21.3 Life of Mine Average Annual Operating Costs		
Average Annual Operating Costs	\$/t	\$/mtu
Mining (per tonne milled)	4.15*	77.7
Milling	7.11	133.2
Waste Management	0.59	11.1
G&A	0.47	8.8
<b>Total Operating Cost to Concentration</b>	<b>12.32</b>	<b>230.8</b>
Molybdenum By-Product Credits	(5.76)	(107.8)
<b>Total Operating Costs</b>	<b>6.56</b>	<b>123.0</b>
APT Costs (including offsite costs)	1.62	30.3
<b>Total APT Cash Costs</b>	<b>8.18</b>	<b>153.3</b>

\*Mining cost per tonne mined is \$2.09/t

### 21.2.1 Mining

Mine operating costs from MMTS are derived from supplier estimates for all major equipment operating parameters, and MMTS estimates for smaller support equipment. These parameters include fuel consumption, consumables, labor ratios, general excise tax (GET) costs, major component repair and replacement, as well as preventative maintenance and general parts costs.

Annual production tonnes, waste tonnes and, loading and hauling hours are calculated based on the capacities and parameters of the loading and hauling fleet. These tonnes and hours provide the basis for drilling, blasting, and support fleet inputs. Based on the tonnes scheduled, a requirement for production drilling hours is calculated based on hole size and pattern, bench height, material density and, penetration rate of the drill. Highwall and pre-shear drilling is included as an allowance for 2,000 m highwall length per year, and uses the same drill rig as specified for production drilling.

Based on quoted rates for explosives, an estimated pattern area and explosive density, the quantity of explosives is calculated, priced, and contractor labor and fees added. An estimate for initiation systems and blasting accessories is provided on a per hole basis. Drilling and blasting inputs (pattern area, powder factor, etc.) are estimated by MMTS.

Fleet requirements for loading, hauling and support are derived from the MSSP loading and hauling hours. To support the loading and hauling, the support fleet of dozers, front-end loaders, graders, service and welding trucks, etc., is added in to complement the required shovel and truck fleet.

All equipment cost is based on quoted fuel consumption rate, consumables cost, GET estimate, and general parts and preventative maintenance costs on a per-hour or per-hour interval basis. The quoted rates are then multiplied by the operating hours of the equipment to derive a constant distributed operating cost per hour.

Repair costs for major components of larger equipment are calculated separately from the distributed hourly cost. Major repairs are clocked with the usage of the piece of equipment so that major repair costs are represented correctly in the year it occurs rather than averaging this cost over many years. This method gives a better representative cash flow. Equipment replacement is clocked in the same manner so that individual equipment unit cumulative operating hours are tracked up to a set limit and then a replacement is introduced, thereby resulting in sustaining capital costs in that year.

Labor manhour ratios are categorized for the different labor categories such as operators, mechanics, electricians, etc., and then assigned to each piece of equipment. The equipment manhour is then multiplied by the operating hours to determine the quantity of hourly personnel required for each position. The mine department is estimated to employ 141 people initially and up to 173 people at the peak of mine operations.

The General Mine Expense (GME) consists of costs for all salaried staff, a consumable and rental allowance, crane rentals, and software and fleet management systems' licensing and maintenance. It is essentially a category for mine operations overhead costs. This category is a fixed cost and therefore does not vary by production or fleet size.

Labor rates are constructed by using a labor rate sheet obtained from Bolu. Processing position rates from this rate sheet are translated to roughly equivalent mining position rates. For example, a Filter and Reagent Operator is an entry position roughly equivalent to a Haul Truck Driver, and Chief Metallurgist is roughly equivalent to the Chief Engineer. Bolu's estimate of 28% payroll burden to account for holidays, sick time, and benefits is used.

## 21.2.2 Processing

Summary of process operating costs are shown in Table 21.4 below. The costs are broken down by major cost center and are based on annual concentrator nominal throughput rate of 10.5M t/a. Annual operating costs by major cost center are further broken down by tonne of ROM processed and per metric tonne unit (mtu) of WO<sub>3</sub> produced in APT.

<b>Table 21.4 Concentrator &amp; APT Plants Combined Operating Cost Summary</b>			
<b>Cost Center</b>	<b>Annual Cost (C\$)</b>	<b>Unit Cost C\$ / t Processed</b>	<b>Unit Cost C\$ / mtu<sup>(1)</sup></b>
Labor	10,400,440	1.00	18.73
Power and Fuel	22,566,521	2.16	40.64
Chemicals & Reagents	36,463,561	3.50	65.67
Consumables	14,514,566	1.39	26.14
Maintenance Repairs & Supplies	4,853,392	0.47	8.74
Other	1,448,800	0.14	2.61
<b>Total Costs</b>	<b>90,247,280</b>	<b>8.66</b>	<b>162.5</b>

<sup>(1)</sup> WO<sub>3</sub> in APT Produced and Before Moly Credits

LOM average on-site operating costs for the concentrator and APT plants are estimated at \$90.2 M per year or C\$8.66/t of ROM ore processed. Cost per mtu WO<sub>3</sub> produced in the form of APT is estimated at C\$162.50. It should be noted that the above operating costs take into account the ramp up period in year 1 when lower than regularly predicted recoveries are achieved. Additionally, unit costs are substantially lower in earlier years when concentrator feed grades, and therefore annual WO<sub>3</sub> production, are higher than LOM average values.

Reagents and chemicals at \$3.50/t of ore processed constitute the largest segment of the overall process operating costs. This is followed by power and fuel oil at \$2.16/t of ROM ore processed. Comminution and other consumable supplies at \$1.39/t and labor cost at \$1.00/t are the other two major cost centers.

Process plant labor roster at Sisson includes the following:

- 21 supervisory personnel,
- 62 operators,
- 36 maintenance personnel, and
- 10 technical and analytical technicians, bringing the total labor roster to 129.

### Scope

Process operating cost estimates include costs related to crushing, grinding, and flotation to produce molybdenum and tungsten concentrates from the ROM ores delivered to the primary crushing dump hopper. Operating costs further include APT plant operations, reclaim water clarification, and reagents and chemicals preparation and handling. Molybdenum concentrates will be directly marketed to buyers

while the tungsten concentrates will be processed onsite into APT before marketing. The process plant has been designed to process 10.5M t/a of ROM ore while the APT plant has been designed to process all tungsten concentrate production peaks at 881k mtu of WO<sub>3</sub> which occurs in year 3 of the current production schedule (at an assumed tungsten concentrate grade of 30%). Operating availability of both plants will be 92%.

#### Basis of estimate

All costs are in constant fourth quarter 2012 Canadian funds and include the costs related to both concentrator and APT plant direct operations, labor, power, fuel, consumables, chemicals and reagents, and maintenance repairs and supplies.

Operating estimates were prepared using metallurgical tests results, process flowsheets, equipment and electrical load lists, plant layouts, and process design criteria. Where necessary, allowances and lump sum estimates were used.

Process plant labor operating schedule is based on two 12-hour shifts per day with two crews providing daily coverage on a 4 days on/4 days off work schedule. All personnel, where possible, will be sourced locally and be bused in from neighboring communities. Operations and maintenance personnel hourly rates are based on 40 h/week, 50 weeks per year plus 10% overtime. All personnel annual base rates, including supervisory personnel, are based on estimates provided by Northcliff. Loaded annual wage rates for all personnel include a 28% overhead to cover benefits such as workers' compensation, employment insurance, disabilities, pension plan, and statutory holidays.

Operating and maintenance supply costs on major items are based on budget quotes provided by suppliers. Reagent and chemical consumption rates are based on metallurgical test results to date. Crushing and grinding media consumption rates are based on industry standards and vendor recommendations where necessary. Cost of comminution media, such as for liners and grinding balls, are based on vendor quotes.

Maintenance repairs and supplies costs are based on factored equipment costs.

Process plant power consumption estimate is based on electrical load lists prepared by SE using major equipment list prepared for the study, and on load and utility factors estimated using in-house data and standards. Power cost estimate is provided by Northcliff and is based on a variable and increasing rate for the first five years as supplied by NB Power Authority.

Fuel oil consumption for the process plant is calculated from first principles and industry standards while the price is based on vendor quote as provided by Northcliff.



Table 21.5 provides a list of LOM major criteria used when developing the operating cost estimates:

Table 21.5 Major Criteria for Life of Mine Process Operating Costs		
Criteria	Quantity	Unit
ROM Mill Feed <sup>(1)</sup>	10,426	kt
ROM Concentrator Feed Grade	0.073	%WO <sub>3</sub>
WO <sub>3</sub> in ROM Concentrator Feed	760,148	mtu
Concentrate Grade	30	%WO <sub>3</sub>
WO <sub>3</sub> produced in APT	555,290	mtu

<sup>(1)</sup> Includes ramp up and short year at end of mine life

### 21.2.3 Waste and Water Management

The total estimated cost for Waste and Water Management per operating year is \$6.2 M or \$0.59/t milled. The breakdown of costs is shown in Table 21.6 below.

Table 21.6 Water and Waste Management Operating Cost Summary			
Cost Center	LOM Cost (\$M)	Unit Cost \$ / t Processed	Average Cost \$M / yr
Pump Power	14.6	0.05	0.5
Water Treatment	12.7	0.05	0.5
Maintenance and Replacement	29.8	0.10	1.1
Quarry (Mining) Cost <sup>(1)</sup>	110.0	0.39	4.1
<b>Total Water and Waste Management Costs</b>	<b>167.1</b>	<b>0.59</b>	<b>6.2</b>

<sup>(1)</sup> Mining cost per tonne of quarried material mined - \$1.54/t

Operating costs for waste and water management have been provided by KP (2012). These costs include pump power, pump and pipe maintenance and replacement, and water treatment.

Pump power usage was estimated based on dynamic pumping head requirements and operating time. Power cost was assumed at \$65/megawatt-hour as per Atlantica Centre for Energy and NB Power.

Pump and pipe maintenance and replacement expenditures are based on fixed allowances to cover one or more full replacement of pumps, pipes, valves and fittings, and pump stations (excluding civil works) during the mine life.

Water treatment cost estimates are based on water quality modeling predictions and scoping level cost estimates prepared by SRK.

In-plant water management costs and tailings reclaim water clarification plant operating costs are included in processing operating costs discussed above.

Mining costs for the mining and movement of quarried material for ongoing construction of the tailings and embankment are also included in Waste and Water Management costs and are estimated at \$1.54/t of quarried material mined, or \$0.39/t milled.

#### **21.2.4 General and Administrative (G&A)**

The G&A costs for Sisson include salaries and benefits for the following department staff:

- General Manager
- Environmental
- Information Technology
- Health, Safety and Security
- Finance

In addition to covering the salaries and benefits for the staffing of these departments, the G&A also includes property tax, site heating and lighting, access road maintenance, and community development costs. The total estimated cost for G&A per operating year is \$4.04 million or \$0.47/t milled. The total number of employees in G&A is estimated to be 37. This brings the total initial manpower estimate to 307 full time employees between the Mine, Processing and G&A departments.

## 22.0 Economic Analysis

### 22.1 Sisson Project Feasibility Results Summary

The Sisson Project's financial results are summarized in Table 22.1 and show a pre-tax net present value (NPV) of \$714 million at an 8% discount rate, an internal rate of return (IRR) of 20.4% and a 4.1-year payback on initial capital expenditures of \$579 million. On a post-tax basis, the Sisson Project has a \$418 million NPV, a 16.3% IRR and a 4.5-year payback on initial capital. All currency values are expressed in Canadian dollars unless otherwise noted.

Table 22.1 Financial Results Summary		
Financial Results	Pre-Tax	Post-Tax
Undiscounted Cash Flows (LOM)	\$2,730 M	\$1,838 M
Net Present Value (5%)	\$1,167 M	\$739 M
<b>Net Present Value (8%)</b>	<b>\$714 M</b>	<b>\$418 M</b>
Net Present Value (10%)	\$509 M	\$272 M
Internal Rate of Return (IRR)	20.4%	16.3%
Payback	4.1 years	4.5 years
Total Capital Costs	\$579 M	

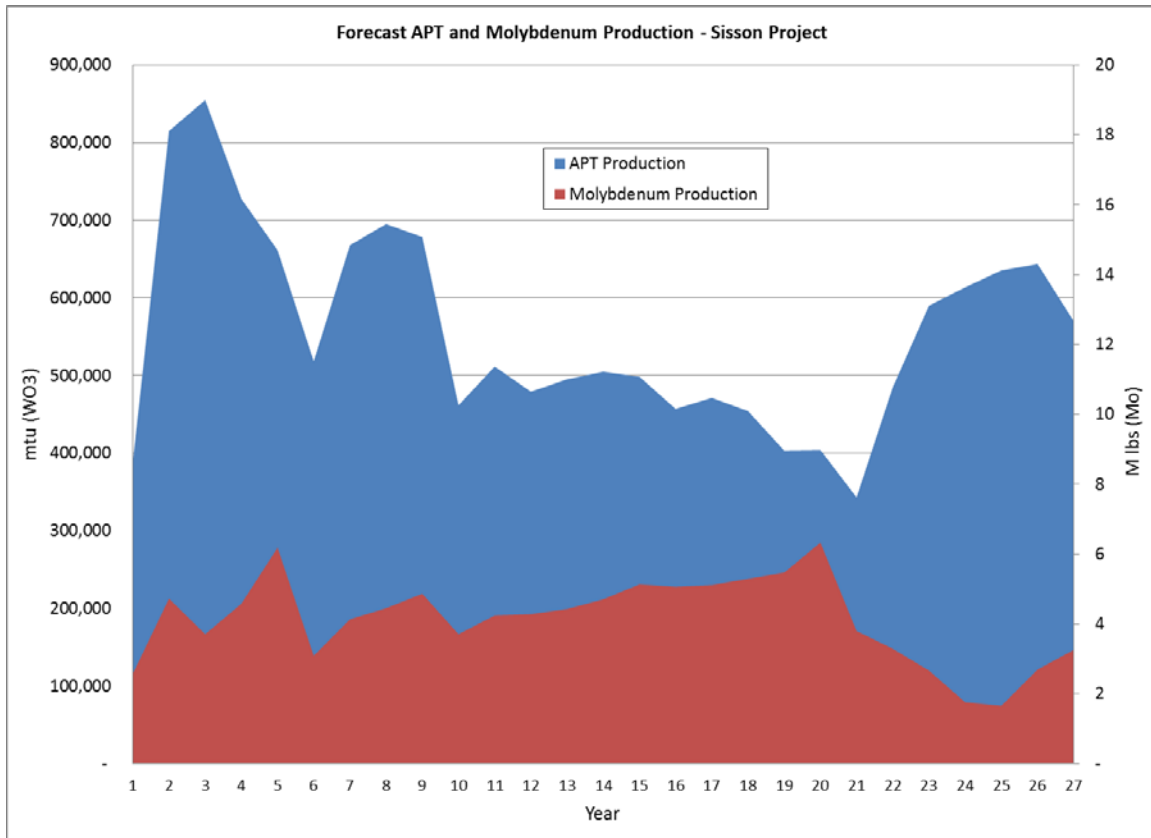
Project financial results are based on long term metal prices of US\$350/mtu for APT and US\$15/lb for Mo.

The Sisson Project's LOM production results are summarized in Table 22.2. The Sisson Project will produce an estimated annual average of 557,000 mtu WO<sub>3</sub> contained in APT and 4.1 million lb Mo contained in concentrates or a total of 15.0 million mtu WO<sub>3</sub> and 111.3 million lb Mo over 27 years of operation. Average annual production in the first five years is forecast at 689,000 mtu WO<sub>3</sub> and 4.4 million lb Mo.

Table 22.2 Production Results			
Production Results	Amount		
Milling Rate (tonnes/day) - nominal	30,000		
Strip Ratio	1 : 1		
Life of Mine	27 years		
	Annual Average	Life of Mine	Years 1 to 5
Tonnes Milled	10.5 M / a	281 M	10.2 M / a
Tungsten (WO <sub>3</sub> ) Production	557,000 mtu / a	15.0 M mtu	689,000 mtu / a
Mo Production	4.1 M lb / a	111.3 M lb	4.4 M lb / a

The Sisson Project’s LOM annual APT and molybdenum production are outlined in Figure 22.1. Peak production of 855,000 mtu WO<sub>3</sub> contained in APT occurs in the third year of operation. Peak production of 6.3 million lb Mo contained in concentrate occurs in the twentieth year of operation.

The economic analysis assumes no additional tungsten concentrate is processed by the APT plant from outside (non-Sisson mine) sources to utilize excess APT plant capacity after peak production in the third year.



**Figure 22.1: Forecast APT and Molybdenum Production**

## 22.2 Methodology Assumptions

### General

SE has prepared a discounted cash flow analysis of the Sisson Project. Technical and cost inputs for the economic model were developed by SE with specific inputs provided by Northcliff, KP, MMTS and Bolu. These inputs have been reviewed in detail by SE and are accepted as reasonable.

The discounted cash flow analysis was performed on a stand-alone project basis with annual cash flows discounted on a mid-year basis subject working capital timing adjustments. The Financial evaluation used a real discount rate of 8% and was performed at commencement of construction (denoted as Year -2 of the Sisson Project) using end of year 2012 Canadian dollars.

This economic analysis is a direct result of the capital cost estimate and is therefore considered to have the same level of accuracy (minus 5 percent and plus 15 percent).

Metal Prices and Exchange Rate assumptions

The Sisson Project’s key financial input assumptions are summarized in Table 22.3.

<b>Table 22.3 Price and Exchange Rate Assumptions</b>	
<b>Metal Prices</b>	
Tungsten (WO <sub>3</sub> ) - \$US / mtu	350
Molybdenum - \$US / lb	15
<b>US to Canadian \$ Exchange Rate</b>	
Capital Costs	1 : 1
Years 1 to 4	0.98:1 to 0.92:1
Years 5 +	0.90 to 1

The basis for the long term metal price assumptions is outlined in Section 19 – Market Studies and Contracts.

Given the Sisson Project’s location in Canada, operating and sustaining costs will be predominately denominated in Canadian dollars with revenues from APT and molybdenum being US dollar denominated. The economics of the Sisson Project will therefore be sensitive to US currency fluctuations relative to the Canadian dollar. As of the date of this report, the US to Canadian dollar exchange rate is close to parity. Historically, the US dollar has been significantly stronger than the Canadian dollar. Since 1980, the average exchange rate has been 0.80 US dollar to 1 Canadian dollar. Over the last 10 years, the average exchange rate has been 0.89 US dollar to 1 Canadian dollar. Capital Cost Estimates have been quoted in the Study based on an exchange rate of 1 US dollar to 1 Canadian dollar. The base case financial evaluation of the Sisson Project assumes a strengthening in the US dollar relative to the Canadian dollar over the first four years of production until an assumed long term US to Canadian dollar exchange rate of 0.90 is reached in the fifth year of production more in line with historical rates. The Sisson Project’s sensitivity to US to Canadian dollar exchange rate movements is further highlighted in the Sensitivity Section below.

### Offsite Charges

<b>Table 22.4 Offsite Charges</b>			
	<b>Rate</b>	<b>\$ / t milled</b>	<b>\$ / mtu</b>
Freight Costs	\$125 / t	0.07	1.3
Insurance, Representation, etc.	0.15%	0.03	0.5
<b>Total APT Offsite Charges</b>		<b>0.10</b>	<b>1.8</b>
Total Molybdenum Offsite Charges	12%	0.79	14.7

Offsite cost assumptions are discussed in Section 19 - Market Studies and Contracts.

### Grade and Recoveries

The Sisson Project's Grade and Recovery assumptions are outlined in Table 22.5. Head grades are estimated each year from the mine model. Concentrate recoveries for WO<sub>3</sub> are variable and are based on the mine plan and head grades. APT plant recoveries are constant and estimated at 97%. Metallurgical recoveries for molybdenum are assumed constant at 82%. Recoveries and grades are incorporated into the financial model, annually.

<b>Table 22.5 Grade and Recovery Assumptions</b>			
<b>Tungsten</b>	<b>Annual Average</b>	<b>Life of Mine</b>	<b>Years 1 to 5</b>
Avg. WO <sub>3</sub> Grade	0.073%	0.073%	0.093%
Avg. WO <sub>3</sub> Recovery	77%	77%	81%*
<b>Molybdenum</b>	<b>Annual Average</b>	<b>Life of Mine</b>	<b>Years 1 to 5</b>
Avg. Mo Grade	220 ppm	220 ppm	240 ppm
Avg. Mo Recovery	82%	82%	82%

\*After ramp-up of concentrator facility

### Working Capital

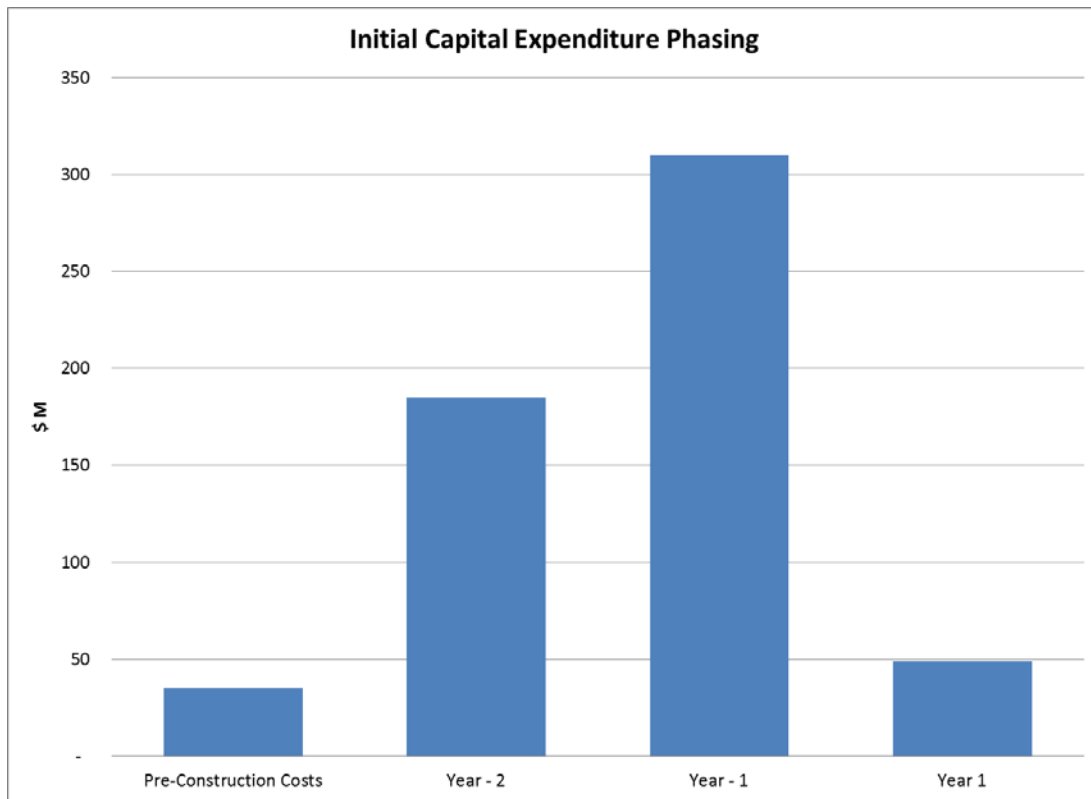
The financial model adjusts annual cash flow for working capital changes. A 60-day payment cycle is estimated from production to collection for APT and molybdenum concentrate sales. Credit terms on operating costs are assumed to be 30 days on average. Working capital requirements in Year 1 factor in concentrator and APT ramp-up assumptions.

### Initial Capital Costs

Total capital costs are estimated at \$579 million as outlined in the Capital and Operating Cost section. Most of the initial capital costs are incurred over a 24-month construction schedule. An estimated \$35



million will be required for lead order items and pre-construction EPCM costs. Initial capital cash outflows assumed in the financial analysis are shown in Figure 22.2 below.



**Figure 22.2: Initial Capital Expenditure Phasing**

Sustaining capital costs for mining equipment and waste management for the Sisson Project are summarized in Table 22.6.

<b>Table 22.6 Sustaining Capital Costs (\$M)</b>			
	<b>Life of Mine</b>	<b>Annual Average</b>	<b>Years 1 to 5</b>
Mining Equipment	127	4.7	39
Waste Management	69	2.6	20
<b>Total</b>	<b>196</b>	<b>7.3</b>	<b>59</b>

Routine maintenance costs and minor equipment replacement for the mill and APT plant are included in the operating cost estimates. No major refurbishment of either the mill or the APT plant is forecasted given the timeframe of the Sisson Project.

### Operating Costs

Total APT Cash Costs (after molybdenum by-product credits) are estimated at \$8.18/t, which equates to \$2,294 million over the LOM. Table 22.7 gives a summary of the operating cost assumptions of the Sisson Project.

<b>Table 22.7 Operating Costs</b>				
<b>Average Annual Operating Costs</b>	<b>\$/t</b>	<b>\$/mtu</b>	<b>LOM (\$M)</b>	<b>Annual Average (\$M)</b>
Mining (per tonne milled)	4.15*	77.7	1,168	43.3
Milling	7.11	133.2	2,001	74.1
Waste Management	0.59	11.1	167	6.2
G&A	0.47	8.8	132	4.9
<b>Total Operating Cost to Concentration</b>	<b>12.32</b>	<b>230.8</b>	<b>3,468</b>	<b>128.4</b>
Moly By-Product Credits	(5.76)	(107.8)	(1,621)	(60)
<b>Total Operating Costs</b>	<b>6.56</b>	<b>123.0</b>	<b>1,847</b>	<b>68.4</b>
APT Costs (including offsite costs)	1.62	30.3	447	16.6
<b>Total APT Cash Costs</b>	<b>8.18</b>	<b>153.3</b>	<b>2,294</b>	<b>85.0</b>

\*Mining cost per tonne mined is \$2.09/t

### Reclamation and Closure Costs

The estimated closure costs and the associated bond contributions have been included in the financial assessment of the Sisson Project. No salvage value has been included in the closure estimate.

### Ramp-Up Assumptions

In Year 1 of production, the concentrator is assumed to reach full throughput capacity and targeted recoveries after 6 months. The APT plant is forecast to reach full throughput capacity and recoveries in 9 months.

### Net Smelter Return (NSR)

Total Net Smelter Return (NSR) including Molybdenum is estimated at \$26.24/t, which equates to \$7,386 million over the LOM. Table 22.8 gives a summary of the NSR assumptions of the Sisson Project.

<b>Table 22.8 Net Smelter Return (NSR)</b>				
	<b>\$/t</b>	<b>\$/mtu</b>	<b>LOM (\$M)</b>	<b>Annual Average (\$M)</b>
APT	20.48	383.6	5,765	213.5
Molybdenum	5.76	107.8	1,621	60.0
<b>Total NSR</b>	<b>26.24</b>	<b>491.4</b>	<b>7,386</b>	<b>273.5</b>

## 22.3 Cash Flow Summary

The estimated annual LOM cash flow for the Sisson Project is summarized in Table 22.9.

Table 22.9 Cash Flow Summary										
Description	Unit	Total	Year - 2	Year - 1	Year 1 to 5	Year 6 to 10	Year 11 to 15	Year 16 to 20	Year 21 to 25	Year 26 to 27
Tonnes Milled	000s tonnes	281,489	-	-	51,220	52,560	52,500	52,500	52,550	20,160
Strip Ratio		1.02	-	-	1.08	1.05	1.07	0.77	1.23	0.64
APT Production	000s mtu	15,029	-	-	3,447	3,022	2,489	2,190	2,667	1,214
Molybdenum Production	M lbs	111.3	-	-	21.8	20.3	22.8	27.3	13.2	6.0
Revenue	\$000s	7,634,132	-	-	1,638,435	1,512,706	1,348,214	1,306,463	1,256,968	571,345
Realization Charges	\$000s	248,496	-	-	48,276	46,059	50,174	58,581	31,265	14,141
Net Smelter Return	\$000s	7,385,636	-	-	1,590,159	1,466,647	1,298,040	1,247,882	1,225,703	557,204
NSR / tonne milled	\$/tonne	26.24	-	-	30.75	27.90	24.72	23.77	23.32	27.65
Operating Cost	\$000s	3,897,045	-	-	699,872	732,824	738,849	735,663	727,327	262,510
Operating Cost / tonne milled	\$/tonne	13.84	-	-	13.66	13.94	14.07	14.01	13.84	13.02
Operating Profit	\$000s	3,488,591	-	-	890,287	733,823	559,191	512,219	498,376	294,695
Initial Capital	\$000s	578,841	220,000	310,000	48,841	-	-	-	-	-
Sustaining Capital	\$000s	195,835	-	-	58,909	37,177	33,068	41,017	23,266	2,397
Post Tax Cash Flow	\$000s	1,837,644	-225,613	-329,749	620,859	504,893	372,500	339,259	321,854	233,641

## 22.4 Taxes and Custom Duties

### 22.4.1 Income and Mining Taxes

Mining operations in New Brunswick are subject to three tiers of taxes: a federal income tax under the Income Tax Act (Canada), a provincial income tax under the New Brunswick Income Tax Act and a provincial mining tax under the New Brunswick Metallic Minerals Act (MMA). The following is a summary of the significant taxes applicable by the Sisson Project.

#### Federal Income Tax

Federal income tax is levied on the mining operations' taxable income (generally being net of operating expenses, depreciation on capital asset and the deduction of exploration and pre-production development costs). The current federal income tax rate in Canada is 15%.

#### Provincial Income Tax

A New Brunswick provincial income tax is based on a similar taxable income as the federal calculation of taxable income. The current provincial income tax rate in New Brunswick is 10%.

#### NB Provincial Mining Tax

The Province of New Brunswick levies a two-tier mining tax: a 2% royalty based on "net revenue" and a 16% levy on "net profits".

The 2% royalty comes into effect two years after a new mine commences. The royalty is based on 2% of the net revenue generated by the mining operation, which is generally equal to the revenue generated from the sale of mine output less allowable transportation and processing costs (including refining and smelting costs). A processing allowance of 8% for milling or concentrating assets plus a 15% processing allowance for smelting or refining assets (such as Sisson’s APT plant) can be deducted from net revenue. The total deduction cannot exceed 25% of the net revenue before the processing allowance has been deducted.

The 16% tax is imposed on annual net profits in excess of \$100,000. The net profit is calculated as the mine’s gross revenue less allowable costs, eligible exploration expenditures and specified allowances for depreciation, financing, and processing. The 2% royalty paid is also deductible in determining net profits.

New Brunswick mining taxes paid under the MMA are deductible for federal and provincial income tax purposes.

Income Taxes Summary

Table 22.10 summarizes the direct income and mining taxes payable by the Sisson Project at base case assumptions.

<b>Table 22.10 Income Tax Summary (\$M)</b>			
	<b>LOM</b>	<b>Annual Average</b>	<b>Peak Year</b>
Federal Income Taxes	367	14	18
Provincial Income Taxes	245	9	18
NB Provincial Mining Taxes	280	10	27
<b>Total Income and Mining Taxes*</b>	<b>893</b>	<b>33</b>	<b>63</b>

\*Excludes any indirect or induced taxes generated by the Sisson Project. These would include income taxes paid by employees, contractors, etc.

**22.4.2 Harmonized Sales Tax (HST)**

The Harmonized Sales Tax (HST) is a consumption tax in Canada used in provinces, New Brunswick included, where both the federal Goods and Services Tax (GST) and the regional Provincial Sales Tax (PST) have been combined into a single value added sales tax. The combined HST in New Brunswick is 13%, or approximately \$69M on the costs in the capital cost estimate. However, since HST or portions thereof can be either deferred or reclaimed, SE has excluded HST from the capital cost estimate.

**22.4.3 Custom Duties**

Customs duties on imported goods are included in the estimate. However, much of the Sisson Project equipment and materials will be available in Canada and would therefore either have no payable duty, or the duty would have already been paid upon import by the seller.

In addition, many of the non-Canadian items will come from the United States, falling under the Most Favoured Nation category of the customs tariff which would have a low duty or be exempt. However,

there will be some expensive specialty equipment items (most notably mills) that will originate from multiple continents. A 3% allowance has been made to address the cost of import duties in the estimate based on origination assumptions and the CBSA 2012 Customs Tariff.

#### **22.4.4 Tax Model Review by PricewaterhouseCoopers LLP**

SE engaged the services of PricewaterhouseCoopers LLP (PwC), an Ontario limited liability partnership, to assist with the review of the tax component of the economic analysis (economic model) conducted for the Sisson Project.

The scope of PwC's review was limited to the following tax components of the economic model (the "Model"):

- Federal corporate income taxes;
- New Brunswick provincial income taxes; and
- New Brunswick metallic mineral taxes

PwC's objective of the analysis was to assist in SE's process of confirming that:

- The Model was constructed appropriately, in so far as the arithmetical accuracy and internal consistency was concerned;
- The key tax assumptions in the Model were materially consistent with existing Canadian and New Brunswick income and mineral tax legislation.

PricewaterhouseCoopers LLP (January, 2013) in its letter to SE on its findings notes, "on the basis of the procedures carried out on your behalf in reviewing the Model, nothing has come to our attention to suggest that the objective of our work described above has not been achieved with regard to the final version of the Model."

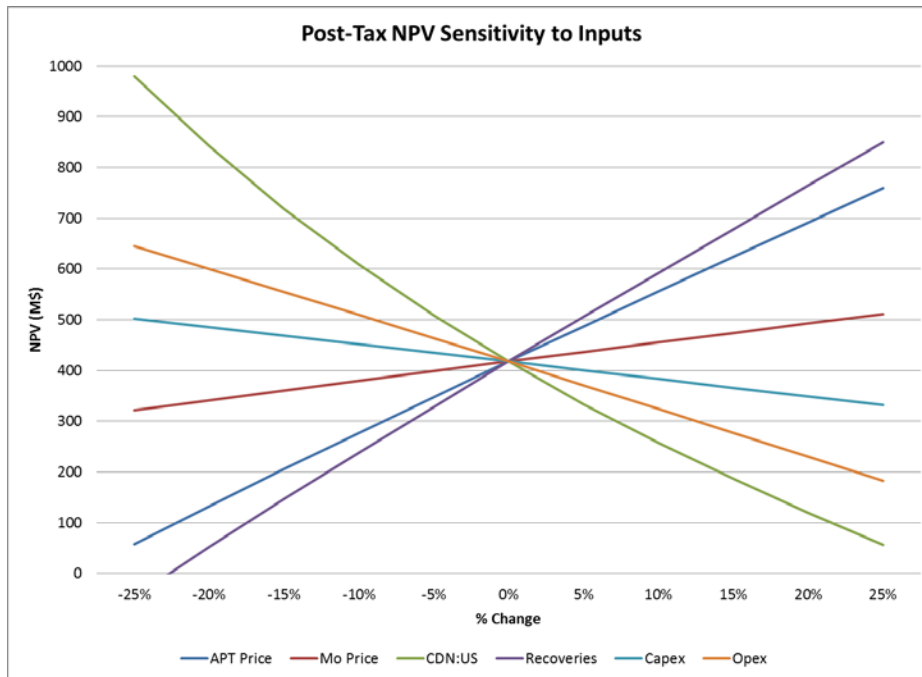
#### **22.5 Sensitivities**

Sisson Project sensitivity analysis was conducted to the following key variables:

- Tungsten Prices
- Molybdenum Prices
- US : Canadian dollar Exchange Rate
- Metallurgical Recoveries
- Operating costs, and
- Capital costs

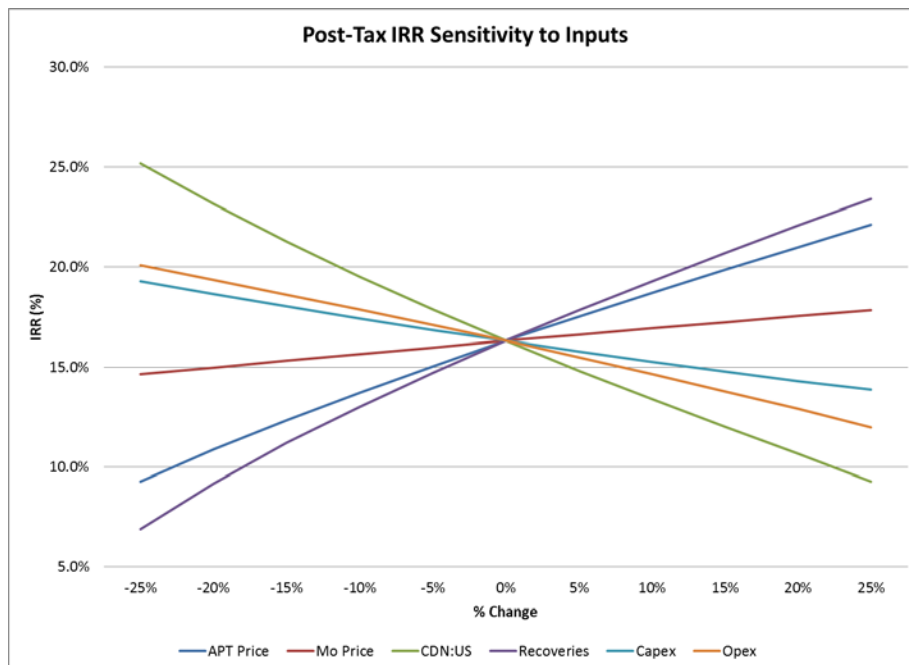
The results of the sensitivity analysis for the key variables on both Post-Tax NPV and Post-Tax IRR are shown in Figure 22.3 and Figure 22.4. The sensitivity analysis shows that the Sisson Project is most sensitive to metallurgical recoveries followed by APT prices and the US : Canadian dollar exchange rate. The Sisson Project financial results are least sensitive to capital costs on a percentage change basis.

**22.5.1 Post-Tax NPV Sensitivity Analysis**



**Figure 22.3: Post-Tax NPV Sensitivity Analysis**

**22.5.2 Post-Tax IRR Sensitivity Analysis**



**Figure 22.4: Post-Tax IRR Sensitivity Analysis**



### 22.5.3 Post-Tax IRR Sensitivity Table – Metal Prices

The results of metal price sensitivity analysis for APT (US\$/mtu) and molybdenum (US\$/lb) on the Sisson Project's Post-tax IRR are summarized in Table 22.11.

Table 22.11 Post-Tax IRR Sensitivity – Metal Prices												
Tungsten Price												
Molybdenum Price		250	275	300	325	350	375	400	425	450	475	500
	12.0	6.0%	8.7%	11.0%	13.1%	15.0%	16.8%	18.5%	20.2%	21.8%	23.3%	24.8%
	12.5	6.3%	9.0%	11.3%	13.3%	15.2%	17.0%	18.7%	20.4%	22.0%	23.5%	25.0%
	13.0	6.7%	9.3%	11.5%	13.5%	15.4%	17.2%	18.9%	20.6%	22.2%	23.7%	25.1%
	13.5	7.0%	9.6%	11.8%	13.8%	15.7%	17.4%	19.1%	20.8%	22.3%	23.8%	25.3%
	14.0	7.4%	9.9%	12.0%	14.0%	15.9%	17.6%	19.3%	20.9%	22.5%	24.0%	25.5%
	14.5	7.7%	10.2%	12.3%	14.2%	16.1%	17.8%	19.5%	21.1%	22.7%	24.2%	25.6%
	15.0	8.0%	10.4%	12.5%	14.5%	16.3%	18.0%	19.7%	21.3%	22.9%	24.4%	25.8%
	15.5	8.4%	10.7%	12.8%	14.7%	16.5%	18.2%	19.9%	21.5%	23.0%	24.5%	26.0%
	16.0	8.7%	11.0%	13.0%	14.9%	16.7%	18.4%	20.1%	21.7%	23.2%	24.7%	26.1%
	16.5	9.0%	11.2%	13.3%	15.1%	16.9%	18.6%	20.3%	21.9%	23.4%	24.9%	26.3%
	17.0	9.3%	11.5%	13.5%	15.4%	17.1%	18.8%	20.5%	22.0%	23.6%	25.0%	26.5%
17.5	9.6%	11.8%	13.7%	15.6%	17.3%	19.0%	20.6%	22.2%	23.7%	25.2%	26.6%	
18.0	9.9%	12.0%	14.0%	15.8%	17.5%	19.2%	20.8%	22.4%	23.9%	25.4%	26.8%	

### 22.5.4 Post-Tax NPV Sensitivity Table – Metal Prices

The results of metal price sensitivity analysis for APT and molybdenum on the Sisson Project's Post-tax NPV<sub>8</sub> are summarized in Table 22.12.

Table 22.12 Post-Tax NPV Sensitivity – Metal Prices												
Tungsten Price												
Molybdenum Price		250	275	300	325	350	375	400	425	450	475	500
	12.0	- 84	30	137	240	341	441	540	638	734	830	926
	12.5	- 70	43	150	253	354	454	552	650	747	842	938
	13.0	- 55	57	163	265	367	466	565	662	759	854	950
	13.5	- 41	71	177	278	379	479	577	674	771	867	962
	14.0	- 26	85	190	291	392	491	590	687	783	879	974
	14.5	- 12	98	203	304	405	504	602	699	795	891	986
	15.0	2	112	216	317	418	516	614	711	807	903	998
	15.5	17	125	229	330	430	529	626	723	819	915	1,010
	16.0	31	138	242	343	443	541	639	736	831	927	1,022
	16.5	45	152	255	356	455	554	651	748	844	939	1,035
	17.0	59	165	268	368	468	566	663	760	856	951	1,047
17.5	72	178	281	381	480	578	676	772	868	963	1,059	
18.0	86	192	293	394	493	591	688	784	880	975	1,071	

## 23.0 Adjacent Properties

There are no disclosures of adjacent properties that are relevant to the resource and reserve estimates, or to the Feasibility Study discussed in this report.

## 24.0 Other Relevant Data and Information

### 24.1 Plan of Execution

A comprehensive plan for development and implementation of the Sisson Project has been conceptualized. The overall project management will be accomplished by an integrated team of Owner project management, and the management group of an Engineering, Procurement, and Construction Management (EPCM) company.

Engineering will be executed in North America by a multi-disciplined EPCM services contractor. The project engineering and procurement will be developed in two distinct phases, a basic engineering phase and detailed engineering phase.

#### Basic Engineering Phase

First, an “at risk” basic engineering phase spanning about 11 months (May 2013 through March 2014) will be implemented. Its intent will be to keep the project advancing in support of the schedule until permitting and full project funding are realized. This phase will also serve to maintain technical continuity with the work done in the feasibility study and will focus on outstanding process issues, general plant layouts, and obtaining enough vendor data to support the detail design to the point where construction contracts can be awarded as soon as permitting allows.

Emphasis will be placed on completing the following specific activities:

- Any remaining concentrator and APT plant metallurgical testwork (an estimated 7-month duration)
- Process flowsheets and mass balance based on new metallurgical data
- Design basis (criteria) for all disciplines
- Major equipment specifications and data sheets
- Purchase orders for critical, long lead time equipment
- Civil designs to allow start of timber clearing and other construction activities immediately upon receipt of permits
- General arrangement layout drawings
- Detailed project execution plans
- Design and sourcing of elements required for construction mobilization (roads/bridges, temporary power, water, sewage, fuel depot, etc.)
- Constructability reviews

Since timber clearing cannot be undertaken from May 1 to August 31 due to bird migration and nesting, it is imperative that clearing begins as soon as permitting allows. Missing this window of opportunity could delay the project by up to four months.

In order to perform the earthwork bench designs, the general plant layouts and arrangements of ancillary facilities will also need to be completed. Vendor data will need to be purchased for some of the larger pieces of equipment in order to complete the layouts, as well as to have them on-hand for the start of detailed engineering immediately following EIA approval. To obtain the required vendor data,

specifications and requests for quotes (RFQs) will likely be prepared and issued for the following long lead equipment:

- Gyratory and cone crushers
- Concentrator HPGR and ball mills
- Flotation cells
- APT Regrind mill
- Autoclave
- Crystallizer
- APT Filters

Purchase orders will be made as necessary to obtain vendor data. But, it is possible that only equipment designs will be purchased in order to delay commitments and expenditures as long as possible, or at least until after the EIA is approved.

#### Detailed Engineering Phase

The basic engineering phase will be followed by a detailed engineering phase which will begin once project permitting and full financing are in place. This phase is expected to last approximately 18 months. As detailed designs and quantity take-offs are completed, they will be transferred to the procurement and contracts groups for buyout. Procurement of equipment and contracting of discrete construction packages will be undertaken according to detailed definitions of the work to be completed. Upon completion of the construction, the project will move to training, commissioning, startup and performance testing prior to the time the plant becomes fully operational.

## **24.2 Project Schedule**

The schedule developed for the Sisson Project is a Level 2, critical-path-method (CPM) schedule using Primavera software to evaluate critical paths to completion associated with the development of the Sisson Project. In addition to CPM analysis, the schedule has been used to complement the review and evaluation of the capital cost estimate and cash-flow expenditures for design, procurement and construction activities.

The schedule shows summary level detail of project development including engineering, major procurement, construction, and start up and commissioning. Schedule development involved identification of key activities, their durations, and proper Gantt logic ties required to determine the critical path, finish date and to assess float. Activities considered low-risk or insignificant have been purposefully excluded from the schedule. Input was provided by New Brunswick Power for the power transmission line, KP for the TSF, and MMTS for mining activities.

The overall project duration from start of detailed engineering to mechanical completion is expected to be approximately 30 months and construction is estimated at 24 months. After mechanical completion, it is anticipated that an additional 3 months will be required for the concentrator to reach full production capacity and an additional 10 months will be required for the APT plant to reach its full capacity.

### Critical Path

The project schedule critical path runs through the EIA approval, construction permitting, detailed design, general construction contract package award, concrete foundations, and construction installations at the process facilities. Other areas with limited total float include the power line, mine and the TSF.

### Permitting

Since the EIA terms of reference were jointly developed by the governments of Canada and New Brunswick, the Sisson Project EIA Report is expected to fully meet the legal requirements of both federal and provincial jurisdictions. The time requirements of the various steps in the harmonized federal/provincial EIA review process indicate that the process will be completed in 19 months from the date of submission of the EIA Report. The timeline starts with the submittal of the EIA Report and ends when the federal Minister of the Environment makes its EIA decision (assumed August 2014).

Following EIA approval, various permits will be needed to begin construction. Permit preparations and related agency discussions will take place during the EIA review process, and a duration of two months following EIA approval has been allowed in the schedule to finalize the initial permits. Since no construction or site disturbance can take place until after permit approvals, the permitting process will remain on the schedule critical path unless the at risk commitments are not made, at which point, the overall schedule duration will need to be extended.

## 25.0 Interpretation and Conclusions

The Feasibility Study confirmed the viability of Sisson as a long-life mining operation. The Sisson Project hosts a Mineral Reserve of 334 Mt at average grades of \$24.14/t (NSR), 0.066% (WO<sub>3</sub>) and 0.021% (Mo) at an \$8.83/t cut-off grade. Financially speaking, the Sisson Project is robust with a post-tax NPV of \$418M and a 16.3% IRR. Technically, the Sisson Project presents no fatal flaws. It is recommended that the Sisson Project proceed to the next level of engineering to enhance the design to pre-construction levels. Below is a summary of specific interpretations and conclusions to advance the Sisson Project from contributing QPs.

RPA has drawn the following conclusions from its audit conducted of the Mineral Resource estimate for the Sisson Project:

- For most of the history of exploration work on the Sisson Project the drill core handling, logging, and sampling protocols have been properly recorded and carried out in a manner consistent with industry best practice.
- Similarly, the assay methods and external QA/QC protocols have been properly documented. Assay methodology has changed somewhat over the history of the Sisson Project, but the various operators have employed conventional protocols, conducted by certified independent commercial laboratories.
- Reasonable and appropriate analytical QA/QC protocols have been observed for all programs since Geodex, and later Northcliff, assumed control, and the QA/QC results have been acted upon in a timely and acceptable fashion.
- Collar and downhole surveys have been conducted using conventional and appropriate methods and equipment.
- The database has been compiled and maintained in a competent manner, using a reasonable level of verification and validation.
- The geology of the deposit, mineralization styles, and controls to mineralization are well-understood and have been applied in a reasonable fashion.
- In RPA's opinion, the assumptions, parameters, and methodology used in the Mineral Resource estimate are appropriate for the style of mineralization and proposed mining method.
- The block model and grade interpolations have been generated using generally appropriate assumptions and parameters.
- The NSR cut-off has been derived using reasonable assumptions.
- The Mineral Resource classification has been done in a manner consistent with the CIM definitions, and as such is consistent with NI 43-101.
- The Mineral Resource estimate shows virtually no change in content from the previous estimate, dated February 2012.

KP's interpretation and conclusions of the TSF and water management design are as follows:

- Additional geotechnical and hydrogeological investigations are required to support the next design phase.



- Best estimates have been made to assess the quantity and quality of surplus water at the project site given the available information.
- Scientific uncertainties that need to be refined in the next stage of design with respect to water treatment are:
  - Target water treatment criteria and plant capacity
  - Acceptable release periods for treated water
  - Duration of operation post-closure
  - Capital and operating expenses
  - Associated closure and reclamation bonding requirements.
- Potential opportunities may exist to decrease the project capital cost, operating costs, bonding requirements, and the associated contingency costs based on the water treatment requirements.

According to MMTS, the Feasibility Study provides a suitable basis for the Economic and Financial analysis of the Sisson Project. The positive results of the Financial Analysis in turn establish the reported tonnes and grades as reserves. Based on the positive economic results the Sisson Project should advance to detailed design and permitting phases.

The Mine plan for the Feasibility Study has the following features:

- The ultimate economic pit limits do not include inferred resources. As the operation progresses, the inferred material will be redefined as mill feed or waste through in-pit drilling several benches ahead. This will result in changes to the material destinations and subsequently redesign of the waste dump and tailings management.
- The detailed pit designs do not include inferred resources, which can readily be added to the lower benches of the Feasibility designs.
- The future mine plans will need to address treating some of the inferred material as mill feed.
- The equipment specifications and costs are suitable for the Feasibility Study. These need to be refined for operations planning. This will be done as part of commercial negotiations with vendors.
- The mine design is at a Feasibility Study level of accuracy. A higher level of design is needed for startup of operations, particularly in the first 6 to 18 months of mill feed.

MMTS concludes that converting the inferred resources within the mine design to ore or waste will affect the pit designs, production schedule, tailings, and waste dump requirements and will potentially positively affect the project economics. A higher level of mine planning detail is needed for internal corporate and operations planning. This mine planning work should incorporate up to date design parameters, including current metal prices, costs and process recoveries.

A higher level of mine planning design will be needed in the EIA and permitting process, including designating the Inferred material into the relevant process stream. The first 6 to 18 months of mill feed in the operating plan, also needs to be defined with close spaced drilling, and metallurgical testing as required.

Bolu's interpretation and conclusions of the process testing and plant design are as follows:

- There has been a substantial amount of metallurgical testing carried out on samples from the Sisson property over the last decade. Metallurgical testing to date covered many aspects of recovery of the metals of economic interest including pre-concentration, gravity separation and flotation.
- Final flowsheet was developed using samples representing areas of mineable areas and detailed testing at SGS Lakefield Research using sampling and assay techniques acceptable for the elements of interest.
- Recovery predictions were prepared from batch and locked cycle test data produced from the same metallurgical program.
- Process and plant design were prepared using criteria, processes, equipment and unit operations widely employed in their specific industries.
- Opportunities exist in improving molybdenum and tungsten concentrate grades and recoveries with further testwork.
- In order to reduce potential risks associated with the use of recycled water in the process, further testing will have to be conducted using aged and recycled treated process water.
- Continued testing on APT metallurgy should be expanded to include data generation for further optimization and detail design engineering.

## 26.0 Recommendations

The Qualified Persons recommend that the Sisson Project be advanced to the Basic and Detailed Engineering phases. It is further recommended that the following work be completed during the early stages of the Basic Engineering phase:

- Mine plan update, including operating level of detail for the startup and commissioning period, and inferred resources for internal planning and EIA permitting use.
- Northcliff is actively advancing the water quality technical aspects of the Sisson Project; however, more resolution is required on several technical aspects before commencing the Detailed Design phase of engineering.
- Comprehensive geotechnical and hydrogeological site investigation programs to support detailed design.
- A fully integrated pilot plant should be undertaken to provide supporting information for basic and detailed engineering of the concentrator. This would require a significant tonnage of ore which could be sampled from the existing test pit on site or from drill core.
- Locked cycle testing on variability composites including evaluation of lower grade composites to cover the full range of feed grades in the mine plan.
- Flotation testing to evaluate the impact of clarified re-cycle process water on metallurgy.
- Additional water clarification testwork using freshly generated and aged process water samples to develop detail design criteria particularly for major equipment such as clarifier, and settling pond sizing.
- Further APT testing to include the following:
  - Testing of feed samples at varying grades and quality to confirm process characteristics and applicability of the process developed to wider concentrate grade ranges and residence times.
  - Optimization of impurity removal steps and testing for varying impurity levels in synthetic sodium tungstate solutions.
  - Optimization of the conversion of sodium tungstate solution to APT.
- Continue to investigate the best disposal options for the APT wastes. While lined pond facilities have been assumed for the 'Feasibility Study', more work is needed to develop a specific reclamation strategy for the APT wastes. Test work to inform the final disposal and reclamation strategy is underway.
- Since the tax system is complex and subject to change, SE recommends that Northcliff engage a tax expert to review the Sisson Project at the earliest stage possible to identify the best methods of conducting business to minimize tax exposure.

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## 28.0 Date and Signature Page

The effective date of the technical report, "Canadian National Instrument 43-101 Technical Report on the Sisson Project, New Brunswick, Canada" is January 22, 2013.

*Original signature and seal on file*

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**Steven Pozder, P.E., Q.P. – Samuel Engineering, Inc.**

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**Daniel Friedman, P. Eng., Q.P. – Knight Piésold Ltd.**

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**David Rennie, P. Eng., Q.P. – RPA, Inc.**

## **29.0 Certificates of Qualified Persons**

The effective date of the technical report, “Canadian National Instrument 43-101 Technical Report on the Sisson Project, New Brunswick, Canada” is January 22, 2013.

**DAVID W. RENNIE, P. ENG.****Principal Geologist****RPA Inc.****CERTIFICATE OF QUALIFIED PERSON**

I, David W. Rennie, P. Eng., as an author of this report entitled "Canadian National Instrument 43-101 Technical Report on the Sisson Project, New Brunswick, Canada", prepared for Northcliff Resources Ltd., and dated January 22, 2013, do hereby certify that:

1. I am a Principal Geologist with RPA Inc. My office address is Suite 388, 1130 West Pender Street, Vancouver, British Columbia, Canada V6E 4A4.
2. I am a graduate of the University of British Columbia in 1979 with a Bachelor of Applied Science degree in Geological Engineering.
3. I am registered as a Professional Engineer in the Province of British Columbia (Reg.# 13572). I have worked as a geological engineer for a total of 34 years since my graduation. My relevant experience for the purpose of the Technical Report is:
  - Review and report as a consultant on numerous exploration and mining projects around the world for due diligence and regulatory requirements.
  - Consultant Geologist to a number of major international mining companies providing expertise in conventional and geostatistical resource estimation for properties in North and South Americas, and Africa.
  - Chief Geologist and Chief Engineer at a gold-silver mine in southern B.C.
  - Exploration geologist in charge of exploration work and claim staking with two mining companies in British Columbia.
4. 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 purposes of NI 43-101.
5. I visited the Sisson Project on February 3, 2012.
6. I am responsible for Section 6.0 through 12.0, inclusive, Section 14.0, and parts of Sections 1.0 and 25.0 of this report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I prepared a report on the Project entitled "Technical Report on the Sisson Project, New Brunswick, Canada", for Northcliff Resources Ltd., dated June 29, 2012.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, 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 this 8th day of March, 2013 at Vancouver, British Columbia.

*/Original signature and seal on file/*

**“David W. Rennie”**

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David W. Rennie, P. Eng.

**JAMES H. GRAY, P. ENG.**  
**Principal Mining Manager**  
**Moose Mountain Technical Services**

**CERTIFICATE OF QUALIFIED PERSON**

I, James H. Gray, of Calgary, Alberta, do hereby certify:

1. I am a Mining Engineer with Moose Mountain Technical Services with a business address at 1975 1st Avenue South, Cranbrook, BC, V1C 6Y3.
2. This certificate applies to the technical report entitled "Canadian National Instrument 43-101 Technical Report on the Sisson Project, New Brunswick, Canada", prepared for Northcliff Resources Ltd., and dated January 22, 2013 (the "Technical Report").
3. I am a graduate of the University of British Columbia, (Bachelor of Applied Science – Mineral Engineering, 1975).
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#11919), and the Association of Professional Engineers, Geologists, and Geophysicists of Alberta (Member #M47177).
5. My relevant experience includes operation, supervision, and engineering in North America, South America, Australia, Africa, Eastern Europe, and Greenland.
6. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
7. My most recent personal inspection of the Property was on November 7, 2011.
8. I am responsible for Sections 15.0 and 16.0 of the Technical Report as well as the Mining Engineering components of Sections 1.0, 18.0, 21.0, 25.0 and 26.0.
9. I am independent of Northcliff Resources Ltd. as defined by Section 1.5 of the Instrument.
10. I have had involvement with the property that is the subject of the Technical Report, acting as a Qualified Person for ongoing Mining Engineering since November 2010.
11. I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.
12. 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 this 8th day of March, 2013 at Calgary, AB.

*/Original signature and seal on file/*

**"James H. Gray"**

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James H. Gray, P. Eng.



**DANIEL FRIEDMAN, P. ENG.**

**Senior Engineer  
Knight Piésold Ltd.**

**CERTIFICATE OF QUALIFIED PERSON**

I, Daniel Friedman, of Vancouver, British Columbia, do hereby certify:

1. I am a Senior Engineer with Knight Piésold Ltd. with a business address at Suite 1400, 750 West Pender Street, Vancouver, British Columbia Canada V6C 2T8.
2. This certificate applies to the technical report entitled "Canadian National Instrument 43-101 Technical Report on the Sisson Project, New Brunswick, Canada", prepared for Northcliff Resources Ltd., and dated January 22, 2013 (the "Technical Report").
3. I am a graduate of McGill University, (B.Eng. - Civil, 2003). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License No. 32571. My relevant experience with respect to mine waste and water management includes over eight years of continuous work in the discipline. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
4. My most recent personal inspection of the Property was September 13, 2011.
5. I am responsible for Sections 18.12, 18.13 and 18.14, and parts of Sections 1.0, 5.2, 20.6, 21.0, 25.0 and 26.0 of the Technical Report.
6. I am independent of Northcliff Resources Ltd. as defined by Section 1.5 of the Instrument.
7. I have no prior involvement with the Property that is the subject of the Technical Report.
8. I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 8th day of March, 2013 at Vancouver, British Columbia.

*/Original signature and seal on file/*

**"Daniel Friedman"**

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Daniel Friedman, P. Eng.

**STEVEN A. POZDER, P.E., MBA**  
**Director of Engineering and Analysis**  
**Samuel Engineering, Inc.**

**CERTIFICATE OF QUALIFIED PERSON**

I, Steven A. Pozder, P.E. do hereby certify:

1. I am the Director of Engineering and Analysis – practicing at Samuel Engineering, Inc., 8450 East Crescent Parkway, Suite 200, Greenwood Village, CO 80111, U.S.A.
2. This certificate applies to the report entitled “Canadian National Instrument 43-101 Technical Report on the Sisson Project, New Brunswick, Canada”, prepared for Northcliff Resources Ltd., and dated January 22, 2013 (the “Technical Report”).
3. I am a graduate of the University of Denver with a B.S. in Mechanical Engineering in 1988. I am a graduate of the University of Denver with an M.B.A. in General Business in 1994.
4. I am registered as a Professional Engineer (P.E.) with the State of Colorado, Registration Number 29144.
5. I have practiced my profession as a Mechanical Engineer and Project Manager in mineral processing and mining for over 24 years. My relevant experience for the purpose of the Technical Report is:
  - I have worked as a consulting engineer on mining projects in roles such a mechanical engineer, project engineer, area manager, study manager, and project manager. Projects have included Scoping Studies, Prefeasibility Studies, Feasibility Studies, basic engineering, detailed engineering and startup and commissioning of new projects.
  - In engineering positions I have estimated and reviewed capital and operating costs and completed economic analyses including power requirements, reagent costs, labor requirements and costs, etc. for 17 years.
6. I have read the definition of “qualified person” set out in National Instrument (NI) 43-101, and do certify that, by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
7. I have not visited the Sisson Project Site.
8. I am responsible for the overall Technical Report preparation, Sections 2.0, 3.0, 4.0, 5.1, 5.3, 5.4, 18.1 to 18.11 (inclusive), 19.0, 20.1, 20.2, 20.3, 20.4, 20.5, 21.0, 22.0, 23.0, 24.0 and 27.0, and parts of Section 1.0, 5.2, 20.6, 25.0 and 26.0.
9. I am independent of the issuer, Northcliff Resources Ltd. applying the tests set out in section 1.4 of NI 43-101.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read the National Instrument 43-101 Standards of Disclosure for Mineral Projects and the Companion Policy 43-101CP, the “Instrument.” The sections of the technical report that I am responsible for have been prepared in compliance with the Instrument.

12. As of the date of this certificate, to my knowledge, information and belief, this Report contains all the scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 8th day of March 2013, at Greenwood Village, Colorado.

*/Original signature and seal on file/*

**“Stephen A. Pozder”**

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Steven A. Pozder, P.E.

**E. J. GRESKOVICH, PHD., P.E.**  
**Principal (Owner)**  
**E. J. Greskovich Consulting**

**CERTIFICATE OF QUALIFIED PERSON**

I, E. J. (Gene) Greskovich of State College, Pennsylvania, U.S.A. do hereby certify that:

1. I am the Principal (Owner) of E. J. Greskovich Consulting, with a business address at 1344 Deerfield Drive, State College, Penna 16803 (U.S.A.).
2. The title and date of the report to which this certificate applies is "Canadian National Instrument 43-101 Technical Report on the Sisson Project, New Brunswick, Canada", prepared for Northcliff Resources Ltd., and dated January 22, 2013 (the "Technical Report").
3. I hold BS, MS, and PhD degrees from The Pennsylvania State University in Chemical Engineering. I also am a registered Professional Engineer in Pennsylvania since 1969 and am/have been a member in good standing. I have practiced my profession for 47 years, holding positions in industry and academia.

Also, I have been a full-time Consultant for the past 21 years in my Field. I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, experience, affiliations with a professional association, and work within the Field, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4. I have not visited the Sisson property or have had any prior involvement that is the subject of this Technical Report.
5. I am responsible for the verification of the Sisson-APT Plant Design which includes parts of Section 1.0, 13.0, 17.0, 21.0, and 26.0.
6. I am independent of the issuer as described in section 1.5 in NI 43-101.
7. I have read the instrument and the Technical Report has been prepared in compliance with NI 43-101.
8. As of the date of this Certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated at Greenwood Village, on this 8th day of March 2013, in State College, Penna.

*/Original signature and seal on file/*

**"E. J. Greskovich"**

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E. J. Greskovich, PhD, P.E.  
Registered Professional Engineer  
State of Pennsylvania (U.S.A.)  
License No. PE 015128E; License status: Active

**H.M. MATT BOLU, P. ENG.**  
**Principal Process Engineer**  
**Bolu Consulting Engineering Inc.**

**CERTIFICATE OF QUALIFIED PERSON**

I, H.M. Matt Bolu, P. Eng. of Surrey, British Columbia do hereby certify that:

1. I am the Principal Process Engineer for Bolu Consulting Engineering Inc., with a business address at #310 – 304 West Cordova St., Vancouver BC, V6B 1E8
2. This certificate applies to the technical report entitled “Canadian National Instrument 43-101 Technical Report on the Sisson Project, New Brunswick, Canada”, prepared for Northcliff Resources Ltd., and dated January 22, 2013 (the “Technical Report”)
3. I graduated with a M.Sc. degree in Minerals Engineering from the University of Birmingham, England (1978)
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#27144).
5. I have practiced my profession continuously for 35+ years and have relevant experience in operations, testing, design and engineering of tungsten (in particular), base and precious metals as well as industrial minerals projects throughout in North America, Europe, South America and Asia.
6. I am a “Qualified Person” for the purposes of National Instrument 43-101 (the “Instrument”).
7. I have not visited the Sisson property but have had prior involvement that is the subject of this Technical Report.
8. I am responsible for Sections 13.0 and 17.0 of the Technical Report as well as the Process Engineering aspects of Sections 1.0, 18.0, 21.0, 25.0 and 26.0.
9. I am independent of Northcliff Resources Ltd. as defined by Section 1.5 of the Instrument.
10. I have read NI 43-101 and the technical report has been prepared in compliance with NI 43-101
11. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 8th day of March, 2013 at Vancouver, BC.

/Original signature and seal on file/

“H.M. Matt Bolu”

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H.M. Matt Bolu, P. Eng.

## 30.0 Illustrations

Illustrations are placed in the Report as they are referenced.