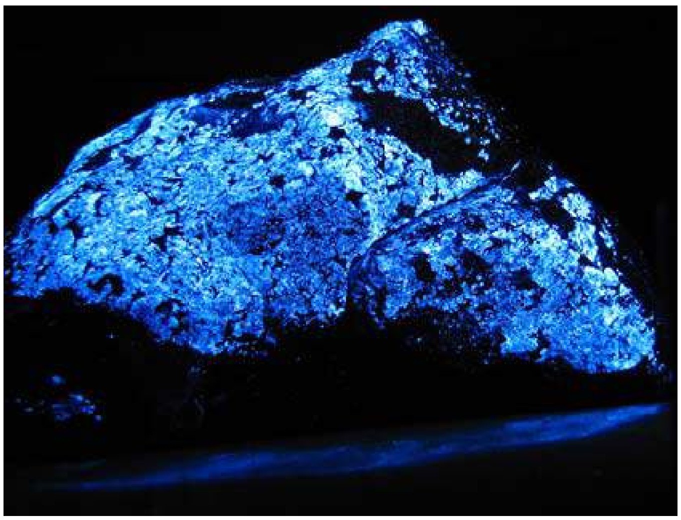
NORTH AMERICAN TUNGSTEN

CORPORATION LTD.

TECHNICAL REPORT ON THE CANTUNG MINE, NORTHWEST TERRITORIES, CANADA



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NORTH AMERICAN TUNGSTEN LTD.

Report for NI 43-101

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1 SUMMARY

The Cantung mine, owned and operated by the North American Tungsten Corporation Ltd. (NATCL), is a tungsten mine in the Northwest Territories, Canada. Cantung is one of the largest operating tungsten mines outside of China. The mine consists of both open pit and underground operations, which extract ore from a scheelite-chalcopyrite bearing skarn. The skarn deposit formed as a replacement to the Ore Limestone and Swiss Cheese Limestone units located on the property.

Production occurs from the seasonally operated open pit and the continuously operating underground mine. The mine currently produces ore at a rate of 1,350 dry short tons per day. Mineral Reserves support a mine life to at least 2017. The primary mining method is sub-level longhole stoping with delayed backfill. Currently longhole methods are planned for the majority of the remaining reserves with cut and fill being employed in areas where the ore zone is too narrow for efficient longhole mining.

Processing is carried out by gravity and flotation circuits. Final products include a premium gravity concentrate (G1), containing on average, 65% WO_3; a flotation concentrate containing, on average, 35% WO_3 and a copper concentrate averaging 28% Cu.

Exploration on the property continues with both in-mine and near-mine exploration programs underway. This has been successful in defining additional ore sources and adding mineral resources and reserves.

When production is taken into account, Mineral Reserve tonnages have increased since the previous NI 43-101 technical report in 2011. These increases are due to successful exploration and mineral reserve definition in new ore zones, combined with the inclusion of lower-grade areas rendered economic by higher tungsten prices. Mineral Reserves now support a mine life to at least 2017. These mineral resources and mineral reserves, calculated as of July 31, 2014, are listed below in Table 1.1, Table 1.2 and

Table 1.3.

Table 1.1 - Probable Mineral Reserves, July 31, 2014

Zone	Probable Mineral Reserve		
	Tons	Grade (WO_3%)	STU's
Amber	442,000	0.85	374,000
Below 3700	44,000	0.87	38,000
E Zone	55,000	0.99	55,000
Pit	190,000	0.77	147,000
Pug	886,000	0.80	707,000
West Extension	159,000	0.77	123,000
Stockpile	41,000	0.77	32,000
Grand Total	1,818,000	0.81	1,476,000

Notes:

1. Mineral Reserves conform to CIM and NI 43-101 requirements.
2. All Mineral Reserves are classified as Probable
3. Mineral Reserves are estimated at a cut-off grade of 0.5% WO_3 (tungsten trioxide).
4. A minimum mining width of 15 feet was used.
5. The Probable Reserve is a subset of the Indicated Mineral Resource.
6. Tons are short tons being 2,000 lbs and STU is Short Ton Unit 20 lbs of WO_3.
7. Numbers may not add up due to rounding.

Table 1.2 - Indicated Mineral Resources, July 31, 2014

Zone	Indicated Mineral Resource		
	Tons	Grade (WO_3%)	STU's
Amber	1,199,000	1.09	1,305,000
Below 3700	30,000	1.43	43,000
E Zone	319,000	1.04	331,000
Pit	192,000	0.85	163,000
Pug	1,851,000	0.86	1,587,000
West Extension	208,000	1.25	260,000
Stockpile	41,000	0.77	32,000
Grand Total	3,839,000	0.97	3,720,000

Notes:

1. Mineral Resources conform to CIM and NI 43-101 requirements.
2. Mineral Resources are estimated at a cut-off grade of 0.5% WO_3.
3. The Indicated Resource includes the Probable Mineral Reserves.
4. Numbers may not add up due to rounding.

Table 1.3 - Inferred Mineral Resources, July 31, 2014

Zone	Inferred Mineral Resource		
	Tons	Grade (WO_3%)	STU's
Amber	730,000	0.7	511,000
Below 3700	140,000	1	140,000
E Zone	120,000	0.9	108,000
Pit	-	-	-
Pug	60,000	0.8	48,000
West Extension	140,000	0.8	112,000
Dakota	170,000	0.8	136,000
Grand Total	1,370,000	0.8	1,096,000

Notes:

1. Mineral Resources conform to CIM and NI 43-101 requirements.
2. Mineral Resources are estimated at a cut-off grade of 0.5% WO_3.
3. Numbers may not add up due to rounding.

Mineral Resources where insufficient work has been completed to date to demonstrate economic viability have been excluded in determining the Mineral Reserves. Additional work may demonstrate economic viability for part of these Mineral Resources. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

In NATCL's opinion the Cantung Mine is positioned to receive the value of the current prices for tungsten and the continued demand for tungsten concentrates. The Mine has the advantages of extensive operating history and established contacts with suppliers and customers. The Mine has operated successfully in the past; however, it should be noted that Cantung has experienced numerous shutdowns during periods of low tungsten prices. In NATCL's opinion, the key risk to mine profitability lies in tungsten price sustainability, USD/CDN exchange rates, metallurgical recoveries, mine head grades over the remainder of the mine life, risks associated with mining and downturns in the World Economy.

It is expected that APT pricing will continue at current levels and may rise in the future, given the forecast for growing consumption and continued demand for tungsten metal and concentrates and increasing supply from existing and new producers as well as secondary recycling of scrap materials. The Company is of the opinion that a base case

G1 (premium gravity concentrate) price scenario of US$312 per MTU is reasonable for the remaining mine life while assuming an exchange rate of CDN$1.09 per US$1.00.

2 INTRODUCTION

This report has been prepared as a National Instrument 43-101 compliant technical report for North American Tungsten Corporation Ltd. (NATCL) by Brian Delaney, P.Eng and Finley J. Bakker, P.Geo

The technical report was prepared at the request of the North American Tungsten Corporation Ltd.'s Board of Directors to determine the current Mineral Resources and Mineral Reserves at their Cantung tungsten mine in the Northwest Territories, Canada. This report is prepared to satisfy the ongoing requirement for continuous disclosure. Information required to create the report was provided by NATCL personnel and, where appropriate, information was also obtained from external resources. These external consultant resources are noted below.

In the course of preparing the Technical Report, the following NATCL personnel provided relevant data as requested, as noted below:

Kurt Heikkila	Chairman of the Board of Directors and C.E.O.
Brian Abraham	Director and legal counsel
Bruce Penich	Senior Manager - Finance
Jason McKenzie	General Mine Manager
Steve Sherwood	Underground Mine Superintendent
Colin Craft	Mill Superintendent
Jenny Penner	Human Resources Superintendent
Deborah Flemming	Environmental Superintendent
Graham Fuslang	Project Manager
Brian Harvey	Maintenance Superintendent

In addition a number of consulting firms provided completed reports, designs, reviews and technical assistance regarding their particular project areas, as listed below:

EBA, TetraTech – tailings dam design, open pit geotechnical report, permitting assistance

Wenck Associates Inc. – tailings dam design, waste water treatment facility design

Veolia Water Solutions and Technology Canada Inc. – waste water treatment facility design, equipment and materials

Kovit Engineering – review of mine backfill plant

Aurora Geoscience – geophysical surveys and interpretation

Wild River Consulting Group, LLC – mill upgrades and assaying

ESG Solutions – underground seismic

Brian Delaney, P.Eng and Finley J. Bakker, P.Geo are both employed by NATCL at the Cantung mine site and spend considerable time on the property.

Units of measurement used in this report conform to the SI (metric) system, with the exception of mine development and production data, noted in Imperial units, as the mine was developed and currently operates using that system of measurement. All currency in this report is in Canadian dollars (C$ or CDN$) unless otherwise noted. Refer to Table 2.1 below.

Table 2.1 - List of Abbreviations Used

μ	micron	kVA	kilovolt-amperes
°C	degree Celsius	kW	Kilowatt
°F	degree Fahrenheit	kWh	kilowatt-hour
μg	microgram	LOMP	Life of mine plan
A	ampere	L	liter
A	annum	L/s	litres per second
bbl	barrels	m	metre
Btu	British thermal units	M	mega (million)
C$	Canadian dollars	m^2	square metre
cal	calorie	m^3	cubic metre
cfm	cubic metres per minute	min	minute
cm	centimeter	mASL	metres above sea level
cm^2	square centimeter	mm	millimetre
D	day	mph	miles per hour
dia.	Diameter	MTU	metric tonne unit
dmt	dry metric tonne	MVA	megavolt-amperes
dwt	dead-weight ton	MW	megawatt
Ft	foot	MWh	megawatt-hour
ft/s	foot per second	m^3/h	cubic metres per hour
ft^2	square foot	opt, oz./st	ounce per short ton
ft^3	cubic foot	oz.	Troy ounce (31.1035g)
G	gram	oz./dmt	ounce per dry metric tonne
G	giga (billion)	ppm	part per million
Gal	Imperial gallon	psia	pound per square inch absolute
g/L	gram per litre	psig	pound per square inch gauge
g/t	gram per tonne	RL	relative elevation
gpm	Imperial gallons per minute	s	second
Gr/ft^3	grain per cubic foot	St	short ton
Gr/m^3	grain per cubic metre	Stpa	short ton per year
Hr	hour	Stpd	short ton per day
Ha	hectare	STU	Short ton unit
Hp	horsepower	t	metric tonne
In	inch	tpa	metric tonne per year
In^2	square inch	tpd	metric tonne per day
J	joule	US$	United States dollar
K	kilo (thousand)	Usg	United States gallon
kcal	kilocalorie	Usgpm	US gallon per minute
Kg	kilogram	V	volt
km	kilometre	W	watt
km/h	kilometer/ per hour	wmt	wet metric tonne
km^2	square kilometre	yd^3	cubic yard
kPa	Kilopascal	yr	year

3 RELIANCE ON OTHER EXPERTS

Brian Abraham, Partner with Dentons Canada LLP was retained as legal counsel.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY SIZE

The Cantung mine property, comprising of all claims and leases, is 24,675.28 acres (9,985 hectares) in size.

4.2 LOCATION

The Cantung Mine is located in the Nahanni area of western Northwest Territories, Canada, approximately 300 km northeast of Watson Lake, Yukon, close to the Yukon border. Refer to Figure 4.1 below. Approximate coordinates of the property are 541,000 E, 6871000 N (WGS84).

Figure 4.1 - Satellite Image Showing Location of Cantung Mine

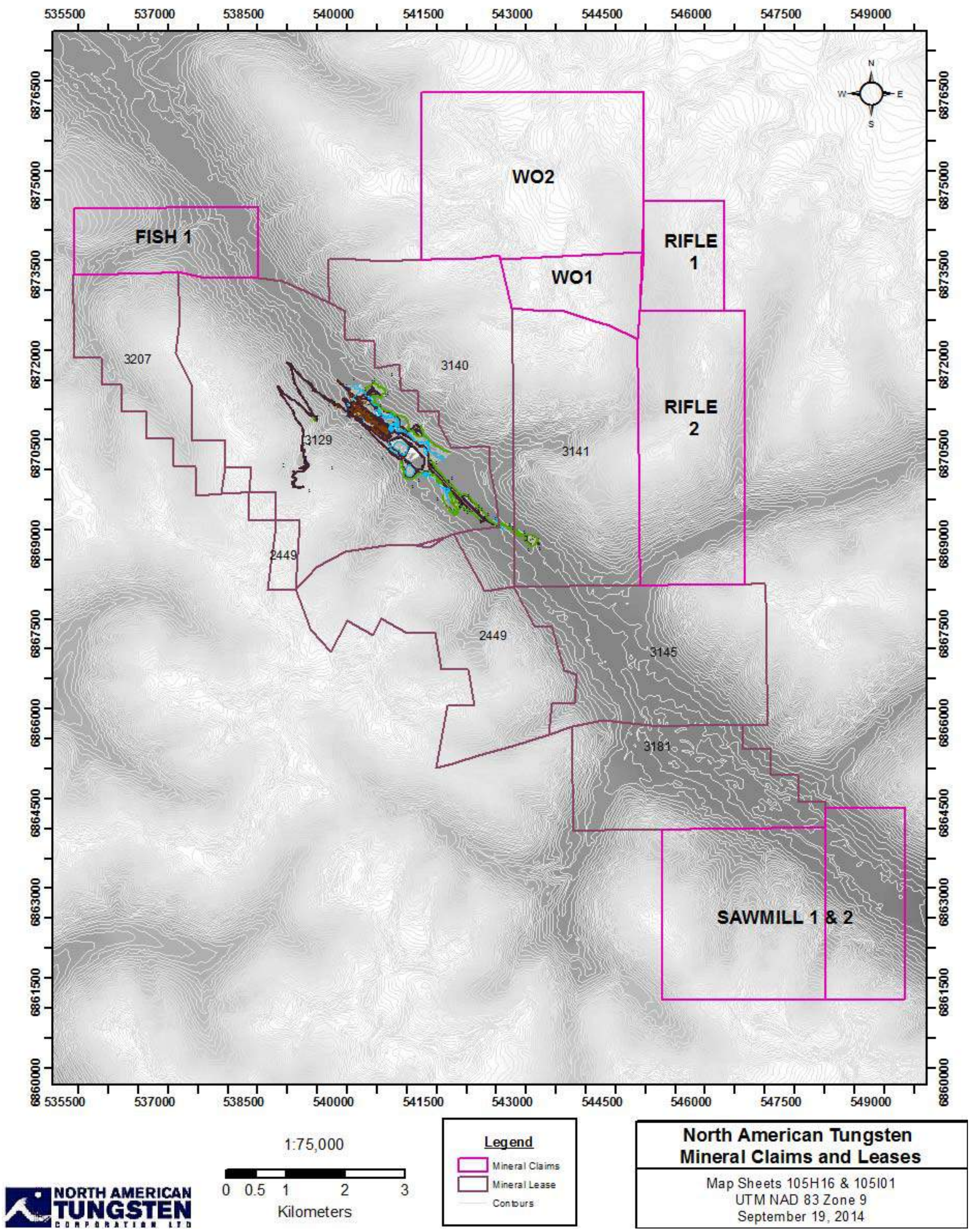
4.3 MINERAL TENURE

NATCL has leases from the Government of the Northwest Territories (GNWT) Department of Industry, Tourism and Investment covering the mine and associated service areas. The current legally surveyed leases and claims are listed and described in Tables 4.1 and 4.2 below and displayed in Figure 4.2.

Table 4.1 - Mineral Tenure Details

LOT	Claim/Lease #	Claim Name	CLAIM OWNER	%	Claim District	NTS 1	AREA (Acres)
CLAIM	F76256	WO 1	NATCL	100	NWT	105H16	531.00
CLAIM	F76257	WO 2	NATCL	100	NWT	105H16/105I01	2487.00
CLAIM	F76260	RIFLE 1	NATCL	100	NWT	105H16	619.80
CLAIM	F76261	RIFLE 2	NATCL	100	NWT	105H16	2066.00
CLAIM	K12453	SAWMILL 1	NATCL	100	NWT	105H16	1859.40
CLAIM	K12454	SAWMILL 2	NATCL	100	NWT	105H16	1084.65
CLAIM	K12455	FISH 1	NATCL	100	NWT	105H16	733.70
LEASE	NT-2449		NATCL	100	NWT	105H16	2258.73
LEASE	NT-3129		NATCL	100	NWT	105H16	3875.00
LEASE	NT-3140		NATCL	100	NWT	105H16	2044.00
LEASE	NT-3141		NATCL	100	NWT	105H16	2338.00
LEASE	NT-3145		NATCL	100	NWT	105H16	2105.00
LEASE	NT-3181		NATCL	100	NWT	105H16	1548.00
LEASE	NT-3207		NATCL	100	NWT	105H16	1125.00

Figure 4.2 - Mineral Title Location Map



4.4 PROPERTY TITLE

Property title details are listed below in Table 4.2.

Table 4.2 - Mineral Tenure Ownership

CLAIM NUMBER	CLAIM STATUS	DISTRICT	RECORD DATE	ANNIVERSARY DATE	AREA (Acres)	CLAIM NAME
F76256	ACTIVE	2	27/09/2004	27/09/2014	531.00	WO 1
F76257	ACTIVE	2	27/09/2004	27/09/2014	2487.00	WO 2
F76260	ACTIVE	2	01/08/2006	01/08/2016	619.80	RIFLE 1
F76261	ACTIVE	2	01/08/2006	01/08/2016	2066.00	RIFLE 2
K12453	ACTIVE	2	22/11/2010	22/11/2014	1859.40	SAWMILL 1
K12454	ACTIVE	2	22/11/2010	22/11/2014	1084.65	SAWMILL 2
K12455	ACTIVE	2	22/11/2010	22/11/2014	733.70	FISH 1
LEASE NUMBER	**LEASE STATUS**	**DISTRICT**	**ISSUED DATE**	**EXPIRES DATE**	**AREA (Acres)**	
NT-2449	ACTIVE	2		11/12/2015	2258.73	
NT-3129	ACTIVE	2		15/11/2015	3875.00	
NT-3140	ACTIVE	2		21/12/2015	2044.00	
NT-3141	ACTIVE	2		21/12/2015	2338.00	
NT-3145	ACTIVE	2		21/12/2015	2105.00	
NT-3181	ACTIVE	2		11/03/2015	1548.00	
NT-3207	ACTIVE	2		22/10/2015	1125.00	

The mineral tenure currently consists of 7 mineral claims totalling 3,796 Hectares (9,381 acres) and 7 mineral leases totalling 6,189 Hectares (15,293 acres). Expiry dates and anniversary dates are shown above in Table 4.2. At the time of the report both claims WO1 and WO2 are in good standing. However it is anticipated that North American Tungsten Corporation Ltd. will not complete the required assessment work and will let these two claim blocks lapse.

North American Tungsten Corporation Ltd. has a 100% interest in the property. Legal access and surface rights are fully granted and pose no issues for the property. To retain all the currently listed claims and leases the only requirement is to spend the money required to maintain the claims and leases.

4.5 ROYALTIES PAYABLE

Royalties of 1% of net revenue, payable to Teck Resources Ltd.

4.6 ENVIRONMENTAL LIABILITIES

A number of mine abandonment and reclamation plans have been prepared for the Cantung site. As required in the water license, the most recent plan for final abandonment and reclamation of the mine was submitted by NATCL to the McKenzie Valley Land and Water Board (MVLWB) in November 2007. In February 2008 the MVLWB informed NATCL that they could not approve the plan as submitted, and as required by the MVLWB, NATC has submitted updated Closure & Reclamation Plans (CRP) annually to the MVLWB. This was done on March 31 in years 2009, 2010, 2011 and 2012. Since 2011 the CRP's have included work plans that provide information and research requirements for the final closure plan.

Commencing in late 2012, the MVLWB changed the format of the updates to the CRP, temporarily suspending the annual updates. At the same time they initiated a collaborative step-wise process to be used to develop an objectives-based summary table that focuses on each mine component. Upon completion of the process the tables developed will be used as base data to produce the next update to the CRP. It is expected that the next CRP produced will be the basis from which the Detailed Interim Reclamation Plan will be developed and then the final CRP. The objectives component was approved in 2013 and the options component is pending approval. Wenck Associates Inc. prepared the updated reclamation cost estimate in October 2014. This cost estimate was $13.1 million.

Mine closure cost estimates were included in all reclamation submissions. Over the years a number of mine closure cost estimates have been prepared for NATCL, the MVLWB and Aboriginal Affairs and Northern Development Canada (AANDC) Indian and Northern Affairs Canada (INAC). As of April 1, 2014, GNWT has taken the role previously held by AANDC, due to devolution. The range of closure cost estimates over the years is shown in the Table 4.3 below. The MVLWB selected the reclamation security amount of $13.1 million for the 2009 license renewal based upon third party estimates up to June 2008. In August 2010, the MVLWB reduced the security deposit required to $11,657,839.

Table 4.3 - Reclamation Cost Estimates

RECLAMATION COST ESTIMATES North American Tungsten Corporation Ltd. Cantung Mine			
Prepared By	Date Prepared	for	Closure Cost Estimate ($M)
EBA	Nov 2001	NATCL	2.3
EBA	Oct 2002	NATCL	2.5 - 3.0
Brodie Consulting	Oct 2002	MVLWB	34.5
NATCL	Jul 2003	NATCL	1.9
MVLWB Security	Dec 2003	MVLWB	7.9
NATCL	Nov 2007	NATCL	3.8
Brodie Consulting	June 2008	INAC	13.1
MVLWB Security	Jan 2009	MVLWB	13.1
NATCL	Mar 2009	NATCL	4.2
NATCL	Mar 2010	NATCL	4.3
MVLWB Security	Aug 2010	MVLWB	11.7
NATCL	Mar 2011	NATCL	5.0
NATCL	Mar 2012	NATCL	7.5
NATCL/ Wenck	Oct 2012	NATCL	8.7
Wenck	Oct 2013	NATCL	8.9
Wenck	Oct 2014	NATCL	13.1

4.7 PERMITS

The Cantung operation is fully permitted to conduct all mining and exploration activities on the property.

4.8 SIGNIFICANT DEFINED RISKS AND FACTORS

The largest identified risks to the ability to perform work include the non-realization of tons or grades from the mining operations, or a significant decrease in the realised price of tungsten. Other risks include, but are not limited to, such things as unexpected capital expenditures; rising future production costs; temporary loss of road access to the mine site due to natural disruptions; variations in mill recovery rates, efficiency or throughput; failure of plant, equipment or processes to operate as anticipated; accidents; labour disputes and other risks associated with mining.

Tailings Pond 4 (TP4) was raised to the present elevation of 3,757 feet above sea level in 2011. Deposition to TP4 ceased in July 2013, at which time deposition was switched to TP5. TP5 has been in use as the tails water exfiltration pond since 2006. This pond was raised to elevation 3,760 feet in 2012. The TP5 Stage 2 raise in 2013 increased the elevation of the dam to 3,765 feet. The TP5 stage 3 raise is underway to bring the dam to the final elevation of 1151.2 meters (3,777 feet). The Stage 3 design will be the final raise on TP5. Capacity has been assessed with respect to the operations of the water treatment facility and the move to dry stack tailings (DST), which is currently under permit application. Tailings deposition commenced into TP5 in July 2013 and the water exfiltration rate has been decreasing significantly since deposition began. NATC has been permitted for and is presently operating a Waste Water Treatment Facility (WWTF) to allow the direct discharge of treated tails water to the environment. This commenced in July 2014.

During geotechnical studies for TP4 berm raise construction in 2011, a potential liquefiable zone was identified. This zone, under the conditions of a one in one thousand year seismic event, may potentially result in instability under a section of the TP4 berm that extends parallel to the Flat River. After considering a number of remediation options, NATC decided, based on dam safety, environmental and economic considerations, to decommission and reprocess the tailings in TP4. By dismantling TP4 and moving the reprocessed tailings to a new storage facilities located further back from the Flat River, the potential instability is mitigated and possible risk to the environment is removed.

Additional engineering and design work has been conducted for long-term tailings storage facilities that would provide up to an additional 10 years of storage capacity for reprocessed tails and the operations at Cantung Mine. The final engineering design and construction work requiring permits for the operation to achieve its life of mine plan will be the Dry Stack Tailings (DST) facility. DST applications are now being assessed for the move toward a more sustainable and environmentally acceptable method of tailings disposal. The MVLWB recently ruled that NATCL’s proposed DST facility will be exempt from an Environmental Assessment, as the work falls within the purview of NATCL’s current plan for tailings storage. NATCL will be requesting proposals for the engineering design, materials procurement and construction management of the DST facility.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 TOPOGRAPHY, ELEVATION AND VEGETATION

For the most part, the area around the Cantung property is a rugged mountain wilderness. The mine site lies in the Flat River Valley, within the Selwyn Mountain Range. Local terrain is characterized by steep mountains and narrow valley bottoms. Elevations vary from 1,130 m above sea level (ASL) at the bottom of the Flat River valley to elevations of 2,750 m ASL on nearby mountain peaks. There are a number of avalanche slide paths in the area, and avalanche monitoring and control is an ongoing requirement in the winter. The Flat River is a tributary of the South Nahanni River, which ultimately drains into the Liard and Mackenzie Rivers.

The ecoregion is characterized by alpine tundra at upper elevations and by subalpine open woodland vegetation at lower elevations. Alpine vegetation consists of crustose lichens, mountain avens, dwarf willow and ericaceous shrubs. Sedge and cottongrass are associated with wetter sites. Barren talus slopes are common. Subalpine vegetation consists of discontinuous open stands of stunted white spruce, and occasional alpine fir and lodgepole pine, in a matrix of willow, dwarf birch, and northern Labrador tea with a ground cover of moss and lichen.

Local alpine glaciers exist in the highest ranges of this ecoregion. Bare rock outcrops and rubble are common at higher elevation. Permafrost is extensive but discontinuous in the western part and continuous, with low ice content, in the eastern part of the ecoregion.

Characteristic wildlife includes caribou, grizzly and black bear, Dall's sheep, moose, beaver, fox, wolf, hare, raven, rock and willow ptarmigan, and bald and golden eagle.

5.2 PROPERTY ACCESS

Access to the property is from Watson Lake, Yukon via Highway 4 (Campbell Highway) and then along Highway 10 (Nahanni Range Road) for a total distance of 310 km. Travel by road between the mine site and Watson Lake takes four to five hours, depending on road conditions. The mine is responsible for maintaining the access road from the mine site to km 134 Nahanni Range Road. The Yukon government is responsible for maintaining the remainder of the road from km 134 to Watson Lake.

In addition to the road access, NATCL also maintains an airstrip. Travel by plane between Whitehorse, Yukon and the mine can take up to two hours, depending upon the type of aircraft used. The airstrip is a 1,219 m long VFR-rated gravel strip and requires periodic grading to maintain operability.

Emergency transportation, particularly medical evacuation, is by whatever means possible in the prevailing weather conditions. Fixed-wing aircraft, helicopter, bus, ambulance or a Company vehicle could be used in an emergency situation.

The Cantung Mine is located approximately 300 km by road northeast of Watson Lake, Yukon. Although the mine is situated in the Northwest Territories, Watson Lake is the staging area for trucking the tungsten concentrates and for supplying the mine site.

5.3 CLIMATE

Cantung is located in the Selwyn Mountains, and climatic conditions vary with elevation. The mean annual temperature for major valley systems is approximately -4.5°C, with a summer mean of 9.5°C and a winter mean of -19.5°C. Mean annual precipitation is highly variable, ranging from 600 mm at lower elevation on the perimeter of the Selwyn Mountains region up to 750 mm at high elevation. Locally at Cantung, severe winter conditions prevail from October to May with temperatures as low as -40°C and substantial snowfall. Total annual precipitation locally is 650 mm, half as rain and half as snow, with an average 1,270 mm snow accumulation in the valley.

Winter conditions limit the operation of the pit to generally 5 months duration per year. Underground operations continue year round as the mine is fully winterized.

5.4 INFRASTRUCTURE AND LOCAL RESOURCES

All surface rights for the mining operation are in place and current.

The site is supplied with electric power from a single powerhouse, equipped with diesel generators. Total installed power at the site is approximately 8.5 MW and the demand in cold weather approaches 4.5 MW. Fuel consumption for power generation is approximately 22,000 litres per day. Waste heat from the genset cooling system is recovered to heat the mill and other building facilities.

Diesel fuel is stored in two 360,000-litre tanks at the site. Diesel fuel from the main tanks is delivered to the powerhouse and roaster by gravity via a five-centimetre diameter pipeline, equipped with a number of control valves. Additionally, there are day tanks in the powerhouse with a capacity of 18,000 litres. Diesel, gas and propane are delivered on a regular schedule by truck via the Nahanni Range Road.

Water is sourced from the Flat River in accordance with the Water Licence. The water usage is restricted by the Water Licence to less than 45,000 m^3 per week. The water is treated and used as potable and process water. The water pump house has its own backup power generator.

The Cantung mine is organized as a single status fly-in/fly-out 365 day non-union operation. NATCL transports Northern employees once a week from Whitehorse and Watson Lake, Yukon, to the mine site via air charter out of Whitehorse. The size of plane depends on the number of people travelling that week. The Southern employees are chartered to/from site with 2 Beech 1900 aircraft on each Wednesday’s crew rotation.

The charter pick up points are Vancouver, Campbell River, Prince George, Kamloops and Smithers, BC. Most employees work on a rotational schedule, nominally three weeks on and three weeks off, 11 hours per day for hourly underground personnel and 12 hours per day for all hourly surface and salaried personnel. An Overtime Averaging Order has been obtained to allow the workforce to work the six-week work cycle and is renewed on a yearly basis through NWT Employment Standards.

The budgeted workforce number for fiscal year 2013-2014 was set at 254. Current workforce levels, as of June 30, 2014, are summarized in Table 5.1 below. Workforce numbers reflect total payroll, with approximately half of the hourly employees and more than half of the staff on site at any given time. In this budgeted workforce, total temporary employees, summer students and summer labourers are included for only a portion of the year.

Table 5.1 - Cantung Operation Workforce Summary, 2014.

Department	Hourly	Staff	Total	Budget
Administration Department	4	15	19	19
Surface Department	37	10	47	44
Mine Department (including Tech Services)	87	23	110	114.5
Mill Department	55	21	76	68.5
Environment Department	5	5	10	8.0
Total	**188**	**74**	**262**	**254**

The Cantung operation is fully permitted and has the required tailings dam facilities in place. Tailings Pond 4 (TP4) was raised to the present elevation of 3,757 feet in 2011. Deposition to TP4 ceased in July 2013, at which time deposition was switched to TP5. TP5 has been in use as the tails water exfiltration pond since 2006. This pond was raised to elevation 3,760 ft. in 2012. The TP5 Stage 2 raise in 2013 increased the elevation of the dam to 3,765 ft. The TP5 stage 3 raise is underway to bring the dam to the final elevation of 3,776.9 ft. The Stage 3 design will be the final raise on TP5, whose capacity has been assessed with respect to the operations of the water treatment facility and the move to dry stack tailings (DST), which is currently under permit application.

Tailings deposition commenced into TP5 in July 2013 and the water exfiltration rate has been decreasing significantly since deposition began. NATCL has been permitted for and is presently operating a Waste Water Treatment Facility (WWTF) to allow the direct discharge of treated tails water to the environment.

Additional engineering and design work has been conducted for long-term tailings storage facilities that would provide up to an additional 10 years of storage capacity, for reprocessed tails and the operations at Cantung Mine.

Permit applications for dry stack tailings (DST) are currently being assessed for the move towards a more sustainable, environmentally responsible method of tailings disposal.

All treated sewage is discharged to the Tailings Containment Area in accordance with the Water Licence provision. This is acceptable under current environmental regulations.

The site has a small garbage incinerator located at the garbage dump. Site garbage is incinerated at this installation and the residue, and other solid non-combustible waste, is buried at the existing garbage dump in a former borrow pit approximately three kilometres southeast of the town site.

Hazardous waste is handled, stored and disposed of in accordance with applicable regulations. PCB materials previously stored in a permitted facility at the site were removed from site to a third party facility for regulatory approved destruction in 2002. There are several transformers in service, with only 2 of these transformers containing PCB’s.

The current mill site was established in 1961 and the existing mill was originally constructed in 1967, after fire destroyed the initial facility. At the present time there are no plans to create a new milling operation. It should be noted that reprocessing of tailings from the older mill tailings is being considered. Although no concrete plans have been made, this reprocessing may require significant modifications or additions to the existing mill.

6 HISTORY

6.1 PRIOR OWNERSHIP

Prospectors discovered the Cantung tungsten deposit in 1954 while looking for copper. In 1959, the Canada Tungsten Mining Corporation Ltd. was formed to acquire and develop the property.

In 1985, Amax Inc. consolidated ownership of the Cantung Mine and transferred all tungsten assets to Canada Tungsten Mining Corporation, including the Mactung Project at Macmillan Pass. Amax Inc. retained majority control. Aur Resources Inc. (Aur) purchased Amax Inc.'s controlling interest in 1995 and Canada Tungsten and Aur merged in 1996. NATCL purchased the Cantung Mine from Aur in 1997, together with the related tungsten assets of the former Canada Tungsten Inc.

6.2 EXPLORATION AND DEVELOPMENT HISTORY

The Cantung mine commenced production in 1962 from an open pit at the rate of 300 tons per day (stpd), with production suspensions in 1963, due to low tungsten prices, and in 1966, due to the destruction of the mill by fire. The construction of a new 350 stpd mill was completed in 1967 and in 1969 the capacity was increased to 450 stpd.

In 1971 deep drilling discovered the E Zone. This zone was accessed through an adit collared at the valley bottom, close to the town site. The mill began to process the underground ore in 1974. In 1975 the mill was further expanded to 500 stpd. A major mill expansion in 1979 increased the mill capacity to 1,000 stpd. In 1986 the mine again ceased operations due to low tungsten prices.

After tungsten prices improved in 2000, NATCL reopened the Cantung mine in December 2001. Underground production and milling resumed at this time. In December 2003, NATCL was placed under the protection of the Companies Creditors Arrangement

Act (CCAA) and the mine was closed. In November 2004, NATCL successfully completed a plan of arrangement to deal with creditors which allowed planning to begin toward reopening the mine. Preparatory work for the reopening began in July 2005, and production resumed in late September 2005. The Cantung mine then again suspended operations in October 2009 due to low metal prices. Production resumed on October 8, 2010, and the operation has remained in continuous operation to the present day. Section 9 provides a summary of exploration work undertaken by NATCL since 2007.

6.3 HISTORICAL MINERAL RESOURCE ESTIMATES.

North American Tungsten Corporation Ltd. is not reporting here any historical resource estimates.

6.4 PRODUCTION HISTORY

Table 6.1 below summarizes the mine production history of the operation since restarting operations in October, 2010.

Table 6.1 - Production History 2010-2014

MONTH	YEAR	TONS	% WO3	STUs
October	2010	22,370	0.87	19,410
November	2010	24,313	1.02	24,743
December	2010	32,260	1.14	36,776
2010 Total		78,943	1.03	80,929
January	2011	27,713	0.99	27,436
February	2011	24,888	0.85	21,155
March	2011	25,431	0.71	18,056
April	2011	33,462	1.23	41,158
May	2011	33,289	1.00	33,176
June	2011	35,122	0.88	30,907
July	2011	31,899	0.64	20,350
August	2011	32,369	0.76	24,747

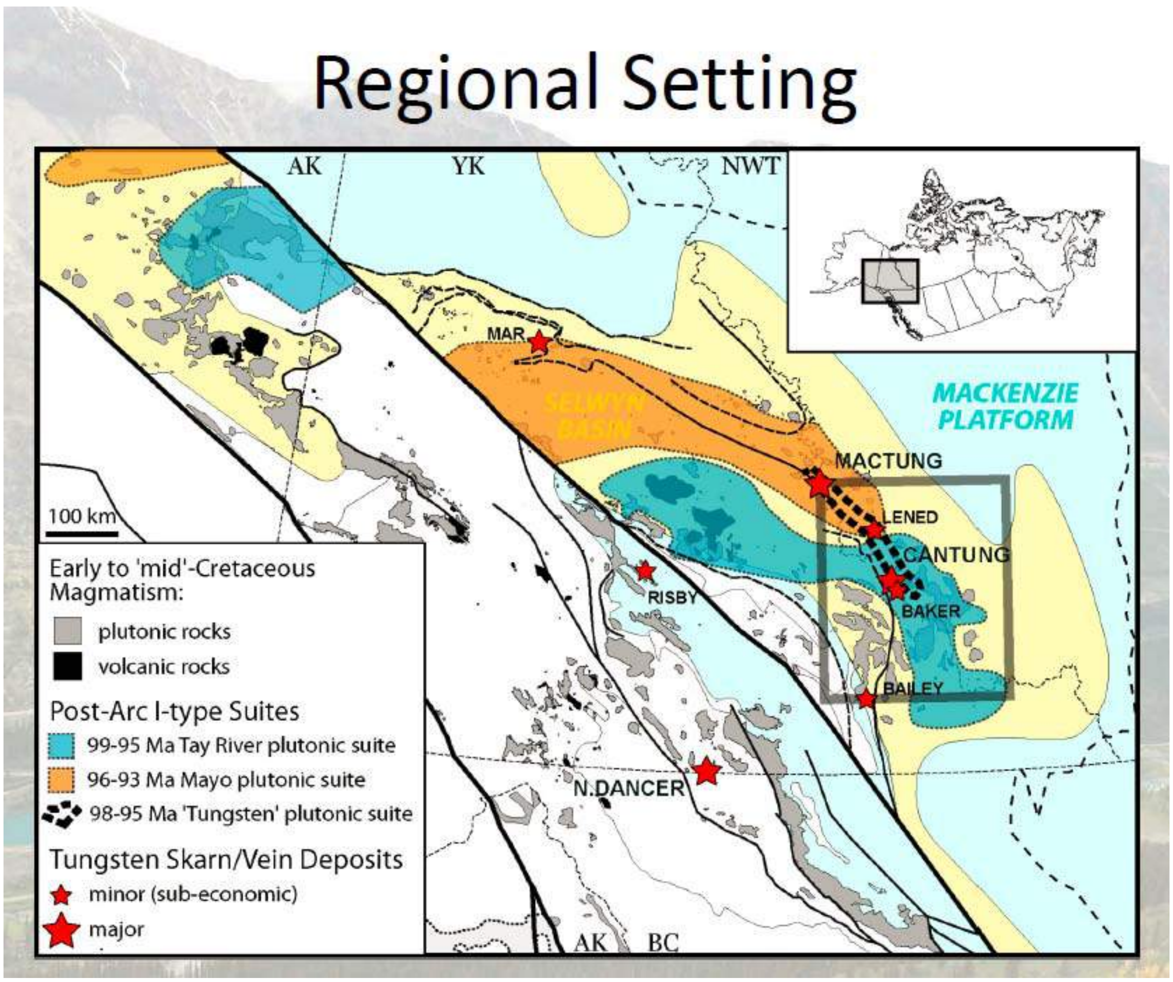
September	2011	32,545	0.97	31,647
October	2011	32,723	1.29	42,276
November	2011	31,945	0.88	27,962
December	2011	32,841	1.19	39,081
2011 Total		341,682	0.95	326,304
January	2012	28,886	0.84	24,229
February	2012	31,701	1.28	40,730
March	2012	32,214	1.23	39,623
April	2012	31,844	0.84	26,686
May	2012	33,786	1.03	34,664
June	2012	18,955	0.91	17,251
July	2012	33,575	0.84	28,288
August	2012	32,817	0.84	27,706
September	2012	32,390	1.32	42,628
October	2012	33,468	1.44	48,309
November	2012	31,870	0.89	28,231
December	2012	34,558	1.03	35,651
2012 Total		376,064	1.05	393,996
January	2013	34,873	0.99	34,472
February	2013	30,667	1.06	32,495
March	2013	34,600	0.99	34,286
April	2013	32,801	0.95	31,213
May	2013	34,267	1.06	36,386
June	2013	33,692	0.85	28,638
July	2013	34,413	0.85	29,168
August	2013	33,975	0.98	33,296
September	2013	32,862	1.10	36,147
October	2013	34,951	0.85	29,590
November	2013	33,351	0.84	28,020
December	2013	36,292	0.91	33,016
2013 Total		406,744	0.95	386,727
January	2014	36,325	1.24	45,111
February	2014	31,794	1.13	35,968
March	2014	34,945	1.26	44,189
April	2014	35,315	1.05	37,003
May	2014	39,405	0.72	28,286
June	2014	36,912	0.63	23,161
July	2014	36,342	0.81	29,479
2014 Total		251,038	0.97	243,197
Grand Total		**1,454,471**	**0.98**	**1,431,153**

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

Cantung is situated on the eastern margin of the Selwyn basin. Refer to Figure 7.1. The region consists of deep sea sedimentary facies rocks; black shale, siltstone, chert and minor carbonate. To the east of Cantung Mackenzie platformal carbonates dominate. To the west, terrane rocks accreted during the Mesozoic dominate.

Figure 7.1 - Regional Geology Map from Rasmussen, 2013



7.2 LOCAL AND PROPERTY GEOLOGY

Geological mapping of the Flat River area was conducted by Blusson during the 1962 and 1963 field seasons. Blusson described the area as underlain by a late Precambrian to Devono-Mississippian succession of miogeocynclinal carbonate and coarse and fine clastic sedimentary rocks. This succession is moderately deformed into a complex synclinal structure the axis of which trends south-easterly down the Flat River Valley. Subsequently the area has been intruded by a series of discordant Cretaceous granitic stocks. Refer to Figure 7.2.

Figure 7.2 - Cantung Property Geology

The following is an outline of the stratigraphy on the west side of the Flat River valley at NATCL's Cantung Mine site:

MIDDLE AND UPPER CAMBRIAN

Unit 9: Rabbitkettle Formation - grey, thin bedded (wavy banded) silty limestone, black shale and graphitic limestone.

LOWER AND/OR MIDDLE CAMBRIAN

Sekwi Formation

Unit 6: Dolomite - grey and buff weathering, sandy and sparry dolomite.

Unit 5: Upper Argillite - dark grey to black slate, siltstone and calcareous argillite, locally pyritic.

Unit 3: Ore Limestone - grey, banded coarse grained crystalline limestone.

Unit 2: Swiss Cheese Formation - grey to purple interbedded argillite and limestone.

LOWER CAMBRIAN AND PROTEROZOIC

Backbone Ranges Formation

Unit 1: Lower Argillite - dark grey slate, siltstone and fine grained quartzite.

7.3 MINERALIZATION

The two main historic scheelite skarn ore bodies, the Open Pit and E-Zone, are spatially located within the Ore Limestone unit on the upper and lower limbs, respectively, of a recumbent anticline on the west side of the Flat River Syncline. Refer to Figure 7.3 and Figure 7.4.

Figure 7.3 - Surface Projection of Ore Zones (Sept 19, 2014)

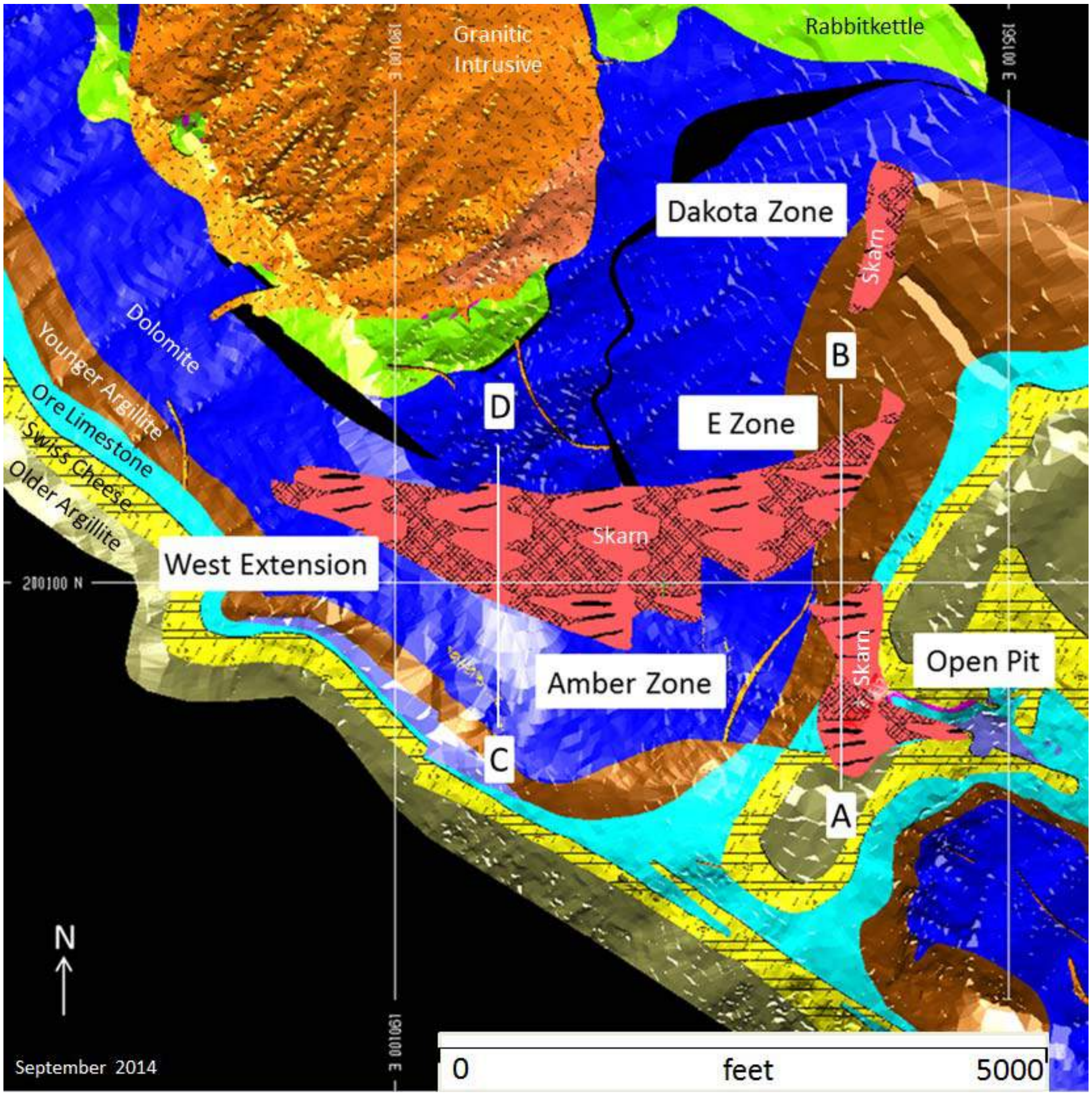
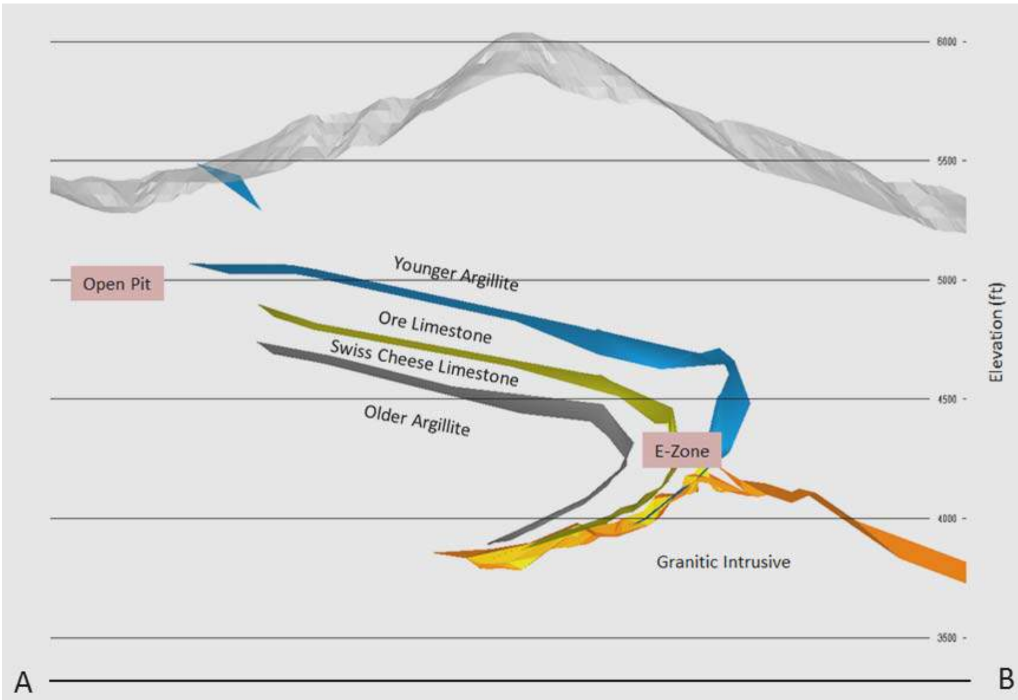
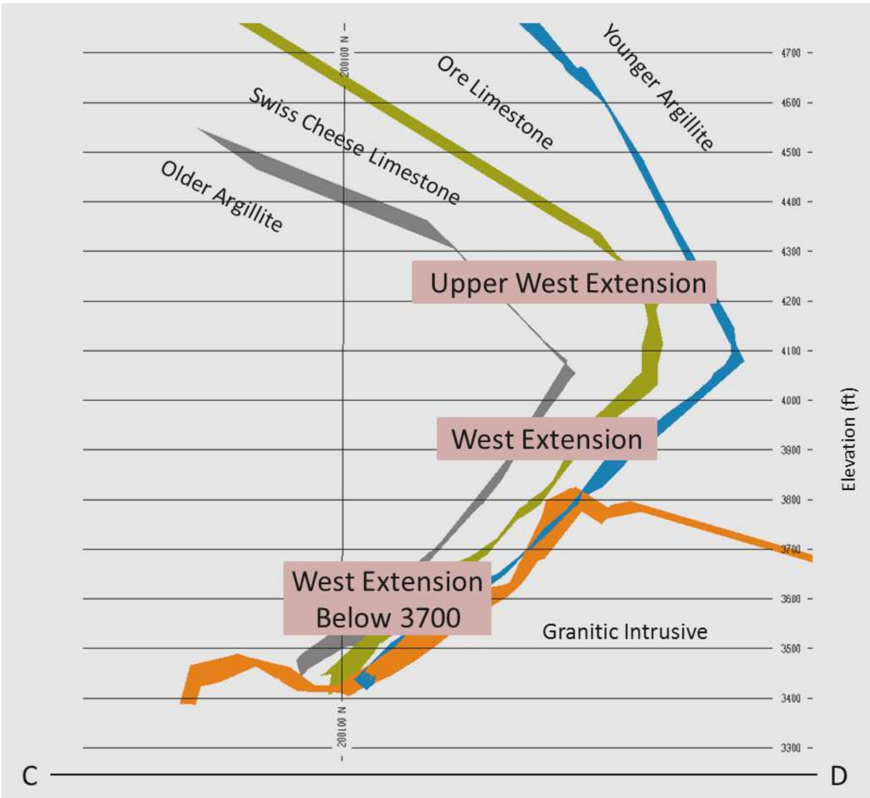


Figure 7.4 - Mine Geology Cross Section, Looking West (Sept 19, 2014)



The major scheelite ore body mined from underground at Cantung historically was the E-Zone ore body. Since reopening in 2010, the mine has expanded the known extent of the scheelite ore horizon down dip along the lower limb of the recumbent anticline to the south and to the west as well as along the upper limb to the west. Refer to Figure 7.5.

Figure 7.5 - Extension Zones Cross Section, Looking West (Sept 19, 2014)



These expanded areas include the West Extension, the Amber zone and Below 3700 elevation level along the lower limb of the fold and the Upper West Extension along the upper limb of the fold. The underground ore zones extend 4,360 ft. along strike, 1,200 ft. down-dip along the lower limb, with some interruption from intrusive sills and dikes, and 200 ft. up dip from the fold hinge along the upper limb.

Throughout most of the underground deposit, three main ore lenses are present: one lens occurs within the Swiss Cheese Limestone, the second lens occurs at the upper contact of the Ore Limestone with the Swiss Cheese Limestone, and the third lens occurs at the lower contact of the Ore Limestone with the Younger Argillite or the granodiorite unit. Intermediate lenses also occur within the Ore Limestone but they tend to be less continuous than either the second or third aforementioned lenses.

The scheelite mineralization in the West Extension and Amber zone is usually fine to medium grained (locally coarse grained), finely to coarsely disseminated and sometimes concentrated in bands, especially within the Swiss Cheese Limestone. These banded concentrations are also observed in the Open Pit Swiss Cheese Limestone ore. Scheelite is usually present in a massive to semi-massive pyrrhotite skarn or a calc-silicate skarn containing abundant pyroxene, garnet and minor pyrrhotite. Both skarn types generally contain chalcopyrite, with some black sphalerite locally. Skarn development in the Ore Limestone and Swiss Cheese Limestone underground is limited by general proximity to the granodiorite intrusive as well as an abundance of fluid transporting fractures and structures. The scheelite mineralization in the Open Pit has been genetically linked by previous researchers to an aplitic dyke and/or quartz vein stockwork that intruded up towards the Open Pit, bringing ore bearing fluids from depth.

8 DEPOSIT TYPES

The Cantung mine is a tungsten-copper exoskarn deposit. The mining operation consists of two areas: the Open Pit and the underground workings. Cantung is a typical skarn-type deposit, albeit of an unusually large size for a tungsten bearing skarn. As with most skarns, the mineralization is related to a granitic intrusion and its associated intrusive dykes. These intrusive units are believed to have given off reactive fluids that have then come in contact with the overlying reactive Ore Limestone and Swiss Cheese Limestone units. As expected, exploration programs have largely focused on the contact horizon between the granodiorite intrusive and various limestone units on the property. This genetic model is summarized below.

Genetic Model Summary - after Rasmussen et al, 2007

• The presence of two high-grade tungsten skarn ore bodies at Cantung necessitates a large magmatic system at depth that was capable of sequestering a large quantity of W through normal crystallization processes.

• Several key factors in the occurrence of high-grade reduced tungsten skarn mineralization at Cantung are:

  - An underlying, evolved or slightly peraluminous, I-type monzogranite (“Mine Stock”)
  - A prolonged cooling period relative to other granites in area (recharging at depth)
  - An average magmatic water/rock ratio of 40 (>90 mol% in equilibrium with late aplite dykes)
  - One or more pre-existing structural traps (the hinge and upper limb of a recumbent anticline), and the focusing of fluids by dykes or faults into these traps
  - A relatively pure crystalline limestone
  - A pre-existing structural and lithological trap (i.e., folded limestone ‘encased’ in argillite)

- A very late-stage over pressuring of incompatible element-rich supercritical fluid, creating subvertical fractures through the plutontic carapace and argillite
- Steeply dipping, highly fractionated aplite dykes (latest residual melt extruded from a solidified magma chamber pervasively metasomatized to a calcic mineralogy
- The interaction between melt, fluid and calcareous country rocks was skarn-forming

9 EXPLORATION

9.1 EXPLORATION PROGRAMS

In recent years the NATCL has extensively explored the mine property through ground and airborne magnetic and electromagnetic surveys, and by geochemical stream sediment and soil sampling. No obvious targets were discovered by these surveys, with the exception of the geochemical anomalies in the upper reaches of Rifle Range Creek. Table 9.1 below is a summary of exploration work completed since 2007.

Table 9.1 - Exploration Programs 2007-2014

Year	Project	Target	Method
2007	Geochemical Soil Sampling	Dolomite Mountain	Field work
2009	Geophysics	Rifle Range	IP, Gravity, and Total Field Magnetics
2009	Soil sampling, subglacial water sampling	Rifle Range	Soil Sampling and Diamond Drill
2009	Geochemical Soil Sampling, Geophysics	Dolomite Mountain and Flat River Valley	IP, Gravity, PROTEM and Total Field Magnetics
2011	Geophysics	Mine area	PROTEM
2011	Surface Drilling	Sheet Mountain	Diamond Drill
2012	Open Pit Surface Drilling and Dakota	PUG and Dakota	Diamond Drill
2013	Surface Drilling	Dakota and Flat River valley	Diamond Drill
2013	Prospecting	Fish and Sawmill	Field work
2013	Geophysics	Dakota Zone	Downhole TDEM, IP, Resistivity
2014	Surface Drilling	Upper Limb and Geophysical Anomalies	Diamond Drill
2014	Prospecting	Fish and Sawmill	Field work

Downhole geophysics was completed during the 2013 surface exploration program on 5 diamond drill holes completed earlier that season in the vicinity of the Dakota Zone. Aurora Geosciences conducted downhole geophysical parameter logging, DC resistivity/chargeability and time domain electromagnetic (TEM) geophysical surveys. Aurora inverted the data and compared geophysical responses to the downhole lithologies noted from geological logging. They outlined two major anomalies at depth to the east of the Dakota Zone corresponding to the prospective contact between the Ore Limestone and Swiss Cheese Limestone units. Refer to Figure 9.1 and 9.2.

Figure 9.1 - Plan View, Recovered Resistivity, 2013

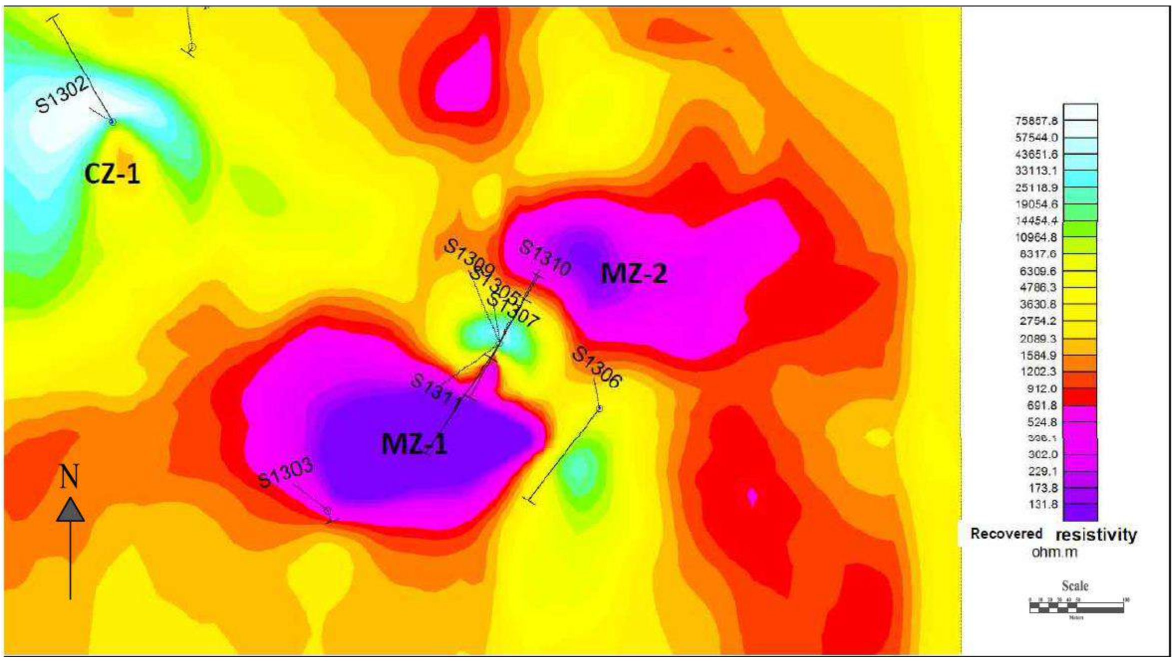
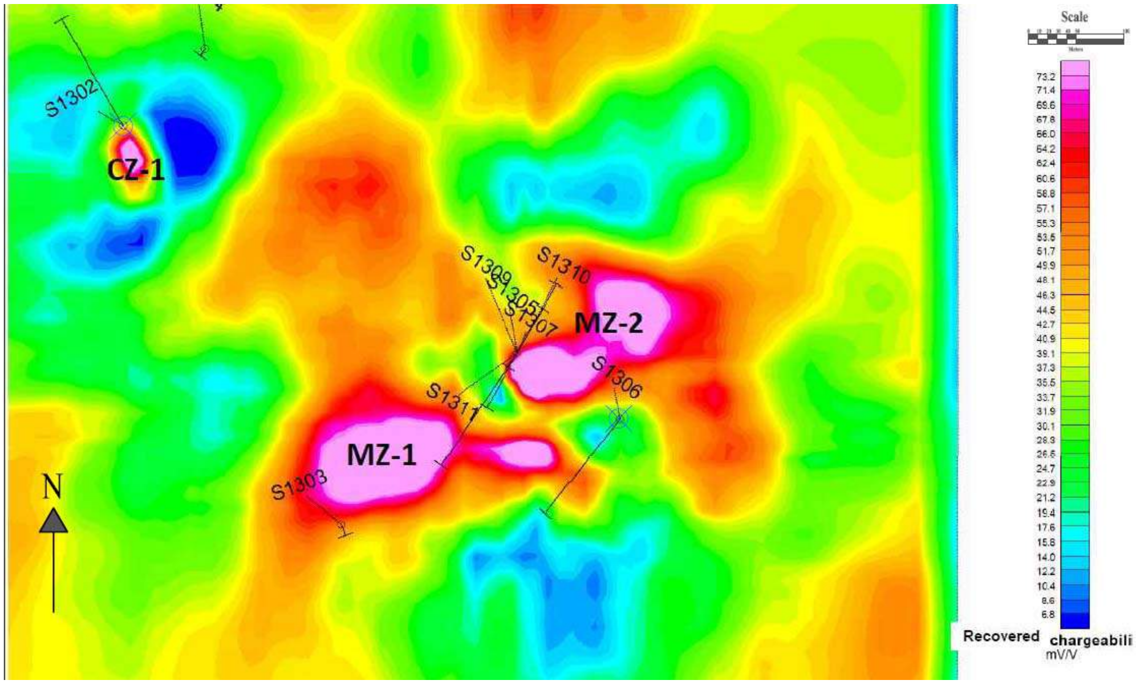


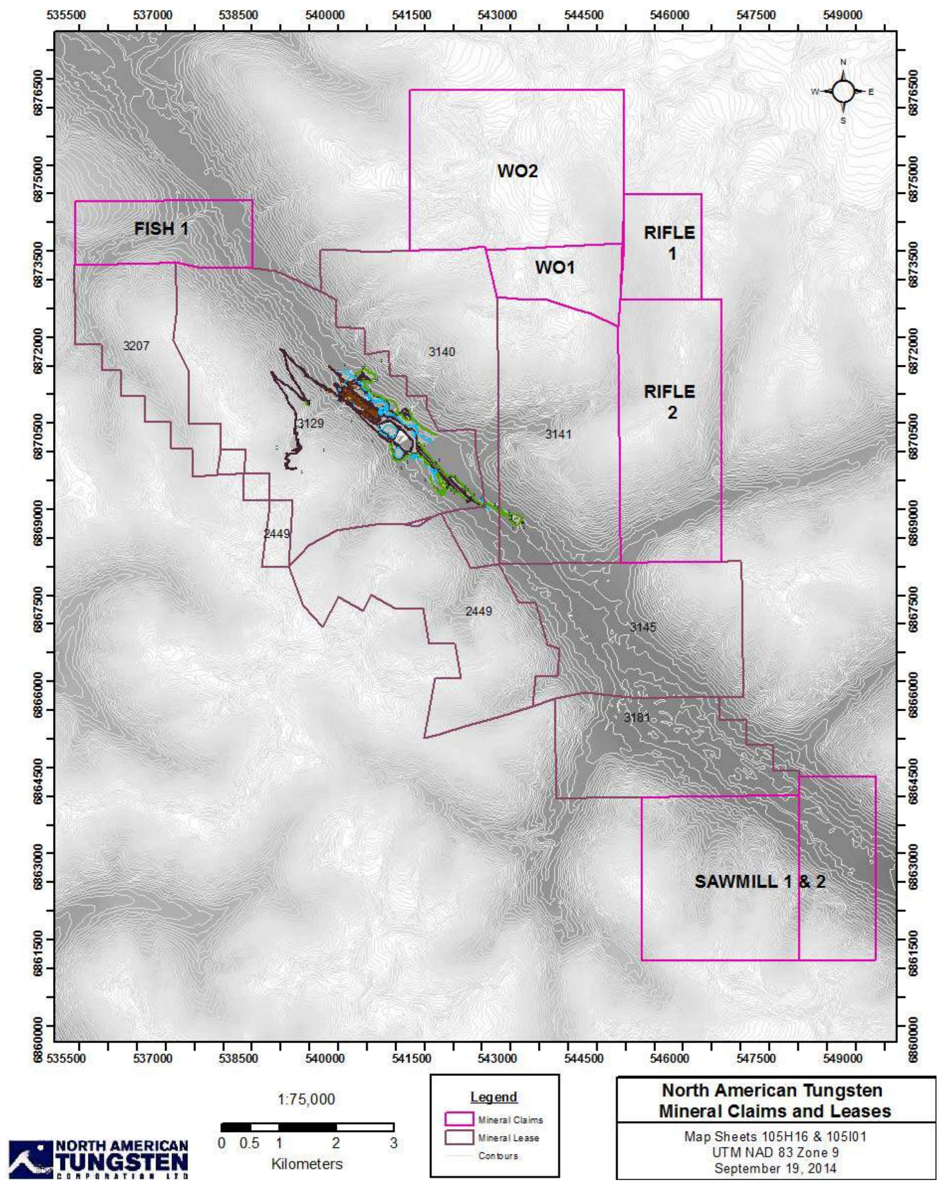
Figure 9.2 - Plan View, Chargeability, 2013



Surface diamond drilling to test these two anomalies is ongoing during the 2014 summer exploration program.

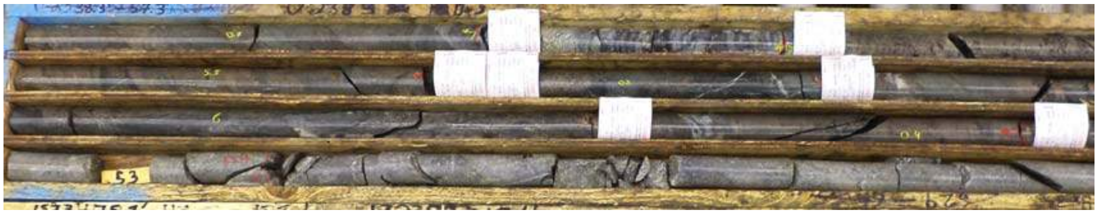
Claims maintenance exploration was completed on the Fish 1, Sawmill 1 and Sawmill 2 claims during the 2013 summer exploration program. These claims are located in the immediate vicinity of the Cantung Mine. Refer to Figure 9.3.

Figure 9.3 - Fish and Sawmill Mineral Claim Locations



The claims had not been the subject of any previous detailed exploration work. In 2013 a small prospecting program was conducted. The goal was to identify areas of mineralization and get an understanding of the underlying stratigraphy. Field work consisted of mapping, prospecting and rock sampling. Follow up work is to be completed on these claims during the 2014 field season with continued mapping and sampling of untested areas.

10 DRILLING



10.1 DIAMOND DRILLING

A total of 2,660 holes drilled have been drilled on the property for a total footage of 691,221 feet. Of these, 295 holes have been drilled from surface for a total of 142,795 feet, with the remaining holes and footage drilled from underground. Over 43,500 drill core samples have been taken and assayed.

Surface diamond drilling up to 2007 was limited to a few drill holes in the Open Pit/PUG Zone since the mine ceased production in 1986. A diamond drill program was reinitiated in mid-2007 including one hole on Sheet Mountain and drilling in the PUG. Further surface drilling was undertaken in 2010 in the area of the Rifle Range, Sheet Mountain and on several geophysics targets on the property. Initial results were not encouraging.

The surface drilling program in 2011 targeted the Circular Stock intrusive observed on Dolomite Mountain and was designed to test the extent of the stock and the nature of its contact to the Mine Stock. Results showed that the original theory of a circular stock was doubtful as an extensive zone of skarn mineralization was encountered. There was no economic mineralization however.

The 2012 surface drilling focused on definition drilling in the Open Pit/PUG as well as exploration drilling targeting the northeastern projection of the E-Zone. The Dakota zone was discovered towards the end of the program. Summer 2013 drilling focused on expansion of the Dakota zone with 12 drill holes totalling 11,312 ft.

The 2014 surface drill program has been designed to test mineralization in the upper limb of the fold north of the Open Pit, to test the local granite model and to expand the known mineralization in the vicinity of the Dakota zone.

Mineral resources and reserves for the area Below 3700 and Amber Zone are based almost entirely upon underground diamond drill core assay data from holes drilled in the mine area by North American Tungsten Corporation Ltd. from 2007 through 2009 and 2012 through 2014.

To date, no drilling, sampling or recovery factors have been identified that could materially impact the accuracy and reliability of results.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 MINESITE SAMPLE PREPARATION, SECURITY AND QUALITY CONTROL

Diamond drill core from the Cantung mine was logged for lithology, mineralization and structure, and when appropriate, geotechnical properties. The drill holes were logged either on log sheets, or directly into computers. The collected data was then transferred to an Access Database customized to store the Cantung diamond drilling data. Lithology was logged according to the parameters specified by the Cantung rock descriptions standard.

Samples, as selected by the geologists, were collected from the drill holes after core logging. Samples collected for assay were sawn, split or whole cored, depending on the purpose of the drilling. Exploration drill holes were sawn, whilst definition drill holes were sawn or split. Production drill holes were typically whole cored. The maximum sample length was limited to 5 feet, with typical sample lengths being in the range of 1-3 feet. Samples were broken on lithological contacts, or at noticeable changes in the concentration of mineralization within the larger skarn unit. Sample pulps and rejects are saved and most of the core has been photographed.

After logging, samples were collected and placed in individual plastic bags and were assigned a unique sample number. The sample numbers were recorded both in the drill log and on a sample transfer sheet which was sent to the lab. At the sample prep stage, the lab would verify that all samples collected by the geologist were received.

To ensure data quality, QA/QC materials were inserted into the sample stream. The QA/QC materials included certified standards, blanks and pulp duplicates and were inserted at a frequency of one certified standard, one blank and one pulp duplicate per assay run. The assay lab also ran core duplicates every ten samples. An in-house, non-certified mill standard was also included in each run for statistical control. The certified standards used at Cantung are commercially available, with the material used to produce the standards sourced from the Cantung mine itself. Table 11.1 is provided below, showing the grades of the certified standards used at the Cantung operation.

Table 11.1 - Cantung QA/QC Certified Standards Used

	Recommended Values	
Reference Standard	Tungsten Concentration	Copper Concentration
CDN-W-1	1.04 ± 0.1%	0.458 ± 0.042%
CDN-W-2	2.78 ± 0.39%	0.45 ± 0.034%
CDN-W-3	1.73 ± 0.19%	0.44 ± 0.036%
CDN-W-4	0.366 ± 0.024%	0.139 ±0.008%

Barren granitic waste rock was used for the blank material. Duplicates were created by taking previously assayed lab pulps and renumbering and resubmitting the samples back to the lab. The resulting data pairs were interrogated to verify the performance of the lab.

11.2 SAMPLE PREPARATION, ASSAYING AND ANALYTICAL PROCEDURES

The majority of the sample analysis was completed at the non-certified Cantung Assay Lab. Sample preparation here involved drying the samples in an oven for 2–4 hours, followed by primary crushing. At that point samples were riffle split down to 200–250g of material. The individual 200 – 250g samples were then pulverized for 125 seconds and mixed by rolling to ensure homogeneity. The samples were transferred to a labelled kraft sample bag to await analysis. Apart from the sample transfer sheet, no special security measures were taken, as it was not deemed necessary to do so.

From 2008 through to 2010, drill holes targeting the West Extension and Amber Zone discoveries were assayed by the non-certified Cantung mine lab using colour analysis. From 2010 to 2014, analysis was performed using two assay procedures. Initially, samples were analyzed by pressed pellet XRF. Assay values in excess of 6% WO_3 triggered an overlimit result and were reanalyzed using acid digestion with colorimetric finish. Assays, which reported low WO_3 values but higher Cu values, were also reanalyzed by colorimetric analysis. A blank, a standard and a pulp duplicate were included within each batch of drill core samples

Colorimetric analysis was performed by taking a 0.2–0.5g subsample from the pulverized, homogenized pulp and digesting the sample in a mix of HCl, H_3PO_4 and HF acids. Distilled water was then added and the sample was bulked to 200 mL and then filtered. A 25 mL aliquot was then taken for colour development with H_2SO_4, HCl, $SnCL_2$ and KSCN and bulked to 100 mL. From there, the aliquot passed to the colour development stage, where the prepared solution was inserted into a HACH DR2800 colorimeter and the %WO_3 calculated. External check assaying was completed regularly by both ALS Canada of North Vancouver and ACME Labs of Vancouver, B.C. Both are highly certified, reputable and independent of Cantung.

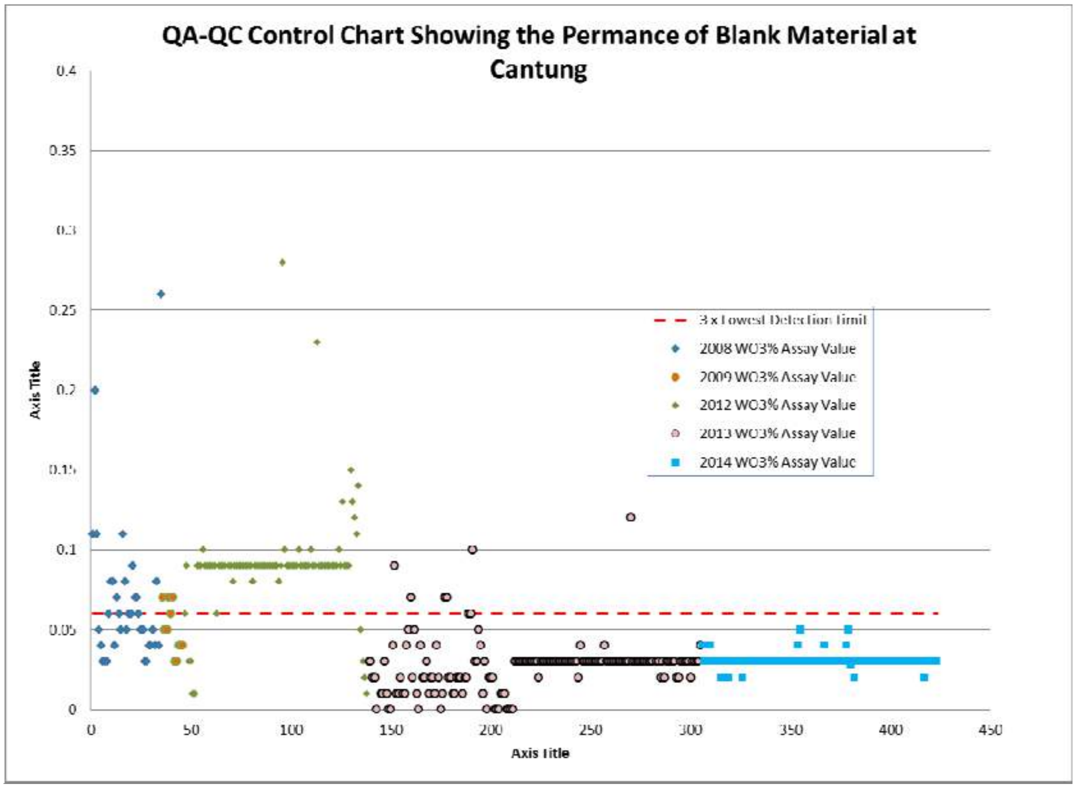
Most drill core samples from the E-Zone were assayed at Rossbacher Laboratories, Burnaby, B.C., utilizing hot hydrochloric acid digestion followed by colorimetric finish. The method of sample preparation and analysis was designed by Amax Inc. for application at its Climax, Colorado mine. Rossbacher used a set of standard samples developed by Amax. Check assays were carried out at ALS Chemex Laboratories Ltd. and Bondar-Clegg Ltd., both in North Vancouver, B.C.

11.3 QUALITY CONTROL MEASURES

A comprehensive set of quality control measures are in place at Cantung. QA/QC materials are inserted, with defined frequency, into the sample stream. These include certified standards, blanks, pulp duplicates, core duplicates and mill standards, as noted in Section 11.1.

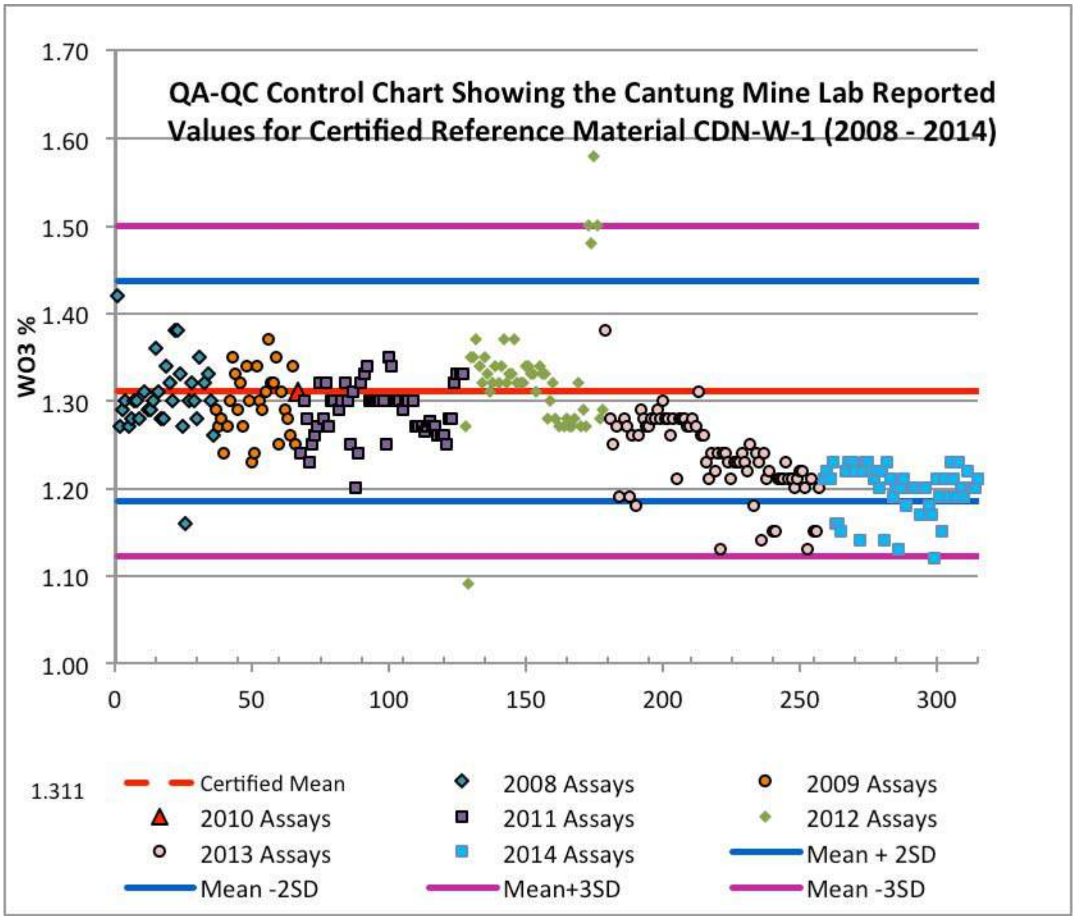
QA/QC Control charts are provided below to summarize the performance of QA/QC materials such as blanks, standards and duplicates. (Figure 11.1 to Figure 11.7)

Figure 11.1 - QA/QC Control Chart for Cantung Blanks, 2008-2014



Analysis of the 423 blanks submitted to August 2014 revealed that from 2013 onwards, the vast majority of blanks passed the QA/QC test, as can be seen in Figure 11.1 above. Three blanks that reported in excess of 1.2% WO_3 (not shown on chart) were most likely a record-keeping error. Reference material CDN-W-1 or a pulp duplicate in a similar range would report such a grade.

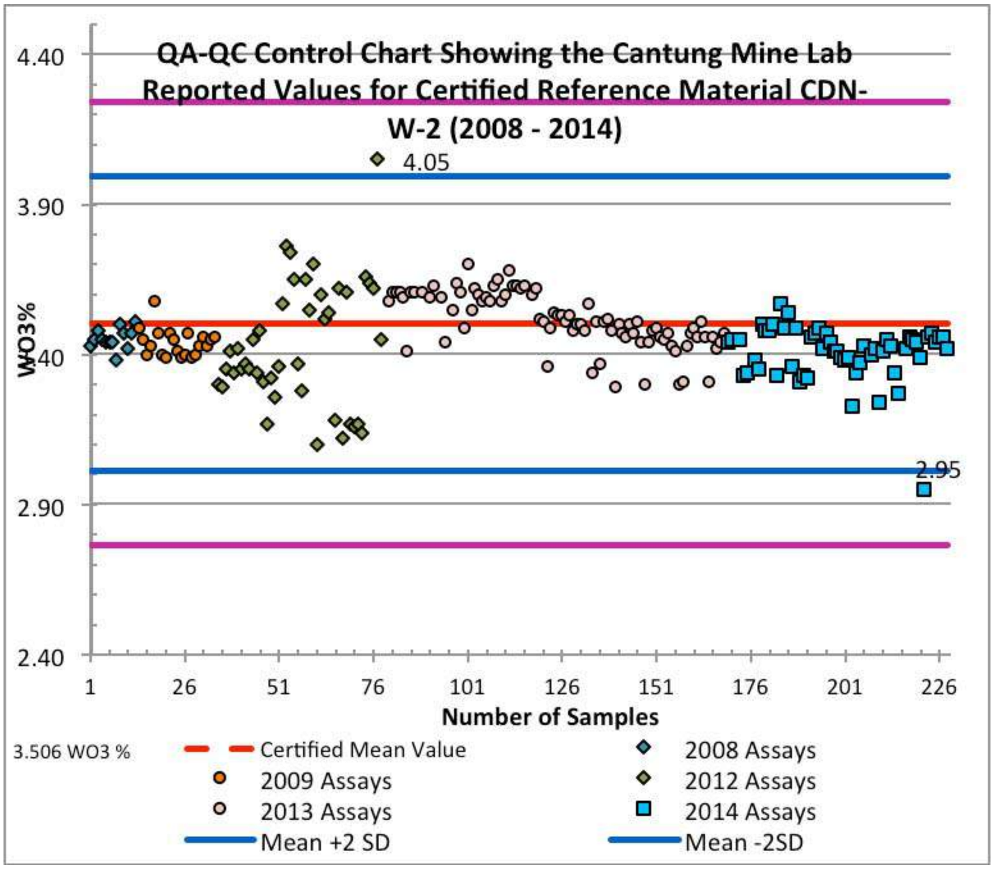
Figure 11.2 - QA/QC Control Chart for Standard CDN-W-1, 2008-2014



From 2008 through to 2010, the lab accurately determined the grade of the CDN-W-1 certified reference material. Assay results from 2012 showed two distinct data populations, but both data populations fell within two standard deviations of the mean certified reference value and were accepted. Refer to Figure 11.2.

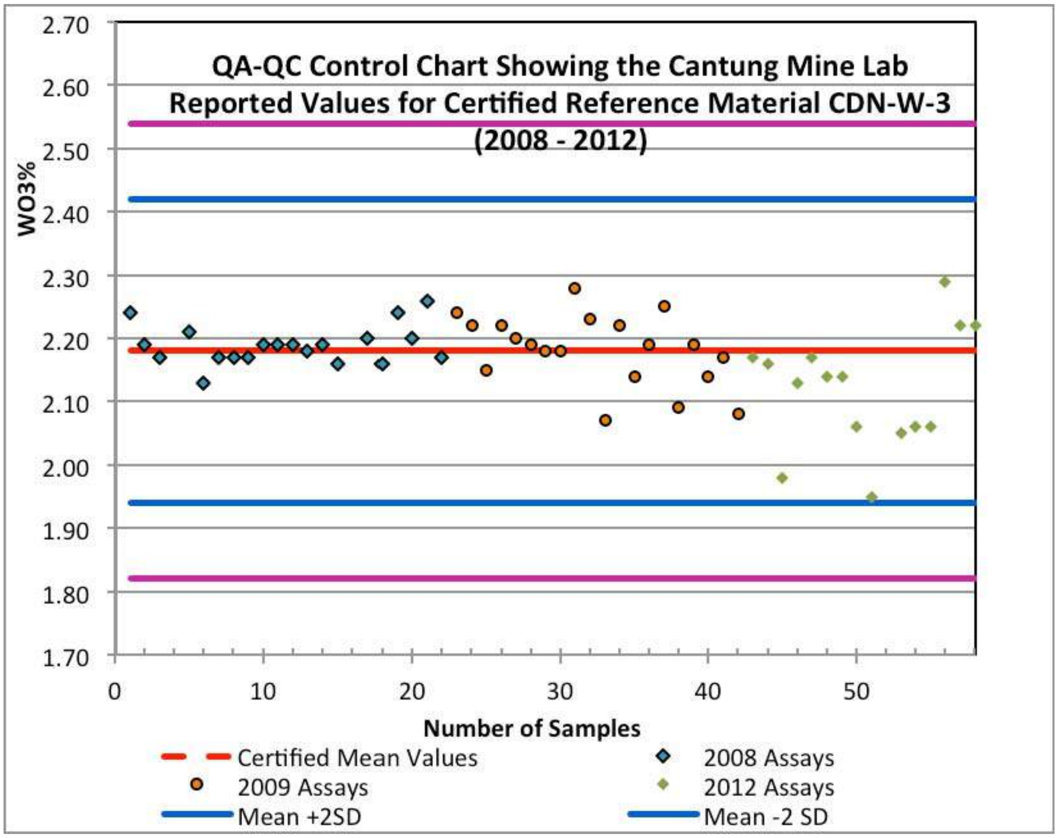
In 2013 we saw a slide in the reported assay value for the W-1 reference material, but the data largely fell within tolerance. The 2014 data showed that the drift in assay values had stopped, although 11 of the 58 assay values reported for 2014 fell below the -2 x Standard Deviation cut off. An additional 4 samples from 2014 (not shown) reported grades around 3.4% WO_3 and were believed to actually be reference material CDN-W-2, not CDN-W-1 as reported and a fifth sample was believed to be a mislabelled pulp duplicate.

Figure 11.3 - QA/QC Control Chart for Standard CDN-W-2, 2008-2014



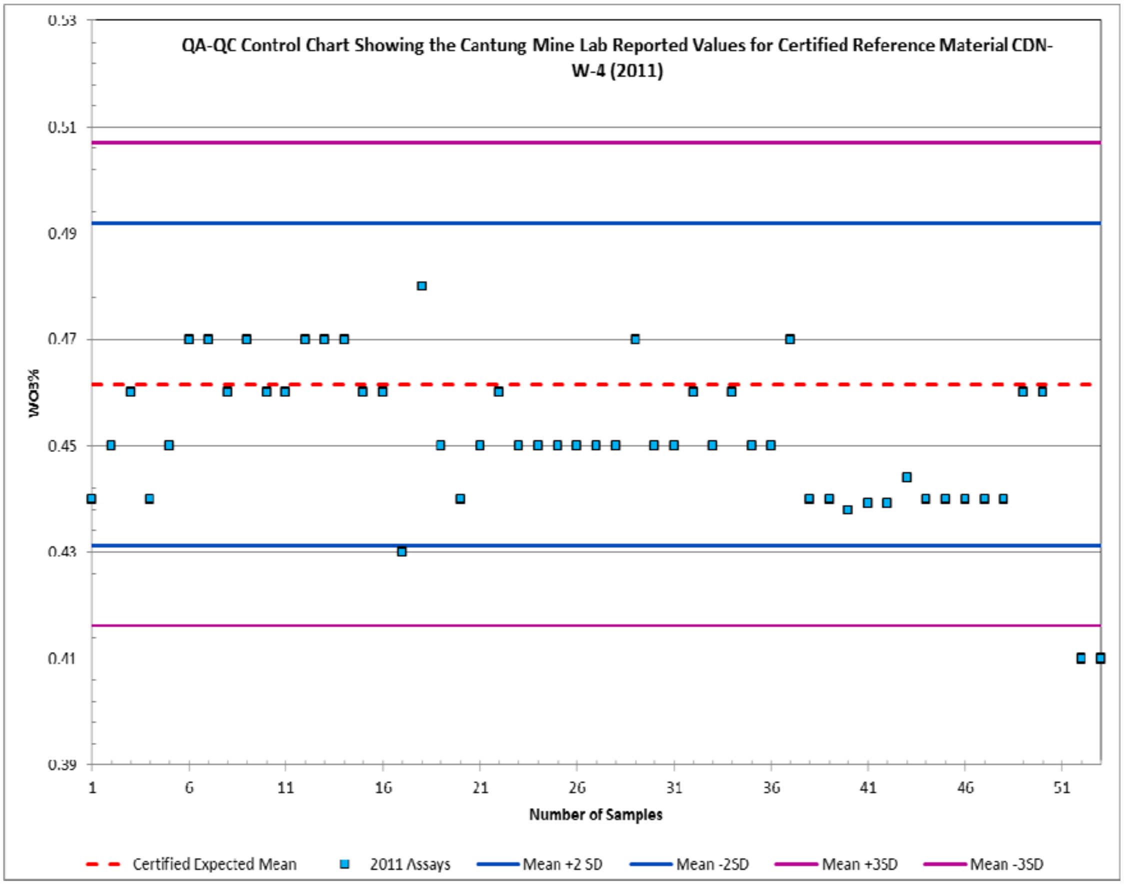
Analysis of the assay values returned from CDN-W-2 analysis showed that the Cantung Mine Lab accurately determined the grade of the reference material. The assay values from 2014 consistently plotted between the Mean and – 2 x Standard Deviation bracket, suggesting that the Cantung Mine Lab analyses have drifted slightly lower over time. Refer to Figure 11.3..

Figure 11.4 - QA/QC Control Chart for Standard CDN-W-3, 2008-2014



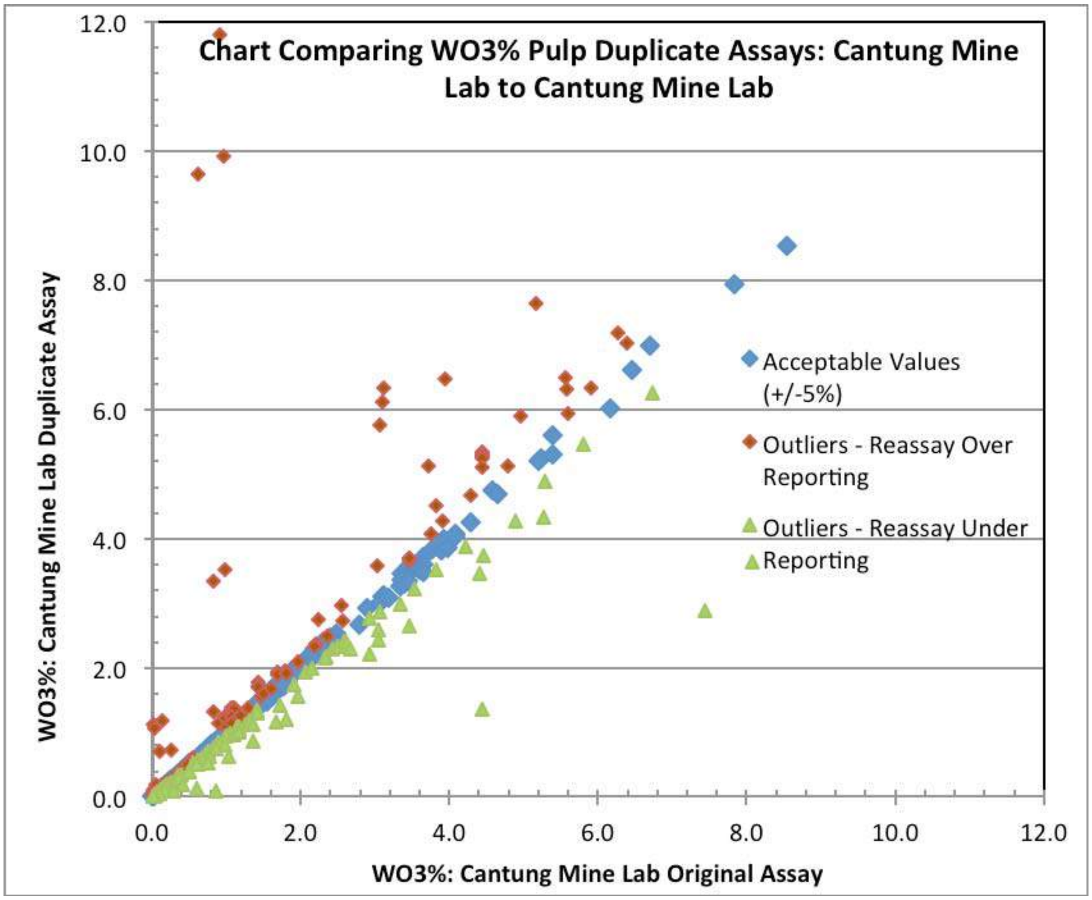
The Certified Reference Material CDN-W-3, which is no longer in use, has reported within tolerance. Refer to Figure 11.4.

Figure 11.5 - QA/QC Control Chart for Standard CDN-W-4, 2011



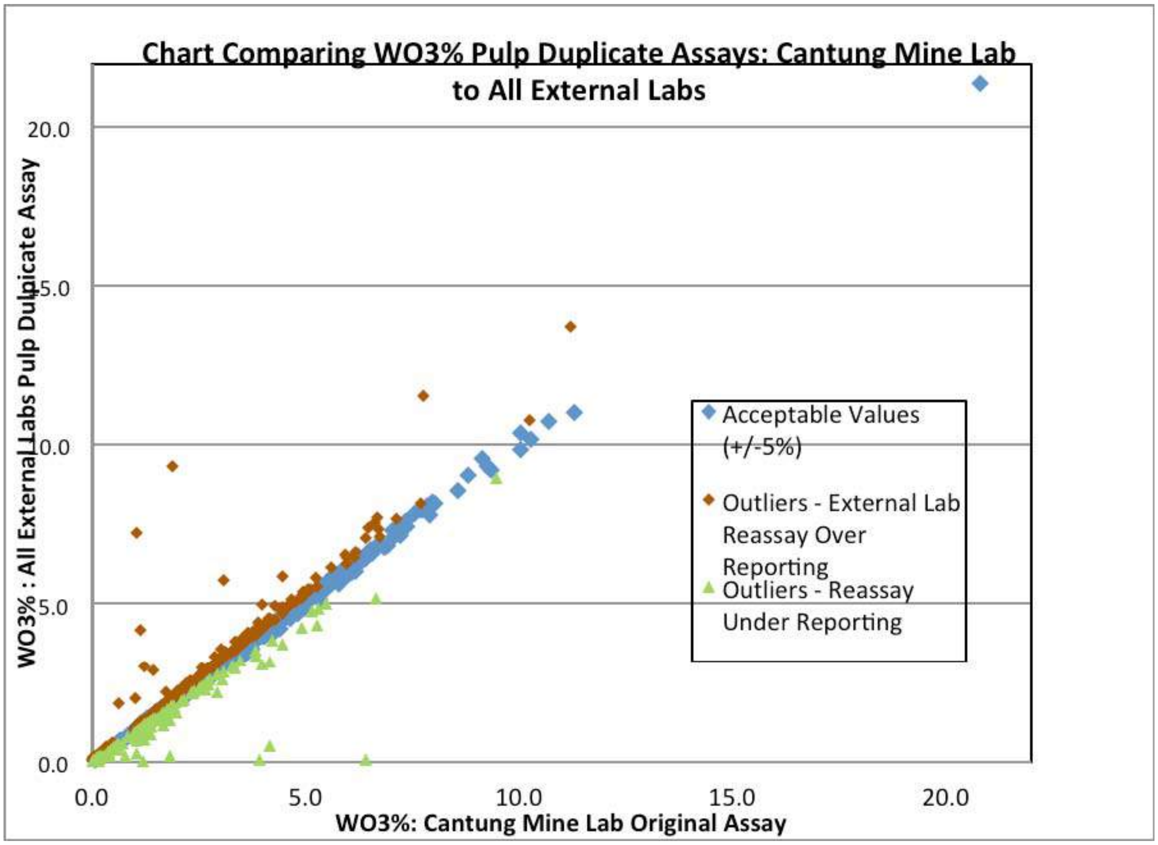
The Certified Reference Material, CDN-W-4 reported well and within expected tolerance during its use in 2011. Refer to Figure 11.5.

Figure 11.6 - Cantung Pulp Duplicates Assay Comparison Chart



The Cantung Mine lab has shown acceptable reproducibility from duplicate samples, as shown in Figure 11.6. Of the 494 sample pairs in the data set, 322 sample pairs plotted within 5%; 119 sample pairs were outliers where the duplicate over reported compared to the original assay; and 53 sample pairs were outliers where the duplicate assay under reported compared to the original assay.

Figure 11.7 - Comparison Chart of Duplicates Assays, Cantung vs External Labs



Analysis of the pulp duplicates QA/QC submitted to both the Cantung Mine Lab and external, independent labs revealed reasonable assay result duplication between the mine and the external labs, as can be seen in Figure 11.7. Of the 946 sample pairs in the data set, 577 sample pairs plotted within 5%; 224 sample pairs were outliers where the external lab assay value over reported compared to the original assay; and 145 sample pairs were outliers where the external lab assay value under reported compared to the original assay. On that basis, the Cantung mine lab can generally be considered accurate, or somewhat conservative, when the internal lab’s assays are compared to external labs.

The authors are of the opinion that the sample preparation, security and analytical procedures currently in place are of a sufficient quality as to allow for the subsequent data processing.

12 DATA VERIFICATION

Both authors work at this producing mine. In order to verify that the data used in the report was accurate, they undertook the following verification procedures:

1) Diamond drill hole data and assays and QA/QC were reviewed.
2) The 3D Block model was checked to ensure that it was properly built and constructed. Test blocks were checked for accuracy.
3) The Reserve blocks were checked by NATCL's Technical staff.
4) As a final step it was felt that the best way to verify data was to compare predicted grade to grades achieved in the mill. This would verify the diamond drill assay data used to create the model, the adequacy of recovery and dilution factors used and the ability of the operation to meet production plans and budgets.

Table 12.1 is a reconciliation/verification of the predictability of the tonnage and grade. Records go back to 2007 when MineSight was once again implemented at Cantung. Totals for the years will not match mill production in some cases, as development rounds were not used in some years. No data is available for 2011, the year that operations resumed. There appears to be considerably more tonnage put through the mill than blasted. In 2007 and 2008 this was almost entirely due to dilution with grade. Pillar recovery of backfilled "waste" rock occurred during this period. A considerable portion of this "waste" rock was economic at this time.

Table 12.2 and Table 12.3 are reconciliations of monthly production comparing Budgeted, Forecast to Achieved Mill grades. When the time lag in ore being processed by the mill and the variability of the ore body are taken into account, the results are considered consistent with expectations.

It is the opinion of the authors that the Resource-Reserve portion of the report meets best practices guidelines.

Table 12.1 - Ore Tonnage/Grade Reconciliation 2008-2014

fiscal year	LEVEL	blasted tons	WO3 reserve grd	Tons Adjusted to Mill	WO3 Grade Adjusted to Mill
	1089 Total	13,521	0.94	17,593	0.97
	1350 Total	4,279	1.04	1,008	0.58
	1600 Total	275	0.80	275	0.11
	3700 Total	15,873	1.04	14,666	0.87
	3740 Total	11,512	1.20	11,708	1.20
	3776 Total	2,404	1.00	2,404	0.81
	3810 Total	8,991	0.99	10,944	1.08
	3812 Total	17,346	0.94	17,734	0.94
	3854 Total	20,383	1.09	27,718	1.08
	3870 Total	6,595	1.01	7,760	1.07
	3880 Total	3,080	1.00	4,638	1.00
	3950 Total	76,647	1.25	103,879	1.15
	4000 Total	6,677	0.91	8,480	0.99
	4050 Total	40,381	1.02	38,527	1.07
	4100 Total	7,664	0.92	9,002	0.76
	4125 Total	32,853	1.29	75,257	0.96
	4170 Total	4,285	1.05	5,323	0.84
	4200 Total	25,017	0.94	31,476	0.88
	4350 Total	-	-	767	0.77
fiscal 2008 Total		**297,783**	**1.10**	**389,159**	**1.03**
	3610 Total	1,390	1.13	1,376	0.57
	3660 Total	9,388	1.20	12,425	1.26
	3700 Total	27,530	1.17	45,787	1.41
	3740 Total	3,267	1.65	6,403	1.49
	3800 Total	690	0.91	808	0.95
	3810 Total	34,258	1.01	46,418	1.09
	3812 Total	5,008	1.11	5,723	0.79
	3850 Total	8,525	0.96	2,887	1.12
	3854 Total	-	-	746	1.97
	3870 Total	12,056	0.91	16,663	0.97
	3880 Total	47,705	0.90	61,924	0.89
	3940 Total	135	1.15	106	1.82
	3950 Total	25,899	1.05	40,309	1.20
	4000 Total	1,266	0.96	938	0.81
	4050 Total	60,937	1.08	81,117	1.04
	4100 Total	5,350	1.04	4,150	1.75
	4125 Total	35,154	1.42	34,870	1.16
	4170 Total	17,906	1.29	31,459	1.36
	4200 Total	1,600	0.93	1,837	0.99
fiscal 2009 Total		**298,064**	**1.10**	**395,945**	**1.13**
	3480 Total	13,321	1.80	12,117	1.53
	3520 Total	62,000	0.80	75,594	0.90
	3560 Total	14,000	1.10	15,809	0.88
	3610 Total	49,077	1.06	64,854	1.06
	3950 Total	21,088	0.50	16,319	0.85
fiscal 2012 Total		**159,486**	**0.95**	**184,692**	**0.99**
	3415 Total	23,230	0.99	16,955	1.38
	3440 Total	16,827	0.74	15,321	0.70
	3480 Total	77,105	1.02	82,125	1.14
	3510 Total	48,154	0.92	56,011	1.02
	3560 Total	58,798	1.05	68,036	1.02
	3610 Total	51,912	1.01	60,873	1.09
	3660 Total	7,843	0.77	6,447	0.56
fiscal 2013 Total		**283,869**	**0.98**	**305,766**	**1.06**
	3360 Total	1,272	1.27	1,272	1.63
	3390 Total	21,512	0.76	23,074	0.61
	3415 Total	2,671	0.90	8,423	0.74
	3440 Total	24,370	1.19	26,720	1.44
	3480 Total	17,993	0.76	25,657	0.77
	3510 Total	14,663	0.73	18,348	0.82
	3560 Total	23,060	0.82	40,732	0.76
	3575 Total	2,813	0.60	3,206	0.68
	3610 Total	7,057	1.08	5,683	0.82
	3640 Total	2,926	1.01	3,046	0.70
	3660 Total	9,948	0.87	11,607	0.90
	3680 Total	11,459	0.73	11,807	0.67
	3800 Total	1,453	0.50	1,453	0.51
	3950 Total	-	-	2,378	0.53
	3480 W To	45	0.50	45	0.49
	3480A Tot	41,318	1.52	36,836	1.52
	3560A Tot	15,643	0.65	8,500	0.55
	3630 E Tot	2,832	0.50	2,832	0.98
	3650 E Tot	3,443	1.00	3,443	0.79
	A 3480 Tot	11,610	1.65	10,440	2.01
	A 3590/35	2,394	0.50	2,394	0.52
	A 3640/36	1,242	0.60	1,242	0.46
	A 3680 To	938	0.80	938	0.52
	A3480 Tot	3,000	1.65	5,564	1.74
	A3510 Tot	988	1.30	1,379	1.03
	A3590 Tot	3,539	0.59	3,539	0.40
	A3620 Tot	9,539	1.39	9,539	1.50
	A3630 Tot	2,995	0.50	2,995	0.51
	A3640 Tot	601	0.50	601	0.29
	A3650 Tot	5,362	0.73	5,362	0.69
	A3680 Tot	3,716	0.72	3,716	0.88
	A3800 Tot	1,298	0.50	1,298	0.60
fiscal 2014 Total		**251,700**	**1.00**	**284,070**	**0.99**
Grand Total		**1,290,902**	**1.04**	**1,559,633**	**1.05**

Table 12.2 - Tonnage Reconciliation, 2013

	Budget			Forecast			Mill Feed Month End Back Calc.		
Month	tons	Grade	STU's	tons	Grade	STU's	Actual	%WO3	STU
October	32,626	1.23	39,987	33,340	1.19	39,696	33,468	1.44	48,309
November	34,832	1.01	35,291	32,451	0.95	30,747	31,870	0.89	28,231
December	32,288	1.00	32,309	33,180	0.96	31,863	34,558	1.03	35,651
January	35,531	0.96	34,108	33,140	0.94	31,041	34,873	0.99	34,472
February	32,114	0.93	29,797	30,477	0.91	27,627	30,667	1.06	32,495
March	33,155	0.92	30,356	32,970	0.91	30,001	34,600	0.99	34,286
April	33,801	0.94	31,731	32,190	0.99	31,753	32,801	0.95	31,213
May	33,269	0.95	31,676	32,620	1.05	34,178	34,267	1.06	36,386
June	34,632	0.84	28,969	32,723	0.90	29,543	33,692	0.85	28,638
July	32,929	0.72	23,649	33,264	0.96	31,890	34,413	0.85	29,168
August	33,343	0.85	28,203	33,067	0.92	30,422	35,883	0.98	33,296
September	35,440	0.85	30,160	32,483	0.95	30,937	32,862	1.10	36,147
Total	**403,960**	**0.86**	**346,076**	**391,905**	**0.97**	**379,698**	**403,954**	**1.01**	**408,292**

Table 12.3 - Tonnage Reconciliation, 2014

	Budget			Forecast			Mill Feed Month End Back Calc.		
Month	tons	Grade	STU's	tons	Grade	STU's	Actual	%WO3	STU
October	34,796	0.87	30,349	33,367	0.88	29,451	34,951	0.85	29,590
November	33,673	0.91	30,642	34,828	0.88	30,649	33,351	0.8401547	28,020
December	34,796	1.09	37,893	35,147	1.04	36,553	36,292	0.9097322	33,016
January	34,796	1.17	40,711	38,664	1.09	42,264	36,325	1.241872	45,111
February	34,286	0.86	29,829	33,366	0.95	31,698	31,794	1.1312826	35,968
March	41,122	0.85	28,572	37,319	0.97	36,228	34,945	1.26453	44,189
April	41,326	0.78	32,317	37,438	0.88	32,945	35,315	1.0477984	37,003
May	42,704	0.78	33,395	42,528	0.80	33,940	39,405	0.7178277	28,286
June	41,326	0.99	40,913	41,333	0.74	30,586	36,912	0.63	23,161
July	40,500	0.91	36,855	36,780	0.77	28,321	36,342	0.81	29,479
August									
September									
YTD	**379,326**	**0.90**	**341,476**	**370,770**	**0.90**	**332,635**	**355,632**	**0.94**	**333,823**

Due to the mill upgrades that occurred throughout 2014 and are expected to continue for a period beyond the date of this report, historical recoveries in the Mill do not match predicted recoveries for Life of Mine. The authors have no reason not to rely on the metallurgical work covered in Section 13 to validate the predictions used in the report based on Rod William's (Wild River Consulting Group, LLC) and Colin Craft's (NATCL) expertise in the this field.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The Cantung operation is fully equipped with its own mill and metallurgy lab. Testing of the mill feed is continuous with a view to ensuring maximum profitability from the milling material. Currently, most metallurgical testing is completed on site, with some material being shipped offsite for specialist studies. Cantung began production in 1962 and the metallurgical properties of the ore bodies identified to date are believed to be well understood. Drill core and mine production samples are readily available to represent all types, tenors and styles of mineralization available to the mill, allowing on-going metallurgical testing with the aim of continually improving mill recoveries.

As an example, the Cantung concentrator scheelite flotation circuit was upgraded this year to include cavitation style flotation columns engineered to recover the fine grained scheelite which was traditionally lost to tails by the existing mechanical flotation cells. A mineralogical assessment of the process streams conducted in early 2011 indicated flotation tails comprise approximately 70% of the losses as liberated fine grained scheelite. In house metallurgical studies demonstrated the feasibility of producing a fine grain, >70 passing 25um, saleable grade product, 35% WO_3. The cavitation style columns installed this summer are of the type demonstrated to recover fine grain material and, while still in the tuning stage of commissioning, produced concentrator yields greater that 80% total.

14 MINERAL RESOURCE ESTIMATES

14.1 SUMMARY

Inferred and Indicated Mineral Resources for the Cantung Mine, as of July 31, 2014, are listed below in Tables 14.1 and 14.2. These Mineral Resources encompass the resources in the underground mine as well as those for the Open Pit. The Amber Zone is a newly discovered zone since the previous technical report and reported in a NATCL news release, dated February 6, 2012, and titled “North American Tungsten Intersects High Tungsten Grades at Cantung Mine; Confirms Mineralization to the East of the known "Below 3700 Level" Ore body”.

The overall model parameters have not changed significantly from previous reports as the reconciliations show good overall comparisons. Block size for the pit/PUG was decreased as it was realized the previous block size was less than ideal for underground mining.

The remainder of the zones include both historical mining areas containing historical reported resources and areas with new in-fill definition and exploration drilling and modeling. The Indicated and Inferred Mineral Resources for the Cantung mine are detailed in Section 14.2, Table 14.1 and Table 14.2 respectively.

14.2 MINERAL RESOURCES REPORTED

Table 14.1 - Indicated Mineral Resources, July 1, 2014

Zone	Indicated Mineral Resource		
	Tons	Grade (WO_3%)	STU's
Amber	1,199,000	1.09	1,305,000
Below 3700	30,000	1.43	43,000
E Zone	319,000	1.04	331,000
Pit	192,000	0.85	163,000
Pug	1,851,000	0.86	1,587,000
West Extension	208,000	1.25	260,000
Stockpile	41,000	0.77	32,000
Grand Total	**3,839,000**	**0.97**	**3,720,000**

Notes:

1. Mineral Resources conform to CIM and NI 43-101 requirements.
2. Mineral Resources are estimated at a cut-off grade of 0.5% WO_3
3. Mineral Resources are not Mineral Reserves
4. The Indicated Resource includes the Probable Mineral Reserves
5. Numbers may not add up due to rounding

This is an increase of 1,386,000 tons and an increase of 990,000 STU’s since 2010, excluding mining.

Table 14.2 - Inferred Mineral Resources, July 1, 2014

Zone	Inferred Mineral Resource		
	Tons	Grade (WO_3%)	STU's
Amber	730,000	0.7	511,000
Below 3700	140,000	1.0	140,000
E Zone	120,000	0.9	108,000
Pit	-	-	-
Pug	60,000	0.8	48,000
West Extension	140,000	0.8	112,000
Dakota	170,000	0.8	136,000
Grand Total	1,370,000	0.8	1,096,000

Notes:
1. Mineral Resources conform to CIM and NI 43-101 requirements.
2. Mineral Resources are estimated at a cut-off grade of 0.5% WO_3
3. Mineral Resources are not Mineral Reserves
4. Numbers may not add up due to rounding

Mineral Resources where insufficient work has been completed to date to demonstrate economic viability have been excluded in determining the Mineral Reserves. Additional work may demonstrate economic viability for part of these Mineral Resources.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.3 ESTIMATION METHODOLOGY

NATCL mine personnel updated the Mineral Resources estimate for the underground and open pit mine. The update was carried out using commercially available software. Minesight/Compass was used to create a 3D block model of the various ore bodies. Mine openings were generated in AutoCad/Promine and entered into the project. The openings were not included in the calculations per se but were used to limit material remaining to be mined.

Model parameters were as follows:

- The model was oriented parallel to the grid EW and NS directions (i.e. not rotated).
- The various ore lens were modeled honouring lithology and geology, utilizing diamond drill data and underground mapping
    - The Pit/PUG used Ore Limestone, Swiss Cheese Limestone and Scheelite rich Argillite
    - The remainder used Hanging Wall Ore Limestone, Footwall Ore Limestone and Swiss Cheese Limestone as limiting boundaries
- Block size was 5 ft x 5 ft x 5 ft
- 200 ft search radius in all dimensions
- Interpolation honoured geological codes i.e. in most cases the search radius was truncated by geological "lens" codes
- Inverse Distance Squared interpolation
- Capping was not applied.
- Minimum composites for Pit/PUG were 1 and 15 respectively
- Minimum composites for the remainder were 2 and 20 respectively
- Measured category was not calculated
- Indicated code was assigned to blocks within 100 feet of a diamond drill hole
- Inferred code was assigned to blocks greater than 100 feet and less than 200 ft of a diamond drill hole
- A grade shell based on a 0.5% WO3 cut-off was applied
- Zones with narrow intercepts, poor understanding of the geology, limited access, poor ground conditions and/or discordant lenses were manually removed from Indicated and placed into the Inferred category. This process specifically affected all of the Dakota and most of the Swiss Cheese Limestone zones.

The database contains 2,660 diamond drill holes, with 2,278 of them being underground diamond drill holes. In detail, 1271 of these holes were used in the calculation of the Amber, Below 3700, E-Zone and West Extension mineral resources with 830 of them averaging above the 0.5% WO3 cut-off grade. Of these, 217 diamond drill holes were used to calculate the Pit/PUG resource, including 21 holes drilled in 2012.

The database contains 45,000 WO3 assays but only 29,340 copper assays. While copper is calculated (and recovered), it is not reported due to the lack of assays compared to the WO3 database. A total of 19,140 WO3 assays were used in the calculation of the Amber, Below 3700, E-Zone and West Extension mineral resources and 5,079 assays were used in the calculation of the Pit/PUG resource.

All listed Indicated Resources fall within the current fully permitted mine workings and there are no factors other than metal prices to impact this resource.

All listed Inferred Resources with the exception of the Dakota Zone fall within the current fully permitted mine workings and there are no factors other than metal prices to impact this resource.

15 MINERAL RESERVES

15.1 SUMMARY

The Mineral Reserve estimate for the Cantung Mine is provided below in Table 15.1 and illustrated in Figure 15.1, both found in Section 15.2. No Mineral Reserves were estimated for the Open Pit/PUG Zone, pending redesign of the pit, and the separation of the underground PUG Zone from the Open Pit resource blocks. These Reserves are reported separately for each zone.

Mineral Reserves for the underground were estimated by applying extraction and dilution estimates to the in situ Mineral Resources reported in Section 14.2. Dilution was applied in a manner specific to the ore body characteristics, configuration and extraction methods, in accordance with the lengthy experience of mine operation. Dilution was added at zero grade, although there are often modest WO_3 values in the diluting material. Some of the Mineral Resources were deliberately excluded from the Mineral Reserves owing to access or design constraints. These Mineral Resources remain in the inventory as material that may become mineable at some future time.

All Mineral Resources in the underground mine are either in the Indicated or Inferred category. All Mineral Reserves have been calculated in the Probable category. The classification of the Mineral Reserves has been done according to the rules and guidelines set forth in NI 43-101 and CIM best practices guidelines.

15.2 PROBABLE MINERAL RESERVES

Table 15.1 - Probable Mineral Reserves, July 1, 2014

Zone	Probable Mineral Reserve		
	Tons	Grade (WO_3%)	STU's
Amber	442,000	0.85	374,000
Below 3700	44,000	0.87	38,000
E Zone	55,000	0.99	55,000
Pit	190,000	0.77	147,000
Pug	886,000	0.80	707,000
West Extension	159,000	0.77	123,000
Stockpile	41,000	0.77	32,000
Grand Total	1,818,000	0.81	1,476,000

Notes:

1. Mineral Reserves conform to CIM and NI 43-101 requirements.
2. All Mineral Reserves are classified as Probable
3. Mineral Reserves are estimated at a cut-off grade of 0.5% WO_3 (tungsten trioxide).
4. A minimum mining width of 15 feet was used.
5. The Probable Reserve is a subset of the Indicated Mineral Resource.
6. Tons are short tons being 2,000 lbs and STU is Short Ton Unit 20 lbs of WO_3.
7. Numbers may not add up due to rounding.

15.3 DISCUSSION OF MINING AREAS AND RESERVES

In the current life of mine plan, the two major mining areas are:

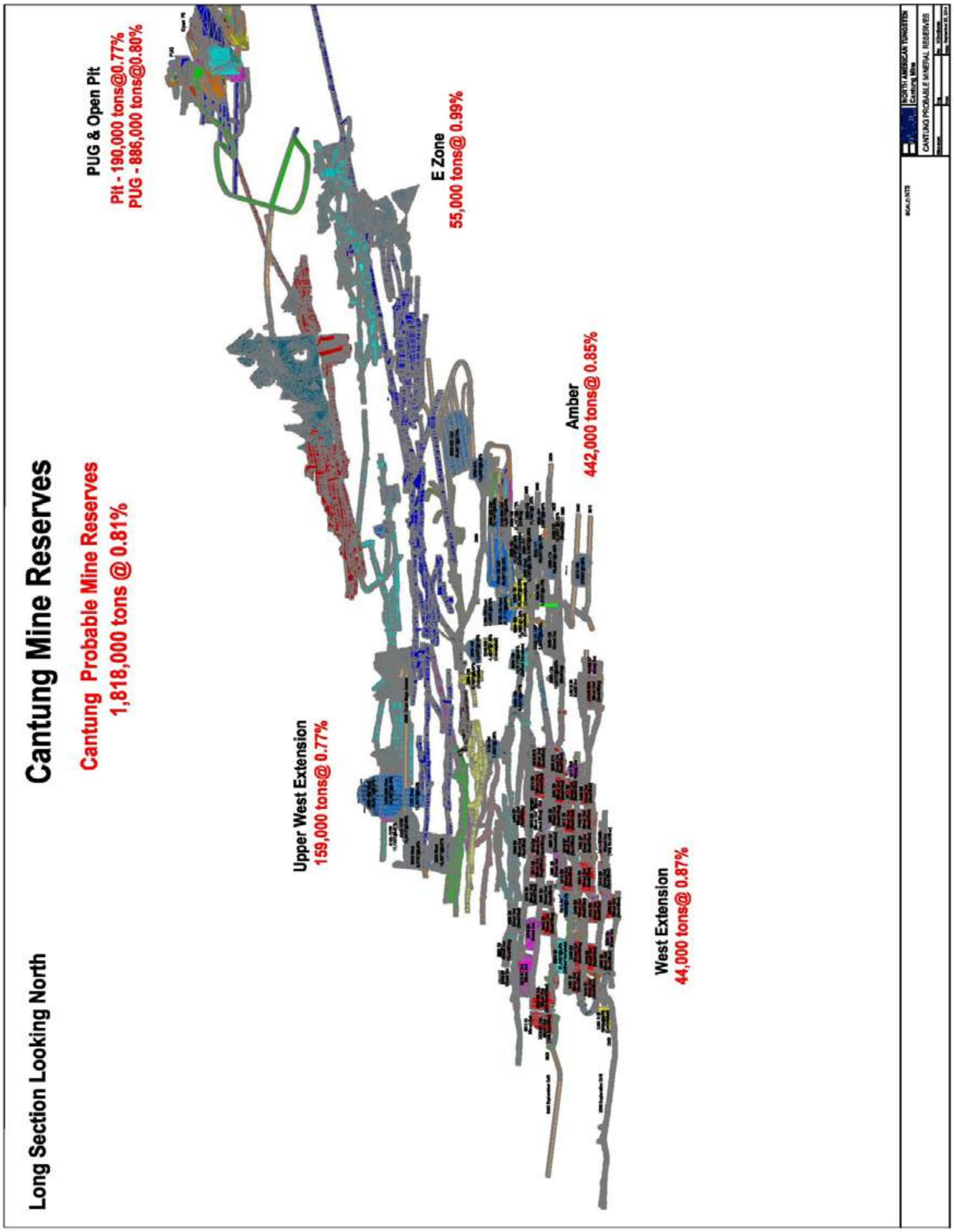
- Amber Zone
- Pit Underground (PUG)

These two mining areas account for 1.328 million tons of the probable mineral reserve. The Amber Zone extends vertically from the 3510 level to the 3800 level of the mine. The PUG zone lies from approximately the 4750 level to the 4960 level.

Another 450,000 tons of mineral reserve ore are included in the current life of mine plan and distributed in the following mining areas:

- Open Pit
- West Extension
- West Extension (below 3700)
- E Zone

Figure 15.1 - Longitudinal Section, Cantung Mine Probable Reserves



The six main mining zones, as shown in Figure 15.1, are described below along with a summary of planned mining methodology.

Amber Area

The Amber Area ore zone is composed of several narrow ore bodies. Refer to Figure 15.2 and Figure 15.3. In the upper part of the Amber zone, the ore is approximately 10 feet true width and is lying at a dip of 20 to 30 degrees. In the lower part of the Amber, the ore body is wider, up to approximately 30 feet width. These ore bodies will be mined independently from each other via separate level accesses driven from the 3680 decline. The Amber ore body, especially in the lower part, is a high value resource. Maximizing mining recovery with minimal external dilution is the priority when selecting the mining method.

Amber indicated resources are an estimated 1,199,000 tons at 1.09%, of which 450,481 tons at 0.87% are designed stopes included in the current life of mine plan reserve.

The ore body has been divided into longitudinal blocks with hanging wall spans limited to 50 feet or less and strike lengths limited to 120 feet or less. These dimensions are estimated to result in a stable hanging wall with minimal sloughing. Between mined blocks, minimum 15 to 20 foot pillars will be designed and the mined out stope void will be filled with waste rock before mining commences on the adjacent block. Mining will be done from the lowest blocks to the highest so that the exposed fill wall is on the low side of the stope. Additionally, 50 foot cable bolts have been installed in areas where the hydraulic radius is potentially large enough to lead to hanging wall instability.

Development is almost complete in the Amber Area, except on the 3780 and 3820 levels. Slot drifts and LH production drilling will be done as mining progresses. Ore production will be delivered from LH Stopes which will be mined concurrently with cut and fill blocks to ensure mill feed at the planned tons and grade.

Figure 15.2 - Amber Area Overview

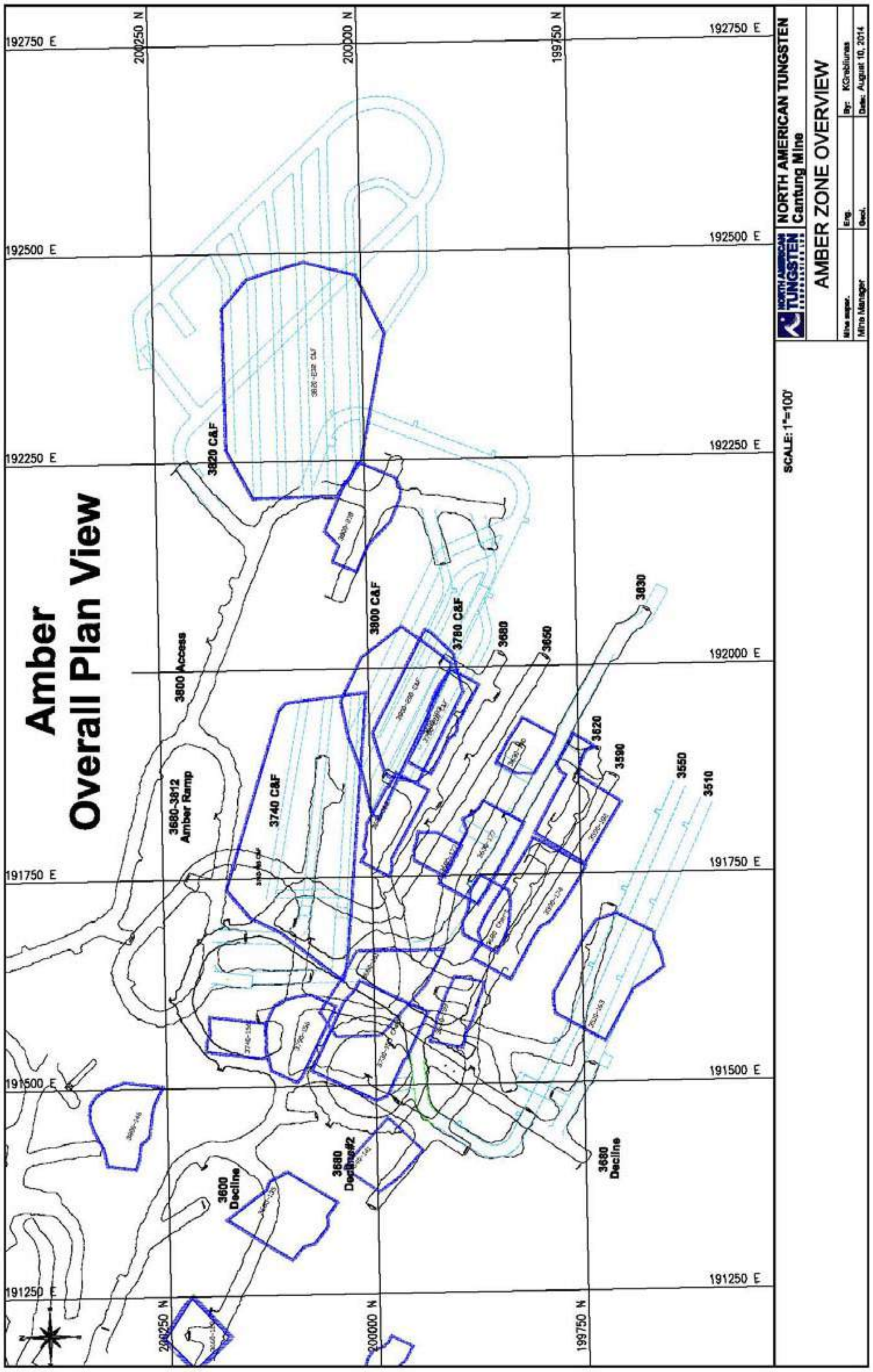
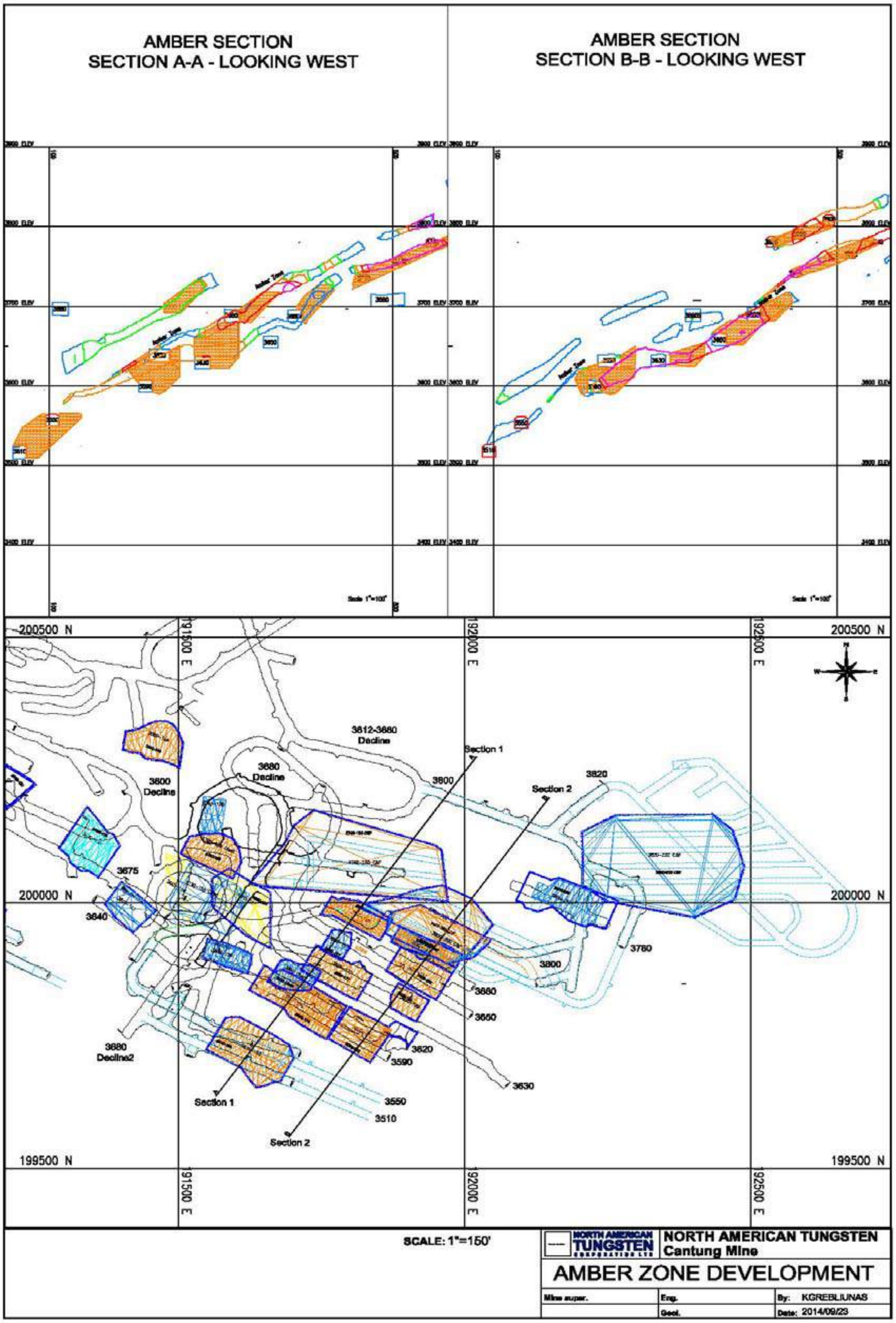


Figure 15.3 - Amber Area Cross Sections



Underground (PUG)

The inability to mine the existing pit highwalls back further into the mountain has resulted in significant mineralization being left in the pit walls and below the pit floor. As a result, it was decided to access this area from underground to mine an estimated reserve of 886,000 tons. Refer to Figure 15.4 and Figure 15.5.

To accommodate year round mining in this area, a 4,000 feet long incline at 15% grade is planned to be driven from the 4200 level to the PUG ore body. This development was initially started in 2011 and then put on hold for two years. The PUG ramp was restarted in June 2014 and is planned for completion by April 2016. Mining of the PUG stopes will commence in October 2016.

Mining will create breakthroughs in the pit floor and in the pit wall. Permanent Pillars will be left in some areas to provide stability and no cemented fill is required in the mine plan.

Ground conditions are expected to be good to very good and ground stress minimal due to the proximity to surface. ITH longhole drilling and haulage to the #2 orepass with large capacity trucks is planned to ensure maximum efficiency.

The indicated resources for this area are 1,851,000 tons with a grade of 0.86% WO_3 and the PUG designed stopes contain a total of 886,000 tons with an average grade of 0.80 % WO_3.

Figure 15.4 - PUG Planned Incline

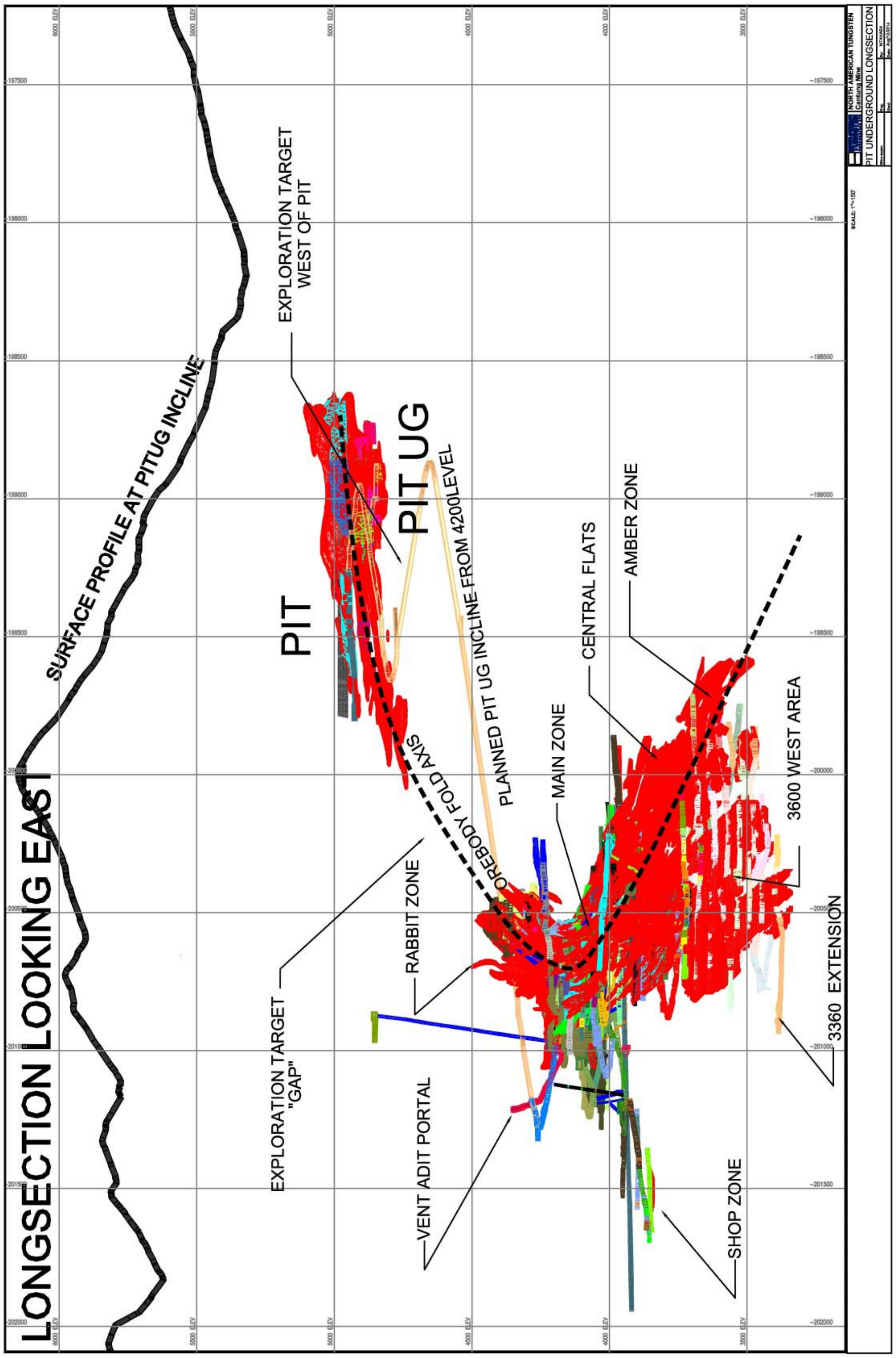
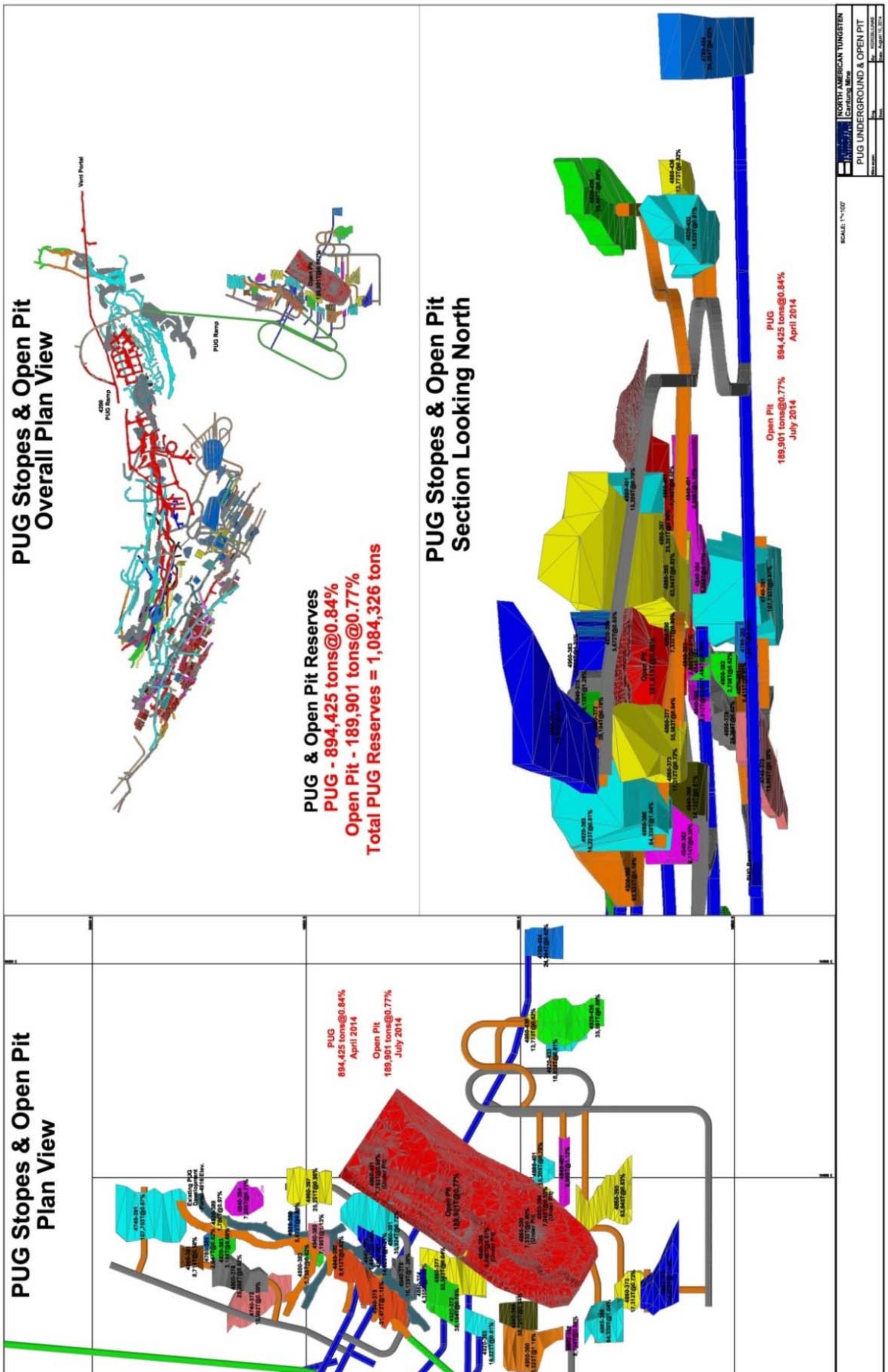


Figure 15.5 - Pit Underground Overview

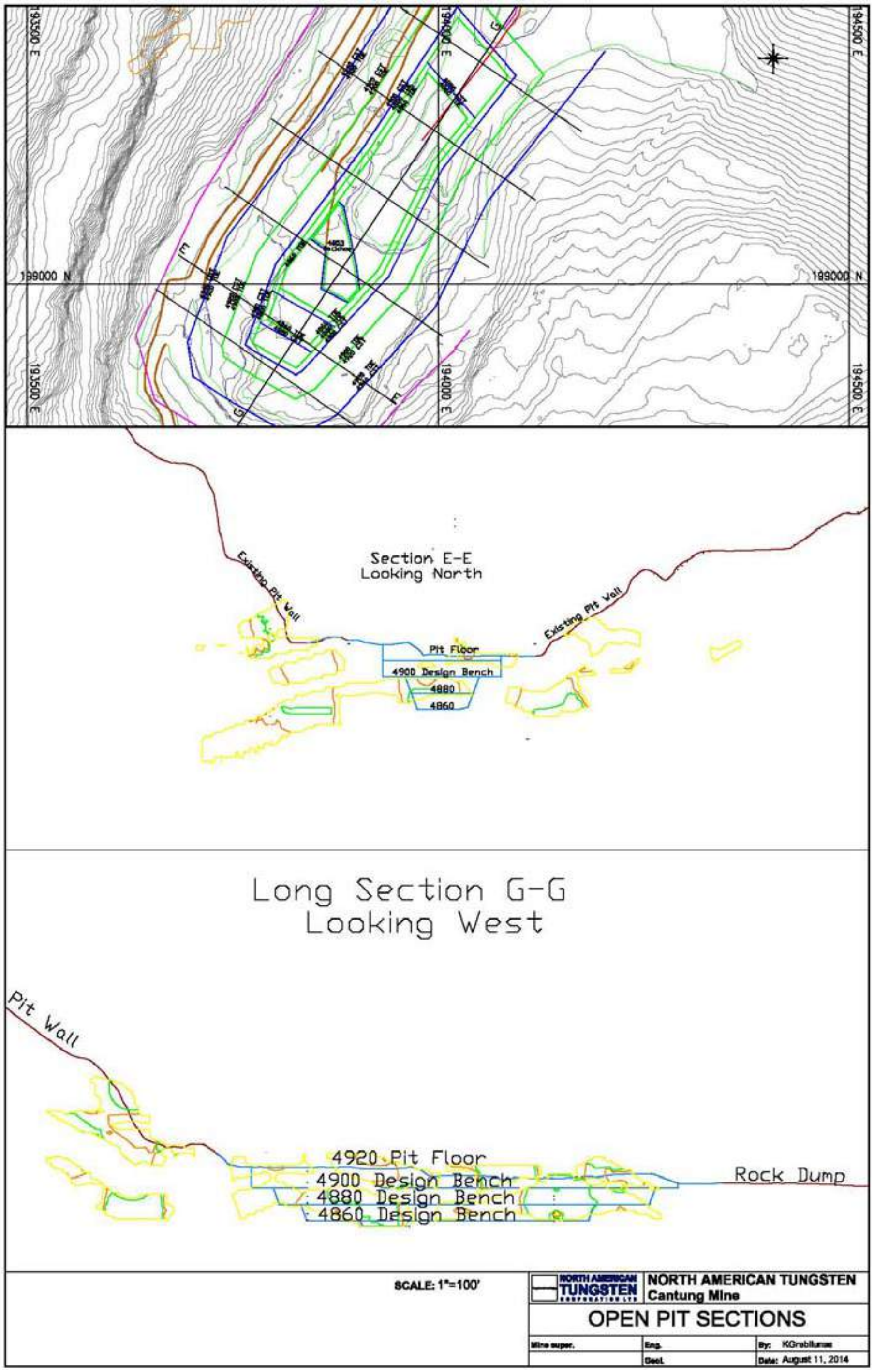


Open Pit

The original open pit ore body, with high grade mineralization in limestone, was mined from 1962 to 1973. The historic open pit is located west of the main underground ore body at the 4900 elevation. Access to the Open Pit is limited to the summer months due to the high mountain location and the winter avalanche risk. The remaining ore is the lower grade mineralization in the underlying chert.

The current designed pit benches are hosted in the chert, and we have a 2-year plan to strip approximate 141,000 short tons of waste and mine 190,000 short tons of ore at an average grade of 0.77% WO3. Mining in the Open Pit occurs from late June or July and finishes at the end of September depending on weather conditions. The mining of the open pit is undertaken by a contractor, Jedway, to ensure planned production is achieved. Refer to Figure 15.6.

Figure 15.6 - Open Pit Mine Plan and Typical Sections



E-Zone

The E-Zone consists of rib, sill and replacement pillars of varying size left in the historic central portion of the mined out E-zone. Some additional sill pillars in the West Extension remaining from the more recent cut and fill mining are also included. The reserve for this area is 55,376 tons at 0.99% WO3. It includes the remaining reserves in the South Flats, near the main portal. This mining area, as well as some small stopes on 3950 level, will be left until the end of mine life due to its proximity to the main access.

The development required for accessing these stopes is minimal and being in the immediate proximity of the main entrance requires only a short haul to the mill. The stopes situated in this area will be mostly mined during the 2016 fiscal year and the voids are to be filled with waste rock.

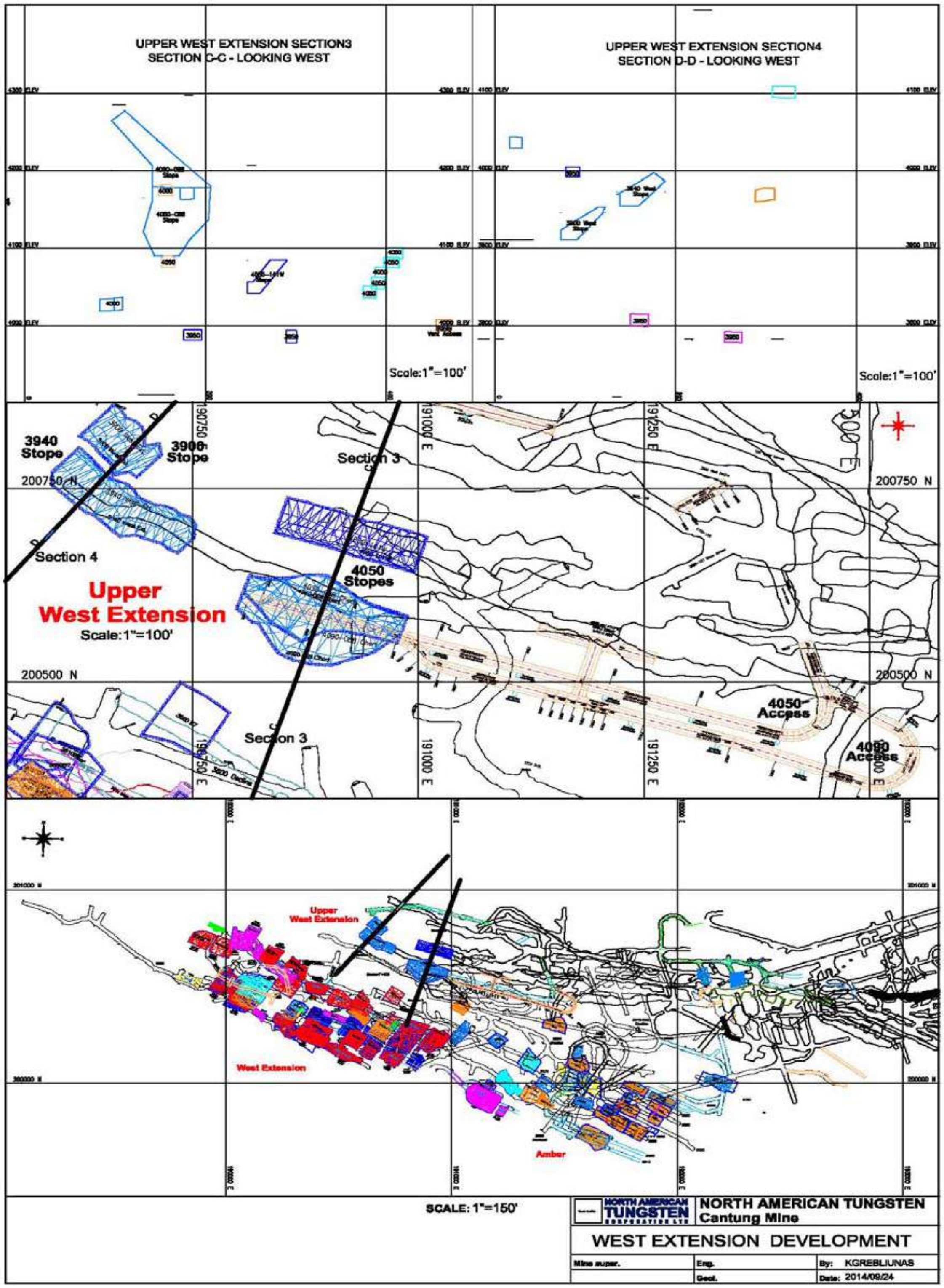
Upper West Extension

Most of the West Extension above 3700 level has already been mined. A few new chert stopes located at the west limit of the ore body were designed to be mined on 3950, 4050 and 4090 levels. These areas were previously considered uneconomic due to the low ore grade. Refer to Figure 15.7.

West Extension

Below 3700 level, the ore body is approximately 30 feet true width lying at a dip of 30 to 45 degrees. Levels have been accessed independently from the 3600 decline. Almost all the stopes have been mined and voids filled with waste rock. This area has provided almost the entire mine production in the last several years and the small reserves left will be mined before December 2014. Exploration drilling will continue on the 3360 and 3520 level towards the west.

Figure 15.7 - West Extension Development



16 MINING METHODS

16.1 SUMMARY OF MINING METHODS

In the recent past, the Cantung mine used longhole stoping, room and pillar, and cut and fill as the main mining methods.

The original ore body (E zone) was accessed by an adit and a decline for traffic flow and ventilation purposes. The West Extension, West Extension Below 3700 and Amber Zone are accessed by a series of declines driven from the 3950 Level adit. This adit is the main equipment traffic road and ventilation exhaust air way. The 4450 level decline is the fresh air ventilation intake and is not used for vehicle traffic. The current main development size is 15ft x 15ft. The drifts are drilled using a two boom jumbo and ground support is installed with a MacLean Bolter or scissor deck.

Longhole stoping is currently the primary mining method used at Cantung in the West Extension area (below 3700) and the Amber Zone. This method is simple, reliable and has a relative lower cost. The majority of the remaining reserve at the Cantung mine will continue to be mined using longhole stoping to meet the current targeted mill feed rate. A small area of the reserve has a shallow dip angle and, depending on the thickness of the mineralization zone, cut and fill or room and pillar will be used to reduce dilution and increase ore recovery.

The longhole stopes are drilled by a Stopemaster drill using 3-inch holes and an ITH (Cubex) drill using 4.5-inch holes. The ore is mucked from the bottom of the stope using an 8 cubic yard remote-controlled LHD (scooptram). The ore is hauled out by 30 and 45 ton trucks. The ore body is divided by levels, with a vertical elevation difference of 40 to 60 feet. Longhole stopes are designed to the maximum extent based on good geotechnical engineering practices. The pillars separating these stopes are generally designed to be in the low grade areas of the mineral resource and are typically 15 to 20 feet wide. These rib pillars are usually not recoverable, as the mine uses unconsolidated waste as backfill. The stopes are sequenced to retreat out of each level to the level access connected to the main ramp.

For the areas where the ore thickness is 6-10 feet, with dip angles ranging from 20-30 degrees, a variation of the room and pillar method will be used in combination with cut and fill stoping. With this method the ore is recovered in horizontal drifts starting from the bottom and advancing upward. The void is backfilled to allow for mining of the next level. Level drifts will be driven 13 feet x 13 feet and mining will be completed using existing equipment.

The internal dilution of longhole stopes varies dramatically depending on the number of lenses included in each individual stope and the thickness of the waste between the lenses. Internal dilution is designed using a cut-off stope grade of 0.5% W03. To increase the grade of a stope above the cut-off grade, one ore lens can be dropped or the stope can be divided into two stopes. An external dilution of 10%, due to overbreak or caving, is used at Cantung. Depending on the shape and inclination of the ore body footwall total dilution, including both planned internal dilution and external dilution, averages 61%. The Amber zone dilution ranges from 40% to over 100% in some stopes.

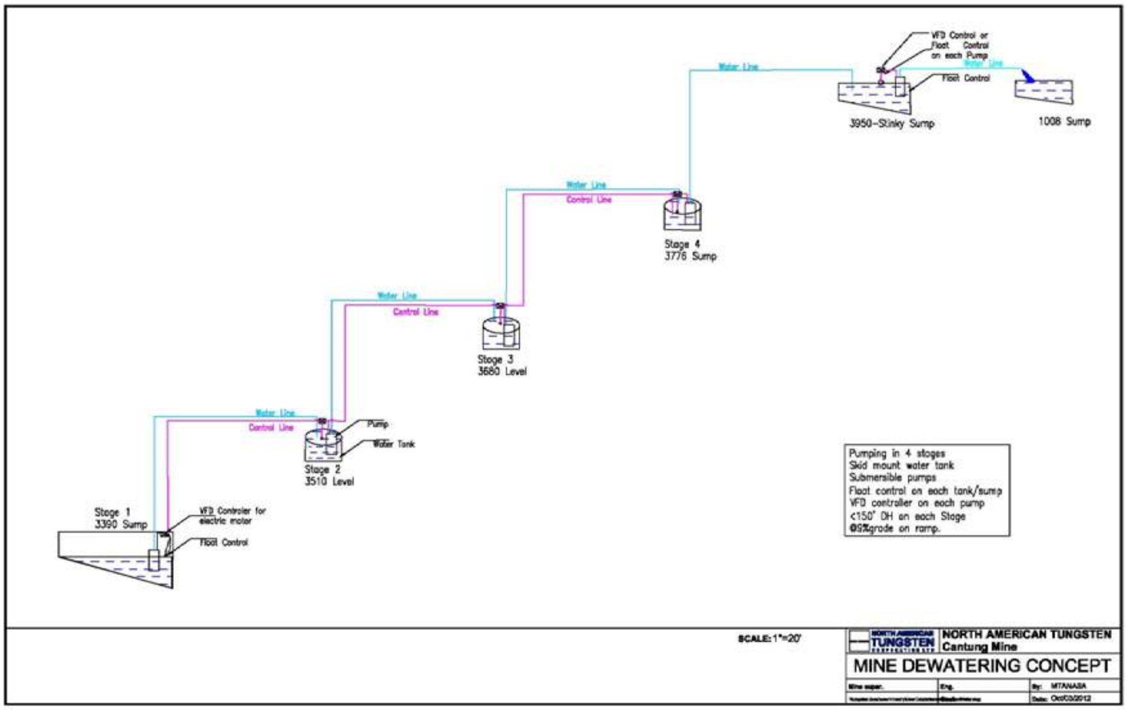
16.2 DEWATERING

The underground portion of the Cantung mine is relatively dry, due to the mine elevation and relatively low permeability of the rocks hosting the ore body. The amount of water inflow is increasing with depth; however, based on the actual location and history of the mine, we are unlikely to encounter a large influx of water. Approximately half of the mine water is recycled and used in the mining process.

The main mine dewatering system at Cantung consists of four Eliminator pumps using 6-inch internal diameter pipeline and reservoirs to pump in stages. The spare dewatering system uses a Tsurumi high pressure pump and 4-inch inside diameter line. An average of

390 US gallons per minute was recorded with the mine discharge flowmeter in the past year. Refer to Figure 16.1.

Figure 16.1 - Cantung Mine Dewatering System

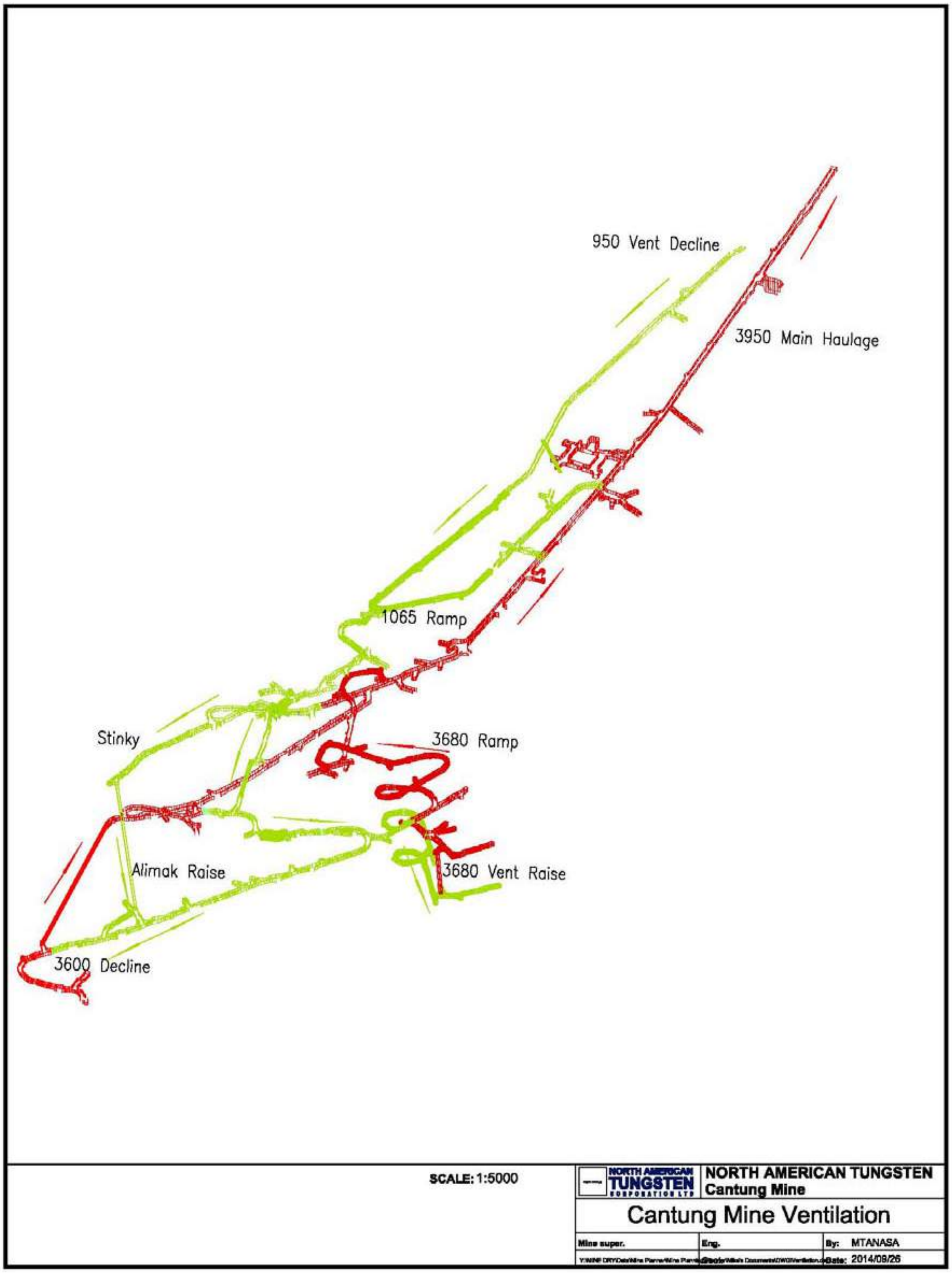


16.3 VENTILATION

The fresh air intake to the underground mine is provided by two 200HP high volume fans installed at the ventilation portal situated at an elevation of 4,450 feet. The fresh air flow of 230,000 CFM is delivered through two ventilation raises to the bottom of the mine. The main ventilation raise to the Western Extension Below 3700 draws fresh air from the 4450 ventilation decline using two 200HP high pressure fans. The levels are ventilated using auxiliary fans and vent ducting, with ventilation doors and regulators strategically placed to prevent short circuits and regulate the air flow. Prior to the Amber zone ramp being developed the air exhaust way was the West decline and 3950 haulage way.

Currently and in future mining, the exhaust air is discharged through the Amber zone, namely the 3812 decline and the 3950 haulage way. Refer to Figure 16.2.

Figure 16.2 - Cantung Mine Ventilation Map



16.4 GROUND CONDITIONS AND ROCK MECHANICS

Rock mechanics and ground stability studies are integral to development and mining methodologies. Extensive underground structural joint set mapping was carried out in 2006, 2007, 2011 and 2012. Together with 138 sets of underground mapping and related data from 2002, a total of 2,937 sets were analysed to identify the dominant structural orientations that control stability throughout the Cantung mine. Rock mass classification and numeric modelling parameters are utilized in the determination of optimal stope dimensions and ground support requirements.

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16.4.1 Joint Sets

A total of 2,937 sets of structure mapping data were input into the stereonet program DIPS TM@ Rocscience. The data was adjusted by the magnetic declination before input into DIPS. The dip direction was adjusted by 27 degrees in the 2002 data and 23 degrees for the rest of the data, as the magnetic declination changes yearly. The ore body shape is complicated, and frequently the argillite, ore limestone and chert have changed orientation dramatically. It can be difficult to identify the exact sedimentary unit from the suite of sedimentary rocks. The granite unit, conversely, is clearly identifiable. The analysis is then carried out using two rock units: sedimentary host rock and granite.

In the sedimentary host rock we identify 4 major and 4 minor joint sets. These joint sets are listed in Table 16.1 and illustrated with the contoured pole plots in Figure 16.3 (major joint sets) and Figure 16.4 (minor joint sets).

Table 16.1 - Joint Sets in the Sedimentary Unit

	DIP	DIP Direction	comments
Joint set 1	41	205	Major
Joint set 2	83	298	Major
Joint set 3	70	331	Major
Joint set 4	75	114	Major
Joint set 5	77	273	Minor, could be conjugated joint set of Major Joint set 2
Joint set 6	76	161	Minor, could be conjugated joint set of Major Joint set 3
Joint set 7	79	074	Minor
Joint set 8	21	347	Minor

In the granite, two major joint sets were identified. These two sets are actually the same joint set family, as the major joint sets 3 and 4 respectively in the sedimentary host rock. Table 16.2 lists the two joint sets in granite.

Table 16.2 - Major Joint Sets in Granite

	DIP	DIP Direction	comments
Joint set 1	69	338	Major
Joint set 2	90	113	Major

As bedding is present everywhere in the sedimentary host rock, this feature is studied separately. Although various bedding dip angles were recorded, the data processing results show good pole concentration. Two major bedding sets were identified by DIPS. Refer to Table 16.3 and Figure 16.5.

Table 16.3 - Bedding Angles in the Sedimentary Unit

	DIP	DIP Direction	comments
Joint set 1	39	205	Major
Joint set 2	09	110	Major

Figure 16.3 - Major Joint Sets in the Sedimentary Unit

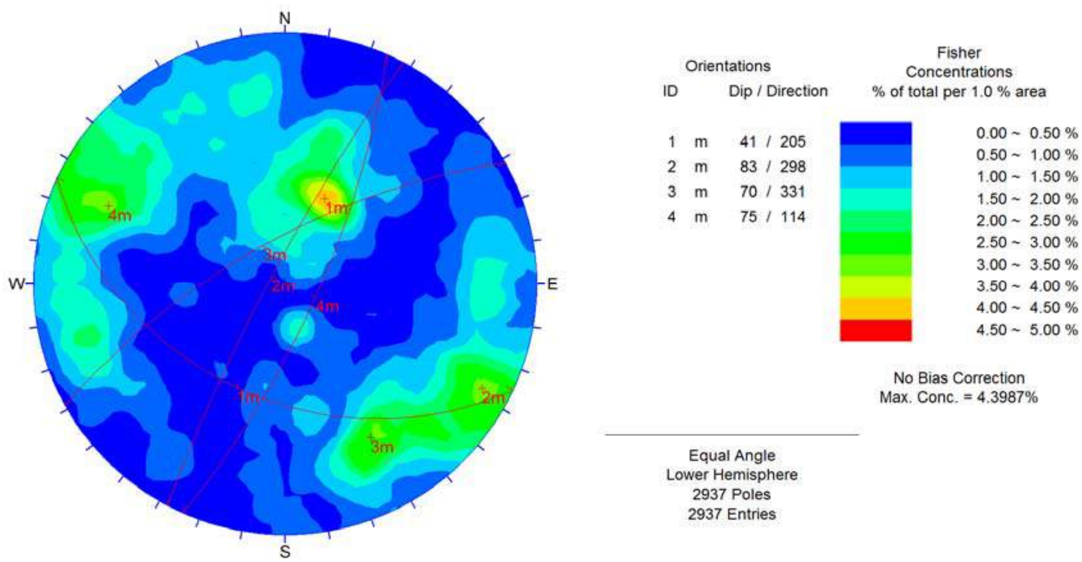


Figure 16.4 - Minor Joint Sets in the Sedimentary Unit

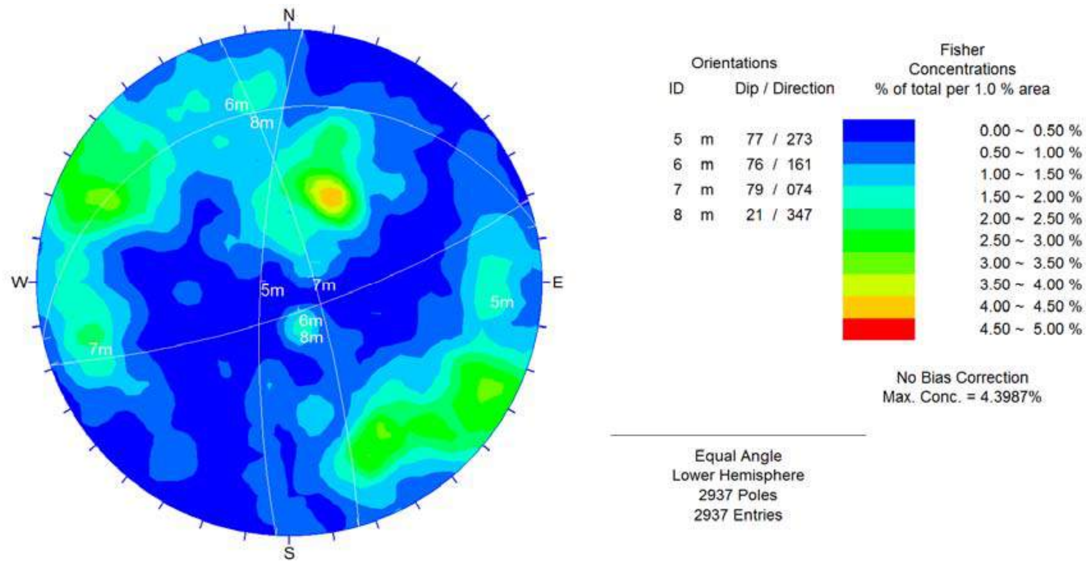
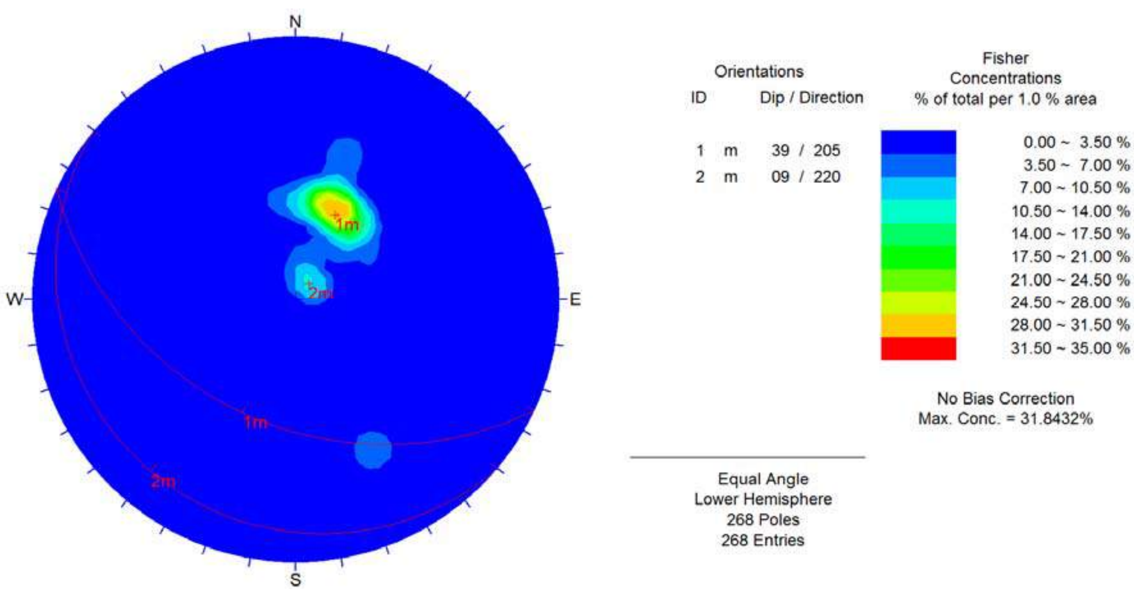


Figure 16.5 - Major Bedding Angles in the Sedimentary Unit



16.4.2 Intact Rock Strength

JSEA (May 28, 1979) conducted rock strength testing on samples of core collected from all of the various rock types in the mine except the granites. The statistical values are shown in Table 16.4. This data shows that Poisson's ratio, Young's modulus and unconfined compressive strength all have significant variation. The JSEA (May 28, 1979) report cannot be located at the site; however Amec (2002) concluded that "Young's modulus is highly variable and is shown to decrease significantly from east to west and also up-dip. The uniaxial strength of skarn also shows similar trends." As more 3D numeric modelling is carried out at Cantung, some triaxial rock strength lab testing is suggested to determine the rock strength factor, e.g. Hoek-Brown parameter m_i.

Table 16.4 - Rock Strength Test Results

Rock Type	Number of samples	Poisson's Ratio	Young Modulus (GPa)	UCS (Mpa)
Ore Limestone	43	0.21±0.05	48±18	159±66
Massive Sulphide	20	0.22±0.04	52±14	137±83
Argillite	22	0.22±0.04	64±24	239±108
Chert (Swiss Cheese Limestone)	24	0.22±0.05	73±38	233±100
Skarn	14	0.19±0.06	48±14	182±98

16.4.3 In Situ Rock Stress

JSEA (1981) carried out the in situ measurements. Amec (2002) made some important comments on the test results during their Rock Mechanics Audit at the Cantung mine. The following is copied from an Amec (2002) report:

> "The Cantung Mine is located near the western edge of the Mid-Plate Stress Province (Adams and Bell, 1990). The principle stress direction in this stress province has been determined mainly from borehole breakout observations and

hydraulic fracturing tests in the Western Canada Sedimentary Basin. The mid-Plate Stress Province is characterized by uniform NE-SW compression."

In the early 1980's, concern was expressed that large HW-FW deformations in the pillars at Cantung were not accompanied by significant stress relief in the chert hanging wall. Due to the very hard brittle nature of the chert, it was thought to be accumulating a large amount of strain energy. For this reason, in situ stress measurements were carried out at Cantung mine in 1981 (JESA, 1981). The door stopper technique was used to determine the in situ state of stress in relatively virgin ground on the 3950 Level to the south and west of the active mining area. From this single test, using three orthogonal drill holes, the principle stress were calculated as shown in Table 16.5.

Table 16.5 - Principal Stress Measurements From Overcoring

Principal Stress	Azimuth	Plunge	Magnitude (Mpa)	Gradient1
Major Principal Stress (σ1)	S87W	-48E	64	$5.9\sigma_v{}^2$
Intermediate Principal Stress (σ2)	N01E	+2N	45	$4.2\sigma_v$
Minor Principal Stress (σ3)	S89E	+37E	26	$2.4\sigma_v$

Notes:

1. These values were measured at Site 1 and are not to be used to estimate the stress level elsewhere in the mine.
2. σ_v is the weight of the overlying rock.

Three additional tests were carried out in the mining area to assess the changes in stress due to mining.

Upon first inspection, the in situ stress appears to be extremely high. Based on the depth of ground cover above the area test (about 400 m), a minor principal stress estimated from the weight of overburden would be 0.027 MPa/per meter depth or 11 MPa. The measured minor principal stress is about 2.4 times this value. It should be recognized that reliance on a single measurement is not recommended since the state of stress can exhibit significant variation with respect to location, proximity to faults, changes in geology and overlying topography. Overcoring site 1 was very close to No.3 Fault, which likely contributed to local stress condition.

JESA reports that they have confidence in the results, noting the general consistency of both the overcoring strain measurements and laboratory strength testing of overcored specimens. However, Amec does not consider that the magnitudes derived from these tests can be used to make estimates about stress level in other parts of the mine. It is noted that at Site 4 which is also relatively unaffected by mining, the major stress parallel to the chert fold axis was 15.4 MPa or about 2 σ_v.

The direction of the major principal stress is almost E-W (mine grid). Adjusting for the rotation of the mine grid relative to true north, the major stress direction relative to true north is ENE, which is roughly consistent with the known regional major stress direction. NE-SW compression is also the direction of compression needed to form the NW-SE trending mountain ridge and Valley of the Nahanni Range.

The major stress direction is roughly parallel to the strike of the chert fold axis. Longitudinal sections of the mine also show that there was subsequent secondary folding produced by compression in EW direction. It is presumed that this secondary folding was the last geologically significant event.

The topography of the Cantung mine area changes dramatically and the effect of the in situ principle stress at different locations is significant. In order to better study the effect of mountain topography on the local mine principal stress Cantung Mine rock mechanic engineers have built a MAP 3D model. Refer to Figure 16.6. In this model the whole mountain above the mine is included. The mine openings in this model are only for geometric reference and do not factor in the calculation. The difference between mine grid north and true north used in the model was 18’45”. In this model, the in situ stress calculation for the mine area assumes that the minor principal stress is the vertical stress

σ_v caused by the weight of the overlying rock. The major principal stress is 2 times σ_v in the E-W direction and the intermediate principal stress is 1.5 times σ_v. The peak of the mountain above the mine (approximately 6,700 feet elevation) is set as the datum.

On the section approximately parallel to the mountain ridge, the minor principal stress is consistent with elevation. Refer to Figure 16.7. On the section perpendicular to the mountain ridge, the minor principal stress decreases as the mountain side slopes down (from SW to NE) while the, ratio $\sigma 1:\sigma 3$ increases. Two points are plotted in Figure 16.8, showing the stress values at an elevation of 3,950 feet. At the same elevation, the minor principal stress changes from 18.00MPa to 14.13MPa and the ratio of $\sigma 1:\sigma 3$ changes from 2.23 to 2.77. This modelling result roughly fits the JSEA in situ stress measurement result 15.4MPa at site 4 on 3,950 level.

This model provides a reliable and easy tool to calculate the in-situ stress at different locations in the mine. The West Extension Zone Below 3700 Level and the Amber Zone were studied by Cantung Engineering. As the West Extension Zone Below 3700 has a small strike angle difference with the mountain ridge (close to parallel), the ratio $\sigma 1:\sigma 3$ remains relatively constant as 2.28 to 2.38 from west to east. The Amber Zone strike angle varies more relative to the mountain ridge, thus the $\sigma 1:\sigma 3$ ratio changes from 2.5 to 2.8 west to east. Table 16.6 and Figure 16.8 illustrate these parameters for the two locations on the 3,950 level.

Table 16.6 - In Situ Stress at Two Underground Locations, 3,950 feet Elevation

	West Extension Below 3700 Zone	Amber Zone
σ1 (Mpa)	40.2	41.0
σ2 (Mpa)	30.8	30.5
σ3 (Mpa)	17.6	15.2
σ1 trend	108◦	106◦
σ1 Plunge	6◦	7◦
σ3 trend	227◦	225◦
σ1:σ2 :σ3	2.3:1.8:1.0	2.7:2.0:1.0

Figure 16.6 - Map3D Whole Mountain Model

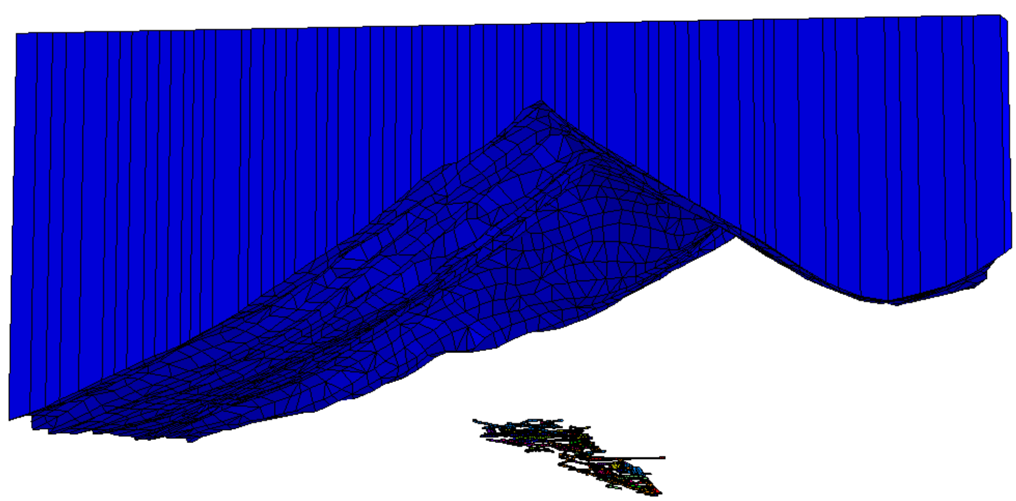


Figure 16.7 - Minor Principal Stress Values Parallel to the Mountain Ridge

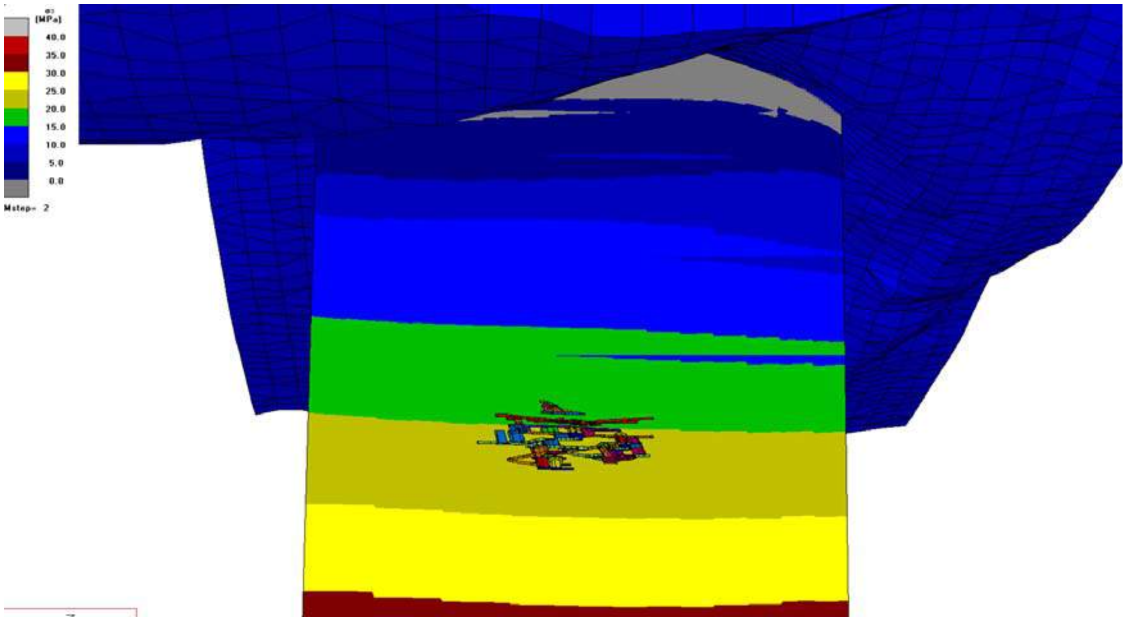
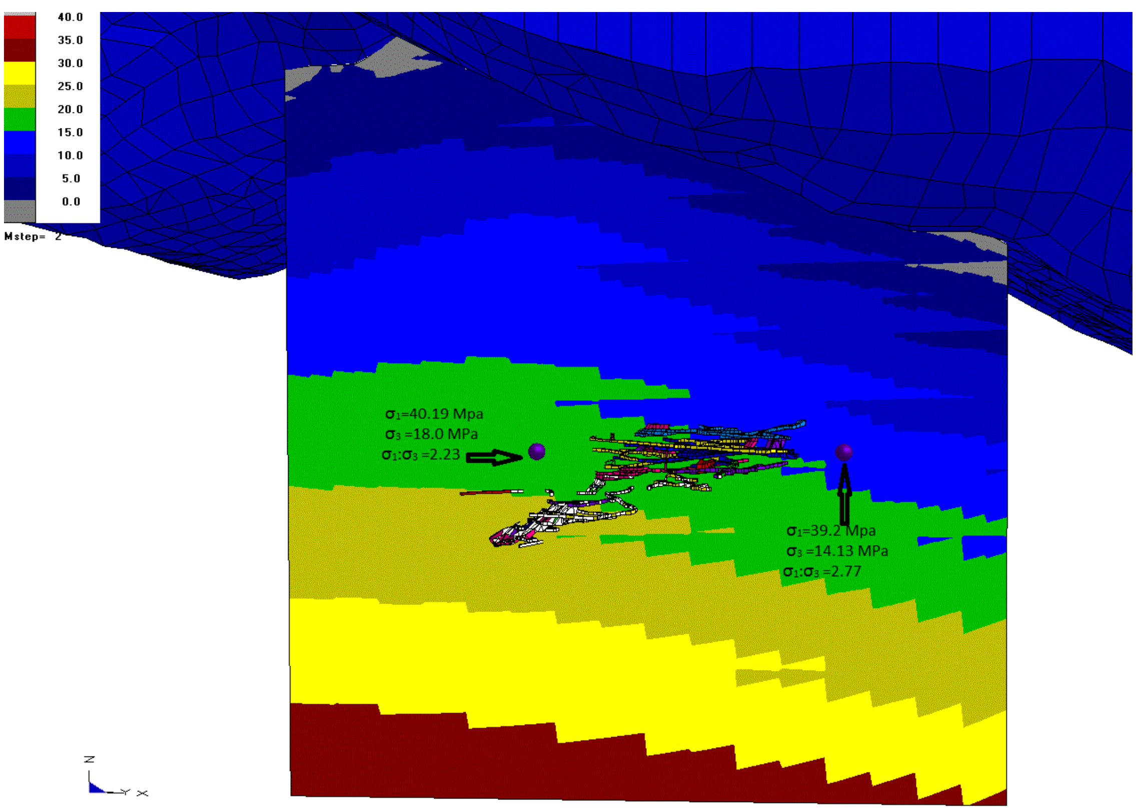


Figure 16.8 - Principal Stress Values & Ratios Perpendicular to the Mountain Ridge



16.4.4 Rock Mass Classification

Rock mass classification is a useful tool to characterise different rock types. Certain empirical methods to evaluate the underground opening stability parameters such as span, stope dilution and pillar strength are based on rock mass classifications, which are well known and widely applied in the mining industry. Rock mass rating (RMR) and Rock Tunnelling Index (Q) systems are commonly used in Canadian mines.

16.4.4(i) RMR

RMR was developed by Bieniawski (1973) and updated in 1976 and 1989. Many empirical methods are based on the 1976 version. RMR 1976 calculations are based on the following 5 parameters:

Strength of intact rock;
Rock Quality Designation (RQD);
Spacing of joints;
Condition of joints; and
Ground water condition.
RMR numbers relate to rock quality is listed in Table 16.7.

Table 16.7 - RMR Relationship to Rock Quality

RMR: 81-100	-RMR: 61-80	RMR: 41-60	RMR: 21-40	RMR: 0-20
I - Very good rock	II - Good rock	III - Fair rock	IV - Poor rock	IV – Very Poor rock

In the rock mechanic audit completed by Amec in 2002, the average RMR for chert and limestone/massive sulphide was determined to be 82 and 70 respectively. Cantung engineering has also evaluated the West Extension drifts below 3700 level. All the evaluations in the appendix are carried out in the ore limestone/massive sulphide production drifts. The analysis is summarized in Table 16.8. RMR in granite was also evaluated by Cantung engineering at an average of 85. All field analysis indicates the Rock Quality ranges from good to very good in the West Extension at the mine.

Table 16.8 - Average RMR, West Extension Below 3700 Zone

Level	RMR
3440	79
3440	79
3440	70
3480 East	76
3480 East	76
3480 East	76
3480 West	67
3480 West	54
3480 West	76
3480 West	74
3480 West	71
3510	82
3510	69
3520	62
3520	71
3520	76
3520	74
3520	65
3520	68
3560	73
3560	71
3560	74
3560	76
Average	72

16.4.4(ii) Barton's Tunnelling Quality Index Q

Using Barton's Tunnelling Quality Index (Q) and the classification parameters based on the block size, shear strength, and active stress. Barton's Q is defined as:

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

Where:

RQD/Jn = Block Size

Jr/Ja = Inter-block Strength

Jw/SRF = Active Stress

RQD = Rock Quality Designation

Jn = Joint Set Number

Jr = Joint Roughness Number

Ja = Joint Alteration Number

Jw = Joint Water Reduction Factor

SRF = Stress Reduction Factor

The Q index is applied at Cantung only to evaluate the stope stability using the Modified Stability Graph method by Potvin (1988) and updated by Nickson (1992).
This method applies Q' which is the modified Q Tunnelling Quality Index and defined as:

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

Cantung engineering has calculated Q' as shown in Table 16.9:

Table 16.9 - Q' Values at the Cantung Mine

	RQD	Jn	Jr	Ja	Q'
Range	60-90	4-9	3	1-2	10-45
Upper	90	4	3	1	68
Average	85	6	3	2	21
Lower	60	9	3	3	7

* Joint set: 2 , joint roughness: wave and rough, joint alteration: clean
* Joint set: 2 plus random, joint roughness: wave and rough, joint alteration: slightly altered
* Joint set: 3, joint roughness: wave and rough, joint alteration: Silty, or sandy-clay coatings

16.4.5 Long Hole Stope Size Design

The primary stoping method chosen for the West Extension Zone below the 3700 level is a typical longhole open stope method. The majority of the Amber Zone will be mined with typical longhole stopes. The Modified Stability Graph method by Potvin (1988) and updated by Nickson (1992) is applied to assist in the determination of the optimal longhole stope dimensions.

The design procedure is based upon the calculation of two factors, N', the modified stability number which represents the ability of the rock mass to stand up under a given stress condition, and S, the shape factor or hydraulic radius which accounts for the stope size and shape. The stability number, N', is defined as:

$N' = Q' \times A \times B \times C$

where

Q' is the modified Q Tunnelling Quality Index

A is the rock stress factor

B is the joint orientation adjustment factor

C is the gravity adjustment factor

Using the appropriate Q’ calculations, representative stability numbers (N') are shown in Table 16.10.

Table 16.10 - Stability Number (N') Calculation

DETERMINATION OF FACTORS A, B, C and N'						
		Hanging Wall	Foot Wall	BACK	E Wall	W Wall
A	UCS (MPa)	250	250	140	140	140
	Sig 1 (MPa)	25	25	50	50	50
	UCS / Sig 1	10	10.0	2.8	2.8	2.8
	Factor A	1	1.00	0.19	0.19	0.19
B	Difference in Dip	30	30	45	30	30
	Difference in Strike	0	0	0	0	0
	Factor B	0.20	0.20	0.50	0.20	0.20
C	Surface Angle	50	50	0	90	90
	Angle (sliding)		45			
	Angle (falling)	50		30	90	90
	Factor C	4.1	6.5	2.8	8.0	8.0
Q' upper		68	68	68	68	68
N' upper		56	88	18	21	21
Q' average		21	21	21	21	21
N' average		17	27	6	6	6
Q' Lower		7	7	7	7	7
N' Lower		6	9	2	2	2

In this calculation the hanging wall is chert, the footwall is granite and back/E Wall/W Wall is massive sulphide. UCS is obtained from JESA rock lab test. The sig1 is taken from Cantung existing MAP3D stress numerical modelling. As the hanging wall and back of the stope is critical, their graphs have been analyzed in Figure 16.9 and Figure 16.10. Designing with average parameters, the hanging wall has a maximum safe hydraulic radius of 7.5 meters and the back has a hydraulic radius of 4.9 meters.

Figure 16.9 - Hanging Wall Stability Graph

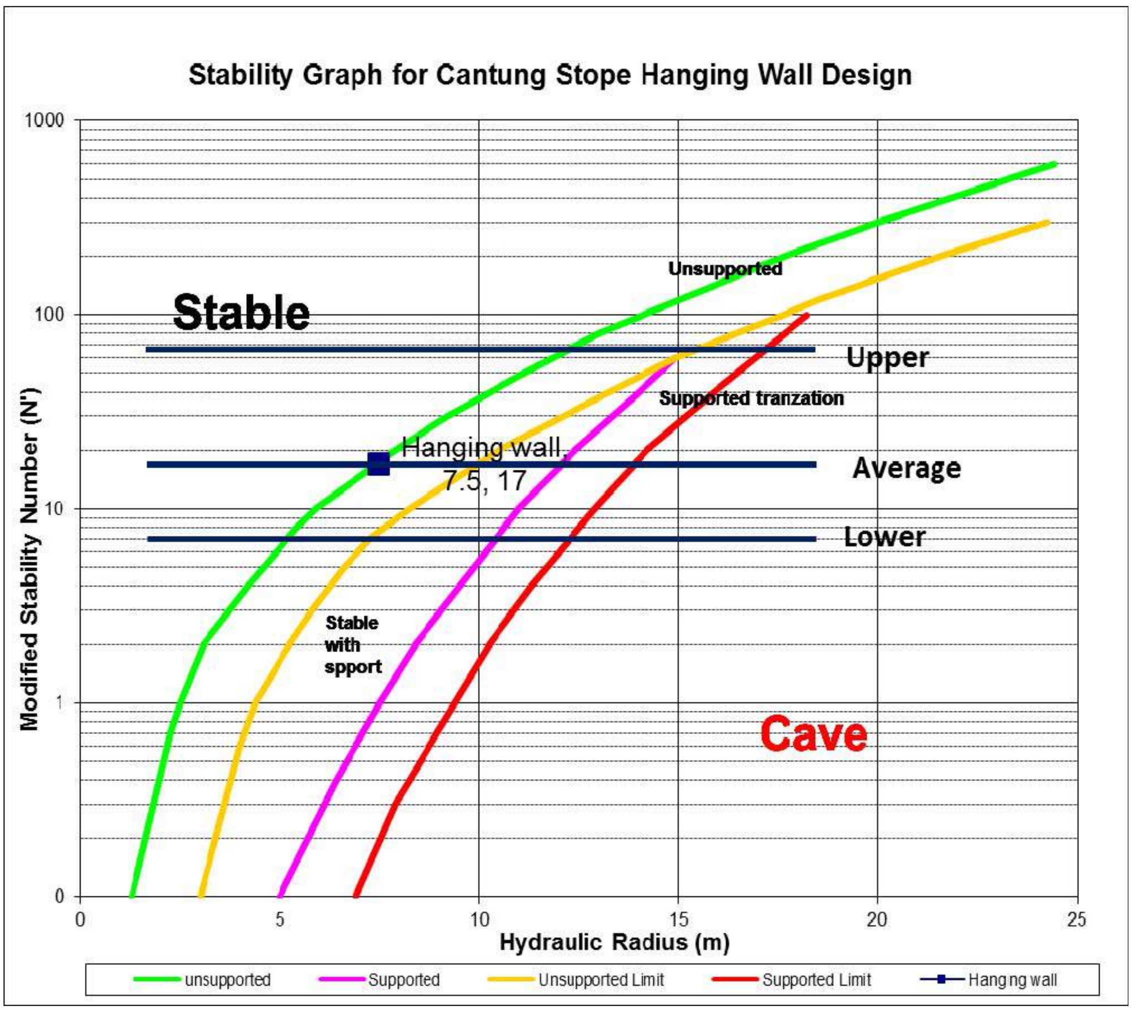
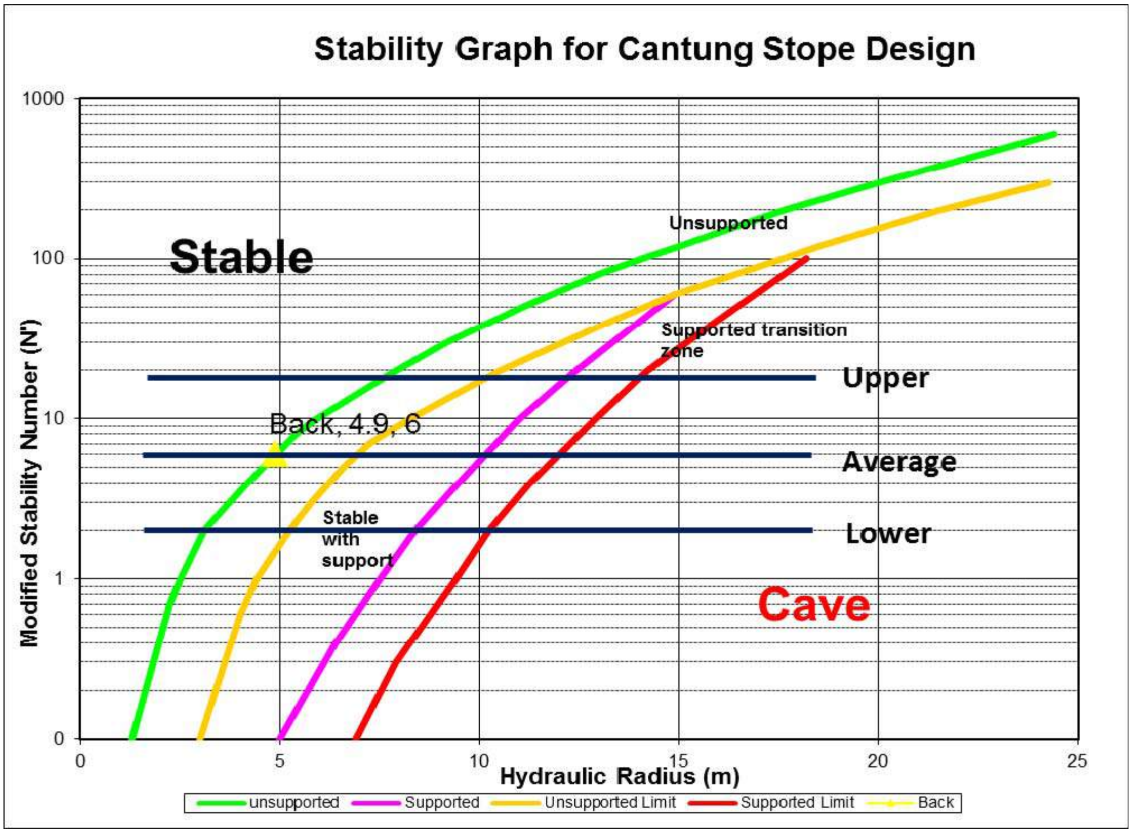


Figure 16.10 - Stope Back Stability Graph



16.4.6 Ground Control System

The ground support protocols and equipment developed at Cantung and detailed below have been used successfully since the mine reopened in 2010.

The Amber Zone is located 400 feet away from the current mining area and the rock mass and structures are the same as in that area. The same support system will be applied in this new zone.

16.4.6(i) Rib Pillar and Sill Pillar

The current rib pillar design for longhole stopes is 20 feet thick at Cantung. Due to over breaking, approximately 2.5 feet on each side, the actual rib pillar thickness is roughly 15 feet. Cantung engineering calculations show these pillars are subjected to 60-100 Mpa stress depending on the height and elevation. Since one side of the stope is waste rock filled before the other side is blasted, the existing rib pillars are all stable for the short-term production mining sequence. Sill pillars are designed for an 11–14 foot thickness.

16.4.6(ii) Waste Fill and Hydraulic/Paste Fill

Currently, waste rock fill is used in both the longhole and cut and fill stopes. Hydraulic back fill studies are planned as there is a new hydraulic sand plant on site. Depending on the results of these studies, Cantung may use both waste rock fill and hydraulic back fill. The longhole mining sequence is designed to mine the stopes from the bottom up and sequence stopes such that juxtaposed stopes are filled before mining the next stope in the sequence.

16.4.6(iii) Ground Support Standard

The current ground support standard is shown in Figure 16.11 below. In ore development, a minimum support type A will be applied. Type A is defined as: 6ft 5/8 inch diameter mechanical bolts on (a) 4ft x 4ft pattern in the back and shoulders, placing screen to 6ft above the floor and anchoring the screen with 6ft split sets.

In permanent development, Type D support is applied. Type D is defined as: 8ft #6 rebar on a 4ft x 4ft pattern in the back and shoulders, placing screen to 6ft above the floor and anchoring the screen with 6ft split sets. Strapping will be used where warranted by rock conditions.

When the span of an intersection is 30 feet or greater, either 10 feet rebar bolts or cable bolts are applied. Cantung engineers use the Unwedge (TM@RockScience Inc.) and

Map3D geotechnical software to do wedge and stress analysis and to make decisions as to whether 10 feet rebar or cable bolts should be used.

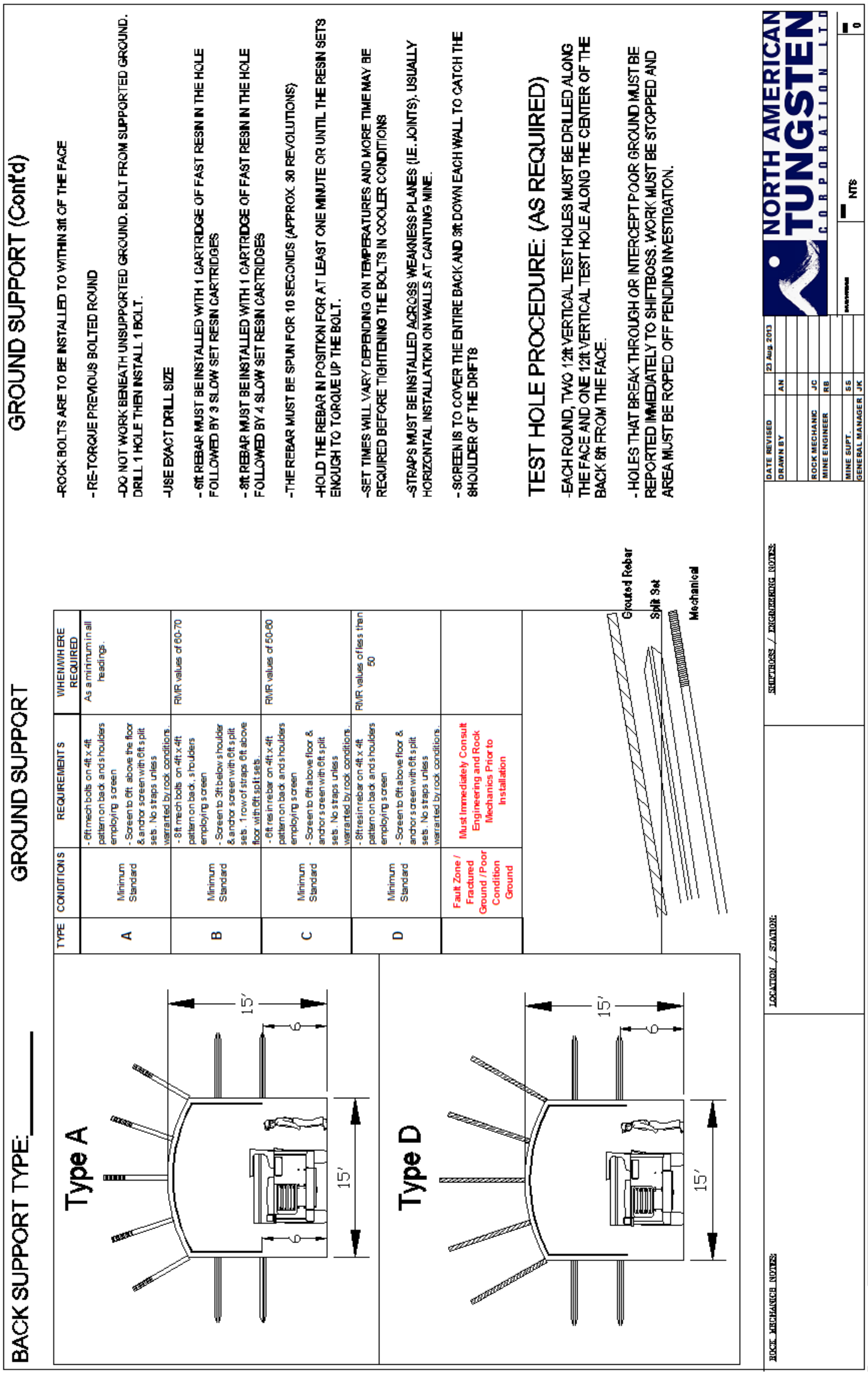
16.4.7 Ground Control Measurement

Cantung has its own mechanical bolt torque test and rebar/split set/cable pull out test equipment. Testing for bolt quality control is performed at least once per year and can be carried out anytime as needed.

Cantung mine uses a 50 foot MPBX (Multi Point Borehole Extensometer, provided by Mine Design Technologies) to monitor the ground movement, and uses Vibrating Wire Stress meter gauges (by Geokon, Inc.) to monitor the stress change in the pillars. Load Cells (developed by RocTek R&D) combined with 15 foot/25 tonne capacity extension bolts will be used in the future to monitor the load on the bolt.

The Cantung mine has installed a micro seismic monitoring system covering the major part of the West Extension Below3700 area. It was installed in August 2012 and after several months of operation, including wave velocity calibration and noise filter adjustment, the system is now providing reliable, accurate data. This system is easy to relocate or expand. When mining is completed in a specific area, the sensors can be moved to a new mining area as seen in the current move to the Amber Zone.

Figure 16.11 - Cantung Ground Support Standard



TYPE	CONDITIONS	REQUIREMENTS	WHEN/WHERE REQUIRED
A	Minimum Standard	- 6ft mech bolts on 4ft x 4ft pattern on back and shoulders employing screen - Screen to 6ft above the floor & anchor screen with 6ft split sets. No straps unless warranted by rock conditions.	As a minimum in all headings.
B	Minimum Standard	- 8ft mech bolts on 4ft x 4ft pattern on back, shoulders employing screen - Screen to 3ft below shoulder & anchor screen with 6ft split sets. 1 row of straps 6ft above floor with 6ft split sets.	RMR values of 60-70
C	Minimum Standard	- 6ft resin rebar on 4ft x 4ft pattern on back and shoulders employing screen - Screen to 6ft above floor & anchor screen with 6ft split sets. No straps unless warranted by rock conditions.	RMR values of 50-60
D	Minimum Standard	- 8ft resin rebar on 4ft x 4ft pattern on back and shoulders employing screen - Screen to 6ft above floor & anchor screen with 6ft split sets. No straps unless warranted by rock conditions.	RMR values of less than 50
	Fault Zone / Fractured Ground / Poor Condition Ground	Must Immediately Consult Engineering and Rock Mechanics Prior to Installation	

16.5 PLANNED LIFE OF MINE PRODUCTION

The mine production schedule is based on providing ore primarily from the Amber zone in the 2015 fiscal year while developing/reopening the E Zone, West Extension and the PUG Zone. In future fiscal years, ore will be delivered from all the aforementioned zones to the processing facilities at the targeted mill feed rate. The E zone and West Extension reserves will be mined during the 2015-2016 fiscal year. After that production will shift entirely to the PUG area in 2017-2018.

Table 16.11 shows the production ore tons for each zone and the relative grades.

Table 16.11 - Mining Production Tons by Zone

		West Ext	*West Ext (below 3700)*	*E Zone*	*Amber Zone*	*PUG*	*PIT*	**Total**
Year	**Grade**	0.77	0.79	0.99	0.87	0.79	0.77	**0.81**
2015	**Tons**	91,000	27,000	11,000	233,000	0	110,000	**472,000**
	Stu's	70,000	21,000	11,000	203,000	0	85,000	382,000
2016	**Tons**	69,000	17,000	44,000	209,000	54,000	80,000	**473,000**
	Stu's	53,000	13,000	44,000	182,000	43,000	62,000	383,000
2017	**Tons**	0	0	0	0	473,000	0	**473,000**
	Stu's	0	0	0	0	374,000	0	383,000
2018	**Tons**	0	0	0	0	359,000	0	**359,000**
	Stu's	0	0	0	0	284,000	0	291,000
TOTAL	**Tons**	160,000	44,000	55,000	442,000	886,000	190,000	**1,777,000**
	Stu's	123,000	35,000	54,000	385,000	700,000	146,000	1,439,000

Note: Production tons do not include stockpile

A comprehensive achievable plan has been completed with limited capital investment to ensure that production rates can be met. The current 4-year Life of Mine Plan is scheduled from October 2014 to September 2018.

Critical Milestones in the Life of Mine Plan are as follows:

1) Completion of the PUG Incline – March 2016
2) Completion of the 3820 Waste Development – Oct 2016
3) Mining 100,000 ton of ore from the open pit during 2015 summer.

The Life of Mine Plan includes substantial mine development in 2015 and 2016 with the PUG incline advancing until the development of current reserves is completed in 2017. Refer to Table 16.12. The development team at Cantung has achieved a linear advance of 11,016 feet in 2014 or 918 feet per month. Following this progress rate, the mining department has planned to complete 11,724 feet of development in 2015 and 11,246 in the 2016 fiscal year. It is anticipated that during the 2017 and 2018 fiscal years the development would decrease unless a new ore body is explored and the mine of life is extended.

Table 16.12 - Life of Mine Development Advance by Year

Life of Mine Development Advance

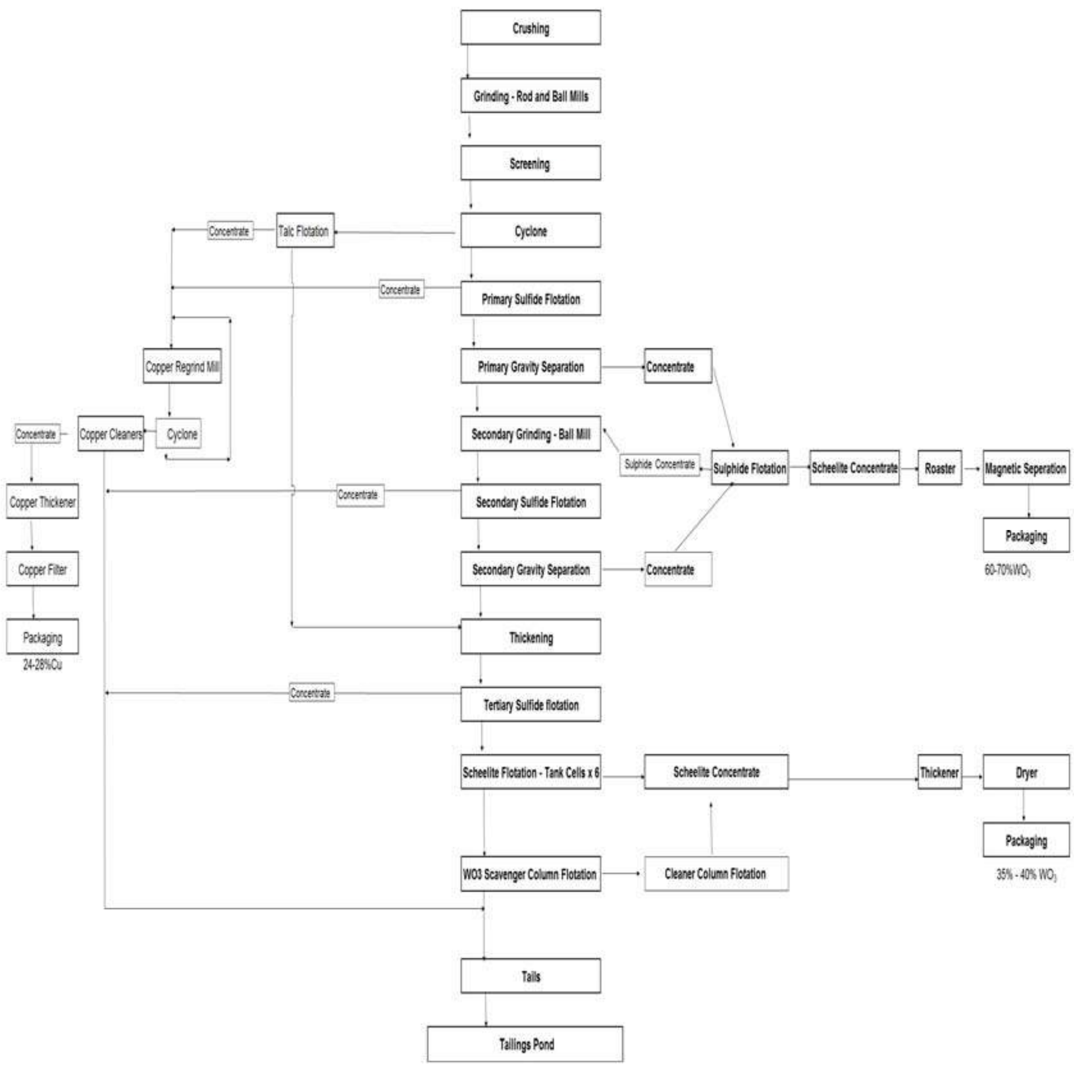
Area	2015	2016	2017	2018	Total ft.
West Extension	3208	1524			4732
Amber	4056	5066			9122
E zone	960	96			1056
Below 3700	1000	560			1560
PUG	2500	4000	5000	990	12490
Total	11724	11246	5000	990	28960

17 RECOVERY METHODS

17.1 PROCESS FLOW CHART

Please see below for a process flow sheet for the Cantung tungsten-copper milling facility. Refer to Figure 17.1.

Figure 17.1 - Mill Process Flow Sheet



17.2 MILL DESIGN AND OPERATION

The mill processing facilities at Cantung comprise primary crushing and coarse ore storage installations, secondary and tertiary crushing, fine ore storage, general gravity and flotation building with offices and a maintenance shop, backfill preparation building (inactive), reagents and supplies storage building and an assay lab. The mill has achieved continuous processing rates of up to 1,300 tons per day. The Life of Mine Plan (LOMP) details an average processing tonnage of 1,350 tons per day at a recovery rate of 83% of WO_3. Final products will be approximately 377 tons per month of gravity concentrate (G1), containing 60% to 70% WO_3 at 54% recovery and approximately 185 tons per month of flotation concentrate containing 30% to 40% WO_3 at 29% recovery. These target numbers appear to be obtainable based on projected ore metallurgical characteristics and past performance attained by the operation. A process flow sheet is shown in Figure 17.1.

Primary Crushing

Ore is handled from the stockpile by a loader or directly dumped from the haulage truck into a 30-ton receiving bin equipped with 42 inch x 10 ft. apron feeder, which in turn feeds a 42 inch x 48 inch jaw crusher. The jaw crusher is set to produce a nominal five-inch crushed product. A conveyor transports crushed ore into a 1,000-ton capacity coarse ore bin. This bin acts as a surge bin for the secondary crushing circuit.

Secondary and Tertiary Crushing

A vibratory feeder and a conveyor feed a 4¼ ft. (secondary) standard cone crusher set at 1 inch. The crushed ore is discharged via conveyor to a vibrating screen equipped with a 7/16 inch x four inch slotted screen. The oversize feeds a 4¼ ft. (tertiary) short-head cone crusher set at 3/8 inch. The tertiary crusher discharge combines with the secondary crusher discharge to feed the screen in a closed-circuit recycle. Screened undersize (minus 7/16 inch) product is conveyed to two fine ore bins ahead of the grinding mills in the concentrator. Dust is controlled by the use of a wet scrubber with the discharge effluent returning to the mill.

Grinding, Classification and Talc Flotation

The grinding circuit consists of a 9 ft. diameter by 12 ft. long rod mill, powered by a 450 hp motor, plus a 7 ft. diameter by 10 ft. long ball mill powered by a 200 hp motor, and a 6 ft. by 6 ft. ball mill powered by a 150 hp motor. The rod mill is fed from two fine ore bins via belt conveyors that discharges ground ore slurry into a sump along with the discharge from the ball mill for pumping to vibrating screens. The screen oversize drops by gravity to a pump that transports the slurry to a dewatering screen. This screen removes excess water prior to further grinding of the solids in the ball mills. As noted above, ball mill discharges join rod mill discharge for presentation to the vibrating screens. Water removed by the dewatering screen is reused for screen feed dilution. Particles larger than the screen openings will circulate through the ball mill until they are sufficiently reduced in size to pass through the screen openings.

Screen undersize also drops by gravity to a pump which then transports the slurry to a set of cyclones that classifies according to particle size. Particles larger than the cyclones cut point drop by gravity to the sands bulk sulfide flotation unit operation. Particles smaller than the cut point feed the slimes bulk sulfide flotation unit operation.

Sulfide Flotation

After conditioning with appropriate promoters, collectors, depressants, and frothers, underflow from the size separation cyclones is directed to flotation cells to remove sulfide minerals which would be deleterious in the downstream scheelite recovery processes. Concentrates can be directed to a copper separation circuit or rejected to tails depending on copper content. Tailings are pumped to the scheelite gravity recovery circuit.

Overflow from the cyclones is directed to the talc flotation cells to remove sulfide minerals in the same manner as in the sands sulfide flotation operation. Concentrates can be directed in the same manner as well. Tailings are pumped to the scheelite flotation recovery circuit as the particle size is too small for effective gravity recovery.

Gravity Separation

Sands bulk sulfide flotation tails are distributed to triple deck shaking tables and spirals for rougher concentration of scheelite. Concentrate from these tables and spirals are cleaned on single deck tables then pumped to a final sulfide mineral removal flotation step, then to a dewatering classifier before going to the roaster.. Rougher table tails then join tails and middlings from other tables to undergo desliming with a cyclone. The cyclone overflow containing particles too small for effective gravity recovery and excess water goes to the scheelite flotation recovery circuit after thickening.

Cyclone underflow, containing particles that remain amenable to gravity separation but are considered to be locked minerals as middlings that require further size reduction to liberate scheelite from gangue, is directed by gravity to a 5 ft. diameter by 8 ft. long ball mill powered by a 150 hp motor. The ball mill discharges to a pump that transports the particles in slurry to another flotation step for removal of liberated sulfide minerals. Concentrate from this flotation step can be directed in the same manner as concentrates from the sands and slimes bulk sulfide flotation operations. Tailings are pumped to triple-deck tables and spirals for scavenging of gravity recoverable scheelite. The concentrate from these tables joins the concentrates from the coarse gravity cleaner tables for final sulfide removal and high temperature drying. Tailings from this fine scheelite gravity recovery operation go to scheelite flotation recovery.

The gravity circuit can be adjusted to produce a range of concentrate grades. Higher grade increases gravity circuit losses which increases the volume of flotation concentrate.

Scheelite Flotation

Flotation feed consists of material that is too fine for gravity separation. This material is first thickened in three parallel 40 ft. diameter three tray stacked thickeners. Thickener underflow is first passed through a third sulphide flotation stage. Concentrate reports to tails, and the tails feeds three agitated conditioner tanks where a pH modifier, depressants, collectors and a frother are added. The slurry stream then passes down a bank of six 5 m^3 agitated forced air tank type flotation cells. The first two cells are roughers, the next two can direct concentrate as rougher or scavenger and the last two are scavengers.

Rougher concentrate is pumped to the scheelite first and second cleaners which is a set of naturally aspirated conventional mechanical flotation cells. Scavenger concentrate is returned to the rougher feed, along with first cleaner tail, by pump. The second cleaner concentrate is the final concentrate. Second cleaner tails return to the first cleaner feed. The final concentrate is thickened in a 20 ft. diameter thickener, filtered, dried, and bagged for shipment to markets. Dried concentrate is weighed and packed in two-ton tote bags. Bagged concentrate is stored in a covered shed while awaiting shipment.

Drying and Magnetic Separation

The gravity concentrate typically contains 60% to 70% WO_3. It is dewatered in a spiral classifier before entering the roaster. The roaster is a propane multiple hearth type with rotating rakes to move material from an upper hearth to a lower hearth and finally to discharge. Material discharging from the dryer is transported in two water jacketed screw type conveyors in series which cool the material prior to downstream transport and processing. Discharge from the second screw conveyor is transported vertically in a bucket elevator to a screen that rejects undesirable material that is recycled via the primary crusher. Undersize then reports to the first stage of dry magnetic separation that consists of two roll type separators in parallel. Each unit has three rolls, one rougher roll that produces a magnetic reject straight away, then two scavenger rolls in series that produce a non-magnetic final product and a magnetic reject that is passed on the second stage of dry magnetic separation.

The second stage is a unit with a rare earth magnet and three electromagnetic units fixed over a moving belt. A non-magnetic final product joins the first stage non-magnetics to be bagged for shipment to markets. The magnetic reject joins the first stage reject and is then returned to main process via the scheelite regrind mill. Dried concentrate is weighed and packed in two ton tote bags. Bagged concentrate is stored in covered areas while awaiting shipment.

Concentrate Handling and Storage

Dried concentrate is packed in two-ton tote bags for shipment to the customers. Bagged concentrate is stored in covered areas before being shipped. The tote bags are handled with a small forklift.

Copper Circuit

The copper recovery circuit can take all or some of the bulk sulfide concentrates from the sands and slimes bulk sulfide flotation and secondary sulfide flotation cells. Only the sands bulk sulfide flotation concentrate is cycloned to separate out material already finer than the requisite optimum particle size. The cyclone underflow containing the material coarser than optimum passes through a 4 ft. x 4 ft. ball mill powered by a 40 hp motor. Mill discharge joins the new bulk sulfide concentrate to be cycloned again. Particles larger than optimum are recirculated through the ball mill until they are reduced in size sufficiently to pass the cyclone's cut size gate and pass on to further processing. The cyclone overflow is passed on to a bank of forced air conventional mechanical flotation cells, known as the first cleaners.

A chalcopyrite specific collector is added to the head end of the bank of cells which promotes chalcopyrite recovery. The pH is taken to ~12 by a lime slurry, $Ca\ (OH)_2$ in water. The lime acts as a depressant for the undesirable pyrite and pyrrhotite in the bulk sulfide concentrate but has no effect on the desirable chalcopyrite.

The tails from the first cleaner cells go to final tails and out. The first cleaner concentrate drops by gravity to the second and third cleaners. Tails from the second cleaner are fed back to the head end of the first cleaners and second cleaner concentrate feeds the head end of the third cleaner. The third cleaner tails feed the head end of the second cleaners. The third cleaner concentrate is the final concentrate.

Final concentrate is thickened and subsequently dewatered on a belt filter then bagged for shipment. Filtered concentrate is weighed and packed in two-ton tote bags. Bagged concentrate is stored in covered areas while awaiting shipment.

Sulphuric acid and coagulant are added to the tails as it leaves the mill to aid in solids settling. Supernatant from No.5 Pond is pumped to a waste water treatment facility (WWTF) for removal of suspended solids before discharge to the environment.

17.3 WASTEWATER TREATMENT FACILITY

The WWTF uses the Actiflo process, designed and built by Veolia Water Solutions and Technologies. The process uses the addition of a coagulant and a polymer, aided by a micro-sand, which is used as a seed for flocculants formation and ballast. The water is then clarified through lamella clarifiers before discharge to the environment. The ballasted sludge is separated from the micro-sand by way of cyclones, sludge returns to tailings pond 5, and the cleaned sand is returned to the head end of the process.

The discharge to environment is limited to a maximum of 4500-8000 cubic meters per day, depending on the flow in the Flat River.

17.4 ASSAY LABORATORY

There is an on-site assay laboratory, which has been used for sample preparation and assaying services for the mine and mill. The laboratory QA/QC testing with other labs is monitored and managed by the Geology Department. Reference materials and control samples are used in each assay run in the lab.

17.5 MILL RECOVERIES

Recovery and grade is back-calculated daily using mill tails assay and weight (mill feed weight in less concentrate weight out) plus concentrates weights and grades to develop a total WO_3 mass in (total out in tails plus total out in product is total in). Total weight of WO_3 out divided by mill feed weight in is the back-calculated mill feed grade. Recovery is then total weight of WO_3 in concentrates divided by total calculated WO_3 in. As a check the sampled mill feed assay is checked against the back calculated feed assay. If a significant discrepancy is noted, conveyor weightometer calibration and sampling systems are immediately checked.

18 PROJECT INFRASTRUCTURE

18.1 SURFACE INFRASTRUCTURE AND EQUIPMENT

As the Cantung operation has progressed into production, all required infrastructure is in place. To date, infrastructure at the site includes:

- Open pit and underground mining operations, including a mechanics shop and all required power, water and air distribution facilities.
- Milling facilities
- Tailings dam facilities
- Water treatment facilities
- Waste disposal facilities
- Power generation facilities - Refer to Table 18.2
- Mechanical repair facilities
- Administration facilities
- Kitchen and accommodation
- Road access
- Airstrip
- Surface mobile vehicles, including a fire truck, ambulance and small trucks, as referenced in Table 18.1
- Sewage treatment facility
- Wastewater treatment facility

Table 18.1 - Surface Mobile Equipment

Type	Units
Champion Grader	1
Volvo Grader	1
D6D Dozer	1
Cat Loaders 950	1
Ambulance	1
Fire Truck	1
Fuel Truck	1
Hiab flat deck	1
Grove 22 t crane	1
5 Ton Steam Truck	1
Small fork lift	4
Excavator / Rock Breaker	1

Table 18.2 - Surface Power Generation Equipment

Type	Units
Caterpillar 3512	4
Caterpillar 3516	4
Caterpillar 3406B	1

18.2 UNDERGROUND INFRASTRUCTURE AND EQUIPMENT

Extensive underground development is in place, including a mechanics shop and all required power, water, air and ventilation facilities. The mobile mining fleet comprises different makes and ages of equipment pieces. The underground mobile fleet is listed in Table 18.3.

Table 18.3 - Underground Mobile Equipment

Type	Units
6 yard LHD	3
8 yard LHD	3
3.5yard LHD (EJC 210)	1
3.5 yard LHD (Elphinestone)	1
2.0 yard LHD (JCI 220)	1
966H Loader (Cat)	1
40 Ton Truck (Cat AD40)	1
50 Ton Truck (Cat AD45)	1
30 Ton Truck (Cat AD30)	3
30 Ton Truck (EJC 430)	1
2 Boom Jumbo (Tamrock)	2
Rock Bolter (Mclean)	1
Stopemaster Long Hole Machine	1
Miller Grader M86	1
Scissor lift JUT	2
Boom Truck Getman	1
Tractor Kubota	2
Telehandler Cat TH220B	1
Telehandler Cat 220	1
Kubota RTV	7

18.2.1 Underground Infrastructure

The main access to the mine is via an adit collared in close proximity to the mill at 3950 level. The adit serves as the main ore haulage drift for the mine and also as a return airway for mine ventilation. Fresh air is delivered to the mine via an intake ventilation adit collared at the 4350 level. The ventilation adit is connected with a decline, serving as an escape way or an emergency access to the mine. There are several declines from the 3950 adit to the lower mining zones.

On both sides of the main haulage drift are five loadout stations, spaced several hundred feet apart. In the past, these loadout stations received broken muck via raises from the mining areas above. Those areas are essentially mined out and only a few ore pillars remain. The current strategy for material handling from the lower mine is to haul the ore, using 30 ton trucks, to the #5 loadout. From there the ore will be hauled out to the crusher or surface stockpile, using a 45 ton truck.

The mine has a number of internal ramps connecting different levels and ore zones. A majority of the ramps are of large cross section and driven on a moderate gradient from 10% to 12%.

The mine has an underground maintenance shop with a warehouse, located on the 3950 level off the main haulage drift. The maintenance shop is a multi-chamber excavation with large service bays and warehousing facilities. The back and walls are well supported and appear to be in very good condition. The shop maintenance bays are equipped with overhead cranes ranging from 5 to 15 ton capacity. Electrical repair and welding stations are part of the shop service infrastructure. The warehouse is large and stocked with a sizable inventory of parts. The shop has a lunchroom equipped for emergency situations and a separate escapeway, in a fresh airway, to surface.

There are open field storage areas and crib facilities located in different parts of the mine. They are stocked with typical mining consumables, such as rock bolts, wire mesh screen, pipe, fittings, and timber needed for construction and ground support applications.

18.2.2 Power Supply

Mine site electric power is obtained from the powerhouse via the site electrical distribution system. The electric power is delivered to the mine substations at 4160V and transformed to 600V. From the mine substations, power is distributed via electric cables to different areas of the mine to feed the underground mobile and stationary equipment.

18.2.3 Water

The mine is generally dry, with flow out the 3950 level adit estimated to be approximately 30 gpm. The water from the mine workings above the 3950 level is drained by gravity via ditches and a number of decant holes and raises down to the 3950 haulage drift. The West Extension workings, below the 3950 level, are dewatered by use of sump pumps and pipelines discharging water to the 3950 adit ditch. The quality of mine drainage water is monitored in accordance with the regulatory requirements.

Mine process water is delivered from the main water storage tank to the mine via a piping system. The mine has an extensive underground water pipe line system feeding the shop and all key production and development areas. Potable water is brought into the mine in small containers.

18.2.4 Compressed Air

Compressed air is delivered to the mine via steel pipe from a compressor house located close to the mine access portal at the 3950 level. Equipment in place at the compressor house is listed in Table 18.4 below.

Table 18.4 - Compressed Air Equipment

Type	Units
Sullair LS-25 1000hp Electric	3
Sullair 1600 PDQ Portable Diesel	1

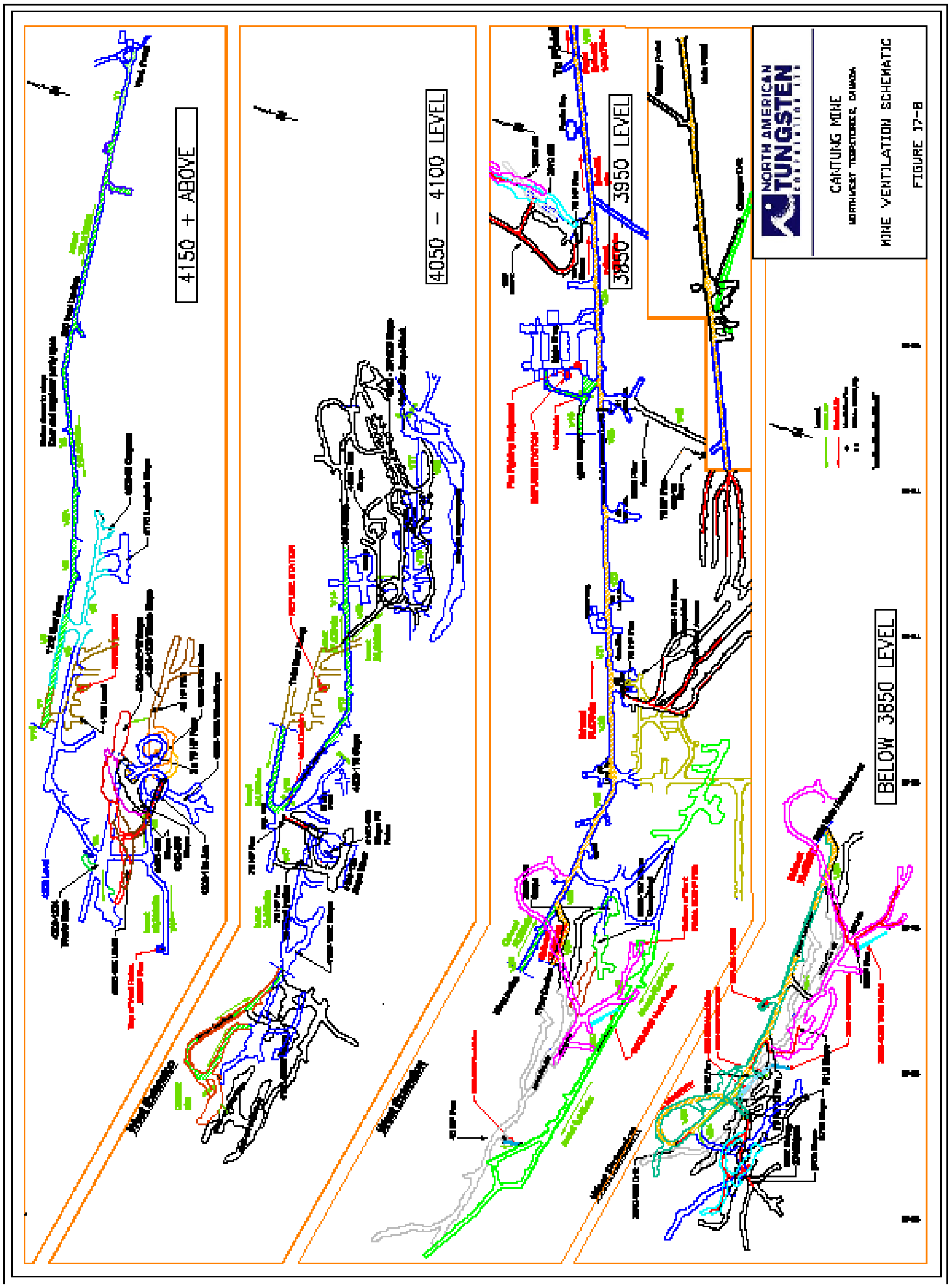
18.2.5 Ventilation

Two intake fans, installed in parallel, deliver 270,000 cfm of fresh air to the mine. These fans are located at the 4350 level adit on surface. The installation includes a set of propane burners for heating the air in winter to prevent freezing of the intake drift. The air heaters are equipped with 16 million BTU burners. The fans push the air down the mine through various levels. The underground airflow is controlled by a number of ventilation doors, raises, and regulators. Auxiliary fans and ducting provide ventilation in development and blind areas of the mine. The mine ventilation air exhausts back to surface via the 3950 level haulage adit. Refer to Figure 18.1.

18.2 6 Vehicle Maintenance

Vehicle maintenance is carried out in the underground shop. The UG maintenance shop is well designed and furbished for providing a wide variety of services for the whole underground equipment fleet. All regular and major maintenance services are done at this location. Major engine rebuilds are done off-site.

Figure 18.1 - Mine Ventilation Schematic



19 MARKET STUDIES AND CONTRACTS

There are many uses for tungsten including:

- Cemented carbides, also called hardmetals, are the most important usage of tungsten today. Hardmetal tools are used for the shaping of metals, alloys, wood, composites, plastics and ceramics, as well as for the mining and construction industries.
- Tool steels, high speed steels, *stellites* and creep-resistant steels and alloys.
- Lighting filaments, electrodes, electrical and electronic contacts, wires, sheets, rods etc. or tungsten alloys.
- Other applications include a widespread variety of chemical uses.

The demand for tungsten, because of its heavy use in machinery and steel, is highly sensitive to worldwide economic conditions. China is the world's largest producer of tungsten, with the amount of tungsten exported from China having the greatest impact on the available supply of tungsten.

It is expected that APT pricing will continue at current levels and may rise in the future, given the forecast for growing consumption and continued demand for tungsten metal and concentrates and increasing supply from existing and new producers as well as secondary recycling of scrap materials.

NATCL sells the G1 and Flotation concentrates based on free market values for ammonium paratungstate (APT). The Company currently has two main customers under contract for tungsten concentrate with one located in Europe and the other in North America. The tungsten supply contracts currently cover the majority of concentrate produced and each of these contracts have similar terms and conditions.

Pricing under the tungsten delivery contracts are based on the London Metal Bulletin ("LMB") European Free Market Quotation, in USD per MTU of APT, using the monthly average of the Low and the High price for the latest completed calendar month. The derived monthly average price for APT is then discounted to arrive at the contract price for the tungsten concentrates produced. The average APT quotation for September 2014 was USD$353.75.

Each contract is for an initial term of the later of 3 years to February 1, 2017, or the completion of the delivery quantity for the initial term. The customer has the right, at its sole discretion, to extend the contract under the initial term until all amounts under customer related loan arrangements have been repaid in full (at June 30, 2014, the outstanding balance of the related customer loans was USD$7.2 million). Under mutual agreement, the contracts could further be extended annually to February 1, 2019. The contract contains quarterly and annual target delivery quantities and do not contain financial penalties for shortfalls in target delivery quantities. The contract contains provisions that could allow the customer to terminate the delivery contract if quantities or concentrate specifications are not achieved for three consecutive months, or if target delivery quantities are not achieved. In the event of default by the Company, under the terms of the delivery contract, the customer has the right to demand immediate full repayment of the outstanding balance under the loan arrangement and any customer advance.

Production in excess of the amounts under contract could be sold to the existing customers under the contracts, on the spot market or to other potential customers.

The terms of these tungsten delivery contracts fall within the industry norms.

NATCL’s competitors are the tungsten producers in China, Russia, Vietnam, Australia, Austria and smaller producers in Africa, Asia, Europe and South America.

20 ENVIRONMENTAL STUDIES AND PERMITTING

20.1 PERMITTING REQUIREMENTS AND RECLAMATION BONDS

As noted in Section 4, the permit for the Dry Stack Tailings Facility is currently under review.

In relation to the water permit, on January 29, 2009 the Company received notification from the Mackenzie Valley Land and Water Board ("MVLWB") of the renewal of the Company's type "A" Water License ("license"). The license was approved for a period of five (5) years commencing January 30, 2009, and expiring January 29, 2014. On October 28, 2011 the MVLWB approved a two-year extension of the water license to expire January 29, 2016.

In relation to the reclamation bonds, the security deposit presently required under the Company's license is $11,657,839 (includes the August 2010 security amount reduction of $1,422,161 received from the MVLWB). As of September 30, 2014, the Company has posted $5.8 million in cash and $6.2 million in the form of secured promissory notes pursuant to the Reclamation Security Agreement ("RSA"). The RSA further provides for:

- the Company to post $100,000 in cash on the 1st of September, 1st of December, 1st of March, and 1st of June (1st payment made on the 1st of September), to reduce the amounts pledged under the promissory notes;
- the cash components payable to DIAND to increase under certain events.

Any security amounts owing under the license and monies owed by way of secured promissory notes are secured by a Security Agreement charging specific assets.

Any funds in excess of ultimate reclamation costs will be returned to the Company.

20.2 MINE CLOSURE REQUIREMENTS AND COSTS

Background information regarding mine closure requirements and costs has been previously covered in Section 4. A summary of the various reclamation cost estimates over time is shown in Table 20.1.

Table 20.1 - Reclamation Cost Estimates, Cantung Mine

RECLAMATION COST ESTIMATES North American Tungsten Corporation Ltd. Cantung Mine			
Prepared By	Date Prepared	for	Closure Cost Estimate ($M)
EBA	Nov 2001	NATCL	2.3
EBA	Oct 2002	NATCL	2.5 - 3.0
Brodie Consulting	Oct 2002	MVLWB	34.5
NATCL	Jul 2003	NATCL	1.9
MVLWB Security	Dec 2003	MVLWB	7.9
NATCL	Nov 2007	NATCL	3.8
Brodie Consulting	June 2008	INAC	13.1
MVLWB Security	Jan 2009	MVLWB	13.1
NATCL	Mar 2009	NATCL	4.2
NATCL	Mar 2010	NATCL	4.3
MVLWB Security	Aug 2010	MVLWB	11.7
NATCL	Mar 2011	NATCL	5.0
NATCL	Mar 2012	NATCL	7.5
NATCL/ Wenck	Oct 2012	NATCL	8.7
Wenck	Oct 2013	NATCL	8.9
Wenck	Oct 2014	NATCL	13.1

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

The total capital costs for the Life of Mine are estimated to be $19.5 million, as shown in Table 21.1.

Table 21.1 - Life of Mine Capital Costs

Summary of Capital Costs		
Costs in CND $ ('000)		
Activities		**Life of Mine**
Maintenance capital	$	8,783
Power / Distribution	$	2,450
Mill	$	481
Tailings Management	$	7,468
Surface	$	322
Total for Life of Mine	**$**	**19,504**

Maintenance capital includes costs to replace or rebuild existing underground mining equipment and maintenance of mill equipment and surface infrastructure. Capital may be necessary to increase power generation capacity to support the dry stacking of tailings and for new power distribution equipment to enable effective distribution of power to the mine site. Tailings management capital cost includes the cost to complete the current raise on an existing tailings pond and for the purchase and construction of a dry stack tailings facility which will enable Cantung to not require further expansion of tailings ponds.

21.2 OPERATING COSTS

Operating costs for the Life of Mine Plan have been based on the requirements of the production schedule. Costs were estimated based on recent and historical site experience and current supplier contracts and pricing. Labour costs are based on the current payroll.

Table 21.2 - Estimated Operating Costs per MTU

Estimated Operating Costs Per MTU Produced by Major Function		
Costs in CND $	Cost Per MTU	
Function	**Produced**	
Mining	$	92
Milling	$	40
Surface and Power	$	59
Administration, Expediting and Support Services	$	42
Environmental	$	7
Estimated Operating Costs Per MTU Produced	**$**	**240**

The estimated operating cost per MTU produced over the Life of Mine is $240 (assuming an exchange rate of CDN$1.09 per US$1.00, equivalent operating cost is USD$220 per MTU produced). This estimate excludes corporate costs and depreciation. Monthly average operating costs are forecast to be in the order of $6.2 million. Operating costs for each function are described below.

21.2.1 Mining Costs

The estimated cost of mining is estimated to be comparable to the costs experienced during recent years as similar levels of explosives, ground support, labour and mine development are anticipated as mining methods will be the same. Underground ore will continue to be supplemented with open pit ore in the mill feed. As shown in Table 16.12 – Life of Mine Development Advance by Year, it is anticipated that feet of mine development may decline in future periods along with associated costs.

21.2.2 Milling Costs

Mill costs are estimated to increase slightly from costs experienced in recent years due to the increase in tons milled per day to 1,350 stpd from the average in recent years of 1,100 stpd. This is forecast to take place in late fiscal 2014 and continue for life of mine.

21.2.3 Surface, Power, Administration and Environmental Costs

Costs for surface, power generation, site administration and environmental are estimated to be similar to the costs experienced in recent years.

21.3 MARKET OUTLOOK

World tungsten supply was dominated by Chinese production and exports. China was also the world's leading tungsten consumer. China's Government has regulated its tungsten industry by limiting the number of exploration, mining, and export licenses; limiting or forbidding foreign investment; imposing constraints on mining and processing; establishing quotas on production and exports; adjusting export quotas to favor value-added downstream materials and products; and imposing export taxes on tungsten materials. To conserve its resources and meet increasing domestic demand, the Chinese Government planned the following: to expand exploration and increase ore reserves in approved mines, to stop illegal mining and close small inefficient mines, to continue to use quotas to control tungsten mine production, to improve its tungsten-processing technology and increase tungsten recovery from ores and tailings, to limit the export of upstream tungsten materials, and to increase the development and sales of value-added downstream tungsten products.

In the next few years, mine production from outside China is expected to increase. Numerous companies worked to develop tungsten deposits or restart tungsten production from inactive mines in Asia, Australia, Europe, and North America. The amount, location, and timing of future production will depend on the companies' abilities to acquire funding. Increased production capacity for ammonium paratungstate and ferrotungsten outside China is also planned. Scrap will continue to be an increasingly important source of raw material for the tungsten industry, worldwide.

World tungsten resources are geographically widespread. China ranks first in the world in terms of tungsten resources and reserves and has some of the largest deposits. Canada, Kazakhstan, Russia, and the United States also have significant tungsten resources.

Potential substitutes for cemented tungsten carbides include cemented carbides based on molybdenum carbide and titanium carbide, ceramics, ceramic-metallic composites (cermets), and tool steels. Potential substitutes for other applications are as follows: molybdenum for certain tungsten mill products; molybdenum steels for tungsten steels; lighting based on carbon nanotube filaments, induction technology, and light-emitting diodes for lighting based on tungsten electrodes or filaments; depleted uranium or lead for tungsten or tungsten alloys in applications requiring high-density or the ability to shield radiation; and depleted uranium alloys or hardened steel for cemented tungsten carbides or tungsten alloys in armor-piercing projectiles. In some applications, substitution would result in increased cost or a loss in product performance.

(Source: U.S. Geological Survey “Mineral Commodity Summaries” – February 2014)

22 ECONOMIC ANALYSIS

As the Cantung Mine is a current producer, an economic analysis is not required for this Technical Report under NI43-101.

23 ADJACENT PROPERTIES

The Northwest Territories mining recorders claim sheets 105H/16 and 105 I/01 show no third party claims tied on to the NATC Cantung property, and none in the immediate vicinity.

24 OTHER RELEVANT DATA AND INFORMATION

No additional information or explanation is required in this technical report.

25 INTERPRETATION AND CONCLUSIONS

In NATCL's opinion the Cantung Mine is positioned to receive the value of the current prices for tungsten and the continued demand for tungsten concentrates. The Mine has the advantages of extensive operating history and established contacts with suppliers and customers. The Mine has operated successfully in the past; however, it should be noted that Cantung has experienced numerous shutdowns during periods of low tungsten prices.

In NATCL's opinion, the key risk to mine profitability lies in tungsten price sustainability, USD/CDN exchange rates, metallurgical recoveries mine head grades over the remainder of the mine life, risks associated with mining and downturns in the World Economy.

When production is taken into account, Mineral Reserves have increased since the previous estimate at year-end 2007. The increase is due to exploration of new zones, inclusion of lower-grade areas rendered economic by higher prices and planned pillar recovery in previously mined areas through the use of longhole mining methods.

As long as development and stope preparation continues in a timely manner, longhole mining should provide steady production at a lower cost than cut and fill mining. Longhole pillar mining carries a risk of lower grades from higher dilution; however, production estimates carry a reasonable and appropriate allowance for expected dilution. Longhole mining is scheduled to provide 81% of the production tons. Cut and fill and/or room and pillar mining will still be employed in the appropriate areas and is expected to provide 9% of production tons. Open pit mining is anticipated to provide the remaining 10% of mill feed.

26 RECOMMENDATIONS

No specific recommendations need be made by the authors as this is an operating mine.

27 SIGNATURES

Brian Delaney, P.Eng

I, Brian Delaney, P.Eng, Assistant Mine Manager as an author of this Technical Report entitled "TECHNICAL REPORT ON THE CANTUNG MINE, NORTHWEST TERRITORIES, CANADA" prepared for

North American Tungsten Corporation Ltd.
#1640 – 1188 West Georgia Street
Vancouver, BC V6E 4A2
Ph: (604)684-5300
Fax: (604)684-2992

and dated September 19, 2014, do hereby certify that:

1. I am Assistant Mine Manager for the Cantung Mine
2. I am a graduate of Queen's University, Kingston, Ontario, Canada, in 2000 with a Bachelor of Science degree in Mining Engineering.
3. I am registered as a Professional Engineer in the Province of Newfoundland and Labrador (Registration # 04350). I have worked as a Mining Engineer for a total of 11 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    a. Engineering design and project management in a wide range of technical projects.
    b. Project work on several North American mines;
    c. Fulfilled the roles of Chief Engineer and Mine Superintendent; and
    d. General Manager of a 500 tpd mining operation.
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 have worked at the Cantung Minesite since December 2013
6. I am responsible for the review of Section 16 and the parts of Section 18 related to mining methods and site infrastructure, of the Technical Report.
7. I am not independent of the issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have been involved with North American Tungsten and the Cantung property that is the subject of this Technical Report since December 2013
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with N43-101 and Form 43-101F1
10. 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.

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

Dated 19th day of September 2014

"signed and sealed"
Brian Delaney, P. Eng

Finley Bakker P.Geo

I, Finley J. Bakker, P.Geo Superintendent of Technical Services as an author of this Technical Report entitled “TECHNICAL REPORT ON THE CANTUNG MINE, NORTHWEST TERRITORIES, CANADA” prepared for

North American Tungsten Corporation Ltd.
#1640 – 1188 West Georgia Street
Vancouver, BC V6E 4A2
Ph: (604)684-5300
Fax: (604)684-2992

and dated September 19, 2014, do hereby certify that:

1. I am Technical Services Superintendent at the Cantung Mine
2. I graduated with a degree in BSc. Honours in Geology from the McMaster University in 1979.
3. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (1991) and a member of the Association of Professional Engineers and Geoscientists of Northwest Territories and Nunavut (2012)
4. I have worked as a geologist for a total of 35 years since my graduation from university.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
6. My relevant experience for the purpose of the Technical Report is:
    a. Chief Geologist at four mines
    b. Have undertaken Resource calculations for 34 years
    c. Five years of employment at the Cantung Mine, Northwest Territories
    d. Have held the positions of Senior Resource Geologist, Exploration Manager
    e. Have used MineSight/Compass software used in calculating the Mineral Resource for 26 years.
7. I am responsible for the preparation of the technical report titled “TECHNICAL REPORT ON THE CANTUNG MINE, NORTHWEST TERRITORIES, CANADA” and dated September 19th, 2014, with the exception of Section 16 and the parts of Section 18 related to mining methods and site infrastructure.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is: I have worked at the site from November 2006 to September 2007, from April 2008 to October 2009, and from January 2013 to present as well as consulting for NATCL.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

10. I am not independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public Company files on their websites accessible by the public, of the Technical Report.

Dated 19th day of September 2014

“signed”
Finley J. Bakker, P.Geo.

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29 APPENDICES

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466
(www.cdnlabs.com)

ORE REFERENCE STANDARD: CDN-W-4

Recommended values and the "Between Lab" Two Standard Deviations

Gold	***0.319 ± 0.040 g/t***	***(*provisional value, RSD = 6.4%)***
Copper	***0.139 ± 0.008 %***	
Molybdenum	***0.110 ± 0.008 %***	
Tungsten	***0.366 ± 0.024 %***	

**** Note:**

Standards with an RSD of near or less than 5 % are certified, RSD's of between 5 % and 15 % are Provisional, and RSD's over 15 % are Indicated. Provisional and Indicated values cannot be used to monitor accuracy with a high degree of certainty.

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: February 15, 2008

METHOD OF PREPARATION:

Reject ore material was dried, crushed, pulverized and then passed through a 200 mesh screen. The +200 material was discarded. The -200 material was mixed for 5 days in a double-cone blender. Splits were taken and sent to twelve laboratories for round robin assaying.

ORIGIN OF REFERENCE MATERIAL:

The tungsten ore was supplied from underground workings at North America Tungsten's Cantung mine in the NWT. It is high sulphide consisting primarily of pyrite containing chalcopyrite. Native gold and bismuth are associated with the chalcopyrite. The tungsten occurs as scheelite. 155 kg of this ore was combined with various Cu / Mo / Au ores to make 800 kg of standard CDN-W4

Approximate chemical composition is as follows:

	Percent			Percent
SiO_2	56.2		MgO	2.3
Al_2O_3	12.7		K_2O	2.4
Fe_2O_3	13.7		TiO_2	0.5
CaO	3.8		LOI	4.5
Na_2O	2.0		S	0.3

Statistical Procedures:

The final limits were calculated after first determining if all data was compatible within a spread normally expected for similar analytical methods done by reputable laboratories. Data from any one laboratory was removed from further calculations when the mean of all analyses from that laboratory failed a t test of the global means of the other laboratories. The means and standard deviations were calculated using all remaining data. Any analysis that fell outside of the mean ±2 standard deviations was removed from the ensuing data base. The mean and standard deviations were again calculated using the remaining data. This method is different from that used by Government agencies in that the actual "between-laboratory" standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Confidence Limits published on other standards.

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466
(www.cdnlabs.com)

ORE REFERENCE STANDARD: CDN-W-1

Recommended values and the "Between Lab" Two Standard Deviations

Tungsten concentration: ***1.04 ± 0.10 %***
Copper concentration: ***0.458 ± 0.042%***

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: February 8, 2006

METHOD OF PREPARATION:
Reject ore material was dried, crushed, pulverized and then passed through a 200 mesh screen. The +200 material was discarded. The -200 material was mixed for 5 days in a rotary mixer. After internal assaying to test for homogeneity, splits were taken and sent to 11 laboratories for round robin assaying.

ORIGIN OF REFERENCE MATERIAL:
The ore was supplied from underground workings at North America Tungsten's Cantung mine in the NWT. It is high sulphide consisting primarily of pyrrhotite containing chalcopyrite. Native gold and bismuth are associated with the chalcopyrite. The tungsten occurs as scheelite.

Approximate chemical composition:
Standard CDN-W-1 is a high sulphide material with approximately 17.5% sulphur.
Whole rock analysis as follows:

	Percent			Percent
SiO_2	25.0		MgO	4.7
Al_2O_3	3.2		K_2O	0.8
Fe_2O_3	46.3		TiO_2	0.2
CaO	10.3		LOI	6.1
Na_2O	1.0			

Statistical Procedures:
The mean and standard deviation for all data was calculated. Outliers were defined as samples beyond the mean ± 2 Standard Deviations from all data. These outliers were removed from the data and a new mean and standard deviation was determined. This method is different from that used by Government agencies in that the actual "between-laboratory" standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Certified Limits published on other standards.

Results from round-robin assaying are presented on the following page:

Assay Procedures: **W:** Variety of methods: a) fusion, XRF b) digestion, ICP c) fusion, ICP-MS d) fusion, ICP

Cu: 4-acid digestion, AA or ICP finish.

CDN Resource Laboratories Ltd.

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ORE REFERENCE STANDARD: CDN-W-2

Recommended values and the "Between Lab" Two Standard Deviations

Tungsten concentration: *2.78% ± 0.39 %*
Copper concentration *0.45% ± 0.034%*
Bismuth concentration *0.32% ± 0.028%*

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CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: July 24, 2006

METHOD OF PREPARATION:
Reject ore material was dried, crushed, pulverized and then passed through a 200 mesh screen. The +200 material was discarded. The -200 material was mixed for 5 days in a rotary mixer. After internal assaying to test for homogeneity, splits were taken and sent to 11 laboratories for round robin assaying.

ORIGIN OF REFERENCE MATERIAL:
The ore was supplied from underground workings at North America Tungsten's Cantung mine in the NWT. It is high sulphide consisting primarily of pyrrhotite containing chalcopyrite. Native gold and bismuth are associated with the chalcopyrite. The tungsten occurs as scheelite.

Approximate chemical composition:
Standard CDN-W-2 is a high sulphide material with approximately 15.6% sulphur.
Whole rock analysis as follows:

	Percent			Percent
SiO_2	27.9		MgO	5.2
Al_2O_3	3.1		K_2O	1.0
Fe_2O_3	43.4		TiO_2	0.1
CaO	10.1		LOI	5.7
Na_2O	0.8			

Statistical Procedures:
The mean and standard deviation for all data was calculated. Outliers were defined as samples beyond the mean ± 2 Standard Deviations from all data. These outliers were removed from the data and a new mean and standard deviation was determined. This method is different from that used by Government agencies in that the actual "between-laboratory" standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Certified Limits published on other standards.

Results from round-robin assaying are presented on the following page:

Assay Procedures: **W:** Variety of methods: a) fusion, XRF b) digestion, ICP
c) fusion, ICP-MS d) fusion, ICP

Cu, Bi: 4-acid digestion, AA or ICP finish.

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ORE REFERENCE STANDARD: CDN-W-3

Recommended values and the "Between Lab" Two Standard Deviations

Tungsten concentration:	*1.73 ± 0.19 %*
Copper concentration	*0.44 ± 0.036%*
Bismuth concentration	*2142 ± 210 ppm*

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: July 26, 2006

METHOD OF PREPARATION:
Reject ore material was dried, crushed, pulverized and then passed through a 200 mesh screen. The +200 material was discarded. The -200 material was mixed for 5 days in a rotary mixer. After internal assaying to test for homogeneity, splits were taken and sent to 11 laboratories for round robin assaying.

ORIGIN OF REFERENCE MATERIAL:
The ore was supplied from underground workings at North America Tungsten's Cantung mine in the NWT. It is high sulphide consisting primarily of pyrrhotite containing chalcopyrite. Native gold and bismuth are associated with the chalcopyrite. The tungsten occurs as scheelite.

Approximate chemical composition:
Standard CDN-W-3 is a high sulphide material with approximately 19.6% sulphur.
Whole rock analysis as follows:

	Percent			Percent
SiO2	25.3		MgO	3.9
Al2O3	2.4		K2O	0.6
Fe2O3	49.5		TiO2	0.1
CaO	8.4		LOI	6.5
Na2O	0.8			

Statistical Procedures:
The mean and standard deviation for all data was calculated. Outliers were defined as samples beyond the mean ± 2 Standard Deviations from all data. These outliers were removed from the data and a new mean and standard deviation was determined. This method is different from that used by Government agencies in that the actual "between-laboratory" standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Certified Limits published on other standards.

Results from round-robin assaying are presented on the following page:

Assay Procedures: **W:** Variety of methods: a) fusion, XRF b) digestion, ICP
c) fusion, ICP-MS d) fusion, ICP

Cu, Bi: 4-acid digestion, AA or ICP finish.