Minto Phase V
Preliminary Feasibility Study Technical Report

Prepared for:

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Executive Summary

Introduction

Minto Explorations Ltd. ("MintoEx") is a wholly owned subsidiary of Capstone Mining Corp. ("Capstone") which owns (100%) and operates the Minto Mine; a 3,200 tonne per day ("tpd") high-grade copper-gold mine approximately 240 km northwest of Whitehorse, Yukon. In 2010, the mine processed circa 925,000 tonnes of ore at a grade of 2.25% Cu, 0.9 g/t Au and 8 g/t of Ag.

A preliminary feasibility study and technical report ("2007PFS") was completed for the Main and Area 2 deposits in November 2007 after a successful exploration program in 2006. In 2007 through to 2009, three other exploration targets, Ridgetop, Area 118, and Minto North were drilled to resource-quality levels and the Area 2 deposit was significantly expanded. These expanded resources formed the basis for the 2009 Phase IV PFS Technical Report.

This Phase V PFS builds upon the 2009 Phase IV PFS and includes the following modifications:

- New mineral resource and/or reserve estimates including:
  - Minto Main;
  - Area 2/118;
  - Minto North;
  - Ridgetop; and
  - Minto East;
- New life-of-mine plan including, underground mineral reserves;
- Updated cost and economic analysis estimates;
- Changed tailings disposal methodology; and
- Processing plant capacity improvements (subject to permit approval).

The Phase V PFS Technical Report was compiled for MintoEx by SRK Consulting (Canada) Inc. ("SRK") with contributions from Ausenco Minerals Canada Inc. ("Ausenco") for all information related to metallurgy and mineral processing and Kirkham Geosystems Ltd. ("Kirkham") for resource estimation of the Minto North and Minto East deposits.

Exploration on the Minto property is ongoing, diamond drilling is currently suspended for the season but is planned to start again in early 2011 and is designed to more fully define and, potentially, expand the mineral resources, as well as to explore additional mineralized targets.

Based on the results of the 2007 PFS, MintoEx applied to the Yukon government for an amendment to its Quartz Mining Licence in order to increase production from the Main deposit to 3,200 tpd, permission for which was granted in July 2008. An application to amend the Quartz Mining Licence to increase production to 3,600 tpd is currently undergoing environmental assessment. A further application to amend its Quartz Mining Licence is expected to be filed by MintoEx in 2011 that enables an additional increase in production and modify operating parameters as presented in this report. .

Geology and Exploration

The Minto Project is found in the north-northwest trending Carmacks Copper Belt along the eastern margin of the Yukon-Tanana Composite Terrain. The belt is host to several intrusion-related Cu-Au mineralized hydrothermal systems. The Minto Property and surrounding area are underlain by plutonic rocks of the Granite Mountain Batholith of Early Mesozoic Age. The component of the batholith represented on the Minto Property is the Minto pluton and is predominantly of granodiorite composition. Hypogene copper sulphide mineralization at Minto is hosted wholly within this pluton in sub-horizontal horizons of structurally prepared rock.

Four deposits of copper-gold-silver mineralization are reported in this document. Each of these deposits closely share a similar style of mineralization hosted by vertically stacked, shallow dipping deformation zones within the intrusion. The Main deposit is currently exposed in an operating open pit mine and this geometry has been confirmed. Three other deposits have drill-delineated mineral resources and/or reserves but mineralization is not exposed.

For the purpose of this report the Area 2 and Area 118 deposits are now considered continuous, and reported as one deposit, namely Area 2/118 located immediately south of Main Minto. The Ridgetop deposit is located just over 300 m south of the Area 2/118 deposit, the Minto North deposit located about 700 m north of the Minto Main deposit, while the most recently discovered deposit with reported mineral resources is the Minto East deposit located about 200 m east of the south end of the Minto Main deposit. These deposits and other mineral prospects define a general north-northwest trend informally called the Priority Exploration Corridor or PEC.

Copper sulphide mineralization is found in the rocks that have a structurally imposed fabric, ranging from a weak foliation through to a strongly developed gneissic banding. The contact relationship between the foliated deformation zones and the massive phases of granodiorite is generally very sharp. These contacts do not exhibit chilled margins and are considered by MintoEx geologists to be structural in nature, separating the variably strained equivalents of the same or similar rock type.

The more highly strained deformation zones form sub-horizontal horizons and can be traced laterally for more than 1,000 m in the drill core. They are often stacked in parallel to sub-parallel sequences and it is postulated that the foliated granodiorite horizons represent healed, shallowly dipping shear zones within the Granite Mountain Batholith; theorized to have formed when the rocks passed through the brittle/ductile transformation zone in the earth's crust in transition from a deep emplacement environment of the batholith to eventual exhumation. There is on-going debate, however, regarding the stratigraphic, intrusive, or structural nature of the zones hosting the foliation and mineralization. MintoEx have engaged the Mineral Deposits Research Unit ("MDRU") of the University of British Columbia to help understand the mineral paragenesis and deformation history. No other recognized deposit type compares directly with Minto mineralization. While an Iron Oxide Copper Gold (IOCG) style for the Minto deposit cannot be unequivocally demonstrated, the authors are of the opinion that this style of deposit provides the most consistent model for the current level of understanding.

The primary hypogene sulphide mineralization consists of chalcopyrite, bornite, euhedral chalcocite, and minor pyrite. Metallurgical testing also indicates the presence of covellite, although this sulphide species has never been positively logged macroscopically.

Texturally, sulphide minerals predominantly occur as disseminations and foliaform stringers along foliation planes in the deformed granodiorite (i.e. sulphide stringers tend to follow the foliation planes). Occasionally, coarse free gold is observed associated with chloritic or epidote lined fractures that cross-cut the sulphide mineralization. The free gold may be due to secondary enrichment during a later hydrothermal process overprinting the main copper sulphide-gold event. Sulphide mineralization is always accompanied by variable amounts of magnetite mineralization and biotite alteration. While these minerals occur in the non-deformed rocks they are present in the mineralized horizons in a much greater abundance in the range of an order of magnitude greater than background.

Supergene mineralization occurs proximal to near-surface extension of the primary mineralization and beneath the Cretaceous conglomerate. Chalcocite is the prime mineral in these horizons along with secondary malachite, minor azurite and minor native copper. Observations of foliated and even copper mineralized cobbles in drilling indicate that “Minto-type” mineralization was exposed, eroded and reincorporated in conglomerate sedimentary deposits by the Cretaceous Age. Other rock types, albeit volumetrically insignificant, include thin dykes (typically less than 1 m) of simple quartz-feldspar pegmatite, aplite, and an aphanitic textured intermediate composition rock.

Structural deformation includes the ore-bearing deformation zones, as well folding present on the regional to micro-scale. Within the deformation zones the foliation exhibits highly variable orientations with the presence of small-scale (several centimetres in amplitude) folds. The ore–bearing zones are also occasionally folded on a scale of several hundred metres. The larger-scale folds appear to be gentle folds with north-south axial traces. Late brittle fracturing and faulting is noted throughout the property area, some of these faults have displacements significant enough to compartmentalize the deposits. For example, the Minto Creek fault bisects the Minto Main deposit, dividing it into north and south areas. The fault is modelled as dipping steeply north-northeast with an apparent left lateral reverse displacement. The DEF fault defines the northern end of the Main deposit. It strikes more or less east-west and dips north-northwest and cuts off the main zone mineralization. The boundary between the Area 2 and Area 118 ore zones is an intermediate NE dipping fault, and at least two parallel structures displace mineralized domains in Area 118. A similar NW striking fault zone appears to define the north-eastern boundary of the Ridgetop deposit, and defines the outcrop of Cretaceous conglomerate.

Pervasive, strong potassic alteration occurs within the flat lying zones of mineralization, and is the predominant alteration assemblage observed in all of the Minto Deposits. The potassic alteration assemblage is characterized by elevated biotite contents and minor secondary k-feldspar overgrowth on plagioclase relative to the more massive textured country rock. Additional alteration includes the replacement of mafic minerals by secondary chlorite, epidote, or sericite observed both in mineralized and waste rock interstitially or fracture/vein proximal, as well as variable degrees of hematization of feldspars. Minor carbonate overprint is occasionally observed associated with secondary biotite. Silicification is present but not pervasive in the Minto deposits.

Mineral exploration on the Minto property has been conducted intermittently since 1971. Subsequent to the discovery of the Minto Main deposit, which is currently in production, the adjacent southern half of the property has undergone systematic brownfields exploration. Exploration on the northern half is more sporadic.

There are currently more than 1,000 drill holes within a roughly 16 square kilometre area. As such, following up on open mineralized horizons in geological models, projecting mineralized horizons into areas of little or no drilling, and drilling near historical drill hole intercepts were the principal exploration tools employed by MintoEx and its geologists. Subsequent to Capstone’s predecessor, Sherwood Copper’s, acquisition of Minto Explorations Ltd. in June 2005, exploration from 2005 to 2010 has concentrated mostly on diamond drilling. However, an extensive historic soil sample survey and some ground based and airborne geophysics have been conducted and are very useful to guide drilling activity.

The current exploration approach by MintoEx is the systematic evaluation of modern electrical (chargeability); geophysical methods by commissioning various “proof-of-concept” surveys over know mineralization and then expanding survey coverage outward into untested areas using these methods that are calibrated to know deposits. An emphasis is placed on looking for signature analogs as opposed to being pedantic about precise measurements of response. The predominant electrical geophysical methods used are Gradient Array Induced Potential (GAIP), Dipole-Dipole Induced Potential, and Titan-24 DC Induced Potential. Drill targeting is predominantly based upon the coincidence of an anomaly in one of the electrical (chargeability) methods with an anomaly in the 1993 total field airborne magnetic survey (MAG).

Within the currently known extent of the Priority Exploration Corridor (“PEC”), future exploration programs will likely be more reliant solely on electrical / chargeability methods as the near-surface potential and discrete magnetic bull’s-eyes have largely been targeted. Magnetic data in areas located north of Minto North plus areas west and east respectively of the PEC may still be useful as these regions are still relatively under explored.

The current highest priority exploration targets are based on the evaluation of geophysics, soil geochemistry, geologic modelling, and diamond drilling. The targets identified as Ridgetop Southwest, Copper Keel (North and South), Airstrip, Connector, DEF, and the newly discovered Wildfire prospect are all located within a 2 km by 2 km area, south of the DEF fault. MintoEx also sees good exploration potential in the area north of the DEF fault, as evidenced by the discovery of the high grade Minto North deposit early in 2009 and the recently discovered Inferno prospect in late 2010.

In 2009, several other historic bedrock copper occurrences discovered in the 1970s north of the DEF fault were relocated and confirmed. In addition various copper-in-soil geochemical anomalies, often coincident with magnetic geophysical anomalies, occur throughout the property and many of them remain untested. However, further understanding of the bedrock geology north of the DEF fault is required before many of these targets can be properly assessed and placed in perspective.

Mineral Resources

A primary objective of SRK's work was to produce a revised independent mineral resource evaluation for the Area 2/118 and for the Ridgetop deposits. The Minto Main resource was reviewed and approved by SRK. The Minto North and East deposits, other integral parts of the Minto system, have been evaluated by Kirkham Geosystems Ltd.

The mineral resource estimate reported herein supersedes earlier mineral resource estimates presented in the 2009 Phase IV PFS Technical Report.

The mineral resource estimate in the Area 2/118 and Ridgetop deposits was completed by Dr. Wayne Barnett, Ph.D., Pr.Sci.Nat., an independent qualified person as this term is defined in National Instrument 43-101. The effective date of this resource estimate is August 30, 2010. Marek Nowak, P.Eng., analyzed the data, reviewed and validated the mineral resource estimates. The Minto North and East deposit resource estimates were completed by Garth Kirkham, P.Geo., of Kirkham Geosystems Ltd., an independent qualified person as this term is defined in National Instrument 43-101.

In the opinion of SRK, the block model resource estimate and resource classification reported herein are a reasonable representation of the mineral resources at Area2/118, Ridgetop, Minto Main, Minto North and Minto East deposits at the current level of sampling. The mineral resources presented herein have been estimated in conformity with generally accepted CIM *"Estimation of Mineral Resource and Mineral Reserves Best Practices"* guidelines and are reported in accordance with Canadian Securities Administrators' National Instrument 43-101. **Mineral resources are not mineral reserves and do not have demonstrated economic viability. Only Measured and Indicated mineral resources have been used in the preliminary feasibility study described in this report**.

The database used to estimate the Area 2/118 and Ridgetop deposits was audited by SRK and the mineralization boundaries were modelled by SRK based on lithological and structural interpretations. Kirkham audited the Minto North and Minto East database and modelled mineralization boundaries.

SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries of the mineralized domains and that the assaying data is sufficiently reliable to support estimating mineral resources.

The "reasonable prospects for economic extraction" requirement for a mineral resource generally implies that the quantity and grade estimates meet certain economic thresholds, and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. SRK considers that the Ridgetop and Minto North deposits are amenable for open pit extraction. The Area 2/118 deposit is amendable to both open pit and underground extraction while the East deposit is suitable for underground mining.

In order to demonstrate the reasonable prospect of economic extraction, SRK constrained the overall mineral resource with Whittle™ pit optimization software using the parameters shown in Table 1. The Cost Factor 1 shell was selected as the constraining surface and resources within the shell were calculated.

Table 1: Whittle Optimization Parameters for Resource Estimate Constraint

Constraining Parameter	Unit	Value
Copper Price	US$/lb	2.85
	C$/lb	3.17
Gold Price	US$/oz	900
	C$/oz	1000
Silver Price	US$/oz	12
	C$/oz	13.33
Exchange Rate	C$: US$	1.11
Mining Cost	C$/t mined	1.50
Processing and G&A Cost	C$/t milled	5.00
Royalty	%	0.5
Slope angles	degrees (overall)	50

The open pit resource is constrained by a Revenue Factor 1 optimized Whittle shell based on the NSR model and the parameters in Table 1. The mineral resource statements for the Main, Area 2/118, Ridgetop, Minto North and Minto East deposits are presented in Tables 2-6. A combined resource from all three deposits is presented in Table 7.

Table 2: Mineral Resource Statement at 0.5% Cu Cut-off for the Main Deposit, SRK Consulting December 31, 2010

Classification	Tonnes (Kt)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lb.)*	Contained Gold (K oz)*	Contained Silver (K oz)*
Measured (M)	2,030	1.18	0.40	4.49	52,704	26	293
Indicated (I)	643	0.86	0.19	4.16	12,217	4	86
Sub-total (M+I)**	**2,673**	**1.10**	**0.35**	**4.41**	**64,921**	**30**	**379**
Inferred	**25**	**0.61**	**0.13**	**2.72**	**337**	**0**	**2**

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

Table 3: Mineral Resource Statement at 0.5% Cu Cut-off for the Area 2/118 Deposit, SRK Consulting August 30, 2010

Classification	Tonnes (Kt)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lb.)*	Contained Gold (K oz)*	Contained Silver (K oz)*
Measured (M)	7,043	1.28	0.49	4.4	198,344	110	996
Indicated (I)	19,411	0.92	0.3	3.32	393,939	186	2,071
Sub-total (M+I)**	**26,454**	**1.02**	**0.35**	**3.61**	**592,283**	**296**	**3,066**
Inferred	**5,573**	**0.83**	**0.26**	**2.89**	**101,519**	**47**	**518**

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

Table 4: Mineral Resource Statement at 0.5% Cu Cut-off for the Ridgetop Deposit, SRK Consulting August 30, 2010

Classification	Tonnes (Kt)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lbs)*	Contained Gold (K oz)*	Contained Silver (K oz)*
Measured (M)	1,531	0.98	0.25	2.14	33,204	12.3	105
Indicated (I)	3,534	0.87	0.3	2.87	67,901	33.8	326
Sub-total (M+I)**	**5,064**	**0.91**	**0.28**	**2.65**	**101,104**	**46.2**	**431**
Inferred	**318**	**0.75**	**0.13**	**1.57**	**5,250**	**1.3**	**16**

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

Table 5: Mineral Resource Statement at 0.5% Cu Cut-off for the Minto North Deposit, Kirkham Geosystems December 1, 2009

Classification	Tonnes (Kt)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lbs)*	Contained Gold (K oz)*	Contained Silver (K oz)*
Measured (M)	1,844	2.15	1.11	7.7	87,530	66	456
Indicated (I)	264	1.04	0.6	5.76	6,055	5	49
Sub-total (M+I)**	**2,108**	**2.01**	**1.04**	**7.46**	**93,585**	**71**	**505**
Additional Inferred	**25**	**0.84**	**0.40**	**4.4**	**457**	**0**	**3**

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

Table 6: Mineral Resource Statement at 0.5% Cu Cut-off for the East Deposit, Kirkham Geosystems October, 2010

Classification	Tonnes (Kt)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lbs)*	Contained Gold (K oz)*	Contained Silver (K oz)*
Measured (M)	688	2.30	1.07	6.30	34,842	24	139
Indicated (I)	489	1.74	0.70	4.60	18,805	11	72
Sub-total (M+I)**	**1177**	**2.07**	**0.92**	**5.57**	**53,647**	**35**	**211**
Additional Inferred	**14**	**1.03**	**0.45**	**2.80**	**316**	**0**	**1**

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

Table 7: Combined Mineral Resource Statement at 0.5% Cu Cut-off for Main, Area 2/118, Ridgetop, North and East Deposits (Effective dates as per Tables 2-6)

Classification	Tonnes (Kt)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lbs)*	Contained Gold (K oz)*	Contained Silver (K oz)*
Measured (M)	13,136	1.40	0.57	4.71	406,624	239	1,989
Indicated (I)	24,341	0.93	0.31	3.33	498,917	240	2,604
Sub-total (M+I)**	**37,476**	**1.10**	**0.40**	**3.82**	**905,540**	**479**	**4,592**
Additional Inferred	**5,955**	**0.83**	**0.25**	**2.82**	**107,879**	**48**	**540**

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

Mine Production and Mineral Reserve Estimate

The Area 2/118, Ridgetop and Minto North (“Phase V”) deposits are proposed to be developed both as open pit (“OP”) and by underground (“UG”) methods, following completion of mining in the Minto Main deposit. The planning for this Pre-feasibility study assumes a start date of January 1, 2011. The proposed Main Pit mine plan (as provided by MintoEx) was incorporated into this pre-feasibility study.

Based on a start date of January 2011, the Main/Phase V open pit and underground mines will produce a total of 12.9 million tonnes (“Mt”) of ore (includes Main Pit stockpile balance as of beginning of 2011) and 58.5 Mt of waste over approximately an 7.5-year mine operating life ending in mid-2018. Approximately 2.4 Mt of ore is planned to be produced from UG mining at a rate of 2,000 tpd. Mill operations continue for an additional 2 years, processing the accumulated 2.0 Mt of ore stockpiled when mining ceases, for a total mill operating life of 9.5 years.

The life-of-mine (“LOM”) plan focuses on accessing and milling high-grade ore first, with lower grade material sent to stockpiles for blending and processing later in the mine life. This is based on repeated exploration success that has supported successive deferrals in the timing of the processing of this lower grade material, as additional higher grade mineralization is discovered and defined.

Mine design for the Phase V open pits and UG mine was initiated with the development of a Net Smelter Return (“NSR”) model. The mine model included estimates of metal prices (US$2.25/lb Cu or C$2.62/lb), exchange rate, mining dilution, mill recovery, concentrate grade smelting and refining payables and costs, freight and marketing costs and royalties. The NSR model was based on a 10 m x 10 m x 3 m block size for Phase V (see Table 8).

For the OP mine, Gemcom Whittle™ software was then used to determine the optimal mining shells for each of the deposits. Detailed mine design, planning and scheduling was then conducted on the optimal pit shells to produce the current pit designs.

Table 8: Summary of Whittle™ Parameters Used for Pit Design

Item	Unit	Value
Metal Prices and Exchange Rate		
Copper	US$/lb	2.25
Gold*	US$/oz	300.00
Silver*	US$/oz	3.90
Exchange rate	C$/US$	1.16
Processing		
Copper recovery to concentrate	max	92%
Gold recovery to concentrate	max	70%
Silver recovery to concentrate	max	80%
Copper grade in concentrate	%	40
Gold grade in concentrate	g/t	variable with Cu
Silver grade in concentrate	g/t	variable with Cu
Concentrate moisture content		8%
Smelter Payables		
Payable copper in concentrate	%	96.75
Payable gold in concentrate	%	per MRI contract
Silver deduction	g/t in conc	30.00
Remaining payable silver in concentrate	%	100%
Other Parameters		
Pit slope angles	overall	As per 2009 PFS
Dilution	%	8%
Mining recovery	%	100
Annual Plant Throughput	Mtpa	1,460,000
Costs		
Waste mining cost	C$/waste tonne	2.25
Ore mining cost	C$/ore tonne	2.25
Processing cost	C$/milled tonne	12.90
G&A cost	C$/milled tonne	11.90
Royalties	%	1.00%
Transport, marketing, ins, etc.	US$/dmt conc	162.40

**Base on terms of royalty stream agreement with Silver Wheaton*

UG mine planning also started with the NSR block model and then assumed a 1.2% Cu equivalent cut-off grade (“COG”) within Datamine™ MRO software to determine economic UG mining shapes. Small mineralized zones distant from any proposed access were excluded from the mine plan.

The mineral reserves estimate for both OP and UG are summarized in Table 9 below. The mineral reserve for Main Pit includes the ore stockpile balance predicted for the beginning of 2011 as well as proposed mining from 2011 going forward. The various estimated copper cut-off grades used are also noted in Table 9.

Table 9: Minto – Mineral Reserves by Class for Phase V

Deposit	Reserve Class	Tonnes (Mt)	Cut-off Grade (%Cu equiv.)	Diluted grade			Contained Metal		
				Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Au (koz)	Ag (koz)
Main*	Proven	2.25	0.62	1.35	0.46	4.79	67	33	346
	Probable								
	Sub-total	2.25	0.62	1.35	0.46	4.79	67	33	346
North	Proven	1.52	0.52	2.36	1.28	8.55	79	63	419
	Probable	0.005	0.52	2.25	0.81	9.38	0	0	2
	Sub-total	1,53	0.52	2.36	1.27	8.56	79	63	421
Ridgetop	Proven	0.63	0.54	1.10	0.25	2.05	15	5	41
	Probable	0.71	0.54	1.11	0.37	3.55	17	9	81
	Sub-total	1.34	0.54	1.11	0.32	2.85	33	14	122
Area 2	Proven	3.37	0.54	1.41	0.53	4.94	105	58	536
	Probable	1.45	0.54	1.08	0.31	3.59	35	15	167
	Sub-total	4.82	0.54	1.32	0.47	4.53	140	72	703
118	Proven								
	Probable	0.49	0.54	1.29	0.09	1.73	14	1	27
	Sub-total	0.49	0.54	1.29	0.09	1.73	14	1	27
Under-ground	Proven								
	Probable	2.44	1.20	1.90	0.82	6.71	102	64	527
	Sub-total	2.44	1.20	1.90	0.82	6.71	102	64	527
Total	Proven	7.77	0.56	1.56	0.63	5.37	266	158	1,343
	Probable	5.09	0.86	1.50	0.54	4.91	169	89	804
	Total	12.87	0.68	1.53	0.60	5.19	435	247	2,146

**includes stockpile balance of 1,631 kt at beginning of 2011 for Main pit but excludes approximately 0.25Mt of partially oxidized material from stockpile.*

The post-2010 mining sequence was divided into various stages. The first stage sees the completion of mining in the Main Pit followed by the first stage of Area 2 (maintaining Pelly as the mining contractor), followed by owner-operated mining of, Minto North, 118, second stage of Area 2, the two stages in Ridgetop, and finally the last 2 stages in Area 2. The underground production supplements the open pit mine ore feed. The stages were designed to provide the required ore per period, to maximize grade and defer stripping waste as long as possible. The Main and Phase V deposits, including the underground ore, are most economical when mined in sequence with the stripping of the Phase V pits beginning at the completion of mining in the current or Main Pit. The LOM mine production schedule is shown in Table 10 with the processing schedule summarized in Table 11.

Table 10: Phase V LOM Mine Production Schedule

Parameter	Unit	Total	2011	2012	2013	2014	2015	2016	2017	2018
Total OP/UG										
Overburden	kt	**14.10**	**3.55**	**2.04**	**0.87**	**0.98**	**0.50**	**1.44**	**4.57**	**0.15**
Rock	kt	**44.42**	5.48	8.51	7.21	4.73	6.96	6.09	3.25	2.19
Total Waste	kt	**58.52**	9.03	10.55	8.08	5.72	7.46	7.53	7.82	2.34
ROM ore	kt	**11.24**	0.82	1.97	1.01	2.53	1.55	1.51	1.01	0.82
Cu Grade	%Cu	**1.58**	1.47	1.56	1.73	2.14	1.36	1.40	1.14	1.07
Au Grade	g/t	**0.63**	0.61	0.58	0.80	0.98	0.51	0.53	0.31	0.33
Ag Grade	g/t	**5.33**	5.50	5.78	5.23	7.21	4.81	4.58	2.74	4.05
Total Mined Cu	Mlbs Cu	**391**	27	68	38	120	47	47	25	19
Total Mined Au	koz Au	**229**	16	37	26	80	26	26	10	9
Total Mined Ag	koz Ag	**1927**	145	366	170	587	240	223	89	107
ROM ore	t/day	**3,848**	2,243	5,392	2,770	6,940	4,260	4,150	2,776	2,252
Total Material	t/day	**22,937**	26,701	32,071	22,834	20,595	23,664	24,768	24,196	8,669

Table 11: Phase V LOM Process Production Schedule

Parameter	UNIT	2011-2020 Total	YEAR 2011	2012	2013	2014	2015	2016	2017	2018	2019	2020
Mill Feed Rate	dmt/day	**3,718**	3,442	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750
Mill Feed Total	Mt	**12.9**	1.256	1.373	1.369	1.369	1.369	1.373	1.369	1.369	1.369	0.653
Feed Grade	Cu %	**1.53**	1.60	1.86	1.70	2.86	1.62	1.68	1.11	0.96	0.78	0.78
	Au g/t	**0.6**	0.6	0.7	0.7	1.5	0.6	0.7	0.3	0.3	0.2	0.2
	Ag g/t	**5.2**	6.0	7.2	5.4	10.0	5.8	5.7	2.9	3.2	2.2	2.2
Recovery to Conc.	Cu	**92.0%**	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%
	Au	**70.0%**	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%
	Ag	**78.0%**	78.0%	78.0%	78.0%	78.0%	78.0%	78.0%	78.0%	78.0%	78.0%	78.0%
Conc. Grade	% Cu	**39%**	41.5%	38.0%	39.0%	38.0%	38.0%	38.0%	38.6%	38.6%	38.7%	38.7%
Conc. Production	dmt	**470,478**	44,633	61,728	54,846	94,937	53,519	55,956	36,066	31,201	25,449	12,134
Conc. Metal	Mlb Cu	**400.4**	40.8	51.7	47.2	79.5	44.8	46.9	30.7	26.6	21.7	10.4
	oz Au	**173,146**	16,807	22,531	22,259	45,118	19,488	20,538	9,375	8,246	5,948	2,836
	oz Ag	**1,673,940**	188,612	246,680	184,654	341,791	197,647	194,509	100,627	110,290	73,897	35,234

Waste Management and In-pit Tailings Disposal

Tailings from the mill will be sent to the currently permitted existing dry-stack location for the life of the Main Pit (to mid-2011). Upon completion of mining in the Main Pit, thickened tailings generated from processing ores from other Phase V deposits will then be deposited into the Main Pit. The permit application for the deposition of tailings into the Main Pit was part of the Phase IV permit that was filed in August 2010 and is assumed to be approved in March 2011. Additional capacity required to store approximately 700,000 cubic metres of water associated with freshet flows, plus incremental storage to meet minimum and maximum operational requirements has been taken into consideration.

Further in-pit tailings storage capacity becomes available once Area 2 is mined and this Area 2 storage capacity will be required in order to hold a portion of the tailings to be produced from the Phase V LOM plan. Ridgetop North is also used as an in-pit storage facility until mining is completed in the final stages of Area 2.

This deposition of tailings into the Area 2 and Ridgetop North Pits will form part of the Phase V permit application to be submitted early in 2011 and is assumed to be approved before the 2nd quarter of 2012.

Although these tailings deposition plans are not yet permitted, they offer a potentially viable solution to tailings disposal that provides backfill material for the Main, Area 2 and Ridgetop North pits, reduces the amount of disturbed land that would normally be required by mining of the Phase V deposits, and provides a significant cost savings over the current dry-stack method.

Waste rock from the current Main pit, as well as a significant portion of the Phase V deposits, will be deposited in an expansion of the existing permitted West Valley Fill waste dump located in the lower valley southwest of the Main pit. In addition, waste rock from Minto North is proposed to be stacked onto the existing Main pit dump, while some waste material from the Phase V deposits will be deposited in a proposed Mill Valley dump to the east of the existing mill facilities. Waste rock material from Area 2 will also be placed in the Main pit to act as a south wall buttress. Backfilling of Ridgetop South and 118 pits will also provide waste storage capacity and will add to the final reclamation plan. Overburden material will be placed in temporary dumps adjacent to the various deposits and used for final reclamation. Any excess overburden will be added to existing Overburden dump.

Metallurgical Test Work

Metallurgy testing, by G&T Metallurgical Services LTD ("G&T") during 2010, was performed on three potential new zones at the Minto mine site. The zones were Copper Keel, Minto East and Wildfire.

The main objectives of the test program were:

- Determine the material content and fragmentation properties of the three deposits;
- Investigate ore hardness properties for the composites;
- Determine bulk density distribution on a select group of core samples;
- Investigate the flotation response for samples using open circuit and locked cycle testing; and
- Determine the concentration of deleterious minor elements in the final copper concentrates.

The test work campaigns conducted by G&T Metallurgical Services Ltd. in 2009 and 2010 have demonstrated performance consistent with the current Main Pit ore flotation characteristics.

Due to their stage of development the Copper Keel, Inferno and Wildfire zones have not been included in the most recent mine plan. The test work results have been reported, however the three zones have not been considered when evaluating the process plant design.

In addition to Minto East the latest mine plan includes material from Minto Main, Minto North, Minto South, Ridgetop East and Area 2/118. Metallurgy test work results for these deposits can be found in the 2009 Phase IV PFS.

Process Plant

The process design for this pre-feasibility study is based on treating ore with similar hardness to the current Minto Main ore being processed, or similar to that tested by DJB Consultants in October 2007.

The throughput selected is a function of the existing Minto plant milling circuit capacity. Ausenco Minerals Canada Inc. ("Ausenco") has modelled the current plant and predicted a throughput of 171 dry metric tonnes per hour based on a portion of the SAG mill feed being crushed to 80% passing 25mm in a pre-crushing circuit.. An average of 3,750 tonnes per day will be processed at a design availability of 91.3%.

The key criteria selected for the plant design are:

- Treatment of an average 3,442 dry metric tonnes per day for 2011, increasing to 3,750 dry tonnes per day for 2012 and beyond;
- Material from Minto Main, Minto North, Mino East, Minto South, Ridgetop East and Area 2/118 will be processed through the Minto plant;
- Design availability of 91.3%, being 7,997 operating hours per year, with standby equipment in critical areas, and
- Sufficient plant design flexibility for treatment of all ore types as per test work completed at design throughput.

Environmental Assessment and Licensing

In the Yukon, mining projects require an environmental assessment prior to the issuance of significant operating permits for mining, including a Type A Water Use License and a Quartz Mining Production Licence. Elements of the Minto Project have undergone environmental assessment under three different federal and territorial assessment bodies. A previous milling and mining rate increase (2008) and the Phase IV expansion (2010) have also been assessed under the current regime, the Yukon Environmental and Socioeconomic Assessment Board (YESAB). The project is currently (February 2010) about to enter the assessment process again for the Phase V expansion project.

The major instruments or authorizations permitting and governing operations for the project include Type A and B Water Use licences, issued by the Yukon Water Board, a Quartz Mining Licence issued by Yukon Government, Energy Mines and Resources, and an Authorization to Deposit a Deleterious Substance under the federal Metal Mining Effluent Regulations.

The expansion of the Minto Mine in the Phase IV development required an environmental assessment under YESAA and major licence amendments all of which are expected to be approved in the 1st Qtr 2011. Water management planning, as expected, is of particular interest to the assessors. The amendment to the Water Use Licence is also expected to be approved in the 1st Qtr 2011.

Selkirk First Nation

MintoEx claims continue to lie within Selkirk First Nation (SFN) Category A Settlement Lands (Parcel R-6A), where both surface and mineral rights are reserved for SFN and the SFN are afforded the rights to exercise certain powers over land use and environmental protection. Therefore, if any of the Minto Exploration claims are allowed to lapse, they cannot be re-staked, and the surface and mineral rights would revert to the SFN. In addition, the mine access road lies within parcels Parcel R-6A and Parcel R-44A, and the east barge landing access point lies on Parcel R-43B.

On September 16, 1997, the company and the SFN entered a Cooperation Agreement concerning the Minto Project with respect to the development of the Minto Mine. This agreement was amended (November 4, 2009). In addition to establishing cooperation with respect to permitting and environmental monitoring, this confidential document deals with other economic and social measures and communication between Selkirk First Nation and the company. This agreement will continue to guide SFN involvement in the project as mine expansion planning and development proceeds.

Environmental Conditions

Environmental conditions pre-mine development have been compiled, assessed and referenced in previous environmental assessments, but the environmental assessment and permitting process for the Phase IV expansion will require that these conditions be further updated based on recent site monitoring program results.

Specifically, baseline environmental conditions of the drainage to the north of the Minto Creek drainage will be of interest to assessors, as the Minto North deposit is located approximately 100 m into the drainage. Although physically there will likely be minimal disturbance in this drainage from the mining activities, there is potential for there to be effects to the aquatic receiving environment downstream.

An updated Environmental Conditions report has been completed that will support the Phase IV development which updated all environmental data for the project area and is being used for the assessment and permitting processes that are projected to be approved in the 1st Qtr 2011.

Water Management and Effluent Discharge

MintoEx, in its original water licence application submitted in 1996, outlined a water management plan based on the limited baseline information and project projections available for the Minto Mine at the time. In the intervening period since the application, screening and issuance of the Type A water use licence, significant additional baseline and operational data have been collected. These data show that the conditions upon which the initial water management and treatment assumptions were predicated were not representative of actual conditions observed.

MintoEx has therefore revised the site Water Management Plan and has submitted an environmental assessment Project Proposal and Water Use Licence amendment request to authorize the implementation of a new water management strategy. This includes the construction and use of storm water diversions, a water treatment plant and revised project effluent discharge standards.

Although the major elements of these water management revisions were designed to be functional beyond the mining of the Main Pit and into mine expansion proposed for the Phase IV and Phase V developments, the plan will require further reassessment during the Phase V development planning process.

The critical consideration with respect to water management for Phase V planning will be contingency runoff storage of water requiring treatment of settling prior to discharge and ensuring that effects to the unnamed drainage for the Minto North deposit are minimized and fully mitigated. Water treatment will continue to be a critical component of the water management strategy into the Phase V expansion, as it is in the currently proposed water management plan.

Closure Planning

Closure philosophies and measures for the Phase V mine plan will mirror those presented in the previously submitted and approved closure plans. Although closure and reclamation concepts will be required for the Phase V environmental assessment and attendant authorization amendments, it is expected that actual details (including closure cost estimates) will be presented in a subsequent revision of the closure plan on the existing Quartz Mining Licence schedule (every 2 years on the anniversary of the mill start up – August 1). Revisions to the closure plan reflecting the Phase V mine plan would not be required until the amendments to the Water Use Licence and Quartz Mining Licence authorizing mining and milling activities in the Phase V deposits are issued, as the closure plan applies to authorized mining activities and plans.

Closure measures for the site following the completion of the Phase V mine plan are expected to generally follow those currently authorized.

Metal Leaching/ Acid Rock Drainage

Characterization of mine rock and tailings from the Area 2/118, Ridgetop, and Minto North deposits has shown that there is sufficient neutralization potential (NP) to offset the acid potential (AP) within the waste materials. Both bulk mine rock and tailings had NP/AP>3 and the majority of mineralized rock samples tested also had NP/AP > 3.

A small proportion of the mineralized waste has lower NP/AP values (a single sample had NP/AP < 1) indicating that localized pockets of potentially acid generating rock do exist. Overall, however, the Phase V characterization results indicate that waste management planning does not need to take prevention of acid rock drainage (ARD) into consideration.

Bulk mine rock has elemental concentrations typical of granitic rocks, therefore metal leaching from bulk waste is not expected to be environmentally significant. Mineralized waste has elevated concentrations of copper and other trace elements. Segregation of mineralized waste with elevated copper and disposal in a way the limits copper leaching (e.g. co-disposal with in-pit tailings) will be required to minimize loadings to the receiving environment over the long term.

Operating Costs

Table 12 presents a summary of the operating costs by major area, while Table 13 summarizes the capital costs.

Table 12: Unit Operating Costs by Major Area

Area	Unit	Cost Estimate
Open Pit Mining	$/t mined	**2.57**
	$/t milled	**13.37**
Underground Mining	$/t milled	**35.17**
Total Mining (weighted average)	$/t milled	**20.04**
Processing	$/t milled	**12.94**
General, administration, camp, royalties	$/t milled	**12.13**
Total	**$/t milled**	**45.11**

Capital Costs

Table 13 shows the capital costs without closure costs. A closure cost allowance of $16M was used in the cash flow analysis, as per an estimation done in 2010. The 2009 PFS closure cost allowance was $20M and was a very preliminary estimate. The 2010 estimate was done in more detail and is considered to be more accurate.

Table 13: Capital Costs by Major Area

Area	Unit	Cost Estimate
OP mining equipment fleet	M$	**32.0**
UG equipment (fixed and mobile)	M$	**18.3**
UG development	M$	**15.8**
Process plant	M$	**5.0**
Contingency	M$	**3.1**
Sustaining Capital	M$	**1.8**
TOTAL CAPITAL COST	M$	**76.0**

Contingency capital is relatively low due to good quality recent mobile equipment and UG development expenditure estimates. Sustaining capital is relative low due to the short mine life that avoids major re-builds on mobile equipment and also replacement of various components in the process pant during the mill expansion project.

Economics

The estimated economic benefit of mining the Minto Phase V deposits is sufficient to continue with the company's expansion plans. While more detailed work will be required to optimize the project, there is adequate economic justification for MintoEx to proceed with further work and, in particular, the application for licence and permit amendments from the Yukon Government.

Table 14 shows the comparison of Phase V PFS economic cases. The Phase V deposits add economic benefit to the mine, yielding pre-tax $NPV_{7.5\%}$ as follows:

	Pre-tax $NPV_{7.5\%}$	After-tax $NPV_{7.5\%}$
• Case A (Base Case) (US$2.75/lb Cu):	$284M	$206M
• Case B (US$2.25/lb Cu):	$180M	$142M
• Case C (variable US$3.60/lb Cu to US$2.25/lb Cu):	$266M	$194M

Case B represents the metal price used in the mineral reserve estimate.

The break-even copper price for the project is US$1.80/lb or C$1.96/lb.

Table 14: Comparison of Phase V Economic Cases

Item	Unit	Case A	Case B	Case C
Waste mined	Mt	58.5		
Ore mined	Mt	11.2		
Total mined	Mt	69.8		
Mill Feed*	Kt	12.9		
Copper millhead grade	% Cu	1.53		
Gold millhead grade	g/t Au	0.60		
Silver millhead grade	g/t Ag	5.2		
Copper in cons	Mlb	400		
Gold in cons	Koz	173		
Silver in cons	Koz	1,674		
Concentrate Grade	% Cu	39		
Base Copper Price (ex. hedging)	US$/lb	2.75	2.25	3.60 to 2.25
Ave. Copper Price (inc. hedging)	US$/lb	2.73	2.25	2.65
Gold price (inc. hedging)	US$/oz	331	324	333
Silver price (inc. hedging)	US$/oz	3.90	3.90	3.90
Exchange rate	C$/US$	1.09	1.16	1.08
NSR (inc. royalties)	C$/t milled	84.29	73.99	80.63
Unit Total OPEX	C$/t milled	45.11		
Unit On-site OPEX	C$/lb Cu payable	1.50		
Unit Off-site OPEX (ex. royalty)	C$/lb Cu payable	0.30	0.31	0.30
Unit By-product Credit	C$/lb Cu payable	(0.16)	(0.15)	(0.16)
Unit OPEX net by-product credits	C$/lb Cu payable	1.33	1.32	1.33
Total Capital (initial & sustaining)	$M	76		
Allowance for closure cost	$M	16		
$NPV_{7.5\%}$ pre-tax	$M	284	180	266

**Note Mill Feed includes Ore Stockpile*

Sensitivity analyses were run on all Cases for Cu grade, Cu price, capital expense (“CAPEX”), and operating expense (“OPEX”). Each variable was changed from -20% to +20% of the base value and the resultant PT -$NPV_{7.5\%}$ values were graphed (Figure 1 for Case A). Each variable was changed independently of the other variables eliminating the compounding effect of multiple variable modifications. As expected all Cases showed high variability to copper price and grade, moderate sensitivity to operating costs and low sensitivity to capital costs.

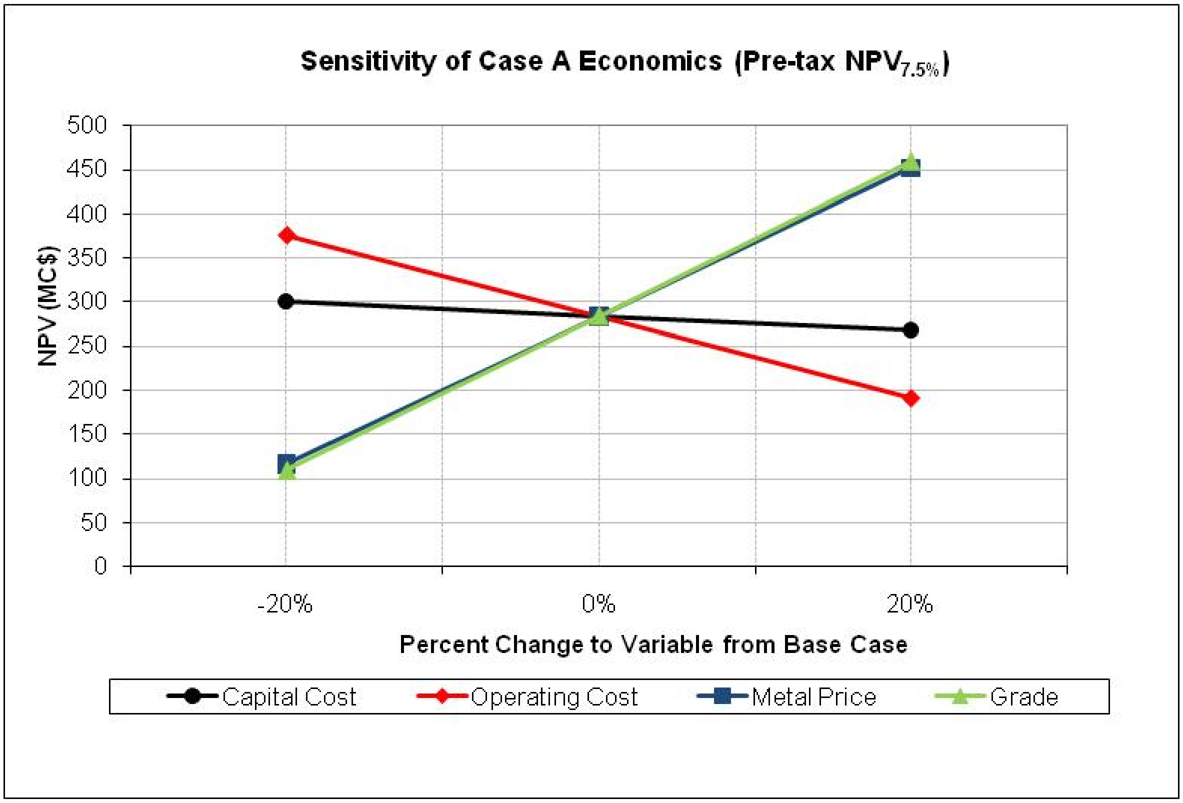

Figure 1: Case A Pre-tax $NPV_{7.5\%}$ Sensitivities

Conclusions

The conclusions of note are:

- The Minto deposits, Phase V pits and underground, contain a significant mineral reserve. The current mining in the Main Pit has helped confirm the expected grade and extent of the mineral reserves and the detailed drilling has provided a further measure of confidence in the mineral reserve estimate.
- The Phase V deposits are estimated to be economic to exploit and, according to the assumptions of this study, add value to the Minto mine by increasing the NPV of the overall project.
- There are strong exploration targets in the immediate vicinity of the known deposits and management has demonstrated its ability and commitment to explore for new deposits.
- Based on the preliminary test work conducted to date, the Phase V waste rock does not appear to have any ARD issues.

The major risk areas identified in this study are:

- Timing and approval of Phase IV and Phase V mine permit revisions;
- Exchange rates, metal prices and external influences;
- The ability to develop the UG mine as per the mining schedule;
- The ability to transform open pit mining operations to an owner-operated fleet;
- Acquisition of experienced personnel for underground and open pit mining; and
- The ability to maintain minimum dilution through effective grade control practices.

The most important opportunities to improve the project are:

- Optimization of mine plan;
- Continued strong demand for copper resulting in sustained high copper prices; and
- Discovering new mineral resources and converting them to mineral reserves.
- Expansion potential

Risks associated with the process plant include:

- The secondary crusher (S4800) installed by MintoEx does not facilitate screening of the feed material prior to the crusher to remove fines. The name plate capacity of the S4800 cone crusher (205 tph) is below the required capacity of 228 tph.
- The design for the plant throughput increase is based on a pre-crushing a portion of the SAG Mill feed to an F_{80} of less than 25mm. This is significantly finer than the current crushing circuit product size of 75 mm. There has not been any material flow test work on this size material. The impact the finer size will have on the draw down angles of the ore into the coarse ore reclaim feeder chute, and therefore the live stockpile capacity are uncertain.

The following measures are proposed to reduce the project risk associated with the process plant:

- Install a scalping screen and a belt conveyor prior to the secondary crusher. This will allow the site to feed divert jaw crusher product (8" maximum size) directly to the SAG Mill and also improves the overall operation and throughput of the crushing circuit. This is planned for 2011.
- An opportunity exists to review the crushed ore properties through further test work and/or experience in operating the recently installed secondary crusher. Stockpile live capacity may be increased by installing a second reclaim feeder. A second feeder will have the added benefit of providing improved blending to the SAG mill and operating redundancy.
- The comminution test work completed is suitable for this level of study. Additional comminution test work is recommended for future stages of the project to confirm the assumptions relating to SAG mill throughput made in this report.

The following opportunities exist to improve the project economics:

- A conceptual level review was completed on a potential Phase V plant upgrade to 7,500 tonnes per day. The review indicated that the plant operating cost could be further lowered to C$9.20/t based on a C$27 million capital expenditure. This estimate excludes capital cost associated with the mine and associated infrastructure, water supply, access roads or tailings storage facility. Both the operating and capital cost estimates are at an accuracy of ± 40% and would require further investigation.

Recommendations

Detailed recommendations of this PFS are contained in Section 27 of this report. The main recommendations of note are:

- Further exploration drilling is recommended to further define drilled targets that indicate anomalous metal values, in particular, deeper targets that could have underground mining potential are under-explored;
- Further tailings settling tests need to be undertaken to confirm assumed deposition densities;
- Optimization of the PFS mine plan should be undertaken to obtain smoother production and grade curve;
- Conduct further waste rock dump geotechnical engineering studies to test all assumptions made in this and other reports.
- Conduct further laboratory tests on representative samples of the tailings to evaluate the initial settled density and the density under loading.

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1 Introduction

This technical report was compiled for Minto Explorations Ltd. (“MintoEx”) by SRK Consulting (Canada) Inc. (“SRK”) to describe:

- New mineral resource and reserve estimates;
- The new life-of-mine plan that includes underground mining;
- Plant capacity improvement information;
- Changed tailings deposition methodology; and
- Updated cost and economic results.

Personal visits to the Minto Mine were conducted by six of the nine Qualified Persons (“QPs”) shown in Table 1.1.

Table 1.1: QP Site Visits

Name of QP	Area of Responsibility	Report Section Responsibility
Scott Carlisle	UG Rock Mechanics	Sections: 17.2
Gordon Doerksen	Overview, Economics and Environment	Sections: Summary, 1 to 4, 14, 20 to 23, 24.1, 24.2, 25, 27.2 and 28 to 31
Dino Pilotto	OP Mining	Sections: 16.7, 16.8, 18.2, 18.3.1 to 18.3.3, 18.4.1, 18.4.2, 24.1.2, 24.2.2 and 26.3
Cam Scott	Waste Management	Sections: 18.4.3 and 26.4
Mike Levy	Open Pit Mining Rock and Soil Mechanics	Sections: 17.1 and 27.3
Wayne Barnett	Geology and Resource Estimation	Sections: 5 to 12 and 26.2
Marek Nowak	Geology and Resource Estimation	Sections 13 and 16.1 to 16.4
Iouri Iakovlev*	UG Mining	Sections: 16.9, 18.1, 18.3.4, 24.1.1 and 24.2.1
Dave Brimage*	Metallurgy and Mineral Processing	Sections: 15, 19, 24.1.3, 24.2.3, 26.1 and 27.1
Garth Kirkham*	Minto North and East Resource Estimation	Sections: 16.5 and 16.6

**No site visit*

This report relies on a broad range of information and data provided to SRK by MintoEx, including the exploration database with detailed assay and geology data from drilling and geophysical surveys. SRK reviewed and performed reasonable independent checks and validations on a portion of the Minto exploration database.

Additionally, MintoEx provided contract details, government agreements, advice on local labour rates and conditions as well as budget operating costs estimated for 2011. SRK has assumed and has no evidence to doubt that MintoEx has acted in good faith and accurately provided all relevant data on the project.

Any previous technical reports or literature used in the compilation of this report are referenced throughout the text.

All units in this report are based on the International System of Units (“SI”) and all currency values are Canadian dollars (“C$” or “$”) unless otherwise noted.

This report uses many common abbreviations and acronyms with explanations found in Section 30.

2 Reliance on Other Experts

The preparation of this report is based upon public and private information provided by MintoEx and on information provided in various previous Technical Reports listed in Section 29 of this report.

The report also relies upon the work and opinions of non-QP experts. The following list outlines the information provided by other experts, who are independent to the authors (the appropriate QPs accept responsibility for the information provided below as defined in their QP Certificates):

- Vivienne McLennan of MintoEx for exploration and land tenure databases and assisting in QA/QC; (Sections 11 to 13);
- Brad Mercer and Taras Nahnybida of MintoEx for assistance with geology, exploration and QA/QC; (Sections 5 to 13)
- Scott Keesey of Access Consulting Group contributed to Section 22 of this report;
- Jaime Delgado, formerly of MintoEx, for the 2011 operating budget;
- Wentworth Taylor, CA of W.H. Taylor Inc., for corporate tax information specific to Minto contributions to Sections 23 and 25 of the report;
- Metallurgical testing conducted by G&T Metallurgical Services Ltd;
- ARD-ML work completed by SGS Canada Inc;
- B. Ross Design Inc. ("BRDI") for processing design and costing; and
- BESTECH for electrical system description and costing.

The authors believe that the information provided and relied upon for preparation of this report is accurate at the time of the report and that the interpretations and opinions expressed in them are reasonable and based on current understanding of mining and processing techniques and costs, economics, mineralization processes and the host geologic setting. The QPs have made reasonable efforts to verify the accuracy of the data relied on in this report.

The results and opinions expressed in this report are conditional upon the aforementioned information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions made herein the authors reserve the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to the authors subsequent to the date of this report.

3 Property Description

The Minto Mine is located in the Whitehorse Mining District in the central Yukon Territory. The property is located approximately 240 km northwest of Whitehorse, the Yukon capital. (see Figure 3.1). The project consists of 164 Quartz Claims covering an area of approximately 2,760 ha.

Figure 3.1: Location Map

The project is roughly centred on NAD 83, UTM Zone 8 coordinates 6,945,000 mN, 385,000 mE. The Minto Mine can be located on the Yukon Government Department of Energy, Mines and Resources 1:30,000 scale Mining Claims Map number 115I11, May 19, 2009. See Figure 3.2 for a portion of the map showing the boundaries of the Minto Explorations Ltd. claims.

The Mine is located on the west side of the Yukon River on Selkirk First Nation (SFN) Category A settlement land (SFN Parcel R-6A).

The 100% registered owner of the claims is Minto Explorations Ltd., a 100% owned subsidiary of Capstone Mining Corp. The current status of the claims is shown in Table 3.1 as per the Yukon Government Energy, Mines and Resources Mining Claims Search website. The status of the claims has been recently confirmed with the Mining Recorder.

The lease, but not the claim boundaries, have been surveyed by an authorized Canada Lands Surveyor in accordance with instructions from the Surveyor General.

There are no known back-in rights, payments or other agreements or encumbrances to which the property is subject other than a recently amended Cooperation Agreement with the Selkirk First Nations ("SFN") and a net smelter royalty payable to the SFN.

Table 3.1: Minto Explorations Ltd. Claim Status*

Grant Number	Reg Type	Claim Name	Claim No.	Operation Recording Date	Claim Expiry Date	Status	Quartz Lease	Ops Number
Y 61620	Quartz	MINTO	1	8/9/1971	5/13/2018	Active	OW00001	500057691
Y 61621	Quartz	MINTO	2	8/9/1971	5/13/2018	Active	OW00002	500057692
Y 61622	Quartz	MINTO	3	8/9/1971	5/13/2018	Active	OW00003	500057693
Y 61623	Quartz	MINTO	4	8/9/1971	5/13/2018	Active	OW00004	500057694
Y 61624	Quartz	MINTO	5	8/9/1971	5/13/2018	Active	OW00005	500057695
Y 61625	Quartz	MINTO	6	8/9/1971	5/13/2018	Active	OW00006	500057696
Y 61626	Quartz	MINTO	7	8/9/1971	5/13/2018	Active	OW00007	500057697
Y 61627	Quartz	MINTO	8	8/9/1971	5/13/2018	Active	OW00008	500057698
Y 61628	Quartz	MINTO	9	8/9/1971	5/13/2018	Active	OW00009	500057699
Y 61629	Quartz	MINTO	10	8/9/1971	5/13/2018	Active	OW00010	500057700
Y 61630	Quartz	MINTO	11	8/9/1971	5/13/2018	Active	OW00011	500057701
Y 61631	Quartz	MINTO	12	8/9/1971	5/13/2018	Active	OW00012	500057702
Y 61632	Quartz	MINTO	13	8/9/1971	5/13/2018	Active	OW00013	500057703
Y 61633	Quartz	MINTO	14	8/9/1971	5/13/2018	Active	OW00014	500057704
Y 61634	Quartz	MINTO	15	8/9/1971	5/13/2018	Active	OW00015	500057705
Y 61635	Quartz	MINTO	16	8/9/1971	5/13/2018	Active	OW00016	500057706
Y 61693	Quartz	DEF	1	8/23/1971	10/7/2028	Active	OW00230	500057707
Y 61694	Quartz	DEF	2	8/23/1971	10/7/2028	Active	OW00231	500057708
Y 61695	Quartz	DEF	3	8/23/1971	10/7/2028	Active	OW00232	500057709
Y 61696	Quartz	DEF	4	8/23/1971	10/7/2028	Active	OW00233	500057710
Y 61697	Quartz	DEF	5	8/23/1971	10/7/2028	Active	OW00234	500057711
Y 61698	Quartz	DEF	6	8/23/1971	10/7/2028	Active	OW00235	500057712
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Y 61706	Quartz	DEF	14	8/23/1971	10/7/2028	Active	OW00241	500057720
Y 61707	Quartz	DEF	15	8/23/1971	10/7/2028	Active	OW00242	500057721
Y 61708	Quartz	DEF	16	8/23/1971	10/7/2028	Active	OW00243	500057722
Y 61709	Quartz	DEF	17	8/23/1971	10/7/2028	Active	OW00244	500057723
Y 61710	Quartz	DEF	18	8/23/1971	10/7/2028	Active	OW00245	500057724
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Grant Number	Reg Type	Claim Name	Claim No.	Operation Recording Date	Claim Expiry Date	Status	Quartz Lease	Ops Number
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Y 61932	Quartz	MINTO	29	8/31/1971	3/1/2013	Active		500057938
Y 61933	Quartz	MINTO	30	8/31/1971	3/1/2013	Active		500057939
Y 61934	Quartz	MINTO	47	8/31/1971	5/13/2018	Active	OW00025	500057940
Y 61935	Quartz	MINTO	48	8/31/1971	5/13/2018	Active	OW00026	500057941
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Y 61939	Quartz	MINTO	52	8/31/1971	5/13/2018	Active	OW00030	500057945
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Y 61981	Quartz	DEF	36	9/8/1971	3/1/2013	Active		500057961
Y 61982	Quartz	DEF	37	9/8/1971	10/7/2028	Active	OW00250	500057962
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Y 61985	Quartz	DEF	40	9/8/1971	3/1/2013	Active		500057965
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Y 61989	Quartz	DEF	44	9/8/1971	3/1/2013	Active		500057969
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Y 61996	Quartz	DEF	51	9/8/1971	3/1/2013	Active		500057976
Y 61997	Quartz	DEF	52	9/8/1971	3/1/2013	Active		500057977
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Y 62016	Quartz	DEF	71	9/8/1971	3/1/2013	Active		500057996
Y 62017	Quartz	DEF	72	9/8/1971	3/1/2013	Active		500057997
Y 62018	Quartz	DEF	73	9/8/1971	3/1/2013	Active		500057998
Y 62019	Quartz	DEF	74	9/8/1971	3/1/2013	Active		500057999
Y 62020	Quartz	DEF	75	9/8/1971	3/1/2013	Active		500058000
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Y 62312	Quartz	MINTO	82	9/22/1971	3/1/2013	Active		500058020
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Grant Number	Reg Type	Claim Name	Claim No.	Operation Recording Date	Claim Expiry Date	Status	Quartz Lease	Ops Number
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Y 62315	Quartz	MINTO	85	9/22/1971	3/1/2013	Active		500058023
Y 62316	Quartz	MINTO	86	9/22/1971	3/1/2013	Active		500058024
Y 62317	Quartz	MINTO	87	9/22/1971	3/1/2013	Active		500058025
Y 62318	Quartz	MINTO	88	9/22/1971	3/1/2013	Active		500058026
Y 62319	Quartz	MINTO	89	9/22/1971	3/1/2013	Active		500058027
Y 66779	Quartz	DEF	79	7/11/1972	10/7/2028	Active	OW00252	500058071
Y 66780	Quartz	DEF	80	7/11/1972	10/7/2028	Active	OW00253	500058072
Y 66781	Quartz	DEF	81	7/11/1972	10/7/2028	Active	OW00254	500058073
Y 66782	Quartz	DEF	82	7/11/1972	10/7/2028	Active	OW00255	500058074
Y 66783	Quartz	DEF	83	7/11/1972	10/7/2028	Active	OW00256	500058075
Y 66784	Quartz	DEF	84	7/11/1972	10/7/2028	Active	OW00257	500058076
Y 76953	Quartz	DEF	1379	8/31/1973	10/7/2028	Active	OW00258	500058311
Y 76954	Quartz	DEF	85	8/31/1973	3/1/2013	Active		500058312
Y 76955	Quartz	DEF	86	8/31/1973	3/1/2013	Active		500058313
Y 76956	Quartz	DEF	87	8/31/1973	3/1/2013	Active		500058314
Y 77310	Quartz	MINTO	94	10/1/1973	3/1/2013	Active		500058315
Y 77311	Quartz	MINTO	95	10/1/1973	3/1/2013	Active		500058316
Y 78024	Quartz	MINTO	96	11/13/1973	3/1/2013	Active		500058317
Y 78025	Quartz	MINTO	97	11/13/1973	3/1/2013	Active		500058318

**All claims are in the Whitehorse District and 100% owned by Minto Explorations Ltd.*

Information taken from the Yukon Government Department of Energy, Mines and Resources Mining Claims Search website.

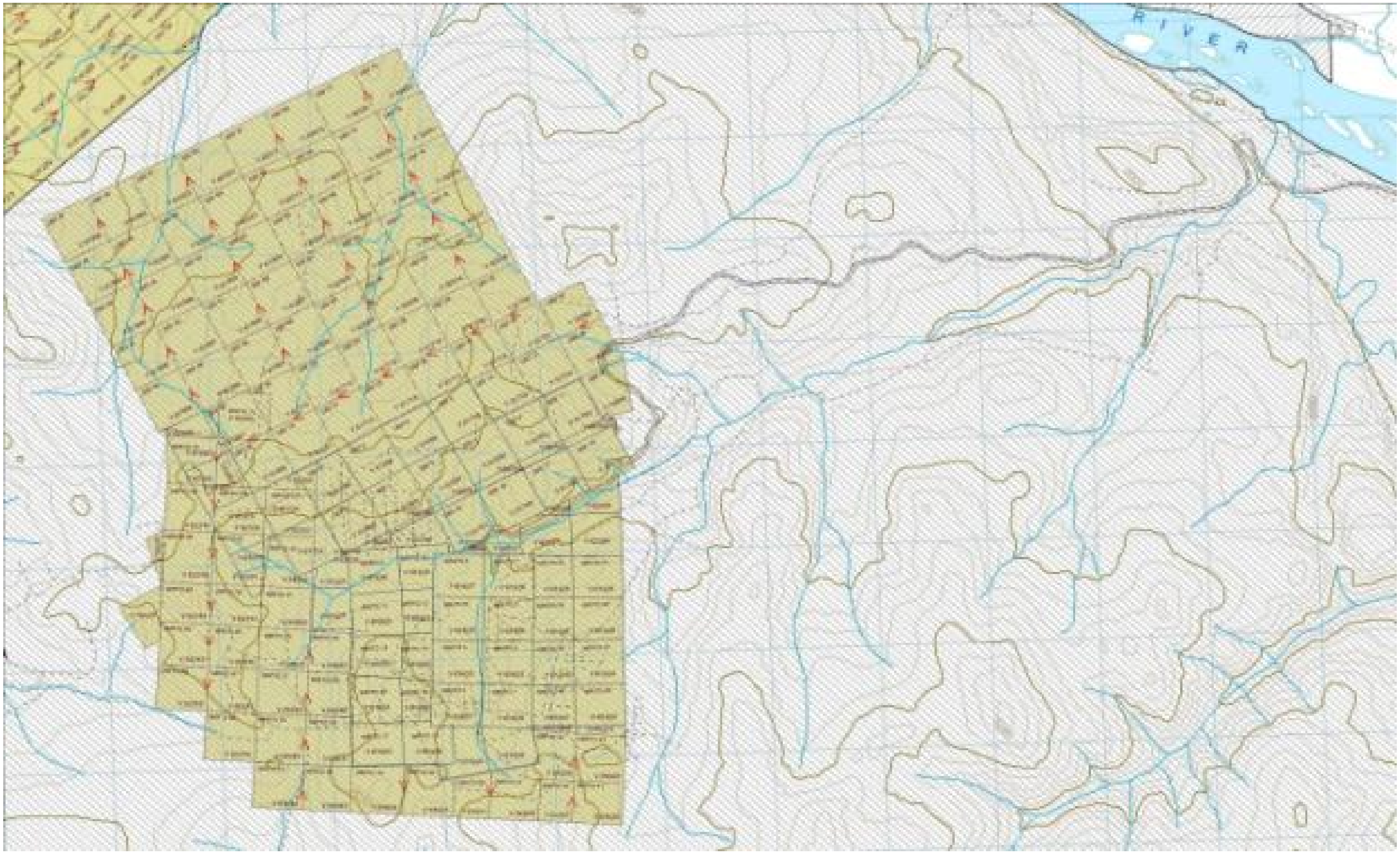


Figure 3.2: Mineral Claims Location Map

4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

4.1 Accessibility

The Minto Mine is accessible via the Klondike Highway (No. 2) to Minto Landing on the east side of the Yukon River, at Minto Landing, the mine operates a barge across the river in the summer months and constructs an ice bridge in the winter. The barge has the capacity to carry one B-train transport trailer and truck (see Figure 4.1). There is typically a 6 to 8 week period during each break-up and freeze-up of the Yukon River when there is no access across the river. A 27 km long, all-weather gravel road provides access from the west side of the Yukon River to the project site. The mine access road crosses one major tributary of the Yukon River, Big Creek, via a single-lane steel span bridge made with reinforced concrete abutments and deck. The highway, river crossing and gravel mine access road are suitable for heavy transport traffic.

When access across the Yukon River is available, operations personnel are transported to the site in commercial buses based out of Whitehorse. During the river freeze and thaw periods, personnel are transported from Whitehorse via charter air services that land on the 1,300 m airstrip located at the mine.

Figure 4.1: Minto Barge Crossing the Yukon River

4.2 Climate

The climate in the Minto area of the Yukon is considered sub-arctic with short cool summers and long cold winters. The average temperature in the summer is 10°C and the average temperature in the winter is –20°C. Average precipitation is approximately 25 cm of rain equivalent per annum in the form of rain and snow.

Like most northern Canadian mines the weather does not impede year round operation of the mine and processing plant except in short periods of harsh cold temperatures which may drop to –50°C, that can cause open pit mining operations to be temporarily suspended.

4.3 Physiography

The property lies in the Dawson Range, part of the Klondike Plateau, an uplifted surface that has been dissected by erosion. Local topography consists of rounded rolling hills and ridges and broad valleys (Figure 4.2). The highest elevation on the property is approximately 1,000 m above sea level, compared to elevations of 460 m along the Yukon River. Slopes on the property are relatively gentle and do not present accessibility problems. Bedrock outcrops can often be found at the tops of hills and ridges. There are no risks of avalanche on the property.

Overburden is colluvium primarily comprised of granite-based sand from weathering of the granitic bedrock in the area and is generally thin but pervasive but can reach plus 50 m in depth. Seams of clay and ice lenses are also present sporadically. South-facing slopes generally provide well-drained, sound foundation for buildings and roads. North-facing slopes in the area typically contain permafrost.

Figure 4.2: Mine Access Road Showing General Relief and Vegetation in the Area

Vegetation in the area is sub-Arctic boreal forest made up of largely spruce and poplar trees. The area has experienced several wildfires over the years, the latest in 2010, and has no old-growth trees remaining. The fire in 2010 led to the partial evacuation of the camp and a short stoppage in production.

4.4 Local Resources and Infrastructure

The nearest services, including fuel, groceries, hotel, restaurant and clinic, are at Carmacks, approximately 75 km south of Minto on Highway 2. Some services are available at Pelly Crossing, 35 km to the east of Minto.

The nearest large community is Whitehorse, the capital of Yukon Territory. Whitehorse has a population of approximately 26,000 and is the transportation, governmental and commercial hub for the region. It is serviced with commercial flights daily from Vancouver, Edmonton and other northern communities. Whitehorse is also connected via paved highways to British Columbia to the south, Alaska to the west and south to the port of Skagway, where Minto concentrate is trucked for loading onto ocean-going vessels.

The Minto mine has been a commercial operation for more than two years and has sufficient power, water, camp and personnel to continue operations through the life of mine plan.

MintoEx is currently preparing to apply for a mining permit revision that considers additional mining areas, higher plant throughput, revised waste and tailings management facilities and other environmental aspects of the project. This report details many of the proposed changes to mine that will be included in the application. Failure to permit the new deposits and waste management facilities will seriously impact the operation viability and mine life.

5 History

Production results for 2007 to 2010 are shown in Table 5.1 (as provided by MintoEx). Commercial production was declared on October 1, 2007 after a 4-month commissioning period. Results for 2008 and 2009 have shown a consistent increase in production and recovery as the mill facility optimization plans are carried out and mill expansion plans are implemented. Operations in 2010 were constrained for an extended period as a result of constraints in the tailings filtration facility, which activity is planned to be eliminated going forward. The positive processing results at Minto have been largely driven by the amenability of the ore to flotation at a coarse primary grind size.

Table 5.1: 2007 to 2009 Operating Results

Parameter	Unit	2007	2008	2009	2010 Projected
Waste mining	Mt	9.26	8.37	11.13	8.09
Ore mining	Mt	0.75	0.83	1.15	1.85
Total material mined	Mt	10.01	9.53	12.28	9.94
Mined copper grade	%	1.7	1.84	2.59	Not available
Mined gold grade – est.	g/t	0.45	0.71	1.14	Not available
Mined silver grade	g/t	6.8	7.65	11.00	Not available
Tonnes processed	Tonnes	238,446	809,426	1,031,190	925,000
Mill head copper grade	%	2.16	2.91	2.59	2.25
Mill head gold grade*	g/t	n/a	1.28	1.14	0.9
Mill head silver grade	g/t	7.7	11.8	11	8.49
Copper recovery	%	85.1	91.9	92.6	91.6
Gold recovery*	%	n/a	77.7	75.3	74.2
Silver recovery	%	77.5	84.6	81.9	84.5
Concentrate produced	Dmt	12,630	53,148	59,863	48,365
Concentrate grade – Cu	%	34.7	40.7	41.4	40.7
Concentrate grade – Au*	g/t	n/a	15.9	14.9	12.7
Concentrate grade – Ag	g/t	113	152	155.8	137
Copper in concentrate	M lb.	9.66	47.69	54.63	43.4
Gold in concentrate*	Oz	n/a	27,202	18,828	18,500
Silver in concentrate	Oz	45,885	217,489	299,767	164,000

** Gold is not assayed on site. Gold values are obtained from smelter returns.*

The following section was taken from Section 8 from the "Technical Report (43-101) for the Minto Project" by Hatch (August 2006) found on the sedar.com website and updated to describe events and information subsequent to the effective date of that report.

Mineral exploration on the Minto property has been conducted since 1971. Exploration efforts by MintoEx since July 2005 are explained in Section 5.4 MintoEx 2005-2010, and a description of drilling during this time is contained in Section 5.2 Drilling.

5.1 Chronology

A history of mineral exploration to production in the area is summarized below.

1970

- Regional stream sediment geochemical survey by the Dawson Syndicate, a joint venture between Silver Standard Mines Ltd. and Asarco Inc.

1971

- Follow-up of stream sediment anomalies and staking of the Minto claims in July;
- Soil sampling, IP geophysical surveys and manual excavated prospect pits on the Minto claims;
- 7 diamond drill holes completed (1,158 m);
- DEF claims staked by United Keno Explorations;
- A joint venture formed with United Keno Hill Mines, Falconbridge Nickel and Canadian Superior Explorations, to cover follow-up prospecting;
- IP and VLF-EM geophysical surveys, soil sampling and mapping on the DEF claims.

1972

- Mapping, airstrip construction and bulldozer trenching, 12 diamond drill holes (1,871 m) on 4;
- zones on the Minto claims;
- Grid soil sampling and bulldozer trenching on the DEF claims.

1973

- 62 diamond drill holes (7,887 m) on the Minto claims;
- Bulldozer trenching, EM and magnetic geophysical surveys and 41 diamond drill holes (7,753 m) on the DEF claims;
- Main mineralized body discovered in June.

1974

- Winter road built from Yukon Crossing and 58 diamond drill holes (11,228 m) on the Minto claims;
- Additional geophysics, rock mechanics, feasibility studies and 52 diamond drill holes (8,238 m) on the DEF claims.

1975-1976

- Joint feasibility studies.

1984

- Silver Standard changed its name to Consolidated Silver Standard and transferred its interest in the Minto claims to Western Copper Holdings, a subsidiary of Teck Corp;
- 5 percussion drill holes (518 m) on the DEF claims.

1989

- Western Copper Holdings transferred its interest in the Minto claims to Teck Corp;
- 84 percussion drill holes (4,897 m) on the DEF claims.

1993

- MintoEx was formed;
- Asarco and Teck sold their interest in the Minto claims (and leases) for shares in MintoEx and provided $375,000 in working capital;
- Asarco and Teck also received a net smelter royalty of 1.5% to be divided evenly;
- Falconbridge, the parent of United Keno Hill, sold its interest in the DEF claims to MintoEx for $1 million, payment due in 1996;
- Falconbridge was granted an option to repurchase the DEF claims on January 1, 2005 if the deposit was not in production by then;
- MintoEx carried out an airborne geophysical survey and drilled 8 diamond drill holes (960 m).

1994

- Initial public offering of shares of MintoEx completed;
- 5,912,501 shares were issued and outstanding with Asarco the majority shareholder with 3,297,500 shares (55.8%);
- 19 diamond drill holes (2,185 m);
- Feasibility study began with engineering and geo-technical studies.

1995

- 6 diamond drill holes (572 m) on magnetic anomalies and 1 condemnation diamond drill hole north of the proposed mill site;
- Feasibility study completed, reserves are 8,818,000T of 1.73% Cu, 0.014 oz/t Au and 0.22 oz/t Ag at 0.5% Cu cut-off grade;
- Recoveries are 95% for Cu and 85% for Au and Ag;
- Mine life was projected to be 12 years at production rate of 477,000 tonnes per year.

1996

- Funding arranged with Asarco to bring the deposit into production whereby Asarco would provide up to US$25 million. Under the funding arrangement, Asarco would acquire a 70% interest in the project, MintoEx would retain a 30% interest and remain as operator;
- MintoEx makes the $1 million payment to Falconbridge for the DEF claims completing the consolidation of the Minto and DEF claims;
- 16 km access road constructed including a barge landing site on the west side of the Yukon River and a bridge over Big Creek;
- 4 diamond drill holes (545 m).

1997

- A further 12.8 km of road construction to complete the new access road;
- Site for camp excavated;
- 72 m water well for domestic water supply;
- Mill site excavated and 2 used grinding mills moved onto site using an ice bridge over the Yukon River;
- Co-operation agreement signed with SFN.

1998

- Mill concrete foundations poured with cement trucks from Whitehorse barged across the Yukon River;
- Type A Water license granted by Yukon government;
- Concentrator design completed;
- Access road completed, camp constructed and the location of the proposed tailings dam was grouted;
- Phase 1 open pit mining plan completed.

1999

- Production license received;
- Five diamond drill holes (957 m) for engineering purposes.

2000

- Minor maintenance of on-site facilities;
- Hatch completes review of 1995 feasibility study.

2001

- Additional maintenance of camp facilities;
- 5 confirmation diamond drill holes (552 m) in the centre of the deposit;
- Most of the Asarco core and all of the Falconbridge core destroyed by time and forest fire;
- Regional airborne magnetic and radiometric surveys carried out by the Yukon government.

2002

- A limited amount of the old Asarco core that could be recovered was re-sampled;
- All the drill and geophysical data compiled in a data base to aid further exploration;
- 3 Landsat anomalies examined and prospected;
- Road maintenance scheduled to keep permits active;
- Asarco bought 100,000 shares of MintoEx to hold a total of 3,397,500 shares.

2004

- MintoEx announces all its shares are for sale.

2005

- Sherwood Copper Corp. acquires the Minto Mine property June 2005;
- 44 confirmation drill holes (5937 m) to confirm the Main Minto Deposit Resources and Reserves.

2006

- Confirmation drilling program executed in order to update the precious metal resource;
- Development of Minto Project and commencement of pre-stripping the Minto Deposit;
- Drill discovery and definition of Area 2 deposit ;
- Copper Keel prospect discovered;
- Mill construction commences;

- C$85 M debt package arranged, forward sales complete, concentrate off-take agreement executed October 2006.

2007

- Power Purchase Agreement for Minto signed;
- Resource estimate for Area 2 deposit completed;
- First copper-gold concentrates at Minto Mine produced;
- 1 exploration and 4 metallurgical drill holes (754 m) at Minto Deposit;
- Area 118 and Ridgetop deposits discovered and partially drill defined;
- Airstrip prospect discovered;
- First concentrates from Minto mine delivered to Port of Skagway, Alaska July 2007;
- Minto Mine declares commercial production and first Minto concentrates shipped from Skagway October 2007;
- Pre-feasibility Study for expansion of Minto copper-gold mine December 2007;
- Phase 2 mill expansion at Minto Mine completed ahead of schedule.

2008

- Minto Mine achieves and exceeds design capacity;
- Reported copper-gold resources increased at Minto Mine June 2008;
- Capstone and Sherwood announce combination to create intermediate copper producer with Sherwood shareholders overwhelmingly approving business combination;
- Closing of precious metal transaction; Silverstone provides upfront payment of US$37.5 M for payable gold and silver from Minto;
- Minto Mine connects to electrical grid;
- Capstone and Sherwood complete business combination November 2008;
- Definition of thick zones of near surface copper mineralization at Ridgetop and deeper mineralization at Area 118.

2009

- High grade Minto North Deposit discovered and defined;
- Increased copper-gold mineral resources at Minto announced in June 2009;
- Dipole-dipole geophysical survey over northern regional targets;
- Titan 24 survey over the Minto Priority Exploration corridor;
- Drill discovery of the Minto East prospect.

2010

- High grade Minto East Deposit defined;
- Expanded copper-gold mineral resources for Area 2/118 and Ridgetop plus preliminary resource for Minto East in August 2010;
- Extended Titan 24 survey over 85% of the Minto property;
- Drill discovery of the Wildfire and Inferno prospects; and
- Updated Mineral Resource for Minto East.

5.2 Drilling

The project has been actively explored since the early 1970s. Companies controlled by ASARCO and Falconbridge drilled on the property in 1973 and 1974. All drill cores collected prior to 1993 were destroyed by forest fires. MintoEx completed further drilling programs between 1993 and 2001 before it was acquired by Capstone Mining Corp. and a further six drill programs from 2005 to 2010 since Capstone's acquisition of MintoEx.

ASARCO and Falconbridge 1972 to 1974

Most of the drilling on the property is recent and was performed by the current operators of MintoEx and has resulted in significant new discoveries and resource additions. However, the initial discovery phase of exploration drilling was performed in the early 1970s by companies controlled by Falconbridge (United Keno Hill Mines Ltd.) and ASARCO (Silver Standard). Subsequent definition drilling by these operators was conducted once the Main deposit was discovered and exploration in the area continued sporadically until 2005 when the project was purchased by the current operators. The early project reports fail to detail their drilling procedures, but basic drilling procedures have unlikely changed little over time.

Early drilling was conducted with BQ drill rods, which return a core diameter of 1.43 inches. Within the main zone of the deposit, the drill hole density is on 100 ft centres on the DEF (Falconbridge) part of the deposit (locally as close as 50 ft), and generally on 150 ft to 200 ft centres on the Minto (ASARCO) side as illustrated in Figure 5.1.

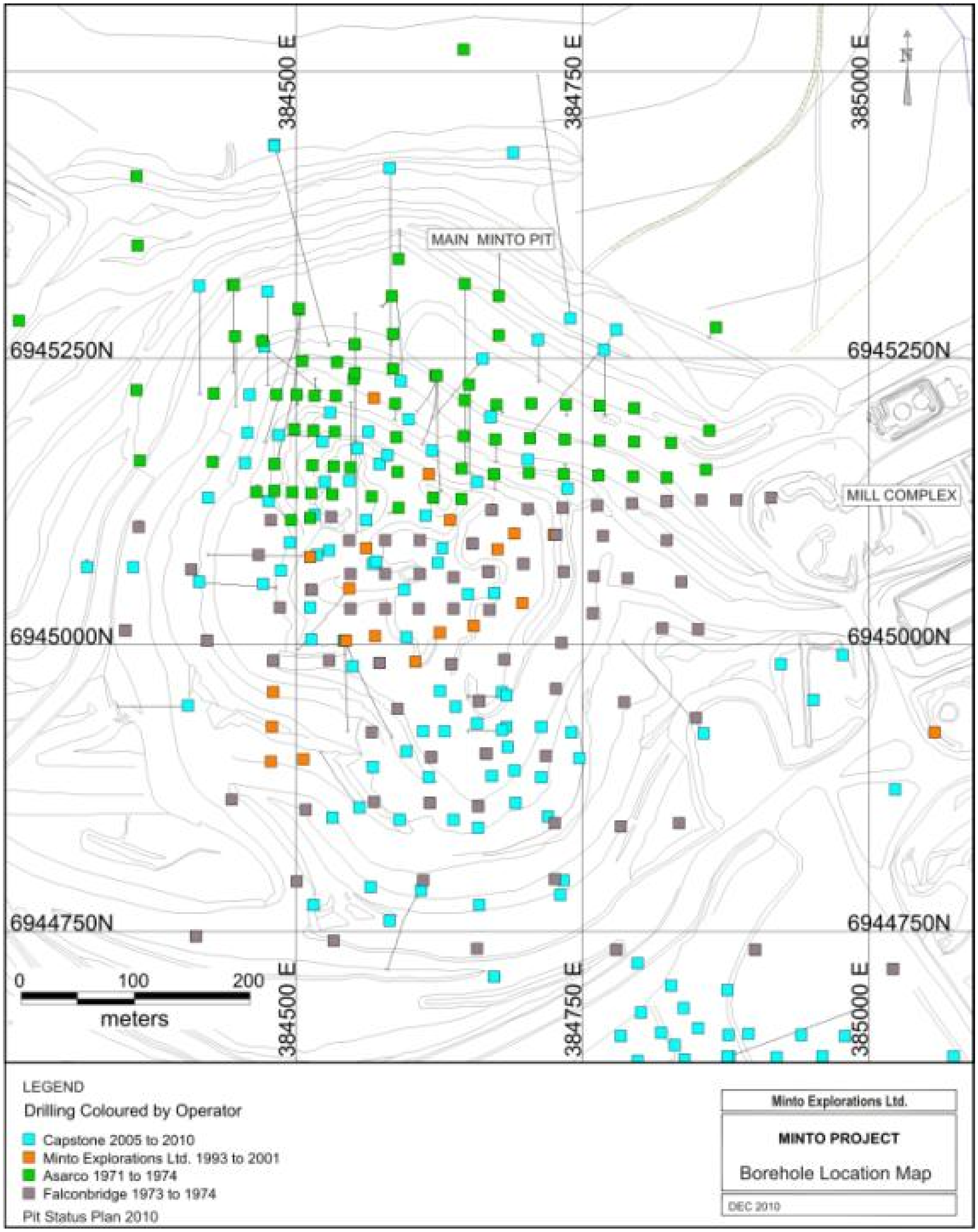


Figure 5.1: Drill Hole Location Map – Minto Main Deposit

Falconbridge drilled 11 angled holes, and all other holes were drilled vertically. The average sample length for ASARCO is 2.4 m with the majority of samples being either 1.5 m or 3.0 m long. The average sample length for the Falconbridge drill holes is 1.5 m.

The locations of the holes were surveyed in by Underhill Geomatics using a local grid controlled by local benchmarks. Prior to the commencement of pre-stripping of the Minto Deposit in 2006, the drill roads and pads for this drilling were still visible and the holes were often identifiable by casing and/or wooden posts protruding from the ground, although the labels were no longer attached or legible.

The core from this drilling was stored onsite in two core sheds. Over time the sheds have collapsed and/or have been burned out by wildfires, rendering most of the core unusable. In addition, the labels on the few remaining intact boxes are missing and/or are not legible.

In their compilation of the results, MintoEx has distinguished the ASARCO drill holes with an ‘A’ prefix and the Falconbridge hole with a ‘K’ prefix.

The results of this drilling have been instrumental in estimating the grade and tonnage of the deposit. The drilling was carried out using accepted practices of the time and is documented well enough to be reliable for the purposes of grade and tonnage estimations, particularly when compared to the results of subsequent infill drill completed by MintoEx in 1993-2001 and in 2005-06.

MintoEx 1993 to 2001

MintoEx has carried out several diamond drilling programs for deposit definition drilling and exploration on the property in general, as follows:

1993

- 960 m drilled in eight holes (93 – A to H) within the deposit area to sample the two main mineralization types (foliated granodiorite and quartzofeldspathic gneiss) for metallurgical test work;
- Six of the holes were located to intersect the lower zone mineralization immediately below the main zone and one was designed to test deeper mineralization indicated in the 1970s drilling;
- The core was used for metallurgical testing and some of it was not split and assayed;
- Four of the holes were logged for magnetic susceptibility.

1994

- 2,185 m drilled in 19 exploration holes to test mineralization south of the main deposit;
- This drilling outlined a mineralized horizon roughly 6 m thick grading 2 – 3% Cu;
- One hole (94-17) filled in a large gap in the deposit area.

1995

- 572 m drilled in 6 holes: 425 m drilled in five exploration holes to test geophysical anomalies; and 160 m completed in one condemnation hole north of the proposed mill site;
- The exploration holes failed to intersect any anomalous mineralization.

1996

- 545 m completed in four condemnation holes in the area of the proposed west waste rock dump.

2001

- 552 m drilled in five confirmation holes within the proposed open pit area.

All the drilling on the project was contracted to E. Caron Diamond Drilling of Whitehorse.

The 1993, 1994, 1995 and 2001 programs utilized HQ core and the 1996 drilling was NQ core. This historical drilling was completed in the 1990s, prior to the legislation for NI 43-101. There was less regulatory scrutiny and results were the focus of reporting, rather than details of data collection. There is little in the way of documentation for the methods used in the pre-1990s drilling and sampling.

The 2001 drilling was subject to a rigorous report by both MintoEx (Minto Explorations Ltd., 2003) and ASARCO (Simpson, 2001), which loaned a geologist to the project to log and sample the core. The results of the 2001 drilling are discussed in the Data Verification section of this report. Some of the core from the 1993, 1996 and 2001 drilling programs is stored in the Ken Bostock Core Library in Whitehorse.

Some of the other core from the exploration on the property (away from the deposit) is stacked on site in behind the camp buildings. Older core was stored in sheds, which were burnt in a forest fire and is now unidentifiable.

5.3 Historic Resource Estimates

The Minto deposit has been subject to several historical tonnage and grade estimations, as summarized in Table 5.2. These mineral resource estimates were based on up to 160 drill holes (totaling more than 25,000 m of drilling).

Table 5.2: Historical Tonnage & Grade Estimates of the Minto Deposit

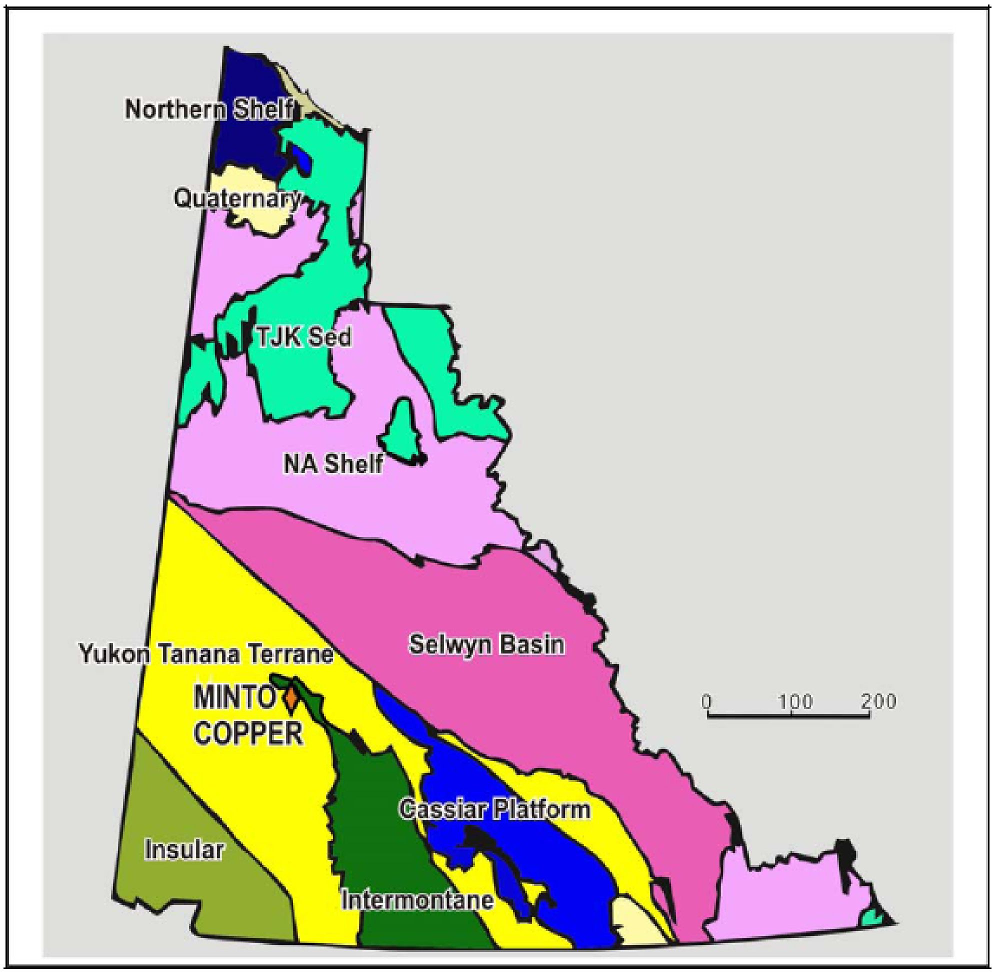
Year	Source	Cut-off (%Cu)	Short Tons	Cu (%)	Au (oz/t)	Ag (oz/t)	Comments
1976	R.T Heard UKHM	unknown	8,219,370	2.04			
1976	L.A. Wigglesworth Falconbridge	unknown	8,210,219	2.03			
1975	R.J. Prevedi ASARCO	0.60	8,441,941	1.74			
1976	R.J. Prevedi ASARCO	unknown	7,220,900	1.86			
1980	D.M. Fletcher ASARCO	2.00	2,968,600	3.24	0.027	0.411	
1989	J. Proc & H.L Klingmann Minto Explorations	0.80	6,368,000	2.11	0.016	0.33	Open Pit and Underground Recovery at 75% and 5% dilution
1990	SRK/Falconbridge	unknown	7,592,318	1.88	0.016		Cut-off Grade 0.0%? Includes Lower Zone
1992	J. Proc & H.L Klingmann Minto Explorations	unknown	6,071,000	2.21	0.018	0.28	Open Pit and Underground UG = 1,600,000 ton @ 3.73% Cu, 0.038 oz/t Au, 0.49% oz/t Ag
1994	G. Giroux Montgomery Consultants	0.50	8,780,000	1.76	0.015	0.223	Pre 43-101 "proven" + "probable"

The estimates in Table 5.2 do not follow the required disclosure for mineral reserves and mineral resources (as outlined in National Instrument 43-101) because they were prepared prior to the inception NI 43-101. The mineral resource estimates have been obtained by sources believed reliable and are relevant but cannot be verified. No effort has been made to refute or confirm these estimates and they can only be described as historical estimates.

6 Geological Setting

6.1 Regional Geology

The Minto Project is found in the north-northwest trending Carmacks Copper Belt along the eastern margin of the Yukon-Tanana Composite Terrain, which is comprised of several metamorphic assemblages and batholiths (Figure 6.1). The Belt is host to several intrusion-related Cu-Au mineralized hydrothermal systems. The Yukon-Tanana Composite Terrain is the easternmost and largest of the pericratonic terrains accreted to the Paleozoic northwestern margin of North America (e.g., Colpron *et al.*, 2005). It is regarded to be the product of a continental arc and back-arc system, preserving meta-igneous and metasedimentary rocks of Permian age on top of a pre-Late Devonian metasedimentary basement (e.g., Piercey *et al*. 2002).



From: Yukon Geologic Survey "Maps Yukon" website (www.geology.gov.yk.ca)

Figure 6.1: Yukon Geology (from Yukon Geologic Survey "Maps Yukon" website (www.geology.gov.yk.ca)

The Minto Property and surrounding area are underlain by plutonic rocks of the Granite Mountain Batholith (Early Mesozoic Age) (Figure 6.2) that have intruded into the Yukon-Tanana Composite Terrain. They vary in composition from quartz diorite and granodiorite to quartz monzonite. The batholith is unconformably overlain by clastic sedimentary rocks thought to be the Tantalus Formation and andesitic to basaltic volcanic rocks of the Carmacks Group, both of which are assigned a Late Cretaceous age. Immediately flanking the Granite Mountain Batholith, to the east, is a package of undated mafic volcanic rocks, outcropping on the shores of the Yukon River. The structural relationship between the batholith and the undated mafic volcanics is poorly understood because the contact zone is not exposed

Geobarometry and geothermometry data (Tafti and Mortensen, 2004) suggests that the Granite Mountain Batholith was emplaced at a depth of at least 9 km, while the presence of euhedral to subhedral epidote, interpreted by Tafti and Mortensen as magmatic in origin, suggests a deeper emplacement depth in the order of 18 to 20 km.

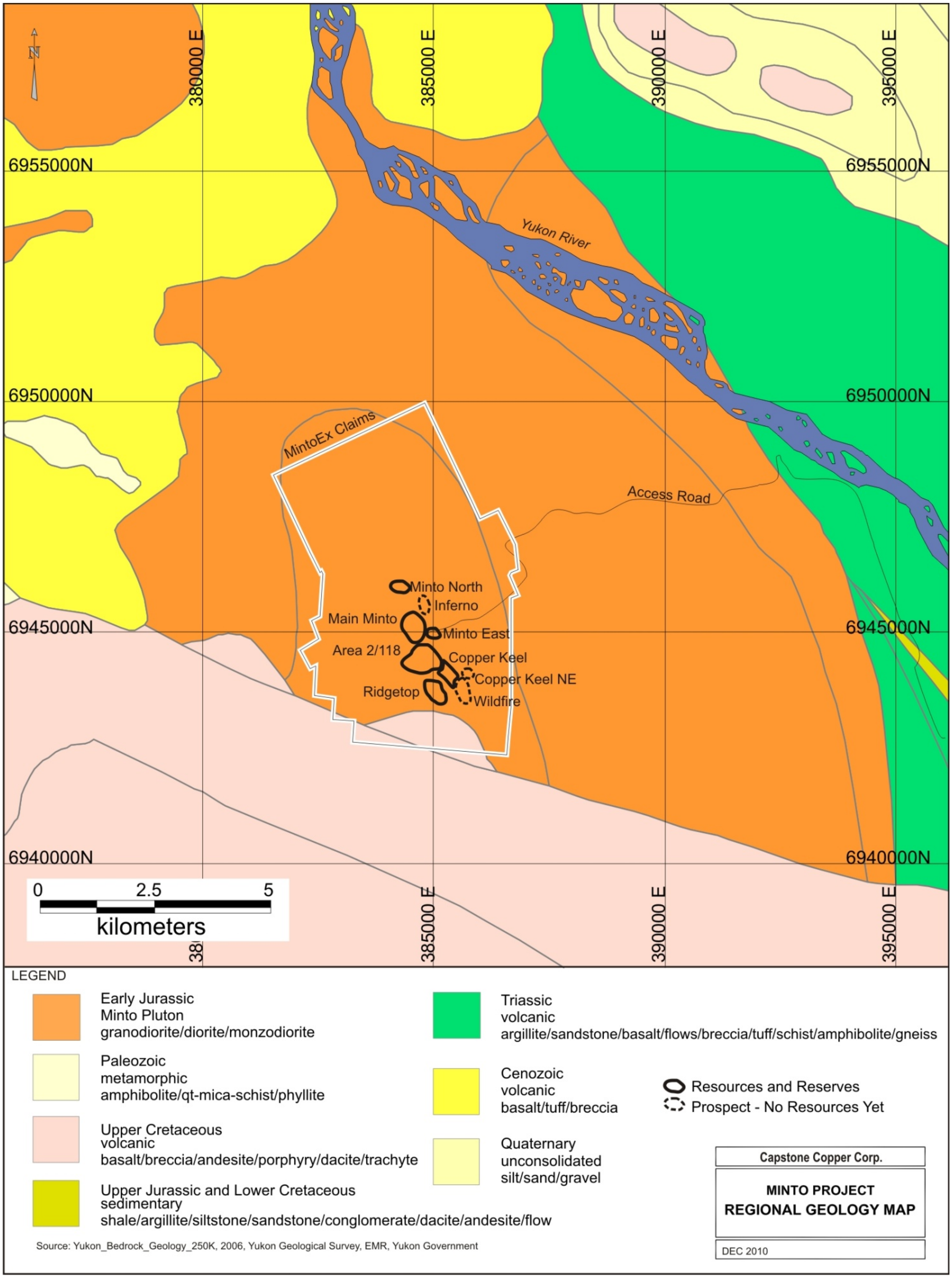


Figure 6.2: Regional Geology

6.2 Property Geology and Lithological Description

Much of the geological understanding of the rock around the Minto deposits is based on observations from diamond drill core and extrapolation from regional observations. The reason for this is poor outcrop exposure (less than 5% coverage), as well as the deep weathering and oxidation of any existing exposed outcrop. The terrain was not glaciated during the last ice age event.

Five deposits of mineralization are reported in this document (Figure 6.2). Each of these deposits closely share a similar style of mineralization of shallow dipping copper sulphide mineralized zones. The Main Minto deposit is already exposed in open pit mining. The Area 2 and Area 118 deposits are considered continuous for the purpose of this report, and reported as one deposit denoted as Area 2/118 located immediately south of Main Minto. The Ridgetop deposit is located just over 300 m south of the Area 2/118 deposit, the Minto North deposit located about 700 m north of the Main Deposit, while the most recently discovered deposit to be reported is the Minto East deposit located about 200 m east of the south end of the Main deposit. In addition to these mineral deposits which have NI 43-101 compliant mineral resources there are several significant mineral prospects. These deposits and prospects define a general north-northwest trend informally called the Priority Exploration Corridor or PEC. The most significant of these prospects are Wildfire; Copper Keel, Airstrip, and Inferno.

The hypogene copper sulphide mineralization at Minto is hosted wholly within the Minto pluton, which intrudes near the boundary between the Stikinia and Yukon-Tanana terrains, however since the contact is not exposed it is unclear if the pluton stitches the two terrains. The Minto pluton is predominantly of granodiorite composition. Hood *et al*. (2008) distinguish three varieties of the intrusive rocks in the pluton. The first variety is a megacrystic K-feldspar granodiorite. It gradually ranges in mineralogy to quartz diorite and rarely to quartz monzonite or granite, typically maintaining a massive igneous texture. An exception occurs locally where weakly to strongly foliated granodiorite is seen in distinct sub-parallel zones several metres to tens of metres thick.

A second variety of igneous rock is a quartzofeldspathic gneiss with centimeter-thick compositional layering and folded by centimetre to decimetre-scale disharmonic, gentle to isoclinal folds (Hood et al., 2008). The third variety of intrusive is a biotite-rich gneiss. MintoEx geologists consider all units to be similar in origin and are variably deformed equivalents of the same intrusion.

Copper sulphide mineralization is found in the rocks that have a structurally imposed fabric, ranging from a weak foliation to strongly developed gneissic banding. For this reason all core logging by the past and present operators separates the foliated to gneissic textured granodiorite as a distinctly discernable unit. It is generally believed by MintoEx geologists that the foliated granodiorite is just variably strained equivalents of the two primary granodiorite textures and not a separate lithology.

While this interpretation, based upon detailed observations from logging of tens of kilometres of drill core is highly likely but it still needs to be conclusively proven. Tafti & Mortensen (2004) noted that the relatively massive plutonic rocks have similar mineral and chemical composition as the foliated rocks. Research in collaboration with the Mineral Deposits Research Unit ("MDRU") of the University of British Columbia is on-going.

The contact relationship between the foliated deformation zones and the massive phases of granodiorite is generally very sharp. These contacts do not exhibit chilled margins and are considered by MintoEx geologists to be structural in nature, separating the variably strained equivalents of the same rock type. Tafti and Mortensen (2004) had interpreted the sharp contacts to be zones of deformed rock within the unfoliated rock (i.e. rafts or roof pendants). Supergene mineralization occurs proximal to near-surface extension of the primary mineralization and beneath the Cretaceous conglomerate.

Conglomerate and volcanic flows have been logged in drill core by past operators. New drilling has confirmed the presence of conglomerate, but not the volcanic flows. The latter cannot be confirmed by the authors as the drill core from historic campaigns was largely destroyed in forest fires and no new drilling has intersected such rocks. However, undated volcanic rocks are mapped by Hood, near the southwest margin of the property, south of a fault that is inferred from geophysics to separate them from the Jurassic Age intrusive rocks. The conglomerate has been dated (unpublished date pers. com. Dr. Maurice Colpron - Yukon Geological Survey) as Cretaceous Age. It is now recognized in outcrop in a borrow pit exposure located west of the airstrip as well as in numerous recent drill holes. Observations of foliated and even copper mineralized cobbles in drilling indicate that "Minto-type" mineralization was exposed, eroded and reincorporated in sedimentary deposits by the Cretaceous Age.

Other rock types, albeit volumetrically insignificant, include dykes of simple quartz-feldspar pegmatite, aplite; and an aphanitic textured intermediate composition rock. Bodies of all of these units are relatively thin and rarely exceed one metre core intersections. These dykes are relatively late, and observed contact relationships suggest they generally postdate the peak ductile deformation event; however some pegmatite and aplite bodies observed in a rock cut located north of the mill complex are openly folded.

It is unclear if this folding is contemporaneous with foliation development in the deformed rocks or post-dates the foliation development. Observations from drill core and open cut benches in the mine show examples where the foliation and the pegmatitic/aplitic intrusions are both folded, as well as examples where the intrusions are not folded, suggesting two populations of minor dykes.

6.3 Structure

There are both ductile and brittle phases of deformation around the Minto deposits. As noted above copper-sulphide mineralization is strongly associated with foliated granodiorite. This foliation is defined by the alignment of biotite in areas of weak to moderate strain and by the segregation of quartz and feldspar into bands in areas of higher strain, giving the rock a gneissic texture in very strongly deformed areas. The deformation zone forms sub-horizontal horizons within the more massive plutonic rocks of the region and can be traced laterally for more than 1,000 m in the drill core. They are often stacked in parallel to sub-parallel sequences. The regular, sub-horizontal nature of the deformation zones allows a high degree of predictability when planning diamond drilling campaigns.

Contrary to some previous reports (Orequest, 2005), the foliated zones do not appear to inter-finger with the more massive rocks. Rather, it appears that blocks of unfoliated granodiorite are sometimes incorporated within the thicker deformation zones that surround them.

The similarity of chemistry and texture of both the deformed and the massive granodiorites suggest the deformation zones are structural in origin and not stratigraphic. Several of these foliated units can be traced in drill holes over long distances at similar elevations.

While this could suggest either a structural or a stratigraphic origin for the foliated rocks it was noted that obvious plutonic textures were found in both the deformed and the massive rocks. However the absence of chill margins or absorption rims at contacts, combined with the great depth of emplacement (Tafti and Mortensen, 2004) likely preclude them from being remnant rafts or roof pendants of metasedimentary or metavolcanic strata, as some workers have postulated. No sedimentary or volcanic features have been observed in these foliated and mineralized rocks. A structural origin remains the best explanation.

It is therefore postulated that the foliated granodiorite horizons represent healed, shallowly dipping shear zones within the Granite Mountain Batholith, and may have formed when the rocks passed through the brittle/ductile transformation zone in the earth’s crust in transition from a deep emplacement environment of the batholith to eventual exhumation. They may represent thrust faults related to regional crustal thickening of the Yukon-Tanana Terrain when the batholith was being exhumed.

Internally, the foliation exhibits highly variable orientations within individual deformation zones with the presence of small-scale folds. The foliation is often observed to be at a high angle to contacts with more massive textured rock units.

Observations by Hood *et al.* (2008) along a transect in the Area 2 deposit suggest that foliation orientations within deformed horizons have a geometry of tight to isoclinal folding with a wavelength on the order of about 30 m.

The observed trend of folds within this area is approximately northwest, parallel to regional structural trends (Tempelman-Kluit, 1984). The ore–bearing zones are also occasionally folded on a scale of several hundred metres. Based upon horizon modeling for resource estimation of Ridgetop the folds have wavelength of about 280 m. The folds appear to be gentle folds with north-south axial traces. Simple shear strain of the foliated zones is also noted adjacent to late cross-cutting fault zones.

Late brittle fracturing and faulting is noted throughout the property area. Some of these faults are significant from an economic standpoint. The Minto Creek fault (MC Fault) bisects the Minto Main deposit, dividing it into north and south areas and is modelled as dipping steeply north-northeast with an apparent left lateral reverse displacement. The northern block moved up and to the west relative to the southern block. Both the vertical and horizontal displacements are evident by offsets in the main zone mineralization and appear to be minimal. A lack of marker horizons in the plutonic rocks, however, makes it difficult to determine the absolute magnitude of the movement (Figure 6.3).

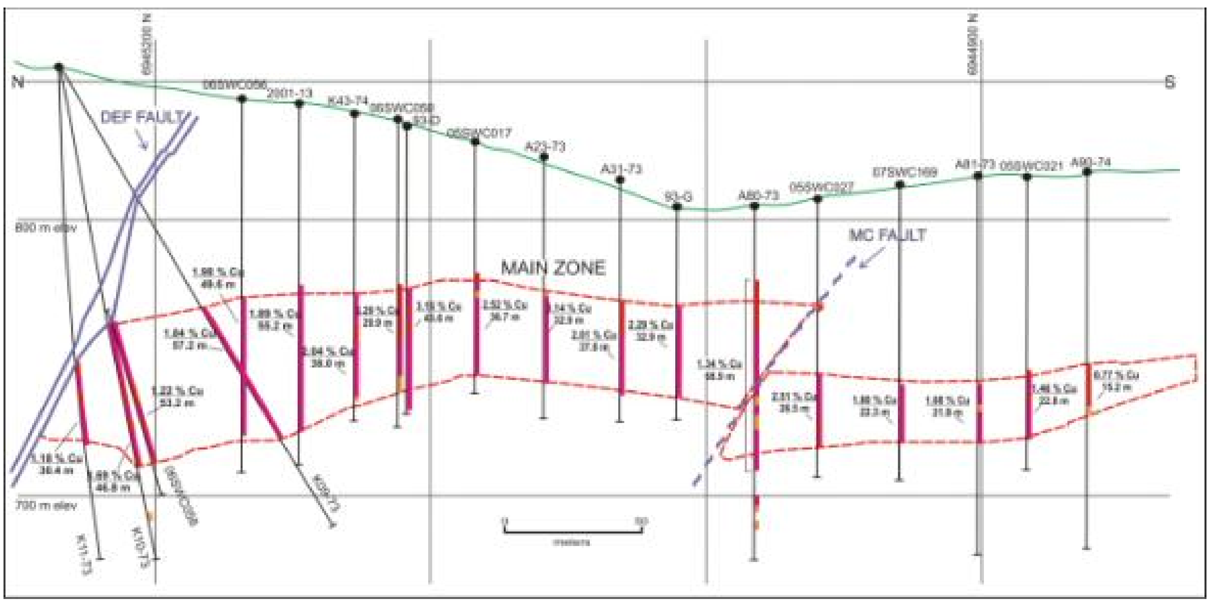


Figure 6.3: North- South Cross Section through Minto Main Deposit showing DEF Fault and MC Fault

The DEF fault defines the northern end of the Main deposit. It strikes more or less east-west and dips north-northwest and cuts off the main zone mineralization, as shown in Figure 6.3. The vertical orientation of most of the drilling is less than optimal to intersect steep to vertical faults. It may share a similar sense of movement to the MC fault, but a significant amount of displacement is inferred. Determining the magnitude of this displacement could lead to locating an extension of the main zone mineralization on the north side of the DEF fault. This late block faulting is noted throughout the Granite Mountain Batholith and in some instances a rotational component is noted as well. Tafti & Mortensen (2004) found the Cretaceous Age Tantalus Formation rotated up to 60 from horizontal in areas located south of the Minto deposit.

A zone of pervasive fracturing on the west side of the deposit limits ore grades in this direction. Limited historical drilling west of this structure did intersect some weak copper mineralization, although foliated horizons do not line up across this fracture zone. It is presumed to be one of the north-south faults that are part of the late brittle conjugate set.

While the limits to Minto Main mineralization on the north and west sides are structural in nature, the southern limit is an erosion channel cutting below the elevation of the mineralization and thereby removing it. This zone of deeper erosion is a paleo-channel that is interpreted to follow another roughly east-west striking fault. Only on the east side does mineralization appear to fade out and have no obvious structural limit.

The boundary between the Area 2 and Area 118 is an intermediate NE dipping fault. The displacement of the mineralization is significant. At least two parallel structures displace mineralized domains in Area 118.

The shear sense on this structure has not been analyzed in detail, but attempts to correlate ore zones across the main boundary fault are complicated by the difficulty in finding a specific characteristic to unambiguously identify the zones. The easiest zone to identify (based on mineralization and texture) is the "N" zone and it has up to 66 m of vertical throw across the boundary fault. Other zones show changes in thickness and orientation, suggesting the presence of pure strain and block rotation. A better structural model is required. A similar NW striking fault zone appears to be present that defines the northeastern boundary of the Ridgetop deposit, and defines the outcrop of Cretaceous conglomerate. The dip of this structure is unknown.

All mineralized horizons exhibit locally pervasive fracturing (typically chloritic or hematitic), which are interpreted to postdate the main copper-sulphide mineralization event. This late structural/hydrothermal event may have potential economic significance, as coarse-grained visible gold has been logged on chloritic fractures.

6.4 Veining

Veins in the Minto Deposit appear to have been emplaced after the copper sulphide mineralization and are therefore not economically significant. The most common veins are very narrow (less than 30 cm) steeply dipping, simple quartz-feldspar pegmatite veins that often contain cavities that are indicative of shallow emplacement. The veins crosscut foliation in the deformation zones and the sulphide mineralization; evidence of their post sulphide mineral emplacement. Other types of late veins found in the deposit include thin (less than 2 mm) calcite, epidote, hematite and gypsum stringers, and fracture coatings. Quartz veining is extremely rare and economically insignificant.

7 Deposit Types

Each of the deposits reported in this technical report are considered to have the same style of mineralization as the Minto Main deposit. The copper sulphide mineralization is associated with sub-horizontal, sub-parallel foliated horizons within a granodioritic pluton. MintoEx have engaged the MDRU of the University of British Columbia to help understand the nature of mineral paragenesis and deformation history at Minto. This research is on-going.

At various times since its discovery the Minto deposit has been described as an example of Porphyry Copper, Volcanogenic Massive Sulphide (VMS), Redbed Copper, Magnetite Skarn (see discussion by Pearson and Clark, 1979) and Iron Oxide Copper Gold “IOCG”(Minto Explorations Ltd., 2003). Based on the preceding paragraph it is reasonable to say that the origin of the Minto deposit is enigmatic. Various workers (including the current authors) appear to have ascribed different interpretations for the most part based on their empirical observations, the background of the observer and the popular models of the day. The abundance of the high Cu/S mineral bornite in a moderately oxidized magmatic system along with the obvious magnetite association suggests that Minto belongs to one of two recognized deposit types: Magnetite Skarn or Iron Oxide Copper Gold (“IOGC”). The lack of a typical calc-silicate skarn mineral assemblage seems to preclude the skarn deposit type, this appears to leave the IOCG model or alternatively it belongs to a previously unrecognized deposit type.

The host rocks to the Minto deposit were emplaced in a deep batholitic setting (exceeding 9 km deep to perhaps as much as 18-20 km deep), which is not considered to be the typical porphyry environment. The host is a moderately oxidized magma (Tafti and Mortensen, 2004) with widespread iron oxide (magnetite and hematite) mineralization. At least some of the hematite is supergene in origin but it is unclear if some hematite is also primary. There are very strong structural controls on ore mineral emplacement and there is no apparent genetic link to a specific phase of intrusion. Typical porphyry-type alteration zoning such as widespread propylitization, argillization, barren silicic core, or large barren pyritic halo is not recognized. Stockwork style, fracture or vein mineralization is also not present.

MintoEx geologists have been advised that some examples of IOCG mineralization (in personal communications) exhibit some similar characteristics and setting to Minto including Copperstone in Arizona, Caldelaria in Chile, and Ernest Henry in Australia (Williams et al., 2005). From a genetic and structural prospective, albeit not size wise, the Sossego Deposit in Brazil may be a reasonable analog. While an IOCG origin for the Minto Deposit cannot be unequivocally demonstrated, MintoEx geologists are of the opinion that this style of deposit provides the most consistent model for their current level of understanding. However, the unique nature of this mineralization style and apparent lack of close analogs elsewhere suggests the Minto Copper-Gold deposits may represent an unrecognized mineral deposit type.

8 Mineralization

8.1 Mineralization

The Minto deposits have essentially no surface exposure with the exception of minimal exposure in historical trenches of the shallow partially oxidized zones associated with the Ridgetop deposit. Observations for the deposits are therefore based almost entirely on hand-specimen and petrographic studies of drill core. The primary hypogene sulphide mineralization consists of chalcopyrite, bornite, euhedral chalcocite, and minor pyrite. Metallurgical testing also indicates the presence of covellite, although this sulphide species has never been positively logged macroscopically. Texturally, sulphide minerals predominantly occur as disseminations and foliaform stringers along foliation planes in the deformed granodiorite (i.e. sulphide stringers tend to follow the foliation planes). Sulphide mineral content, however, tends to increase where this foliation is disrupted by intense folding. In addition, semi-massive to massive mineralization is also observed; this style of mineralization tends to obliterate the foliation altogether. Silver telluride (hessite) is observed in polished samples but has not been logged macroscopically. Native gold and electrum have both been reported as inclusions within bornite and accounts for the high gold recoveries in test copper concentrates. Occasionally, coarse free gold is observed associated with chloritic or epidote lined fractures that cross-cut the sulphide mineralization. The free gold may be due to secondary enrichment during a later hydrothermal process overprinting the main copper sulphide-gold event. Sulphide mineralization is almost always accompanied by variable amounts of magnetite mineralization and biotite alteration. While these minerals occur in the non-deformed rocks they are present in the mineralized horizons in a much greater abundance in the range of an order of magnitude greater than background.

The Minto Main deposit exhibits crude zoning from west to east. The bornite zone is dominant in the west while a thicker, lower grade chalcopyrite zone is dominant on the east side of the deposit. The bornite zone is defined by the metallic mineral assemblage magnetite-chalcopyrite-bornite. Bornite mineralization is conspicuous, but chalcopyrite is the dominant sulphide species. Stringers and massive lenses of chalcopyrite with various quantities of bornite are typical. Massive mineralization occurs locally over intervals exceeding 0.5 m in thickness and semi-massive mineralization over several metres in thickness may occur. In these sulphide rich areas, textures often resemble those seen in magmatic sulphide zones with sulphide mineralization interstitial to the rock forming silicate minerals. The higher grade portion of the Minto Main deposits roughly corresponds to the bornite zone. Local concentrations of bornite up to 8% are seen. The precious metal grades are elevated in the bornite zone (very fine gold and electrum occur as inclusions in bornite) and occurrences of coarse grained native gold are noted almost exclusively in bornite-rich material. The chalcopyrite zone is characterized by the metallic mineral assemblage of chalcopyrite-pyrite +/- very minor bornite and magnetite.

Empirical observations indicate the highest concentrations of bornite are associated with coarse grained, disseminated and stringer-style magnetite mineralization, up to 20% by volume locally. The stringers of magnetite are often folded or boudinaged, suggesting that at least some of the magnetite mineralization predates peak ductile deformation.

Sulphide mineralization on the other hand, shows both evidence and absence of ductile deformation locally and is interpreted to have formed contemporaneous with, or late in the ductile deformation history.

The Minto North and Minto East Deposits also exhibit a zoning from west to east. High-grade bornite-dominant mineralization is observed in the west with lower grade chalcopyrite-dominant mineralization in the east. The bornite zone is defined by the metallic mineral assemblage bornite-magnetite-chalcopyrite. Bornite mineralization occurs as strong disseminations and foliaform stringers locally >10% to occasional semi-massive to massive lenses up to 2 m in thickness. Chalcopyrite concentrations are typically within the 1 to 2% range, but locally can reach concentrations of 10%. Precious metal grades are elevated in the bornite zone, and visible gold has been observed on occasion.

Mineralization at Area 2/118 is distinct in that mineralization is predominantly disseminated (plus occasional foliaform stringers) and that semi-massive to massive sulphide mineralization is absent; as a whole, the mineralization is more homogenous and consistent as compared to Minto Main and Minto North. The primary mineral assemblage at Area 2/118 includes chalcopyrite-bornite-magnetite with minor amounts of pyrite; and a crude zoning is present in that the higher grade northern half of the deposit shows increased bornite concentrations up to 8% locally.

Mineralization at Ridgetop is subdivided into the near surface horizons that have been affected by supergene oxidation and the more typical primary sulphide mineralization of the deeper zones. The lower zones are defined by a mineral assemblage of chalcopyrite-magnetite with minor amounts of pyrite. Chalcopyrite is the dominant sulphide in the lower zones, and bornite is only observed in minor amounts. Texturally, chalcopyrite occurs as disseminations and foliaform stringers, and is rarely observed as semi-massive to massive bands. Magnetite is coarse grained, disseminated, stringer-style, and can occur in bands up to 0.3 m in thickness, up to 20% volume locally.

These empirical observations of bornite/chalcopyrite relative abundances are supported by a copper and gold grade trend in mineral resources discovered to date within the PEC where the Ridgetop deposit sits at the lower grade southern end and Minto North sits at the much higher grade northern end of the currently defined trend.

8.2 Alteration, Weathering, and Oxidation

Pervasive, strong potassic alteration occurs within the flat lying zones of mineralization, and is the predominant alteration assemblage observed in all of the Minto deposits. The potassic alteration assemblage is characterized by elevated biotite contents and minor secondary k-feldspar overgrowth on plagioclase relative to the more massive textured country rock. Biotite concentrations range up to 30 to 70% by volume locally, compared to about 5 to 8% in waste rock. Additional alteration includes the replacement of mafic minerals by secondary chlorite, epidote, or sericite observed both in mineralized and waste rock interstitially or fracture/vein proximal, as well as variable degrees of hematization of feldspars. Uncommon but locally pervasive sericite-muscovite alteration is observed associated with post-mineral brittle faults; this type of alteration is most common in the Area 2/118 Deposit.

Hematization is the most pervasive at the Minto Main deposit proximal to the DEF fault, whereas in the other deposits it is predominantly fracture controlled within narrow alteration selvages. It is interpreted to be supergene in origin. Minor carbonate overprint is occasionally observed associated with secondary biotite. The contacts between the altered and unaltered rocks are sharp, as are the contacts between mineralized rocks and waste rocks.

Silicification is present but not pervasive nor uniform in distribution in the Minto deposits. At Minto Main, Minto East, and Minto North it is sporadic within the bornite zone (west) and lacking in the chalcopyrite zone (east). At Area 2/118 silicification intensity is variable in all ore zones. On rare occasions, silicification is pervasive enough to almost entirely overprint both primary and deformation textures (Area 2) while it is essentially absent at Ridgetop. The relationship between silicification and the mineralization is unclear due to inconsistent core logging over three decades, although in most cases higher grade sulphide mineralization is coincident with silicification.

Copper oxide mineralization, like the hematization seen at surface in float, trenches, and in the upper mineralized zones at Ridgetop is the result of supergene oxidation processes. This surface mineralization at Minto Main and Area 2/118 represents either the erosion remnants of foliated horizons that are located above the deposits or is vertical remobilization of copper up late brittle faults and fracture zones that intersect primary sulphide mineralization at depth. Chalcocite is the prime mineral in these horizons along with secondary malachite, minor azurite, and rare native copper. The mineralization is found as fracture fill and joint coatings and more rarely interstitial to rock forming silicate minerals.

At the Ridgetop deposit and the Wildfire prospect, the upper near surface mineralized zones are unique in that the dominant oxide facies mineral is the sulphide chalcocite rather than chalcopyrite or bornite, and it is believed to be a secondary supergene enrichment associated with a paleo water table, or fault proximal oxidation via circulating groundwater. Minor malachite, azurite, remnant chalcopyrite-bornite, and native copper are also present within these near surface mineralized zones.

Cobbles and pebbles of this supergene chalcocite mineralization in Cretaceous age (unpublished data) conglomerate that unconformably overlies the plutonic rocks of the Granite Mountain Batholith indicate that the upper parts of the Minto System were on surface and being partially oxidized and eroded in the Late Cretaceous.

In addition to the obvious copper oxide minerals, oxidation is also evident by pervasive iron staining (limonite), earthy hematite, clay alteration of feldspars, and a significant loss in bulk density. The degree and distribution of copper oxide minerals appears to be directly related to the depth of the water table. For the most part this is confined to about -30 m (but up to -60 m) beneath the surface and is generally sub parallel with the present topographic surface. The Minto Main zone has experienced relatively little oxidation since it is generally more than 60 m below the surface except at its southern end where it crops out directly beneath unconsolidated overburden in the Minto Creek Valley. Very locally this oxidation may be drawn deeper along late brittle faults cutting primary sulphide mineralization.

8.3 Additional Mineralization Targets

The most favorable exploration targets (based on the evaluation of geophysics, soil geochemistry, geologic modelling, and diamond drilling are summarized below. The targets identified as Ridgetop Southwest, Copper Keel (North and South), Airstrip, Connector, DEF, and the newly discovered Wildfire prospect are all located within a 2 km by 2 km area, south of the DEF fault. MintoEx also sees good exploration potential in the area north of the DEF fault, as evidenced by the discovery of the high grade Minto North deposit early in 2009 and the recently discovered Inferno prospect in late 2010 as well as the presence of multiple Titan-24 DCIP anomalies.

Also in 2009, several other historic bedrock copper occurrences discovered in the 1970s north of the DEF fault were relocated and confirmed. In addition various copper-in-soil geochemical anomalies, often coincident with magnetic geophysical anomalies, occur throughout the property and many of them remain untested. However, further understanding of the bedrock geology north of the DEF fault is required before many of these targets can be properly assessed and placed in perspective. Various exploration targets that MintoEx geologists identify as having potential are identified in Figure 8.1 and are described in more detail below.

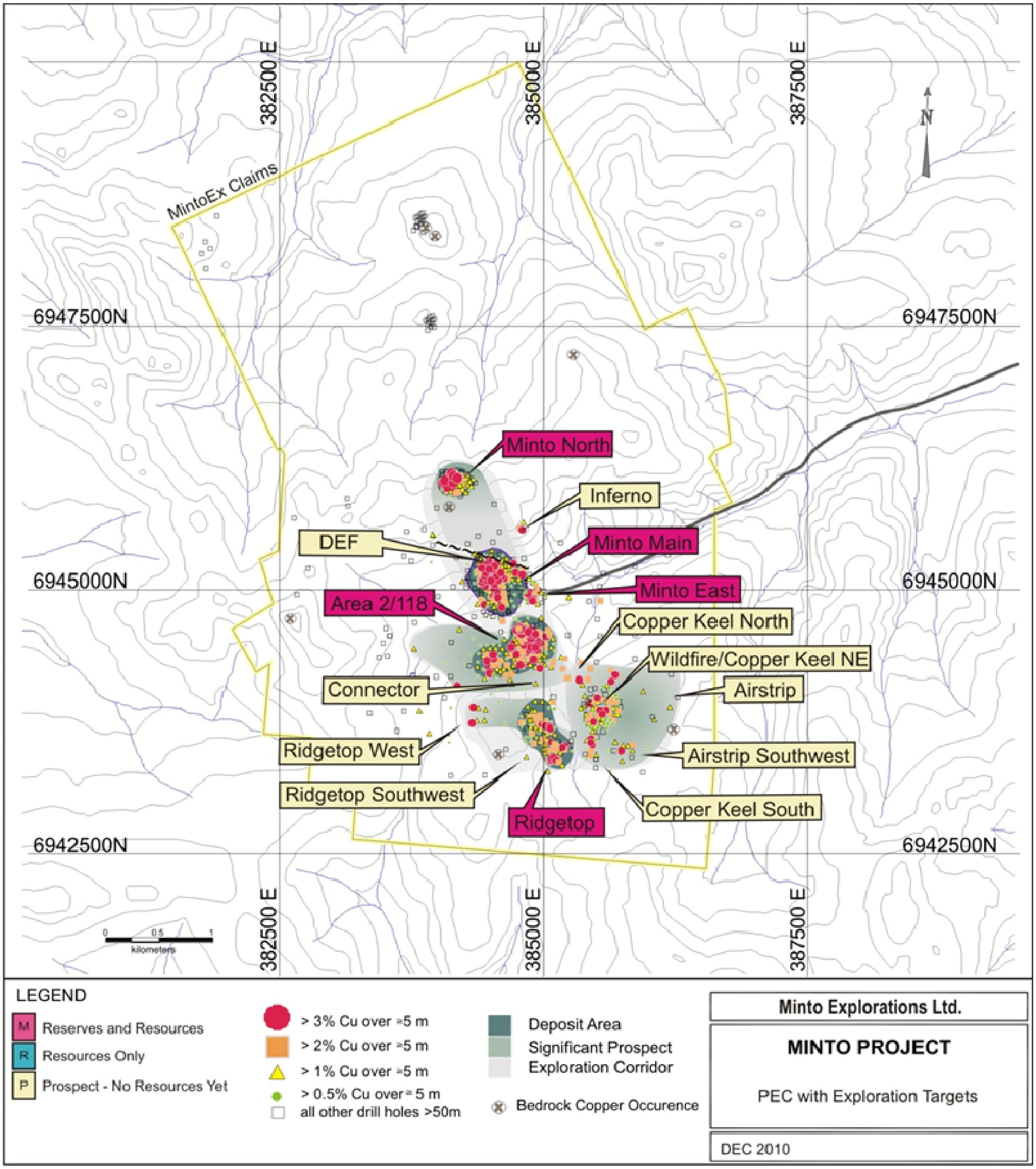


Figure 8.1: Exploration Targets

Wildfire / Copper Keel

This target, which lies east of the Minto Main-Area 2/118 – Ridgetop trend, was identified by a Titan-24 anomaly in 2010. The Wildfire chargeability anomaly measures more than 900 m north-south by more than 250 m east-west. Previous drilling at Copper Keel North, Copper Keel South, and Airstrip SW had skirted the perimeter of the Wildfire anomaly, but had not tested the main portion of the anomaly. Discovery drill hole 10SWC635 targeted the bulls-eye of the northern half of the Titan-24 anomaly at 700 m elevation, and intersected both shallow, high-grade mineralization and moderate-depth, moderate grade mineralization. Follow-up drill holes 10SWC639 and 10SWC640, stepping out 40 m to the north and south respectively, successfully intersected the same horizons as the discovery hole solidifying Wildfire as a bona fide exploration target. Shortly thereafter, systematic drilling on east-west oriented sections across the Wildfire target commenced with the goal of establishing the geographic extent of the mineralization and to attempt to link Wildfire to the Copper Keel prospect to the to the northwest.

A total of 88 holes (23,419 m) were drilled at the Wildfire target in 2010; the results of the systematic drilling indicate that the shallow mineralization at Wildfire is stacked on top of Copper Keel mineralization with the best grades at Copper Keel trending north-northwest and at Wildfire north-northeast (see Copper Keel discussion below). The shallow mineralization at Wildfire is principally comprised of chalcocite, along with lesser bornite, chalcopyrite, and trace native copper; similar chalcocite-dominated mineralization was defined in the upper mineralized zones at Ridgetop. Deeper mineralization is comprised of chalcopyrite and bornite, typical of Minto-style hypogene mineralization.

The shallow Wildfire zone was not intersected in some 2010 drill holes because of the presence of a conglomerate wedge truncating the zone. Cobbles of mineralized foliated granodiorite were observed in the Cretaceous Age conglomerate clearly showing the Wildfire horizon was on the surface and is clearly affected by erosion in the Cretaceous Age. Exhumation and erosion at some time before the Late Cretaceous Age appears to have removed sections of mineralization at the Wildfire prospect (similar to Copper Keel South and Airstrip SW).

Geological modeling is currently in progress and assaying is currently behind schedule so a NI 43-101 compliant mineral resource estimate is not anticipated until early 2011. The additional potential of the Wildfire prospect will be evaluated when drilling resumes in early 2011 including further step-out and infill drilling, and potentially establishing continuity across the current gap between the Wildfire and Ridgetop Deposits.

Significant assay results for Wildfire are presented in Table 8.1.

Table 8.1: Select Assay Interval Highlights from Wildfire Drilling

DDH ID	From (m)	To (m)	INT (m)*	Cu (%)	Au (g/t)
10SWC635	87.5	98.3	10.8	2.41	0.81
10SWC639	76.7	93.7	17.0	1.60	0.72
10SWC640	86.8	96.1	9.3	1.75	0.84
10SWC668	80.6	88.3	7.7	2.68	1.02
10SWC686	89.8	105.2	15.4	2.05	0.70

**Geological modelling shows that the best continuity between drill holes indicates horizontal to sub-horizontal mineralized horizons. Therefore the intervals indicated in Table 8.1 are to be near true widths.*

Copper Keel, is located southeast of the Minto deposit, and is subdivided into Copper Keel North and Copper Keel South. Copper Keel North is connected to the southeast edge of the Area 2 deposit and Copper Keel South is located approximately 180 m east of the southeast edge of the Ridgetop deposit (Figure 8.1). Moving to the east, the targeted mineralized horizons of Copper Keel North are located beneath the shallow mineralized zones of the Wildfire prospect. MintoEx geologists believe that the Copper Keel target is in the axis of a syncline, and that Copper Keel North is connected to Copper Keel South along the plunge of this open fold nose over a distance possibly up to 800 m; preliminary modeling of recent 2010 drilling appears to support this theory.

Copper Keel North roughly corresponds to an airborne magnetic anomaly approximately 600 m long by 200 m wide, and is defined by drill hole 06SWC164. Based on the analysis of the 3-D geological model from Area 2, MintoEx geologists interpreted a synformal structure, and positioned test hole 06SWC164 to intersect both the magnetic anomaly and the inferred keel of the fold. 06SWC164 intersected high grade copper mineralization (chalcopyrite + bornite + magnetite) at moderate depth within 3 m of the predicted intersection based on the geological model. Prior to 2009 five drill holes in a broad area had intercepted good grade copper mineralization at similar elevations.

Since then, further drilling in 2008, 2009, and 2010 comprising an additional 28 drill holes have been completed at the Copper Keel North target proper (not including holes denoted as Area 2 south or holes drilled to Copper Keel level depth at the Wildfire prospect). Highlights of the drilling are presented in Table 8.2. To date, all drill holes have intersected copper mineralization at a similar elevation as discovery drill hole 06SWC164, but with variable zone thickness and copper-gold grade.

The Copper Keel North target remains open essentially in all directions, but further drilling is required to increase the understanding of geology and any possible controlling structures on mineralization. More specifically, to the west the target horizon is open but is abruptly pinching off, to the east there is an approximately 250 m gap that may be connected to successful Titan-24 test drilling along a northeast extension of Copper Keel North (“Copper Keel NE”) and while still open the grade is diminishing to the north and south.

The Copper Keel South target corresponds to a Gradient Array Induced Potential (GAIP) chargeability anomaly approximately 600 m long by 240 m wide, and may be linked to the Ridgetop deposit in the west, crudely to the Airstrip Southwest target to the east, and Copper Keel North. Initial drilling at Copper Keel South was conducted in 2007 when drilling (971 m) identified high grade, chalcocite dominant, copper mineralization at shallow depths in 3 of 4 holes. In hole 07SWC242, the prospective zone was not intersected because of the presence of a conglomerate wedge truncating the zone, although cobbles of mineralized foliated granodiorite were observed in the conglomerate. Exhumation and erosion at some time before the Late Cretaceous appears to have removed sections of mineralization at the South Copper Keel and adjacent Airstrip prospects (similar to Wildfire). Follow-up drilling in 11 drill holes as part of the 2008 (229 m), 2009 (646 m), and 2010 (1,037 m) drill programs returned variable results for this same reason. Exploration here will need to be cognizant of this reality and further drilling is required to increase the understanding of geology and any controlling structures that may be removing or displacing the mineralized horizon.

SRK recommends that down hole geophysical surveys be carried out in any future drill holes in order to better vector exploration in the area. Highlights of the drilling at Copper Keel South during 2007 to 2010 are presented in Table 8.2.

Table 8.2: Select Average Assay Interval Highlights from Copper Keel North and South Drilling

DDH ID	From (m)	To (m)	INT (m)*	Cu (%)	Au (g/t)
A100-74	198.73	220.07	21.34	0.33	-
08SWC312	234.2	245.8	11.6	2.13	0.8
08SWC389	188.3	212.8	24.5	2.07	0.86
09SWC394	230.3	233.8	3.5	1.42	1.06
09SWC395	241.2	245.5	4.3	3.12	2.44
09SWC399	202.9	217.2	14.3	1.31	0.67
09SWC451	203.2	218.6	15.4	0.56	0.23
07SWC217	71.2	77.8	6.6	1.96	1.11
07SWC241	88.2	90.3	2.1	2.84	1.79
07SWC243	68.2	72.3	4.1	3.1	2.27
07SWC442	40.2	42.5	2.3	1.13	1
07SWC447	70.4	90.7	20.3	1.84	1.61
07SWC450	71.8	80.9	9.1	0.4	0.12
10SWC642	105.5	115.3	9.8	2.49	1.88

**Geological modelling shows that the best continuity between drill holes indicates horizontal to sub-horizontal mineralized horizons. Therefore the intervals indicated in Table 8.2 are to be near true widths.*

Airstrip Southwest

The Airstrip Southwest target corresponds to a GAIP chargeability anomaly approximately 300 m long by 300 m wide, and was initially defined by 2 historic drill holes A114-74 and A117-74. Between 2007 and 2008, MintoEx drilled 12 holes (3,323 m) in the Airstrip Southwest target returning encouraging copper mineralization results. Similar to the Copper Keel South area, the presence of a chalcocite dominant mineralization at shallow depths is confirmed. It is presumed that Airstrip Southwest was once connected and continuous with the Copper Keel South chalcocite horizon before deposition of the conglomerate, however erosion during the Cretaceous Age removed parts of the targeted horizon and the conglomerate wedge has replaced significant extents of the zone. However, promising chalcopyrite dominant copper mineralization at moderate depths was observed in almost all 2007 and 2008 drilling. The Airstrip Southwest target remains open in the east, south, and north (towards Wildfire) directions, and further drilling is required to determine the extent of mineralization.

SRK also recommends that down hole geophysical surveys be carried out on any future drill holes in order to vector exploration in the area. Select highlights of historical and current assays results are presented in Table 8.3.

Table 8.3: Select Assay Interval Highlights from Airstrip Southwest Drilling

DDH ID	From (m)	To (m)	Interval (m)*	Cu Grade (%)	Au Grade (g/t)
A114-74	141.12	157.89	16.77	1.04	-
A117-74	57.30	84.73	27.43	0.38	-
07SWC213	99.90	104.00	4.10	2.79	0.93
07SWC213	186.30	189.40	3.10	5.75	1.88
07SWC215	176.70	182.60	6.00	1.00	0.13
07SWC219	183.30	199.80	16.50	0.43	0.07
07SWC221	164.40	175.60	11.20	0.72	0.16
07SWC225	175.80	194.00	18.20	0.64	0.08
07SWC227	219.30	230.00	10.70	0.81	0.06
07SWC229	156.90	164.70	7.80	0.62	0.25
07SWC231	181.10	189.60	8.50	1.50	0.07
07SWC235	162.80	169.70	6.90	0.90	0.12
08SWC290	262.60	265.60	3.00	1.11	0.14

Inferno

This target, which lies immediately north of the DEF fault and about 150 m northeast of the Minto Main deposit, was initially identified by a Titan-24 anomaly in 2009. Initial test drilling of the target in 2 holes did not intersect any significant copper-gold mineralization, and MintoEx geologists believed that the position of the anomaly was in question similar to what was observed at Minto East. Upon the completion of the expanded 2010 Titan-24 survey, the center of the anomaly was better constrained and shifted about 150 m to the northeast of the original location.

The first drill hole 10SWC718 targeted the center of the 500 m by 200 m anomaly at 600 m elevation and intersected high-grade copper-gold mineralization at depth. A mise-a-la-masse down hole IP survey was immediately carried out, and the results indicated a strong southern vector. Follow-up drilling in 3 separate holes confirmed the continuity of a shallow, moderate grade zone underlain by a narrower, high grade zone in a 40 m by 40 m area.

MintoEx favours this area as a drill target as it is positioned in the gap between the Minto Main and Minto North deposits. Furthermore, this appears to be the best candidate for the off-faulted northern extension of the Minto Main deposit. The Titan-24 survey suggests that the four holes completed to date have tested the northern end of a chargeability anomaly that extends more than 300 m south of this drilling. This additional potential will be evaluated when drilling resumes in early 2011.

Significant assay results for Inferno are presented in Table 8.4.

Table 8.4: Select Assay Interval Highlights from Inferno Drilling

DDH ID	From (m)	To (m)	INT (m)*	Cu (%)	Au (g/t)
10SWC723	61.9	75.9	14.0	1.14	0.59
10SWC746	31.6	61.4	29.8	0.82	0.57
10SWC746	278.3	282.3	4.0	3.19	1.83
10SWC750	274.2	276.3	2.1	19.39	5.49
10SWC750	330.6	345.0	14.4	1.07	0.57

**Geological modelling shows that the best continuity between drill holes indicates horizontal to sub-horizontal mineralized horizons. Therefore the intervals indicated in Table 8.4 are to be near true widths.*

9 Exploration

Mineral exploration on the Minto property has been conducted intermittently since 1971. Subsequent to the discovery of the Main deposit, now being mined as an open pit, the adjacent southern half of the property has undergone systematic brownfields exploration. Exploration on the northern half is more sporadic. There are currently more than 1,000 drill holes within a roughly 16 square kilometre area. As such, following up on open mineralized horizons in geological models, projecting mineralized horizons into areas of little or no drilling, and drilling near historical drill hole intercepts were the principal exploration tools employed by MintoEx and its geologists. Subsequent to Capstone’s predecessor, Sherwood Copper’s acquisition of Minto Explorations Ltd. in June 2005, exploration from 2005 to 2010 has concentrated mostly on diamond drilling. However, an extensive historic soil sample survey and some ground based and airborne geophysics have been conducted and are very useful to guide drilling activity.

The current approach by MintoEx is the systematic evaluation of modern electrical (chargeability), geophysical methods by commissioning various “proof-of-concept” surveys over known mineralization and then expanding survey coverage outward into untested areas using these methods that are calibrated to known deposits. An emphasis is placed on looking for signature analogs as opposed to being pedantic about precise measurements of response. The predominant electrical geophysical methods used are Gradient Array Induced Potential (GAIP), Dipole-Dipole Induced Potential, and Titan-24 DC Induced Potential. Drill targeting is predominantly based upon the coincidence of an anomaly in one of the electrical (chargeability) methods with an anomaly in the 1993 total field airborne magnetic survey (MAG). Within the currently known extent of the PEC, future exploration programs will likely be more reliant on deep penetrating electrical / chargeability methods as the near-surface potential and discrete magnetic bull’s-eyes have largely been targeted. Magnetic data in areas located north of Minto North plus areas west and east respectively of the PEC may still be useful as these regions are still relatively under explored. Local test surveys of Bouger gravity over the Main deposit and horizontal loop electromagnetics (HLEM) over the Area 2 deposit failed to detect the mineralization and proved to be of little use, they were not conducted over other areas.

In a cycle of discovery and definition, new deposits have now been identified by diamond drilling in five separate areas outside of the original or Minto Main deposit that was known when the project was acquired in 2005. The new deposits include Area 2 discovered in 2006, Area 118 discovered in 2007, Ridgetop drilled for the first time by MintoEx in 2007, Minto East discovered in 2007, Minto North discovered in 2009, Wildfire and Inferno discovered in 2010. Also, as described in the previous section there are multiple other prospects distributed throughout the property. The focus of exploration since 2005 involves systematic exploration of the property area both south and north of the current open pit mine in a south-southeast to north-northwest striking trend MintoEx calls the Priority Exploration Corridor described (“PEC”) (Figure 9.1).

A brief chronological summary of work conducted on the property is contained in the history section of this report and is also described in the "Technical Report (43-101) for the Minto Project" by Hatch (August 2006) and "Area 2 Pre-feasibility Study Minto Mine, Yukon" (November 2007) found on the sedar.com website.

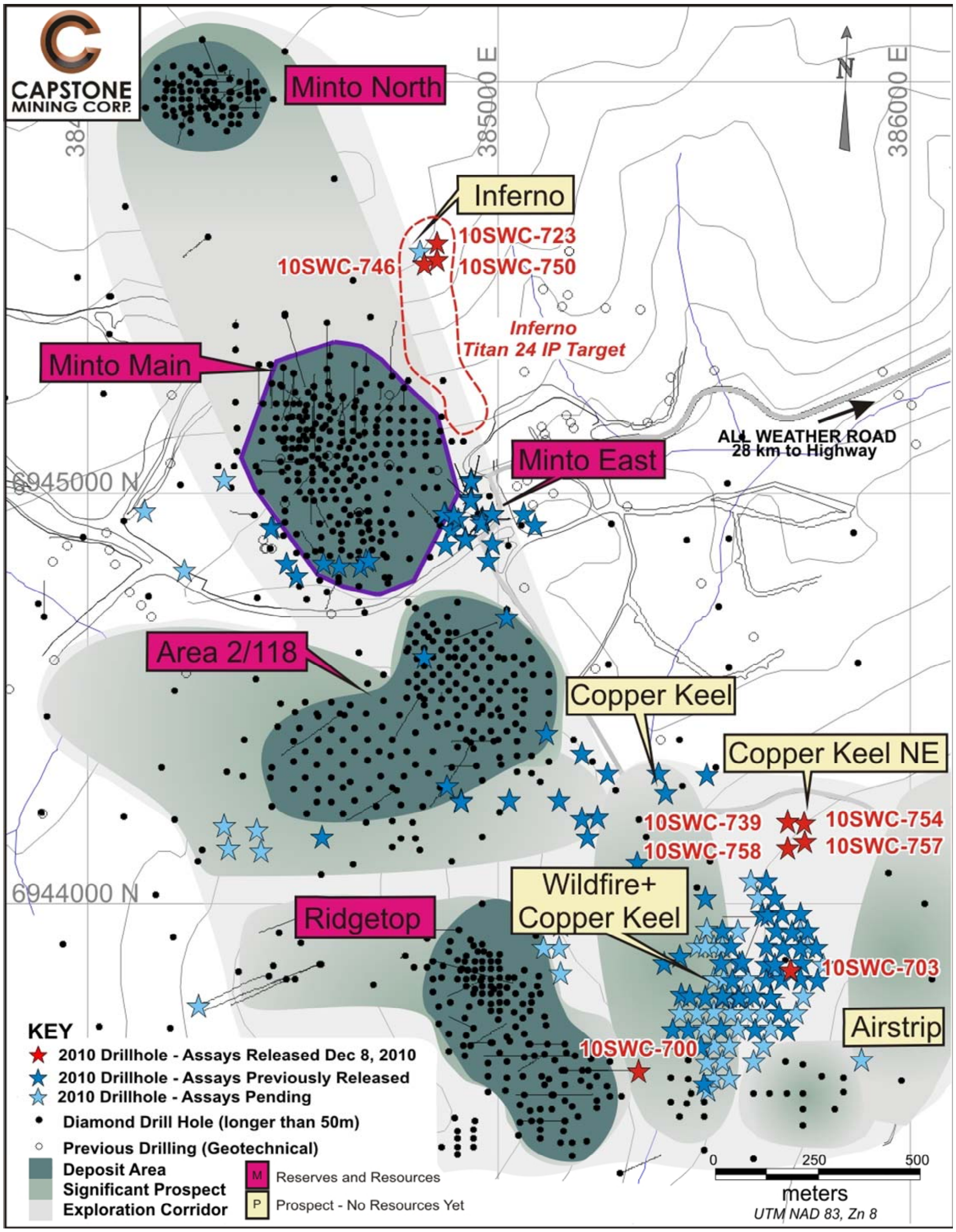


Figure 9.1: Priority Exploration Corridor (PEC) with Drill Collars Current to December 9, 2010

In 2008 and 2009, 61 additional infill and margin step-out drill holes into the Area 2/118 deposit allowed for the completion of a NI 43-101 resource estimation that was released June 9, 2009. In 2010, 22 additional infill and southern margin step-out drill holes into the Area 2/118 deposit lead to a more robust NI 43-101 resource estimation that was released August 30, 2010. The results of the 2010 drilling effectively linked the deeper mineralization in the southeast portion of Area 2/118 to the mineralized zones at Copper Keel North.

MintoEx geologists reassessed the Ridgetop area in 2007 (ASARCO's original Area 1 or Main discovery area) and drilled 25 new diamond drill holes, following up on 16 historical holes between the 1970's and early 1990's. The subsequent interpretation and drill density allowed for the completion of an NI 43-101 compliant resource estimate for Ridgetop East released December 12, 2007. In 2008 and 2009, 116 additional infill and step-out drill holes into the Ridgetop Deposit lead to a more robust NI 43-101 compliant resource estimation, which was released June 9, 2009. Subsequent follow-up drilling in the fall of 2009 totalling 40 drill holes initiated another NI 43-101 compliant resource estimate that was released August 20, 2010.

Early in 2008, a limited program of drilling in the overburden filled upper area of the Minto Creek valley identified several previously unknown areas of copper-gold mineralization now considered prospective. These discoveries are totally blind to surface, not discernable with GAIP surveys, have very muted magnetic high signatures and are essentially wildcat discoveries. Geological modelling at the western edge of the PEC at West Ridgetop and the western margins of Area 118, suggested the mineralized horizons may continue westward and dip beneath upper Minto Creek, expanding the Priority Exploration Corridor.

In 2009, MintoEx geologists followed up on two historic drill holes K88-74 and K91-74, that were originally collared to test a historic geophysical anomaly with a similar signature to the Minto Main deposit. Both drill holes failed to intersect any significant copper mineralization. The current 3D model now shows that one angled hole from 1974 drilled from the north passed beneath the main Minto North horizon, narrowly missing the discovery. A geology report dating from 1974 in the MintoEx archives, indicates the two holes were designed to test an IP feature.

The 1974 report suggests that the geophysical anomaly must have been misallocated in error. A more modern (2007) GAIP survey places the chargeability anomaly approximately 90 m further south than the historic anomaly. Drill testing based upon this new data resulted in the discovery of Minto North in 2009.

The first drill hole at Minto North 09SWC390, collared in the center of both the GAIP and MAG anomalies, intersected high-grade, near surface, Minto-style mineralization. The discovery drill hole was followed up by two additional preliminary step-out holes 09SWC392 and 09SWC393, that also intersected significant mineralization. MintoEx geologists now understand that the 1974 vertical drill hole K88-74 completely missed the deposit, and that angled drill hole K91-74 drilled underneath the deposit.

Upon the confirmation of the high-grade mineralization by assays, the new northern target was denoted as Minto North, and plans were made for additional step-out and possible infill drilling. After the first phase of step-out and infill drilling was completed April 13, 2009 a preliminary resource estimated was released on June 9, 2009. Shortly after, another infill program was completed by August 6, 2009 leading to the NI 43-101 compliant resource estimate completed June 9, 2009 contained herein. A total of 87 drill holes are included in the resource estimate reported herein.

The drilling at Minto North in 2009 returned some of the best copper mineralization intersected to date on the property. Similar to the Minto Main deposit, Minto North displayed a zoning from high-grade bornite dominant mineralization in the west to lower grade bornite + chalcopyrite mineralization in the east. The high-grade bornite-rich core also returned excellent gold grades, and in some cases visible gold was observed along epidote lined fractures.

The Minto East target was initially identified during the 2007 drilling in the gap between the Minto Main deposit and Area 2/118 deposit. A drill program was designed drill hole 07SWC176 collared approximately 200 m east of the southeast corner of the Minto Main deposit intersected 11.7 m of high grade copper-gold mineralization that looked remarkably similar to the Minto Main deposit mineralization, including abundant stringers of massive chalcopyrite. At the time, MintoEx geologists suspected that this intersection was the extension of the deep mineralization seen at Area 2. In 2008, a second drill hole (08SWC286) was collared approximately 120 m south-southeast of 07SWC176. This hole intersected mineralization at the anticipated depth although it was narrow in width and only moderate grade. The target remained dormant until 2009 when a geophysical survey (Titan-24) identified a sizable DCIP chargeability anomaly in the area at the right elevation.

The deep penetrating Titan-24 survey returned a chargeability anomaly spanning a minimum of 180 m long by 180 m wide being strongest at 600 m elevation. However because the anomaly was located only on one line on the easternmost flank of the survey it was poorly constrained. The first drill hole in 2009 drilled nearly on the geophysical survey line 09SWC583 intersected only a narrow zone.

Because the Titan-24 survey was a localized test of the technology, it was suspected the source of the anomaly was due to mineralization located some distance off the survey line. Drill holes 09SWC584, 09SWC586, and 09SWC591 were collared further east and returned excellent copper grades and thickness’ confirming Minto East as a bona fide exploration target. A down hole geophysical survey in 09SWC584 produced a southern vector that was the focus of initial follow-up drilling in 2010.

In 2010, an additional 27 holes at Minto East defined the deposit and lead to a robust NI 43-101 compliant resource estimate released August 30, 2010. This drilling essentially cut-off the mineralization in all directions, but is marginally still open in the south towards the east side of Area 2. Similar to the Minto Main deposit and Minto North, Minto East displayed a zoning from high-grade bornite dominant mineralization in the west to lower grade bornite + chalcopyrite mineralization in the east.

The high-grade bornite-rich core also returned excellent gold grades, and in some cases visible gold was observed along epidote lined fractures or directly associated with bornite.

Company geologists proposed, in 2006, that the separate prospects and deposits mentioned above comprise a single large continuous to contiguous mineralized system that has subsequently been deformed; openly folded and cut by late regional faults (some with vertical displacements and some with inferred lateral displacements). The sum of MintoEx's drilling and geological modelling since 2005 to date continues to support the single system thesis and upcoming exploration work in 2011 and beyond will focus on creating a unified geological model for the property south of the DEF fault, and possibly extending north of the DEF fault to Minto North.

Projecting 3-D geological models based on drill hole data into untested areas and then following up on promising targets remains the most important exploration tool at Minto. A significant portion of exploration work in 2008 and 2009 concentrated on infill drilling followed by stepping out from the Area 2/118 deposit and Ridgetop deposits. At Minto North, 2009 drilling evolved from exploration, to delineation, to infill. A similar pattern was followed for Minto East in 2009 and 2010. Infill drilling for all deposits yielded statistically more robust resource calculations, supporting the current PFS study, while step-out drilling continued to test for further extensions of the deposits. During 2009, two separate deep penetrating geophysical surveys were completed in order to fill in gaps not covered by the 2006-2007 GAIP survey, to test areas with deep overburden or permafrost, and to test deep ground under known deposits in the PEC. The first program of Dipole-Dipole Induced Polarization (DDIP) was completed by Aurora Geosciences of Whitehorse, Yukon over areas northwest, north, and northeast of the Minto deposit. The second program of Titan-24 DCIP and MT was completed by Quantec Geosciences of Toronto, Ontario over the PEC. In 2010, an expanded Titan-24 DCIP survey was completed covering ~85% of the property. The Titan-24 surveys are discussed in more detail in section 9.3.

In 2010, drill testing of Titan-24 anomalies successfully identified two new high priority targets including the Wildfire prospect and the Inferno Prospect. Step-out and infill drilling at both prospects is slated to continue in 2011. An initial resource estimate for the Wildfire prospect is anticipated for early 2011 and a more robust estimate should be completed later in that same year after the seasonal cessation of drilling activities.

The discovery of eight new copper-gold deposits or significant prospects (Figure 9.2) in five years attests to the validity of the exploration methods being used at the Minto Mine by Capstone Mining Corporation and its subsidiary MintoEx.

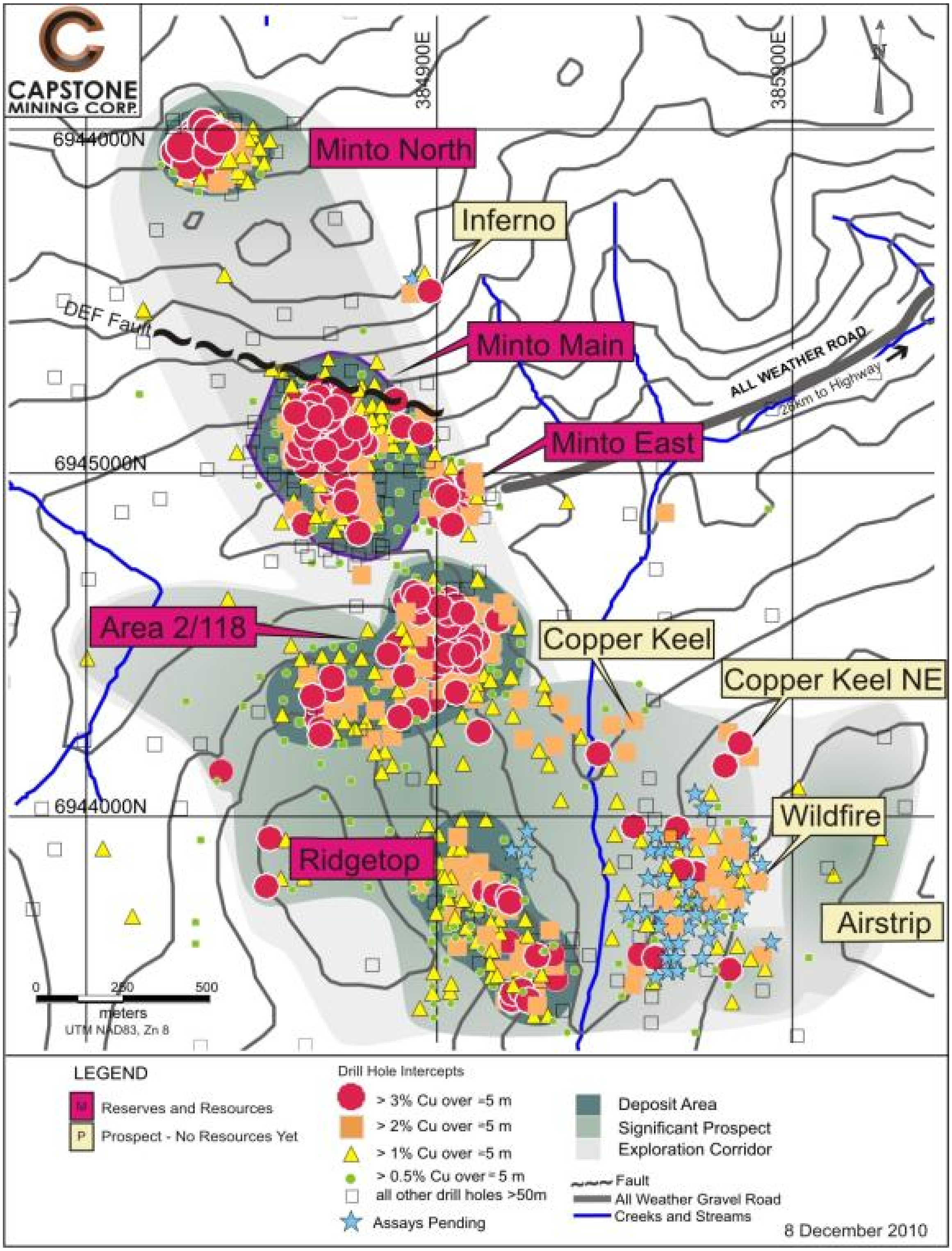


Figure 9.2: Priority Exploration Corridor (PEC) with Drill Results Showing the Highest Copper Grade over a Minimum Continuous 5 m Interval

9.1 Gradient IP Geophysical Surveying

An important component of the 2007 exploration program included increasing the coverage of the Gradient Array Induced Polarization ("GAIP") survey at Minto. A total of 138 line kilometres of GAIP surveys were completed in 2007, a four-fold increase over the 33 km completed in the 2006 program, bringing the total GAIP kilometres surveyed by MintoEx for both years to 171 km. The GAIP surveying for 2006 and 2007 was conducted by Aurora Geosciences of Whitehorse, Yukon Territory, using the following specifications:

- Array: Gradient
- Dipole Spacing: 50 m
- Tx: Time domain, 50% duty cycle, reversing polarity, 0.125 Hz
- Stacks: Minimum 15
- Rx Error: 5 mV/V or less, otherwise repeated several times
- Grid Registration: Handheld GPS points minimum every 300 m and at line-ends; (<10 m accuracy)

The 2007 survey was completed on ten separate blocks expanding upon the 2006 survey area to provide near seamless coverage over a total area of approximately 10 km^2. Areas with extensive mining activity or infrastructure could not be surveyed. The 2007 GAIP program was much more extensive than the 2006 pilot survey because drilling of the chargeability anomalies generated in the 2006 survey was positive. The GAIP survey showed a coincidence of significant copper sulphide mineralization with chargeability anomalies and suggested MintoEx had developed an additional exploration tool for prioritizing exploration drill targets.

The focus of the 2007 geophysical program was two-fold; firstly, to evaluate areas south of the main Minto deposit, expanding coverage into areas of known prospectivity that was not covered in the 2006 program and secondly, to begin evaluating areas north of the Minto mine, where there are multiple coincident copper-in-soil and magnetic anomalies, but very little core drilling. After positive drill results were obtained late in the drill program on a changeability anomaly, located at the Airstrip SW and Copper Keel prospects on the southern limit of the 2006 survey area, a decision was made to expand the GAIP survey to an area south of the drill discovery.

The additional survey at Airstrip-Copper Keel defined a large chargeability anomaly in an under-explored region located to the south of the diamond drilling. This area was previously thought to be not prospective due to the presence of Cretaceous age cover rocks. These cover rocks are thought to represent a significant down throw and burial of the prospective host Jurassic age granodiorite. The new drilling had indicated the cover sequence was shallower than expected and granodiorite is locally exposed beneath overburden in small erosion windows through the conglomerate.

Drill discoveries of high-grade copper-gold mineralization at Airstrip and Copper Keel in 2007 are on the northern edge of a much larger chargeability feature than shown by the 2006 GAIP survey, suggesting additional potential beyond the range of recent drilling. This large chargeability anomaly remains a high priority drill target for future drill programs.

Several other chargeability anomalies identified in the 2007 GAIP survey are located to the north of the main Minto Main open pit mine, indicating exploration potential north of the mine. This is an area where total field magnetic data and soil geochemistry indicate a prospective exploration environment but it has had only very cursory exploration drilling by past operators. Two anomalies identified in the 2007 program (both coincident with total field magnetic highs and positive copper-in-soil geochemistry) included a strong east-west linear chargeability feature located approximately 600 m north of the Minto Main deposit (now known as the Minto North deposit) and the very large horseshoe shaped anomaly to the northeast of the Minto Main deposit. Based on the success in 2009 drilling the coincident anomalies at Minto North, the horseshoe shaped anomaly northeast of Minto Main deposit is considered a priority drill target for future exploration drill programs.

Not all anomalies have produced positive results. A chargeability anomaly from the 2006 GAIP survey was drill tested in 2007 with negative results. No significant copper-gold mineralization was encountered despite the intersection of multiple, thick sequences of foliated favourable host rock. Minor pyrite and trace chalcopyrite was sporadically encountered in four drill holes but it is believed that the low concentration of this mineralization does not satisfactorily explain the chargeability results.

Despite excellent correlation of copper-gold mineralization with GAIP anomalies at other locations on the Minto property, the survey does not yield a unique correlation with high grade mineralization. The GAIP survey is a tool that is more efficient when used in conjunction with other corroborating data suggestive of buried mineral deposits. For example, at Copper Keel and Airstrip, direct targeting of GAIP anomalies was considered instrumental in their discoveries. However, at Ridgetop and Area 2/118, breaks in the GAIP and Magnetic anomalies were helpful in inferring some limiting structures but the projection of nearby 3D models and previous drilling provided the strongest rationale for 2007 drilling.

Drilling in 2008 and 2009 has shown that the GAIP method is less effective in areas of deep overburden with variable permafrost conditions. In 2008, three new areas of mineralization were discovered in the upper Minto Creek valley under permafrost bearing overburden in areas that did not show any significant GAIP anomalies. Total Field Magnetic data was of some use in these areas, but drilling magnetic anomalies also produced inconsistent results. Future success in areas of deep overburden will rely heavily on geological modelling or deep penetrating IP surveys such as dipole-dipole and Titan 24 DCIP.

9.2 Modified Pole-Dipole Geophysical Surveying

A new exploration tool implemented in 2009 included the completion of a modified pole-dipole geophysical survey over areas west and north of the DEF fault from July 18 to August 10, 2009. The survey targeted areas of known historical geophysical anomalies, and well as overlapping GAIP coverage were permafrost or deep overburden ground conditions returned poor results (Figure 9.3). A total of 20.6 line kilometres were completed by Aurora Geosciences of Whitehorse, Yukon Territory, using the following specifications:

- Array: Modified Pole-Dipole Array
- Dipole Spacing: 50 m on all lines
- Dipole Read N = 1 through 10 (10 Channels)
- Tx: Time domain, 50% duty cycle, reversing polarity, 0.125 Hz
- Stacks: Minimum 15
- Rx Error: 5 mV/V or less, otherwise repeated several times
- Grid Registration: Handheld GPS points minimum every 250 m and at line-ends
- <10 m accuracy; all coordinates in UTM NAD83 Zone 8V North

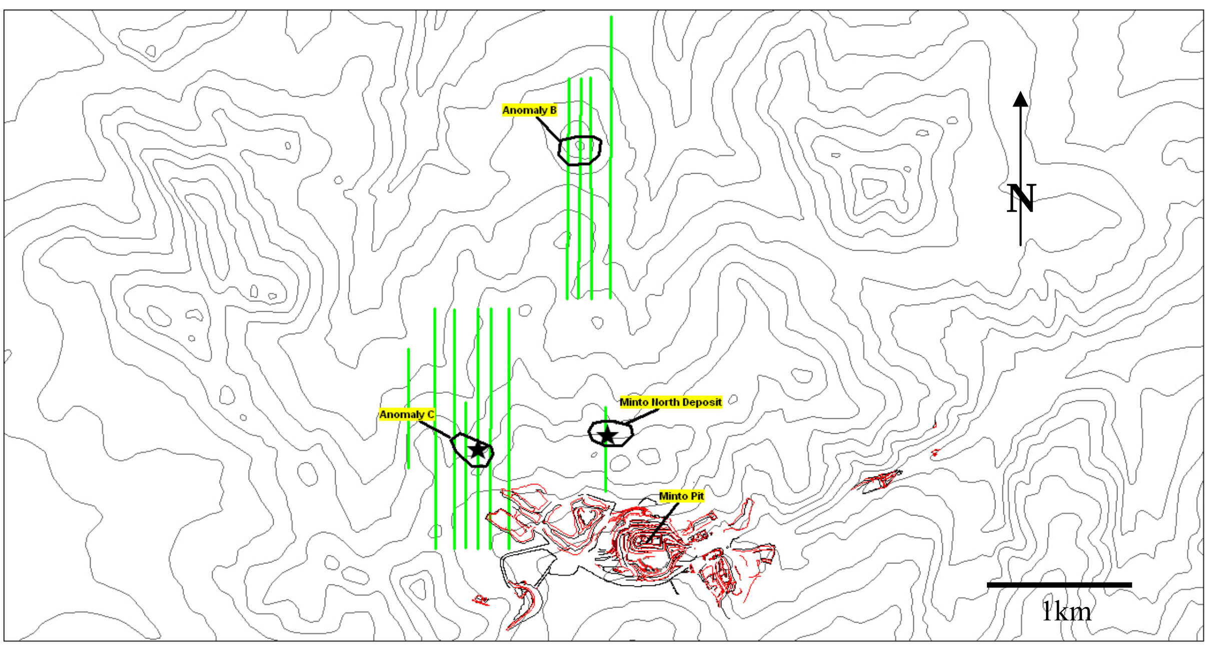


Figure 9.3: Modified Pole-Dipole 2009 Survey Grid Location Map (Green Lines) and Location of modified Mise-a-la-Masse Drill hole Surveys (Black Stars)

The results of the 2009 modified pole-dipole survey indicated two separate anomalies, one approximately 1,000 m due west of Minto North, and the second approximately 2,400 m due north of Minto North.

These 2009 anomalies were in good agreement with the historical pole-dipole survey anomalies denoted as Anomaly B (north) and Anomaly C (west) identified by ASARCO in 1974 (Figure 9.3). Similar to the historical Minto North anomaly ("Anomaly A"), ASARCO geologists believed that both of these anomalies were promising targets since the chargeability results were in similar magnitude to that of the Minto Main deposit. Due to the positive results of drilling at Minto North in 2009, MintoEx executed 1 drill hole into Anomaly B and 2 drill holes into Anomaly C. Drill results were enigmatic in that no significant copper-gold mineralization was encountered despite the intersection of multiple, thick sequences of foliated favourable host rock.

Minor pyrite and trace chalcopyrite or bornite was sporadically encountered in the 3 drill holes but it is believed that the low concentration of this mineralization does not satisfactorily explain the chargeability results.

Since the 2009 modified pole-dipole test line over Minto North with known high-grade copper mineralization confirmed a similar chargeability response to Anomalies B and C, MintoEx geologists felt that the results of the preliminary drilling were inconclusive. Thus, a single down hole mise-a-la-masse survey was completed at Anomaly C in hopes of further vectoring follow-up drilling (see below for details of the survey). Preliminary field results of this down hole survey were again in agreement with a calibration survey at Minto North suggesting that Anomaly C was still an intriguing exploration target. Both Anomalies B and C remain priority targets for future drill programs, and follow-up drilling will be focused using the results of the combined 3-D modelling of survey and incorporated down hole survey results.

As part of the 2009 modified pole-dipole geophysical survey, one calibration (Minto North) and one follow-up (Anomaly C) mise-a-la-masse drill hole IP survey were completed by Aurora Geosciences of Whitehorse, Yukon Territory, using the following specifications:

- Array: Radial Array
- Dipole Spacing: 25 m on all lines
- Tx: Time domain, 50% duty cycle, reversing polarity, 0.125 Hz
- Stacks: Minimum 15
- Rx Error: 5 mV/V or less, otherwise repeated several times
- Grid Registration: Handheld GPS points at line-ends and the center of each line
- <10 m accuracy; all coordinates in UTM NAD83 Zone 8V North

In 2010, mise-a-la-masse drill hole IP surveys were completed in single drill holes at Copper Keel NE and the Inferno prospect by Aurora Geosciences of Whitehorse, Yukon Territory, using the following specifications:

- Array: Radial Array
- Dipole Spacing: 50 m on main lines, 25 m for infill lines
- Tx: Time domain, 50% duty cycle, reversing polarity, 0.125 Hz
- Stacks: Minimum 15
- Rx Error: 5 mV/V or less, otherwise repeated several times
- Grid Registration: Handheld GPS points at line-ends and the center of each line
- <10 m accuracy; all coordinates in UTM NAD83 Zone 8V North

9.3 Titan-24 Geophysical Surveying

This section is summarized from the "Quantec Titan-24 Distributed Acquisition System DC Resistivity, Induced Polarization and MT Resistivity Survey over the Minto Mine Interpretation Report" by Quantec Geoscience (September 2009) and from "Titan-24 DC/IP Survey Geophysical Report, Minto Mine" by Quantec Geoscience (August 2010).

Another new exploration tool implemented in 2009 included the completion of the deep penetrating Titan-24 geophysical survey over the Minto priority exploration corridor from July 29 to August 8, 2009. The survey included three double spread direct current resistivity/induced polarization (DC/IP) and magnetotelluric ("MT") lines totalling 21 line kilometres.

Each line was positioned on an azimuth of 341 degrees extending from south of Ridgetop to north of Minto North. Each line was surveyed with pole-dipole geometry with a dipole spacing of 100 m. The array length was 2.4 km and two arrays were used with 400 to 500 m overlap to measure the 4 km long line. The data were inverted using 2D inversion algorithms to produce plan and section maps of DC and MT resistivity and chargeability of the subsurface. Data quality was very good, especially for an active mine site; typical measurements errors for DC were well below 0.5% and approximately 5% for the IP data with MT data in the quality range of 10 kHz to 0.01 Hz for most of the sites. The Titan-24 surveying for 2009 was conducted by Quantec Geoscience of Toronto, Ontario.

The 2009 DC/IP surveys used the following specifications:

- Survey Array: Dipole-Pole-Dipole (combined PDR and PDL)
- Receiver Configuration: 24-25 Ex = Continuous in-line voltages
  13 Ey = Alternating (2-station) cross-line voltages
- Array Length: 2400-2500 metres

- Number of Arrays/Line: 2
- Dipole Spacing: 100 metres
- Sampling Interval: Ex = 100 metres
  Ey = 200 metres
- Rx-Tx Separation: N-spacing (Pn-Cn min) = 0.5 to 39.5
- Infinite Pole Location: UTM: 392344E, 6948844N (NAD 83, Zone 08V North)
- Spectral Domain: Tx = Frequency-domain square-wave current
- Spectral Domain: Rx = Full wavefrom time-series acquisition
- Transmitter Waveform: 30/245 Hz square wave (~ 4 s. pos/neg), 100% duty cycle
- The 2009 MT surveys used the following specifications:
- Technique: Tensor soundings, remote-referenced
- Base Configuration: 24-25 Ex = Continuous in-line E-fields
  13 Ey = Alternating (2-station) cross-line E-fields
  1 pair LF coils
  1 pair HF coils
- Remote Configuration 1 Ex = in line E-fields
  1 Ey = cross-line E-fields
  1 pair LF coils
  1 pair HF coils
- Array Length: 2400-2500 metres
- Number of Arrays/Line: 2
- Dipole Spacing: 100 meters
- Sampling Interval: Ex = 100 metres
  Ey = 200 metres
- Ex/Ey Sampling Ratio: 2:1
- E/H Sampling Ration: Ex = 24:1 and 25:1
  Ey = 13:1
- Remote Measurements: 1Hx/Hy set (1 Ey/Ex for verification/monitoring)
- Remote Position: 424855E, 7001518N (NAD 83, Zone 08V North)
- Frequency bandwidth: 0.01 to 10000 Hz

- Data Acquisition: Full-waveform time-series acquisition
  Data processing/output in frequency-domain

An expanded Titan-24 DC/IP survey covering about 85% of the property was completed from May 19 to July 15, 2010; MT was not included in the expansion. In total, ten additional parallel lines expanding outwards to the east and west from the 2009 PEC coverage with 600 m current extensions on either end were surveyed as well as one northern extension to an existing 2009 line. Each line was positioned on an azimuth of 341 degrees, and line spacing varied between 200 m and 400 m. The survey included ten double spread and one single spread direct current resistivity/induced polarization (DC/IP) lines totalling 62.5 line kilometres.

Each line was surveyed with pole-dipole geometry with a dipole spacing of 100 m. The array length was between 1.7 and 2.7 km and multiple arrays were used with 400 to 600 m overlap to measure lines up to 6.6 km long (including current extensions).

The data were inverted using 2D inversion algorithms to produce section and plan maps of DC resistivity and chargeability of the subsurface. Data quality was very good, especially for an active mine site; typical measurement errors for DC were well below 0.5% and approximately 5% for the IP data. The Titan-24 surveying for 2010 was conducted by Quantec Geoscience of Toronto, Ontario.

The 2010 DC/IP surveys used the following specifications:

- Survey Array: Dipole-Pole-Dipole (combined PDR and PDL)
- Receiver Configuration: 24-25 Ex = Continuous in-line voltages
  13 Ey = Alternating (2-station) cross-line voltages
- Array Length: 2400-2500 metres
- Number of Arrays/Line: 2
- Dipole Spacing: 100 metres
- Sampling Interval: Ex = 100 metres
  Ey = 200 metres
- Rx-Tx Separation: N-spacing (Pn-Cn min) = 0.5 to 39.5
- Infinite Pole Location: UTM: 392344E, 6948844N (NAD 83, Zone 08V North)
- Spectral Domain: Tx = Frequency-domain square-wave current
- Spectral Domain: Rx = Full wavefrom time-series acquisition
- Transmitter Waveform: 30/245 Hz square wave (~ 4 s. pos/neg), 100% duty cycle

The 2009 Titan-24 survey was completed over the Minto PEC in order to first test the geophysical response over the known deposits Ridgetop, Area 2/118, Minto Main, and Minto North; and secondly to evaluate the possibility of deep mineralization lying beneath these known deposits to a depth of approximately 750 m. Thirdly, using the maps of the resultant resistivity to possibly identify and characterize large scale structures over the Minto Mine area. Where the survey grid was positioned over the Minto Pit, the west and east flanking lines were bent around the pit and the central line was executed by using rafts to position electrodes across the flooded pit bottom. The expanded 2010 Titan-24 DC/IP survey was completed in order to constrain the positioning of anomalies identified in 2009 as well as to evaluate areas of the property proximal to and well outside the PEC. The combined 2009-2010 Titan-24 survey grid is presented in Figure 9.4.

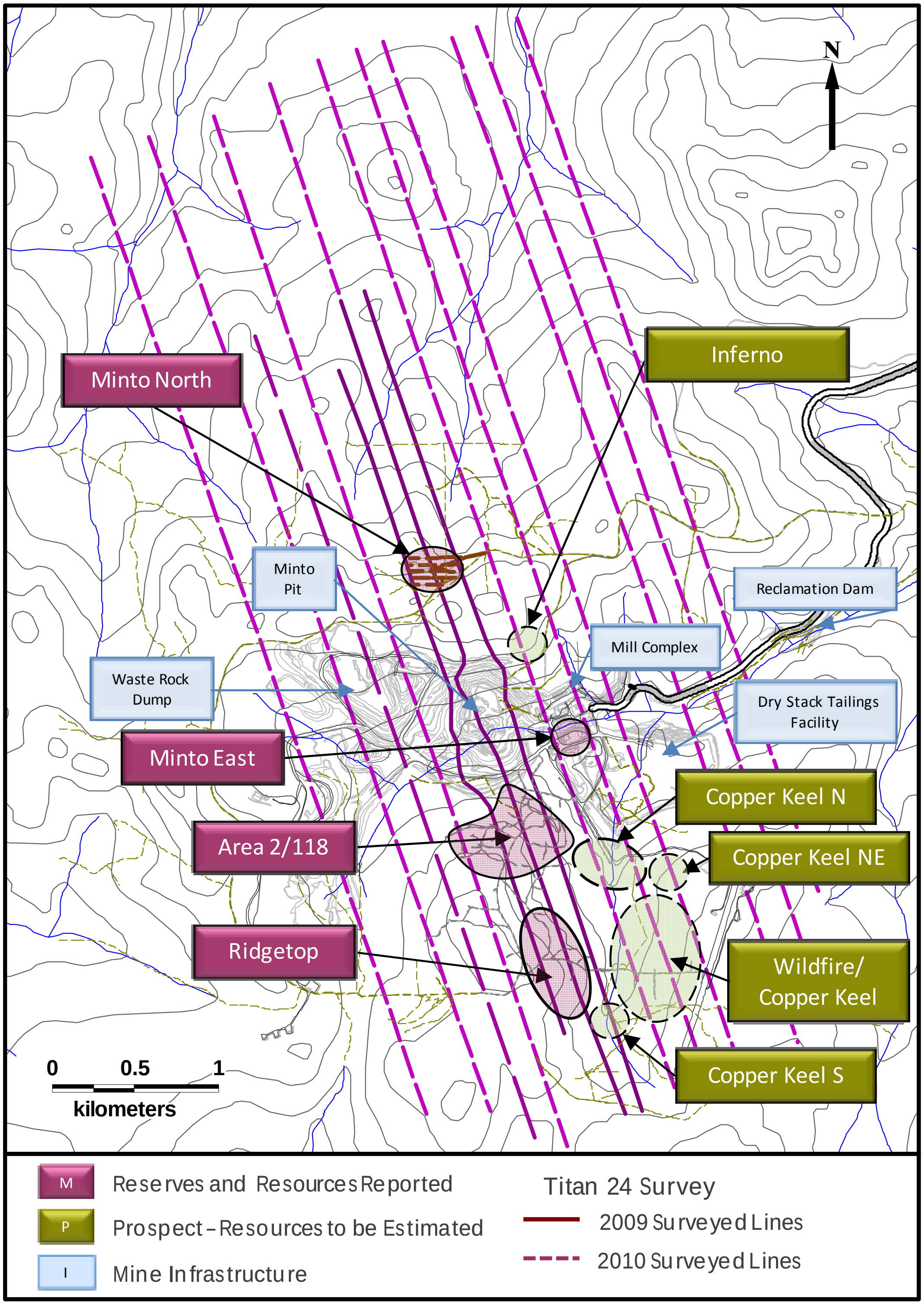


Figure 9.4: Titan-24 Combined 2009-2010 Survey Grid

The 2009 Titan-24 survey showed a coincidence of significant copper sulphide mineralization of known deposits with chargeability anomalies as well as several previously unknown deep anomalies, suggesting that MintoEx had developed an additional exploration tool for prioritizing exploration drill targets. The most attractive deep targets were located south of Ridgetop, flanking the Minto Main Pit (west, southeast, northwest, and northeast), and flanking the Minto North deposit (east, west, and north). The survey also identified a near surface target southwest of Ridgetop. MT results indicated steeply dipping fault-like structures with an estimated 70^o dip to the north, the most prominent being the DEF fault.

Preliminary drill testing of the Titan-24 chargeability targets spanned from September 4 to October 17, 2009. Results of the drilling were variable returning promising copper mineralization intersections in 9 drill holes at Ridgetop Southwest and significant copper-gold mineralization in 2 holes southeast of Minto Pit (Minto East discovery), but in 9 holes at 8 other separate targets no significant copper-gold mineralization was encountered despite the intersection of multiple, thick sequences of foliated favourable host rock. Based upon discussions with representatives of Quantec Geosciences and upon the experience gained at Minto East where the first hole missed and a second hole drilled more than 130 m east of the actual survey line confirmed the discovery, the lack of success at some of these other anomalies was attributed to at least in part due to the limited coverage of the survey. In other words, the method appeared to be able to “see” anomalous features that actually sit well to the side of the survey area. Because the initial proof-of-concept survey was only three lines wide and because all significant and unexplained anomalies lay on either of the two flanking lines these anomalies were considered to be poorly constrained.

Minor pyrite and trace chalcopyrite was sporadically encountered in the nine unsuccessful 2009 test holes, but it was believed that the low concentration of this mineralization did not satisfactorily explain the chargeability results. MintoEx geologists suspected that the poor intersections into the various targets may have reflected a positioning problem with these specific anomalies; as mentioned above these anomalies flanked either the eastern or western survey lines and the exact locations were thus poorly constrained. Follow-up down hole DC/IP surveys were completed in five holes where drilling results were in question in order to guide follow-up drilling, and the decision was made to complete additional parallel survey lines positioned to the east and the west of the PEC coverage to further vector in on the precise locations of the anomalies using more constraining data to provide better resolution and more precise locations of chargeability anomalies.

Similar to the 2009 Titan-24 survey, the expanded 2010 survey identified previously unknown moderate to deep anomalies (Figure 9.5). The most attractive new targets were located east of the Copper Keel trend (Wildfire prospect), at Copper Keel NE, southwest of Ridgetop, at Airstrip SW, and northeast of the Minto airstrip. The positioning of the 2009 anomalies flanking the Minto Main Pit (west, southeast, northwest, and northeast) and flanking the Minto North deposit (east, west, and north) shifted as expected due to better constraining data. For example, the discovery hole for the Inferno prospect targeted the 2010 position of the anomaly northeast of the Minto Pit that shifted by 180 m to the northeast as compared to the 2009 anomaly location.

However, the positioning of the Minto East anomaly did not shift to the east as was expected, and the conflict between the anomaly location and the deposit location remains unexplained.

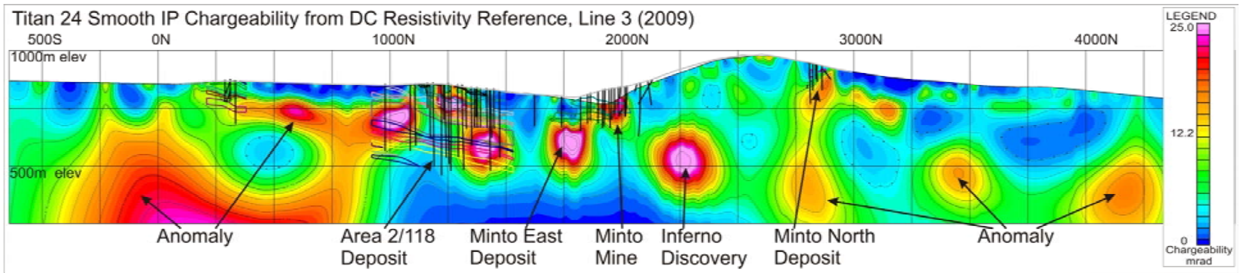


Figure 9.5: North-South Cross-Section (Line 1 from Figure 9.4) showing Titan-24 Anomalies)

Drill testing of the 2010 Titan-24 chargeability targets spanned from June 25 to November 5, 2010.

Results of the drilling were variable returning significant copper mineralization intersections in more than 70 plus drill holes east of the Copper Keel trend (Wildfire discovery) and in 4 holes northeast of Minto Pit (Inferno discovery); promising copper-gold mineralization was observed in 3 holes southwest of Area 118, 4 holes at Copper Keel NE, and in 1 hole at Ridgetop NE; no significant copper-gold mineralization was encountered in 5 holes at three other separate targets despite the intersection of multiple, thick sequences of foliated favourable host rock. In some cases, the new 2010 anomalies flanked the survey lines and it is suspected as with the 2009 survey that the positioning of these anomalies are again questionable. However, those anomalies drill tested that were well constrained that did not intersect significant mineralization remain unexplained. Based upon discussions with representatives of Quantec Geosciences the lack of success at some of these other anomalies may be attributed to at least in part to the presence of magnetite or platy minerals (i.e. biotite) which are present in the foliated granodiorite horizons at Minto.

Testing of new and verified Titan-24 targets as well as revisiting 2009 anomalies that have been better constrained is slated to continue into 2011 with the prime focus of follow-up drilling at Wildfire, Inferno, Copper Keel NE, and other targets north of the DEF. The authors recommend that all initial holes testing new anomalies should be followed up by DHIP to vector future drilling.

10 Drilling

MintoEx drilled a total of 47,084 m in 167 drill holes on the Minto Property in 2010. The 2010 drilling program was conducted between January 22 and November 5, 2010 and was contracted to Kluane Drilling Ltd. of Whitehorse, Yukon (up to February 15, 2010) and Driftwood Diamond Drilling Ltd. (after February 15, 2010) of Smithers, British Columbia under the direct supervision of MintoEx and Capstone Mining Corporation staff geologists. Forty-nine 2010 drill holes (15,263 m) were used in the resource estimations discussed in this report, however 92 drill holes (25,152 m) completed in 2010 are associated with the Wildfire and Inferno prospects that are still being explored and as such are not incorporated into the mineral resource estimates used in this report.

In 2009, MintoEx drilled a total of 31,479 m in 201 diamond drill holes at the Minto North, Area 2/118 and Ridgetop deposits, and at various other prospects. Drilling was conducted from January 27 to October 17, 2009 and was contracted to Driftwood Diamond Drilling of Smithers, BC under the direct supervision of MintoEx and Capstone Mining Corporation staff geologists. The median length of 2009 MintoEx drill holes was 123 m (average 157 m), with the shallowest hole being 54 m in length and the deepest, 752 m.

In 2008, MintoEx drilled a total of 23,840 m in 120 diamond drill holes at the Area 2/118, and Ridgetop deposits, and at various other prospects. Drilling was conducted between March 6, to August 29, 2008 and was contracted to Peak Drilling Ltd. of Courtney, BC under the direct supervision of MintoEx and Capstone Mining Corporation staff geologists. The median length of 2008 MintoEx drill holes was 198 m (average 199 m), with the shallowest hole being 26 m in length and the deepest, 385 m.

A total of 49 holes (22 Area 2/118 and 27 Minto East) or 15,263 m of the 2010 drilling were incorporated into the four resource models described in this report. 118 holes for 31,821 m were drilled specifically at exploration prospects outside of these resource models. The median length of 2010 MintoEx drill holes was 279 m (average 282 m), with the shallowest hole being 33 m in length and the deepest, 693 m. MintoEx diamond drill holes by year and deposit, from 2005 through 2010, are summarized in Table 10.1 below.

Table 10.1: Summary of MintoEx Drill holes by Deposit (2005 to 2010)

Company	Deposit	Year	No. DDH	Type	Core Size	Metres	Angled	Vertical
MintoEx	Minto	2010	14	DDH	NQ	1,090	12	2
MintoEx	Minto	2009	2	DDH	(1) HQ, (1) NQ	591	1	1
MintoEx	Minto	2008	-	-	-	-	-	-
MintoEx	Minto	2007	5	DDH	(3) HQ, (2) NQ	754	3	2
MintoEx	Minto	2006	25	DDH	NQ	4,119	-	25
MintoEx	Minto	2005	44	DDH	NQ	5,369	8	36
MintoEx	Area 2	2010	21	DDH	(7) NTW, (14) NQ	6,104	6	15
MintoEx	Area 2	2009	5	DDH	NQ	568	-	5
MintoEx	Area 2	2008	14	DDH	NQ	3,594	-	14
MintoEx	Area 2	2007	26	DDH	NQ	7,672	2	24
MintoEx	Area 2	2006	79	DDH	NQ	18,134	-	79
MintoEx	Area 118	2010	1	DDH	NQ	267	-	1
MintoEx	Area 118	2009	10	DDH	NQ	3,299	3	7
MintoEx	Area 118	2008	32	DDH	NQ	6,998	-	32
MintoEx	Area 118	2007	23	DDH	NQ	6,437	-	26
MintoEx	Ridgetop	2009	71	DDH	NQ	7,855	3	68
MintoEx	Ridgetop	2008	45	DDH	NQ	5,786	-	45
MintoEx	Ridgetop	2007	25	DDH	NQ	3,432	-	25
MintoEx	Minto North	2009	88	DDH	NQ	11,548	17	71
MintoEx	Minto East	2010	27	DDH	(4) NTW, (23) NQ	8.892	17	10
MintoEx	Minto East	2009	3	DDH	NQ	1,080	2	1
MintoEx	Minto East	2008	1	DDH	NQ	385	0	1
MintoEx	Minto East	2007	1	DDH	NQ	380	0	1

The Area 2/118 resource estimation incorporates the majority of 2008-2009, all 2010 drilling within Area 2/118, and 23 drill holes completed by ASARCO in the 1970s.

At Area 2, MintoEx drilled at total of 6,104 m in 15 vertical and 6 angled diamond drill holes from January 22 to March 25, 2010. The 2010 drill holes range from 198 m to 384 m in length, with a median length of 286 m, and an average length of 291 m. The size of the drill core is NQ and NTW. MintoEx drilled a total of 4,162 m in 19 vertical diamond drill holes from May 11, 2008 to September 10, 2009.

A total of 21 vertical holes and 2 angled holes drilled by ASARCO in 1973 and 1974 are also included in the resource estimation. The size of the historical ASARCO drill core was not recorded but is believed to be BQ size, based on observation of core found in core storage sheds destroyed by forest fire. Drill collars are spaced at approximately 28 m to 60 m centers on a northeast striking grid. Mineralized zones, shown in Figure 10.1, undulate and dip shallowly to the northwest.

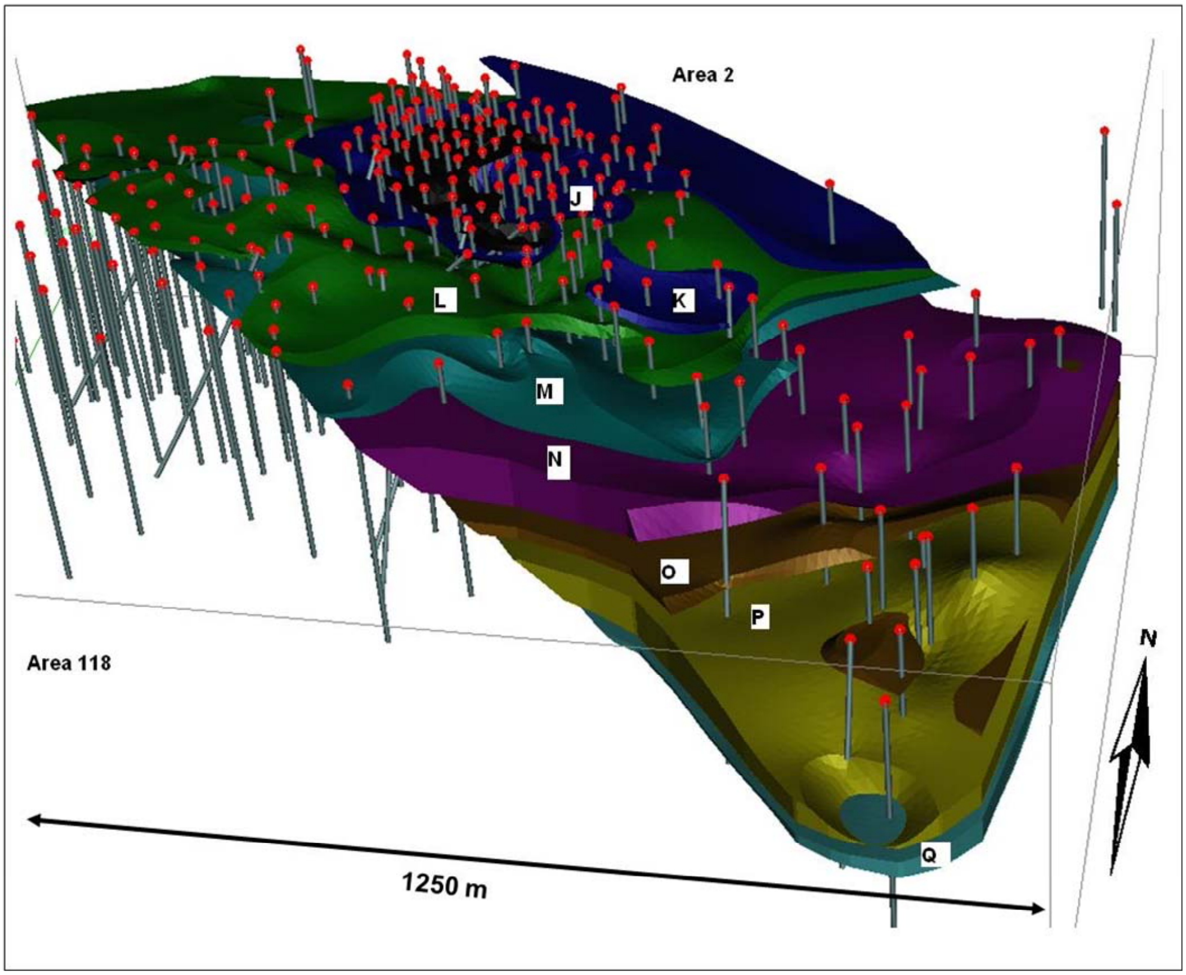


Figure 10.1: Wireframes of Mineralized Domains with Drill Holes, Area 2. A Fault Separates Area 2 from Area 118. View Northwest

At Area 118, MintoEx drilled a total of 10,297 m in 39 vertical and 3 angled diamond drill holes from May 6, 2008 to March 12, 2009. In 2010, MintoEx drilled a total of 267 m in 1 vertical diamond drill hole from February 27 to March 1, 2010. The size of the drill core is NQ2 and NQ. The median length of the 2008 to 2009 drill holes is 215 m (average 245 m); the shallowest hole was 162 m long and deepest hole was 393 m. All 43 drill holes were used in the Area 2/118 resource estimation. 6 vertical holes drilled by ASARCO in 1974 were included in the Area 118 resource estimate. ASARCO core is assumed to be BQ. Drill hole collars are spaced at approximately 40 m centers. Mineralized zones, shown in Figure 10.2, undulate and dip shallowly to the northwest.

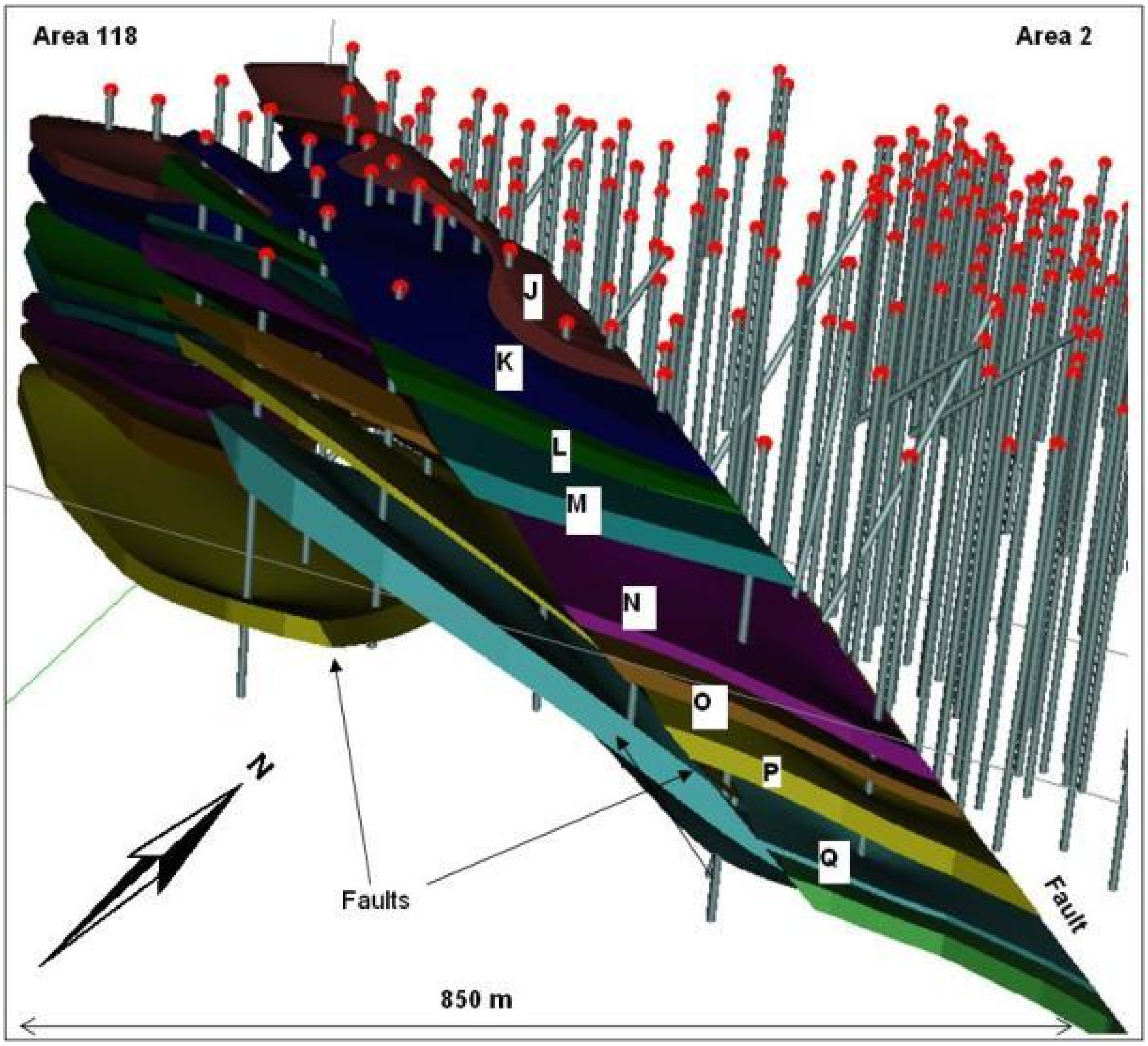


Figure 10.2: Wireframes of Mineralized Domains with Drill Holes, Area 118. Faults Separate Area 2 from Area 118, and Subdivide Area 118 into Three Sub-domains

At Ridgetop, MintoEx drilled a total of 13,641 m in 113 vertical drill holes and 3 angled diamond drill holes from June 21, 2008 to September 20, 2009. The size of the MintoEx drill core is NQ. The median length of the 2008 to 2009 Ridgetop drill holes is 111 m (average 118 m); the shallowest hole was 54 m long and the deepest hole was 322 m. One vertical hole (150 m) and three angled holes (468 m) drilled by ASARCO in 1971, and three vertical (462 m) holes and four angled holes (571.5 m) drilled in 1972 were included in the resource. Size of the ASARCO drill core is assumed to be BQ. In 1994, four vertical holes (520 m) and five angled holes (654 m) of HQ-sized core were drilled; these holes were used in the resource estimate. Drill hole collars are spaced at approximately 20 to 60 m centers. Mineralized zones are dipping moderately to the northeast (Figure 10.3).

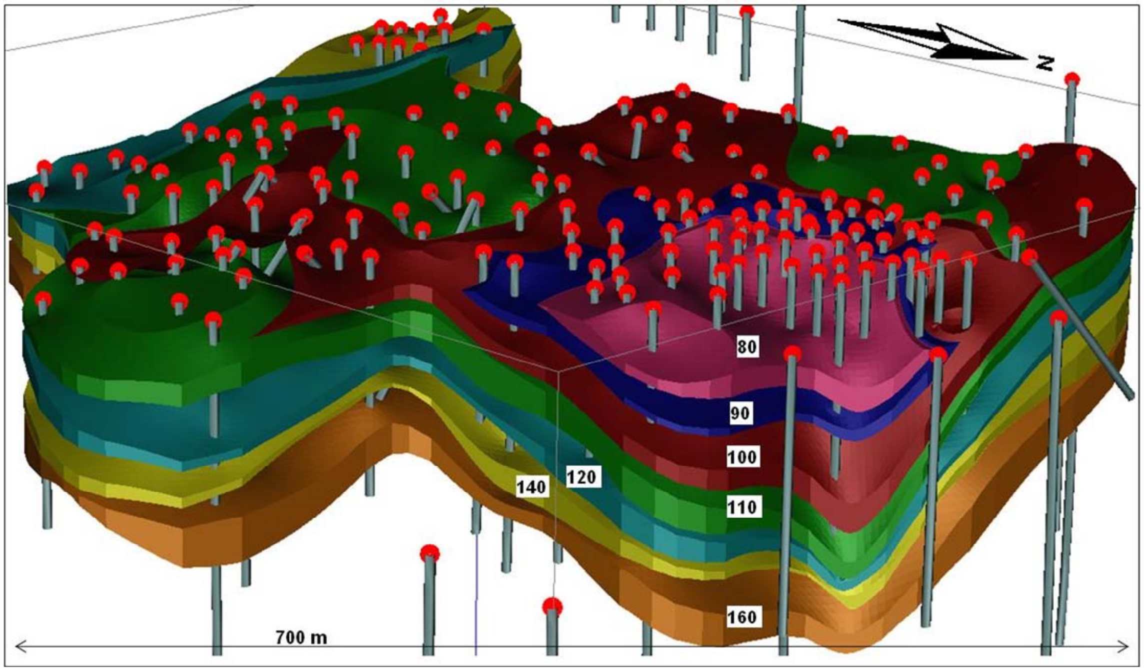


Figure 10.3: Wireframes of Labelled Mineralized Domains with Drill Holes, Ridgetop

At Minto North, MintoEx drilled a total of 11,433 m in 71 vertical and 17 angled diamond drill holes from January 27 to October 4, 2009. In total, 87 drill holes are included in the resource model; one drill hole is excluded because it is located well outside the currently defined deposit boundaries. No historical drill holes are included in the resource model. The size of the MintoEx drill core is NQ. The median length of the 2009 Minto North drill holes is 120 m (average 130 m); the shallowest hole was 57 m and the deepest hole was 342 m. Drill hole collars are spaced at approximately 15 to 20 m centers. Mineralized zones are shallowly dipping to the northwest (Figure 10.4).

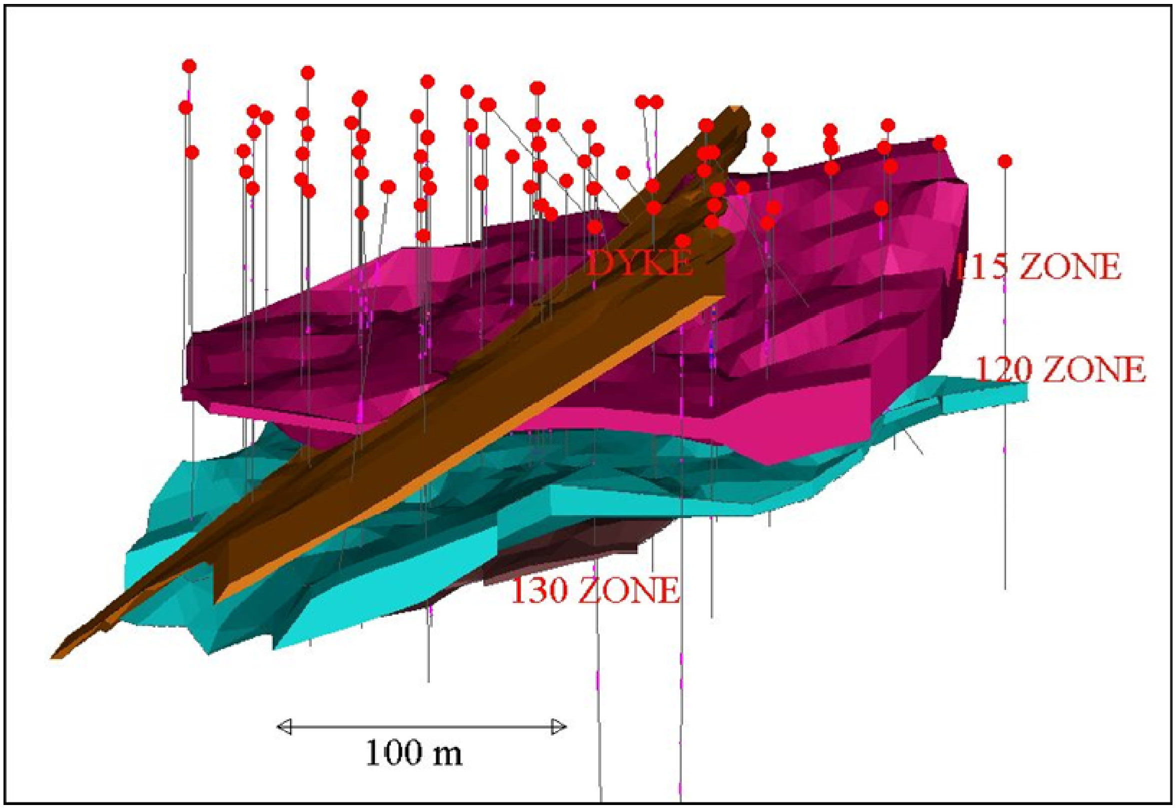


Figure 10.4: Wireframes of Mineralized Domains with Drill holes, Minto North

At Minto East, MintoEx drilled a total of 10,737 m in 13 vertical and 19 angled diamond drill holes from April 18, 2007 to August 21, 2010. In total, 32 drill holes are included in the resource model. No historical drill holes are included in the resource model. The size of the MintoEx drill core is NQ with the exception of 4 drill holes in NTW. The median length of the Minto East drill holes is 332 m (average 336 m); the shallowest hole was 179 m and the deepest hole was 408 m. Drill hole collars are spaced at approximately 40 m centers. Mineralized zones are shallowly dipping to the northwest (Figure 10.5).

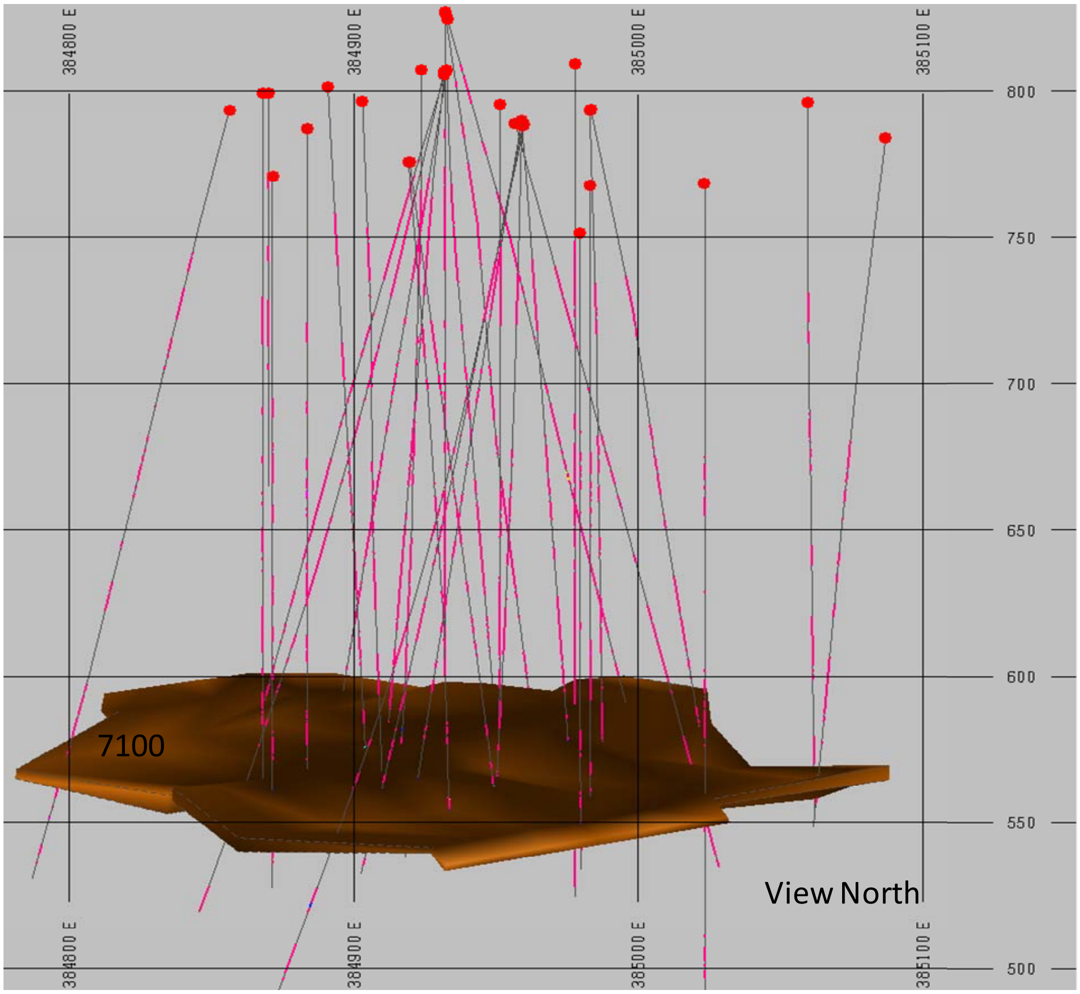


Figure 10.5: Wireframe of Mineralized Domain with Drill holes, Minto East

Prior to 2008, all drilling for MintoEx was completed using the imperial system, and footages were converted to metres by MintoEx personnel who logged and recorded all data in metres. Since 2008, drilling for MintoEx was completed using the metric system. Drill hole collar locations were initially located using a differential GPS unit, and more precise location coordinates were surveyed after completion of drilling by the Minto Mine survey team using a Trimble R8 GPS unit.

Acid tests were performed at the end of each hole or at various depths down the hole in the winter of 2008 and in some occasions during 2010 drilling. Minimal deviations were typical in all holes which were predominantly drilled at a vertical inclination.

Since the spring of 2008, down hole surveys were primarily performed using a FLEXIT down hole survey tool. Although local magnetite concentrations sometimes prevented measurement of azimuth deviations, the tool overall provided realistic readings showing minor deviation in azimuth and dip. In 2010, a maxibor tool was utilized in 22 drill holes in areas known to be highly magnetic. Mineralized intervals measured in the vertical drill holes are believed to represent very close to the true widths of mineralized layers within the deposit because of the sub-horizontal attitude of the mineralized zones.

The core was transported from the drill rig to the logging facility by the drilling contractor, where MintoEx personnel logged it for geological, sampling, and geotechnical purposes. Geological data including lithology, structure, alteration, and mineralization was recorded for all drill holes.

All drill core was photographed for easy reference when constructing geological models for resource estimation.

Geotechnical data was collected on all drill holes in 2008 to 2010, including RQD, core recovery, fracture density and orientation, hardness and joint data. Recovery was typically very good to excellent. Orientation data for individual joints and structures was not measured for most holes as they were drilled vertically, but the approximate alpha angle was recorded. Orientation data for individual joints and structures were recorded in 10 oriented geotechnical drills totalling 2391 m, including 3 holes at Area 118 (981 m), 3 holes at Ridgetop (525 m), 2 holes in the DEF area of the Minto Main deposit (591 m), and 2 holes at Minto North (294 m).

Magnetic susceptibility data was also collected for each drill hole in 2008 to 2010. No direct correlation between the degree of magnetic susceptibility and grades of mineralization can be made, but a marked increase in the magnetic susceptibility is noted in mineralized intervals. This is not surprising since increased magnetite content is frequently logged in all mineralized horizons. However, magnetite is often more pervasive than sulphide mineralization and magnetite concentrations are not directly proportional to copper grade. Elevated levels of magnetite are found within the mineralized horizons, but where sulphide mineralization has a sharp transition from foliated to unfoliated domains, magnetite alteration can persist, although at much lower concentrations into unmineralized domains. In some instances, the presence of hematite or hematite/magnetite combinations in unmineralized domains corresponds to brittle structures, suggesting some remobilization of iron after mineralization and is thought to be due largely to supergene processes. In such case, the magnetic susceptibility readings are muted somewhat.

Sample intervals were marked on the core and a cut line was drawn with a china marker for the diamond saw cutter to follow. Half of the core was placed in a sample bag and the other half was returned to the core box. Sample intervals were nominally taken at 1.5 m in the mineralized zones, with a minimum of 2 shoulder samples taken into the waste contact. Waste material between successively stacked mineralized zones was sampled at 3 m intervals to avoid gaps in assay data.

Sample intervals from the vertical holes approximate the true width of the mineralized zones, whereas FLEXIT or Maxibor down hole survey data was used to determine the true width of mineralized zones in angled drill holes. Sampling results are described in detail in subsequent sections.

Bulk density measurements were taken from nearly all holes drilled from 2005 through 2010 in both mineralized and waste material. Measurements were taken at approximately every 1 to 3 m intervals in ore, corresponding to 1 to 3 measurements per run in strongly mineralized material, 1 every 5 m in poorly mineralized material, and at least 1 measurement every 20 to 30 m in waste.

Pieces of core were weighed both in air and in water using an Ohaus triple beam balance. Spot checks on the field data were undertaken internally by MintoEx, where 159 samples from 66 drill holes were analyzed. Measurements were recorded on a triple beam scale on the same piece of core that was originally measured.

Bulk density data obtained prior to 2005 were not used in the resource estimations because the data was constructed by correlating bulk density to copper grade based upon too few actual measurements and because the core upon which this method was constructed was destroyed in forest fires and the methodology could not be audited.

For additional information regarding drilling and bulk density measurements obtained prior to 2008 for the Minto, Area 2, Area 118, and Ridgetop Deposits, please refer to Section 7 in "*Technical Report (43-10 1) for the Minto Project*" by Hatch (August 2006) and to Section 11 in "*Area 2 Pre-feasibility Study Minto Mine, Yukon*" (November 2007) and to Sections 11 and 12 in "*Technical Report Minto Mine, Yukon*" prepared by SRK Consulting (Canada) Inc. (June 2008) found on the sedar.com website.

11 Sampling Method and Approach

11.1 1973 to 2001

The sampling programs in place for the historical samples were implemented by geological employees of large Canadian, American and International mining companies. No reports or data detailing the sampling methods, analyses methods, quality control measures or security procedures used by the previous lessee companies were available to the authors for review and verification during the time of this report preparation.

Based on the information available, most of the samples sent for analysis were obtained by splitting the core using a mechanical wheel core splitter (in contrast to a diamond saw in 2005-2010). In the case of two holes drilled in 1993 for metallurgical grinding testing, the entire core through the mineralized interval was utilized to improve the validity and reliability of the metallurgical tests and hence no assay data are available.

In the early drilling, sample intervals were consistently 1.5 m or 3.0 m long, except in areas of complicated geology or contacts. The 2001 drill program utilized a 1.5 m sample interval, with smaller samples taken at contacts or mineralization variations. The mineralization is quite obvious and contacts between mineralized and non-mineralized material are generally sharp.

In the deposit, the intensity of sulphide mineralization is generally consistent and evenly distributed, so no inadvertent biasing of the results, depending on what part of the core was sampled, is expected.

11.2 2005 to 2006 (MintoEx)

The mineralized intervals intersected in core have been sampled in lengths ranging from 0.3 m to 3.0 m and averaging 1.0 m to 1.5 m. The sampling intervals were typically 1.5 m in mineralized material and 3.0 m in longer waste intervals within the mineralized zones. Two shoulder samples were taken in waste at both the upper and lower contacts, consisting of a 1.5 m sample and a 1.0 m sample. Samples did not cross geological contacts. This approach is appropriate for this style of mineralization and the objectives of the program.

MintoEx analyzed 1,391 sawn core samples in 2005 and 1,354 in 2006. The samples were tagged and then split in half using a rock saw on site. One half of the core was put into sample bags and then packaged into rice bags with security zip seals and sent to Vancouver for assaying. Manitoulin Transport was used to send the samples by ground in 2005 and Air North was commissioned in 2006 to air freight the samples. The remaining core was returned to the boxes and remains on site as a record of the hole.

In 2005 and 2006, the core was photographed after the sample tags were stapled to the boxes at the down hole end of each sample. Sample tags for standards were also stapled to the box in the order they were taken.

11.3 2007 (MintoEx)

The mineralized intervals in core were sampled in lengths ranging from 0.24 m to 3.49 m and averaging 1.33 m with a median of 1.5 m from 7,450 sawn core samples. Sampling intervals were typically 1.5 m in mineralized material and 3.0 m in longer waste intervals between mineralized zones. Drill core assay samples were collected from all foliated granodiorite horizons and, typically, sampling extended into the surrounding massive, unfoliated and unmineralized rock for at least 3.0 metres. Individual samples do not cross the geological boundary between foliated and unfoliated rock which is generally a sharp contact. The sampling methodology is appropriate for this style of mineralization.

In 2007, MintoEx cut 7,450 core samples by diamond saw, located on site adjacent to the exploration camp. One half of the core was put into sample bags and then packaged into large rice bags with security zip seals and transported to the laboratory for assaying. From July 5 to 15, 2007, 485 samples were transported by truck to SGS Laboratories (under contract agreement) at the Minto Mine Site, Yukon for assaying for copper and silver. Lab capacity was unsuited to a large, ongoing influx of exploration samples so no further samples were submitted. The coarse rejects for the 485 samples and sawn core for all subsequent samples were sent to ALS Chemex in Terrace for processing and on to Vancouver for assaying and ICP multi-element analysis. Samples were transported initially to Whitehorse by Small’s Expediting Ltd and then to Vancouver or Terrace by bonded carrier; either Manitoulin Transport or Air North Ltd. The remaining half of the core was returned to the wooden boxes and remains on site as a record of the hole.

Drill core was photographed after the sample tags were stapled to the boxes at the down hole end of each sample. Sample tags for standards were also stapled to the box in the order they were taken.

11.4 2008 (MintoEx)

The mineralized intervals in core were sampled in lengths ranging from 0.25 m to 4.20 m and averaging 1.29 m with a median of 1.3 m from 12,538 sawn core samples. Sampling intervals were typically 1.5 m in mineralized material and 3 m in longer waste intervals between mineralized zones. Drill core assay samples were collected from all foliated granodiorite horizons and, typically, sampling extended into the surrounding massive, unfoliated and unmineralized rock for at least 3 m. Individual samples do not cross the geological boundary between foliated and unfoliated rock which is generally a sharp contact. The sampling methodology is appropriate for this style of mineralization.

In 2008, MintoEx cut 12,538 core samples by diamond saw, located on site adjacent to the exploration camp. One half of the core was put into sample bags and then packaged into large rice bags with security zip seals and transported to the laboratory for assaying. From March 8 to September 25, 2008, 6,450 samples from outside the Ridgetop area were transported by truck to SGS Laboratories (under contract agreement) at the Minto Mine Site, Yukon for assaying for copper and silver.

During mid-July, MintoEx requested quality control copper reanalysis at the SGS Lakefield, Ontario facility after a switch failure at the Minto Mine Site facility. From July 27 to September 30, 2008, 6,087 samples were sent to ALS Chemex in Terrace for processing and on to Vancouver for assaying. The samples were transported initially to Whitehorse by Small's Expediting Ltd and then to Vancouver or Terrace by Byers Transport. The remaining half of the core was returned to the wooden boxes and remains on site as a record of the hole.

Drill core was photographed after the sample tags were stapled to the boxes at the down hole end of each sample. Sample tags for standards were also stapled to the box in the order they were taken.

11.5 2009 (MintoEx)

The mineralized intervals in core were sampled in lengths ranging from 0.19 m to 4.50 m and averaging 1.47 m with a median of 1.5 m from 13,026 sawn core samples. Sampling intervals were typically 1.5 m to 2.0 m in mineralized material and 3 m in longer waste intervals between mineralized zones. Drill core assay samples were collected from all foliated granodiorite horizons and, typically, sampling extended into the surrounding massive, unfoliated and unmineralized rock for at least 3.0 metres. Individual samples do not cross the geological boundary between foliated and unfoliated rock which is generally a sharp contact. The sampling methodology is appropriate for this style of mineralization.

In 2009, MintoEx cut 13,026 core samples by diamond saw, located on site adjacent to the exploration camp. One half of the core was put into sample bags and then packaged into large rice bags with security zip seals and transported to the laboratory for assaying. From February 4 to October 29, 2009, 13,026 samples were sent to ALS Chemex in Vancouver for processing and assaying. The samples were transported initially to Whitehorse by Small's Expediting Ltd. and then to Vancouver by Byers Transport. The remaining half of the core was returned to the wooden boxes and remains on site as a record of the hole.

Drill core was photographed after the sample tags were stapled to the boxes at the down hole end of each sample. Sample tags for standards were also stapled to the box in the order they were taken.

11.6 2010 (MintoEx)

The mineralized intervals in core were sampled in lengths ranging from 0.22 m to 3.90 m and averaging 1.41 m with a median of 1.5 m from 18,739 sawn core samples. Sampling intervals were typically 1.5 m to 2.0 m in mineralized material and 3 m in longer waste intervals between mineralized zones. Drill core assay samples were collected from all foliated granodiorite horizons and, typically, sampling extended into the surrounding massive, unfoliated and unmineralized rock for at least 3.0 metres. Individual samples do not cross the geological boundary between foliated and unfoliated rock which is generally a sharp contact. The sampling methodology is appropriate for this style of mineralization.

In 2010, MintoEx cut 18,739 core samples by diamond saw, located on site adjacent to the exploration camp. One half of the core was put into sample bags and then packaged into large rice bags with security zip seals and transported to the laboratory for assaying. From January 28, 2010 to May, 5, 2010, 4,437 samples were sent to ALS Chemex in Vancouver for processing and assaying; samples were transported Whitehorse by Small's Expediting Ltd and then to Vancouver by Byers Transport. When drilling resumed after a short break in the spring, 14,302 samples were sent to ALS Chemex in Whitehorse for processing and then to ALS Chemex in Vancouver for analysis from July 3 to December 15, 2010. The samples were transported initially to Whitehorse by Small's Expediting Ltd. and then in custody of ALS Chemex to Vancouver. The remaining half of the core was returned to the wooden boxes and remains on site as a record of the hole.

Drill core was photographed after the sample tags were stapled to the boxes at the down hole end of each sample. Sample tags for standards were also stapled to the box in the order they were taken.

12 Sample Preparation, Analyses and Security

12.1 Historic Samples

ASARCO 1971 to 1974

No detailed descriptions of historical sampling methods, preparation and analyses by ASARCO were recorded, however, based on observation, 5 and 10 foot long samples were favoured. Very few ASARCO holes are used in the resource and all are near MintoEx holes, limiting the effect of the ASARCO data on the resource calculation. No usable core survives from that period. It is inevitable that company employees would be involved in sampling but the exact activities and names of these ASARCO employees are unknown. It is not known whether officers or directors of ASARCO were involved in the sample preparation, but this is considered unlikely given the minor nature of the project. Subsequent sample preparation such as crushing, pulverizing and sample splitting would have been the responsibility of the laboratory.

Chemex in Vancouver is believed to have been responsible for the 1970s analyses (Simpson, 2002). At the time, copper analyses were typically performed by digesting a 2 g sample pulverized to 100 mesh, in perchloric and nitric acid with an atomic absorption spectroscopy (AAS) finish. Modern practices use a 0.4 g 150 mesh samples and aqua regia digestion. Gold analyses in the 1970s probably used a 10 g pulp digested in aqua regia and an AAS finish. Electronic microbalances and improvements in AA analysis have combined to reduce detection limits in the past 25 years.

Some of the early samples were not analyzed for precious metals. Most samples were analyzed solely for total copper, resulting in an incomplete data set of gold and silver. Copper oxide mineralization is confined typically to the upper level of the deposit and, historically, non-sulphide copper was not universally quantified by analysis of soluble copper.

TECK 1993 to 2001

From 1993 to 2001, TECK (now part of Teck Cominco) drilled 48 diamond drill holes on the Minto property. Sample lengths vary from 0.55 m to 2.75 m, averaging 1.59 m with a median of 1.53 m. Sampling protocols and information regarding security of samples, as required in NI 43-101, were not well documented during the 1993 to 2001 drill programs. The historic samples would likely have been prepared on site from split core under the supervision of TECK and MintoEx geologists, bagged and shipped to the laboratory. As in 1974, it is assumed company employees would be involved in the sampling process but it is not known exactly who would have been involved other than the project manager, F.T. Graybeal. It is considered unlikely officers or directors of TECK or MintoEx were involved in sample preparation. Subsequent sample preparation such as crushing, pulverizing and sample splitting would have been the responsibility of the laboratory.

Northern Analytical Services of Whitehorse, Yukon conducted the analyses for copper, gold and silver. Analytical methods are not documented in the certificates of analysis for this work, but are believed to be equivalent to the methods listed on the certificates for check analysis performed by Chemex, detailed below. Non-sulphide copper was not initially quantified by analysis of soluble copper at Northern Analytical Services.

Bondar-Clegg of North Vancouver carried out the analyses of the 2001 samples. Each 0.25 gm sample was digested with HCL, HNO3, HCLO4 and HF acids with final copper determination by AAS. Gold and silver were determined by fire assay using a 30 gm sample and AAS finish.

No useable mineralized intersections of the 1994 TECK Ridgetop East drill holes remain on-site. A few stacks of 1994 core were discovered at the old location of the Minto Exploration camp site and at the Yukon Geoscience core library but the bottom of the holes containing mineralized intervals were not present. No other useable drill core from the 1993 to 2001 period remains on-site.

12.2 MintoEx Samples

MintoEx 2005 and 2006 Samples

During 2005 and 2006, drill core samples, Standard Reference Materials ("SRM") and blanks were submitted to the Vancouver Chemex laboratory for copper and gold analysis in North Vancouver, Canada. In addition, Chemex was also instructed to perform analysis on pulp duplicates injected into the sample stream at regular intervals. In 2005, all samples were processed in Vancouver. In 2006, some samples were processed at other Chemex locations. Chemex-Elko, NV, USA processed 9% of the total number of samples and Chemex-Thunder Bay, ON processed 11%. The samples submitted to Chemex were first crushed in a jaw crusher to reduce the material to greater than 70% -10 mesh (2 mm). A 100 to 250 g subsample was then split and pulverized to better than 85% passing -75 µm.

Copper was determined through a four acid digestion method (HF, HNO3, HCLO4 digestion and HCL leach) with final copper determination by AAS. Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination.

Gold was determined by one assay-tonne fire assay analysis. During 2005, all sample analysis was completed by gravimetric finish. During 2006, the first 17% (1,955) of the sample analysis was completed by gravimetric finish. For the remaining samples (9,182), the gold analysis was determined using AAS method. Silver was analyzed using aqua regia digestion and AAS finish.

MintoEx 2007 Samples

The 2007 drill core samples, blanks, SRMs and duplicates were submitted to the Vancouver Chemex laboratory for copper and gold analysis in North Vancouver, Canada. Some samples were processed at other locations. SGS Laboratories under agreement with MintoEx processed 485 samples (6% of the total number of samples); assays were all performed at the Vancouver Chemex Lab. Sample preparations were performed at Chemex at Elko, NV, USA, 4% of the total number of samples, Chemex at Reno, NV, USA 10%, and Chemex at Terrace, Canada50%.

The samples submitted to Chemex were first crushed in a jaw crusher to reduce the material to greater than 70% -10 mesh (2 mm). A 100 to 250 g subsample was then split and pulverized to better than 85% passing -75 μm.

Copper was determined by the four acid digestion method (HF, HNO3, HCLO4 digestion and HCL-leach) with final copper determination by AAS. Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination. Gold was analyzed by one assay-tonne fire assay followed by AAS. Silver was analyzed using aqua regia digestion and AAS finish.

MintoEx 2008 Samples

Two laboratories were used in 2008. Drill core samples, blanks, SRMs and duplicates were submitted to SGS Laboratories under agreement with MintoEx, and to the Vancouver Chemex laboratory for copper and gold analysis in North Vancouver, BC after processing at the sample preparation facility in Terrace, BC. SGS Laboratories under agreement with MintoEx processed 61% of the total number of samples from areas outside of Ridgetop. The remaining 39% of the samples were analysed at the Vancouver Chemex Lab.

The samples submitted to SGS were first crushed in a jaw crusher to reduce the material to greater than 85% -10 mesh (2 mm). A 250 g subsample was then split and pulverized to better than 90% passing -75 μm. The pulp was split with one part analysed for copper and silver at the SGS facility at the Minto site and one part analysed for gold and non-sulphide copper at SGS Red Lake, ON operation. During mid-July, silver analyses were performed by SGS at Lakefield, ON and Don Mills, ON after a switch failure in SGS Minto ICP-AAS equipment. Copper reanalysis due to SRM failures were done by SGS at Lakefield and Don Mills in Ontario.

Copper was determined by aqua regia digestion method with final copper determination by atomic absorption spectroscopy ("AAS"). Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination. Samples were assayed for gold using a fire assay procedure on a thirty grams sub-sample with atomic absorption spectroscopy finish. Silver was analyzed using aqua regia digestion and AAS finish.

The samples submitted to Chemex from July 27 to August 19 were first crushed in a jaw crusher to reduce the material to greater than 85% -10 mesh (2 mm). A 250 g subsample was then split and pulverized to better than 90% passing -75 µm. The sample turnaround time increased to nearly 7 weeks after implementing the finer crush, so subsequent samples were first crushed in a jaw crusher to reduce the material to greater than 70% -10 mesh (2 mm) with a 250 g subsample split and pulverized to better than 85% passing -75 µm.

At Chemex, copper was determined by the four acid digestion method (HF, HNO3, HCLO4 digestion and HCL-leach) with final copper determination by atomic absorption spectroscopy ("AAS"). Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination. Gold was determined by one assay-tonne fire assay analysis followed by AAS. Silver was analyzed using aqua regia digestion and AAS finish.

MintoEx 2009 Samples

The 2009 drill core samples, blanks and SRMs were submitted to the Vancouver Chemex laboratory for copper and gold analysis in North Vancouver. In addition, Chemex was also instructed to perform analysis on pulp and coarse reject duplicates injected into the sample stream at regular intervals.

The samples submitted to Chemex were first crushed in a jaw crusher to reduce the material to greater than 70% -10 mesh (2 mm) with a 250 g subsample split and pulverized to better than 85% passing -75 µm.

Copper was determined by aqua regia digestion method with final copper determination by atomic absorption spectroscopy ("AAS"). Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination. Gold was determined using a fire assay procedure on a thirty grams sub-sample with atomic absorption spectroscopy finish. Silver was analyzed using aqua regia digestion and AAS finish.

MintoEx 2010 Samples

The 2010 drill core samples, blanks and SRMs were analyzed at the Vancouver Chemex laboratory for copper and gold analysis in North Vancouver. In addition, Chemex was also instructed to perform analysis on pulp and coarse reject duplicates injected into the sample stream at regular intervals. After August 2010, the pulp and coarse reject duplicates were returned to the MintoEx office in Vancouver, where they are transferred to fresh Kraft paper bags, assigned new sample numbers and resubmitted to Chemex as "blind duplicates".

The samples submitted to Chemex were first crushed in a jaw crusher to reduce the material to greater than 70% -10 mesh (2 mm) with a 250 g subsample split and pulverized to better than 85% passing -75 µm.

Copper was determined by aqua regia digestion method with final copper determination by atomic absorption spectroscopy ("AAS"). Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination. When native copper was logged in drill core, a screen metallic copper method was used. Gold was determined using a fire assay procedure on a thirty grams sub-sample with atomic absorption spectroscopy finish. Silver was analyzed using aqua regia digestion and AAS finish.

12.3 Quality Assurance and Quality Control Programs

Quality control measures are typically set in place to ensure the reliability of exploration data. Exploration work by MintoEx was conducted using a quality assurance and quality control program generally meeting industry best practices. All aspects of the exploration data acquisition and management including surveying, drilling, sampling, sample security, and assaying and database management were conducted under the supervision of appropriately qualified geologists and include written field procedures and verifications.

Analytical control measures typically involve internal and external laboratory control measures to monitor the precision and accuracy of the sampling, preparation and assaying. Insertion of certified Standard Reference Material ("SRM") and blank material ("blanks") monitors the reliability of assaying results and is also important to prevent sample mix-up and monitor potential contamination of samples.

Assaying protocols typically involve regular duplicate and replicate assays to monitor the reliability of assaying results throughout the sampling and assaying process. Umpire assaying is typically performed as an additional reliability test of assaying results by re-assaying a set number of sample rejects and pulps at a secondary laboratory.

ALS-Chemex and SGS implemented internal laboratory measures consisting of inserting quality control samples (blanks and certified reference materials and duplicate pulp) within each batch of samples submitted for assaying.

Quality control procedures used during the 1971 to 2001 drill programs are not known, with the exception of 10 samples submitted for umpire analysis in 1994. The 2001 sample shipments were accompanied by 4 types of quality control samples, namely: a blank (granodiorite from the site), an ASARCO coarse standard, prepared pulp samples and duplicate splits (coarse ground rejects and the pulverized rejects).

MintoEx inserted one each of an SRM, blank, pulp reject duplicate and coarse reject duplicate (for Chemex only) with every 16 sawn core samples. Umpire assaying of pulps at a secondary laboratory was conducted periodically, typically involving analysis of 0.5% or more of the sawn core samples. The analytical quality control data produced by MintoEx from 2006 to the beginning of December 2010 are summarized in Table 12.1.

Quality control data analysis was done on all 2010 data and found to be acceptable, but is not included in this report. Refer to the 2009 Phase IV PFS Technical Report for older data.

Table 12.1: Quality Control Data Produced by MintoEx in 2006 through 2009

<table>
<tr><th colspan="2"></th><th colspan="2">2006</th><th colspan="2">2007</th><th colspan="2">2008</th><th colspan="2">2009</th><th colspan="2">2010</th></tr>
<tr><td colspan="2">Total Samples Collected</td><td colspan="2">13,121</td><td colspan="2">13,552</td><td colspan="2">15,119</td><td colspan="2">13,056</td><td colspan="2">18,762</td></tr>
<tr><td colspan="2" rowspan="13">SRM Used</td><td>CGS-5</td><td>36</td><td rowspan="3">SRM-95</td><td rowspan="3">4</td><td rowspan="3">SRM-95</td><td rowspan="3">13</td><td rowspan="3">SRM-95</td><td rowspan="3">3</td><td rowspan="2">SRM-1</td><td rowspan="2">84</td></tr>
<tr><td>Cu-115</td><td>47</td></tr>
<tr><td>Cu-116</td><td>48</td><td rowspan="2">SRM-2</td><td rowspan="2">58</td></tr>
<tr><td>CGS-10</td><td>116</td><td>CGS-10</td><td>120</td><td>CGS-10</td><td>24</td><td>SRM-1</td><td>27</td></tr>
<tr><td>CGS-7</td><td>103</td><td>CGS-7</td><td>137</td><td>CM-3</td><td>27</td><td>SRM-2</td><td>27</td><td rowspan="2">SRM-3</td><td rowspan="2">33</td></tr>
<tr><td>CGS-12</td><td>54</td><td>CGS-12</td><td>139</td><td>CGS-12</td><td>31</td><td>SRM-3</td><td>24</td></tr>
<tr><td>GSP-5</td><td>19</td><td rowspan="2">CGS-8</td><td rowspan="2">17</td><td rowspan="2">CGS-17</td><td rowspan="2">56</td><td rowspan="2">CGS-17</td><td rowspan="2">12</td><td>CGS-22</td><td>152</td></tr>
<tr><td>GS-2A</td><td>17</td><td>CGS-23</td><td>372</td></tr>
<tr><td>CM-1</td><td>6</td><td>CM-2</td><td>8</td><td>CM-2</td><td>99</td><td>CM-2</td><td>117</td><td>CGS-24</td><td>126</td></tr>
<tr><td>CGS-9</td><td>109</td><td>CGS-9</td><td>51</td><td>CGS-18</td><td>120</td><td>CGS-18</td><td>190</td><td>CM-5</td><td>150</td></tr>
<tr><td>CGS-11</td><td>52</td><td>CGS-11</td><td>175</td><td>CGS-11</td><td>156</td><td>CGS-11</td><td>123</td><td>CM-8</td><td>100</td></tr>
<tr><td>Cu-132</td><td>50</td><td>CGS-13</td><td>17</td><td rowspan="2">CGS-15</td><td rowspan="2">177</td><td rowspan="2">CGS-15</td><td rowspan="2">191</td><td rowspan="2">CGS-15</td><td rowspan="2">100</td></tr>
<tr><td>Cu-128</td><td>40</td><td>Cu-128</td><td>15</td></tr>
<tr><td colspan="2">Total SRM</td><td colspan="2">697</td><td colspan="2">682</td><td colspan="2">703</td><td colspan="2">714</td><td colspan="2">1175</td></tr>
<tr><td colspan="2">Blanks</td><td colspan="2">595</td><td colspan="2">674</td><td colspan="2">685</td><td colspan="2">698</td><td colspan="2">1186</td></tr>
<tr><td rowspan="2">Paired Data</td><td>Coarse Reject Duplicate</td><td colspan="2">404</td><td colspan="2">556</td><td colspan="2">590</td><td colspan="2">590</td><td colspan="2">1137</td></tr>
<tr><td>Pulp Reject Replicate</td><td colspan="2">597</td><td colspan="2">702</td><td colspan="2">568</td><td colspan="2">568</td><td colspan="2">1168</td></tr>
<tr><td colspan="2">Total QC samples</td><td colspan="2">2293</td><td colspan="2">2614</td><td colspan="2">2582</td><td colspan="2">2570</td><td colspan="2">4666</td></tr>
<tr><td colspan="2">Frequency (percent)</td><td colspan="2">17</td><td colspan="2">19</td><td colspan="2">17</td><td colspan="2">20</td><td colspan="2">25</td></tr>
<tr><td colspan="2">Umpire checks (percent)</td><td colspan="2">2</td><td colspan="2">1</td><td colspan="2">0.5</td><td colspan="2">0.5</td><td colspan="2">1</td></tr>
</table>

Summary of Quality Assurance and Quality Control Programs in 2006 and 2007

Of the 1,269 blank samples analyzed in 2006 and 2007, eleven returned elevated gold and copper results. Internal review by MintoEx indicated six of these erroneous values may have been the result of sample switches. No systematic or long term contamination during sample preparation is evident.

Varying grades of copper and gold SRM were purchased from CDN Resource Laboratories of Delta, BC ("CDN") and WCM Sales Ltd of Burnaby, BC ("WCM"). In 2007, a copper-only SRM with mean value of 2.59% Cu was purchased from Analytical Solutions Ltd. of Toronto, ON. The SRM were submitted in sequence with the sawn core samples. A total of 1,379 SRM were analyzed. The performance of copper and gold standards was acceptable overall. Performance of SRM for gold improved part way through 2006 when AAS finish was used instead of gravimetric finish after fire assay.

Analyses of the pulp reject and coarse reject laboratory duplicates indicate the 2006 and 2007 sample preparation protocols were excellent for copper analysis and acceptable for gold analysis. To optimize the reproducibility of gold analysis, MintoEx considered increasing the amount of material passing fine meshes during sample prep in 2007. However, no adjustments were made to the sample preparation protocol as any change to the standard preparation procedure was anticipated to increase turnaround time for results to lag times that would have been unacceptable.

In 2006, five percent of the Area 2 samples (approx 2% of all 2006 sawn core samples) were submitted back to Chemex for blind analysis. Results in all grade ranges were reproducible. For copper, more than 95% had absolute differences of less than 10%; for gold 79% of the pairs were within 15% of each other. One gold outlier pair was removed from the data set.

Umpire pulp check samples representing 1% of the sawn core samples submitted in 2007 were sent to Inspectorate Laboratories in Richmond, Canada. Inspectorate analyzed the check samples using the same analytical procedure as Chemex. Overall, the gold and copper values exhibit unbiased scatter about the mean. No outliers were removed from the dataset. The target for pulp samples analyzed at different labs should have a relative difference not exceeding 15% at the 90th percentile. Copper results for the check samples in 2007 had a relative difference of 12% for the 90% of the population. Gold results for the check samples in 2007 had a relative difference of 15% for 65% of the population. The level of precision is excellent for copper. The level of precision for gold is acceptable but warrants improvement. However, results are shown to be reproducible.

For additional information regarding performance of quality control samples in 2006 and 2007 please refer to "*Technical Report (NI-43101) for the Minto Project*", Hatch, August 2006 and to "*Area 2 Pre-feasibility Study Minto Mine, Yukon*", SRK Consulting (Canada) Inc., November 2007 and to "*Minto Mine Technical Report*" SRK Consulting (Canada) Inc., June 2008.

Performance of Blanks in 2008, 2009 and 2010

MintoEx personnel inserted one field blank sample into the sample stream for every 16 drill core samples submitted for analysis. The blank sample was inserted to ensure sample preparation procedures did not introduce any contamination of gold or copper to the sawn drill core samples. The field blanks consisted of pieces of local, barren granodiorite, void of any gold or copper values. A total of 685 blanks were submitted with the sawn core samples from the Minto, Area 2/118 and Ridgetop 2008 drilling campaign, In addition, 698 blanks were submitted with sawn core samples from the Minto North, Area 2/118, Copper Keel and Ridgetop 2009 drilling campaign. In 2009, 1186 blanks were submitted with sawn core samples from drilling campaigns covering Minto East, Minto Main, Copper Keel, Area 2/118, Wildfire and Inferno. Blanks performed very well, showing only very minor, sporadic contamination during sample preparation. The results indicate adequate control procedures during the laboratory’s preparation stages in the assaying process.

Performance of SRM in 2008, 2009 and 2010

Standard reference material (SRM) control samples provide a means to monitor the precision and accuracy of the laboratory assay deliveries. SRMs of varying grades for copper and gold were purchased from CDN of Delta, BC and Analytical Solutions Ltd. of Toronto, ON in 2008 and 2009. Three custom SRMs of varying grades for copper and gold were created from Area 2 coarse reject materials in 2009. The custom SRMs were certified for copper, gold and silver by Dr. Barry Smee of Smee and Associates Consulting Ltd. of North Vancouver, BC. Details of the results from the SRM assays for 2010 are available but not provided in this report. Refer to the 2009 Phase IV PFS Technical Report for older data.

MintoEx personnel inserted one SRM sample within every group of 20 samples. MintoEx considered a copper or gold SRM sample to have failed if a single value exceeded three standard deviations or if more than two consecutive standards fell outside of the two standard deviation limit. When a sample failed, MintoEx reviewed the data and if a re-assay was warranted, the assay laboratory was contacted and instructed to re-assay the failed sample batch. The laboratory was instructed to review the samples for sufficient material for re-analysis. If an SRM had insufficient material left in the sample bag, then the laboratory was supplied with a new standard before re-assaying of the batch began. Some re-assayed samples and internal lab investigations requested by MintoEx are outstanding at the time of this report. For silver SRM, any values outside of the three standard deviations limit or periods of bias were reported to the lab. Re-assays were not ordered unless the SRM also failed for copper or gold.

In 2009 and 2010, the purchased SRM samples typically performed well for gold analysis. The results are distributed about the mean with periods of bias above and below the mean. The bias is within acceptable limits.

In summary, performance of the SRM samples is acceptable. For copper and gold, most of the charts for each of the SRM show good distribution about the mean with little or no bias. Periods of some bias are evident on some of the charts but all are within acceptable limits. For gold, all SRM assays generally quite closely follow the mean and, as with copper, there is little or no bias.

Performance of Pulp Reject and Coarse Reject Duplicates in 2008, 2009 and 2010

Within every batch of 20 samples, a pulp reject and a coarse reject (for Chemex only) samples were selected for reanalysis by the geologist logging the borehole to test whether lab methods were sufficient to homogenize material for reproducible analysis. Copper and gold results were shown to be reasonably reproducible from pulp and coarse reject duplicates, using current sample preparation protocols. Values are acceptable for resource estimation purposes although the gold in the duplicates is elevated.

During the second half of 2010, the pulp materials for pulp reject and coarse reject samples were not analyzed in sequence with the parent samples. Instead, the samples were placed in fresh envelopes, given new sample numbers and SRM were inserted every 20 samples. Copper and gold results were again shown to be reasonably reproducible compared to the parent materials, although gold in the duplicates was slightly elevated.

A graphical analysis of duplicate quality control data was undertaken but is not provided in this report.

Performance of Umpire Analyses in 2008, 2009 and 2010

Umpire assaying was done to further check reliability of assay results by re-assaying a set number of sample pulps at a secondary laboratory. The pulps were selected across all grade ranges and repackaged into newly numbered pulp bags with SRM inserted every 20 samples. The target for pulp samples analyzed at different labs was a relative difference not exceeding 20% at the 80th percentile.

Generally, the copper and gold values exhibit unbiased scatter about the mean on Q-Q plots. In addition, the target relative differences were met for copper, gold and silver from 2008 to date in 2010 and to a lesser extent for gold and silver in 2008 and 2009. This level of precision is excellent for copper during the period and improvement was seen for gold and silver in 2010. In short, the results were shown to be sufficiently reproducible for resource estimates.

13 Data Verification

13.1 Verification by MintoEx

1973 to 2001

Independent data verification consisted of drilling by MintoEx, 2005 through 2007, in the Minto Deposit. No confirmation drilling was undertaken in the Area 118 and Ridgetop East. At Ridgetop East, however, two 2007 drill holes were drilled within 30 m of a historic hole, five vertical 2008 drill holes were drilled along the trace of two historic holes and one 2009 hole was drilled within 30 m of a historic hole. At Area 118, three 2008 drill holes were drilled within 40 m of two historic holes. No additional data verification was carried out on historic work. The historic work on the property has been carried out by reputable companies and there does not appear to be any reason to question the validity of the information. Core from the early drilling programs is not useable since both the Falconbridge and ASARCO core sheds have either collapsed and/or burned during regional forest fires, i.e. much of the old core is now in piles on the ground. The core boxes appear to have been labelled by felt pen, rather than metal or plastic tags and the labels on core boxes that remain intact are not legible.

2005 and 2006

Of the 79 drill holes in the 2006 Area 2 database, eleven collars (13%) were selected at random in the area of the resource estimation boundaries and were checked by a handheld Garmin GPS. Table 13.1 compares the results of the collar locations as documented by SRK and Sherwood Copper. MintoEx sighted the drill hole collars by differential GPS, which were later surveyed by the Minto Mine Survey team. The recorded values show good agreement and differences lie within the error of the handheld GPS.

Table 13.1: Comparison of Selected Drill Hole Collars by SRK and MintoEx

Hole ID	Collars – SRK Handheld GPS				Collars – Minto Mine Survey		
	Easting	Northing	Elevation	Accuracy	Easting	Northing	Elevation
06SW068	384948	6944463	860	7	384949	6944461	854
06SW095	384914	6944522	858	7	384917	6944523	851
06SW114	384975	6944503	851	5	384979	6944499	844
06SW115	384854	6944467	872	3	384855	6944465	865
06SW116	384878	6944521	864	6	384880	6944519	857
06SW122	384938	6944379	870	3	384940	6944378	861
06SW133	385037	6944601	829	3	385039	6944600	821
06SW151	384980	6944622	834	3	384981	6944621	825
06SW153	384918	6944603	845	5	384919	6944600	835
06SW168	385081	6944561	827	3	385083	6944558	818

2008

In December 2008, MintoEx conducted a review of the drilling data from Area 2/118 and Ridgetop deposits. A total of 10% of the values in the database were checked against primary sources including the borehole collar surveys against survey records, lithology and mineralization data against core logs and assays for copper and gold against signed certificates of analysis. No significant errors were found.

13.2 Verification by Kirkham Geosystems

In November of 2009, Kirkham Geosystems manually compared the Minto North Deposit database assays against original assay certificates. A total of 15% of the values were checked and no errors or omissions were found. In addition, a spreadsheet check was run against the Area 2, Area 118 and Ridgetop database.

13.3 Verification by SRK

Site Visits

In 2007, In accordance with NI 43-101 guidelines, MintoEx commissioned SRK to provide an independent verification of exploration data for Area 2. Data verification consisted of a site visit, examination of drill hole collars, examination of selected drill core and a check of the assay database against original laboratory certificates. Andrew Ham of SRK visited the Minto property between the 24th and 26th of January, 2007. Dr. Ham personally inspected drill core storage facilities, drill collars and selected drill core from mineralized zones within the Area 2 resource. In addition, he personally checked collar coordinates in eleven drill holes with a handheld Garmin GPS (see Table 13.1).

In 2009, Wayne Barnett visited the Minto property between the 4th and 6th of March. Dr. Barnett personally inspected the drill core logging and storage facilities and a drill site. Mineralized and non-mineralized drill core was reviewed and the geological logging procedure was discussed with the core loggers. Sample bags were inspected for tags and the sampling tagging process was reviewed.

Verification from Electronic Lab Files

SRK compared electronic lab files from 2008 and 2009 drill campaigns in 2009 as described in the SRK Technical Report, December 2009. The electronic lab files for 2009 and 2010 were sent directly to SRK by Chemex and SGS Labs. Approximately, 90% of the Cu and Au assays were checked. The assays were found accurately compiled, i.e., current assay database is an accurate reflection of Cu and Au assay grades generated by the labs. Ag assays were spot checked and were not extensively verified. No problems were found.

Comparison of Assays from Historical and New Drill Holes

All assays older than 2006 in Area 2/118 and 2007 in the Ridgetop area have been designated as historical (see Table 13.1). The comparison was carried out on 3.0 m composite Cu assay grades within mineralized domains. To compare the data, a nearest neighbour block model was created. Only the blocks estimated from both datasets within a maximum distance of 30 m from the nearest sample were compared. Figure 13.1 show Q-Q plots of the block estimates from the historical and the MintoEx data. Overall, the historical data compare well with the new data, indicating no bias between the two data sets. Based on the results, the historical data have been included in the resource estimates.

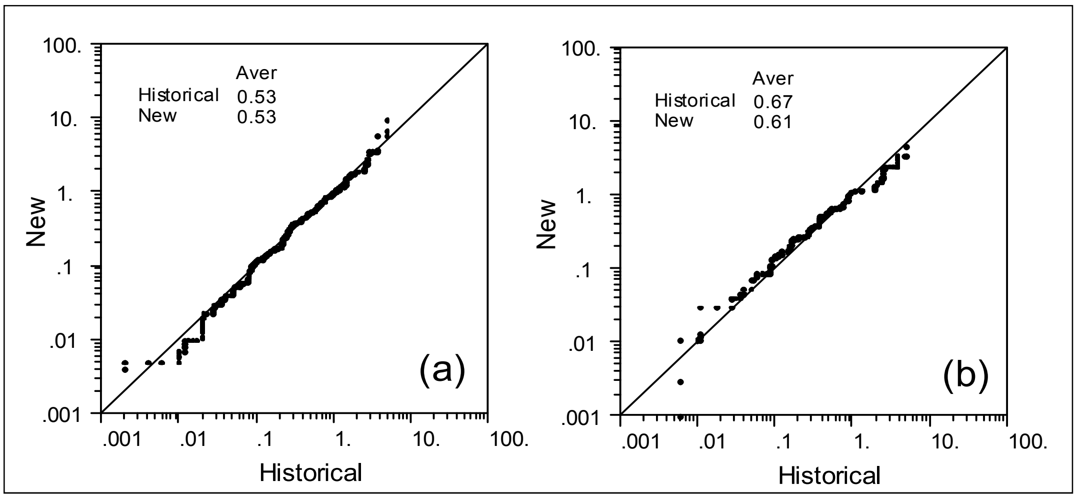


Figure 13.1: Comparison of historical and new data in: (a) Area 2/118 and (b) Ridgetop

14 Adjacent Properties

No references to any adjacent properties, other than general regional geology comments, are used in this report. The mineral resource estimation, mineral reserve estimation and exploration targets described in this report are based solely on work done on the Minto Property and are not influenced in any way by any potential mineralization on adjacent properties.

15 Mineral Processing and Metallurgical Testing

15.1 Introduction

Metallurgy testing by G&T Metallurgical Services LTD ("G&T") was performed on three potential new zones at the Minto mine site. The zones were Copper Keel, Minto East and Wildfire.

The main objectives of the test program were;

- Determine the material content and fragmentation properties of the three deposits;
- Investigate ore hardness properties for the composites;
- Determine bulk density distribution on a select group of core samples;
- Investigate the flotation response for samples using open circuit and locked cycle testing; and
- Determine the concentration of deleterious minor elements in the final copper concentrates.

Findings from the test work program are summarised in the proceeding section and supporting data found in the G&T 2010 report "Preliminary Metallurgical Testing Wildfire, Copper Keel, & Minto East Zones; Minto Mine; KM2751".

Due to their stage of development both the Copper Keel and Wildfire zones have not been included in the most recent mine plan. The test work results have been reported however the two zones have not been considered when evaluating the process plant design.

In addition to Minto East the latest mine plan includes material from Minto Main, Minto North, Minto South, Ridgetop East, Area 2, and Area 118. Metallurgy test work results for these deposits can be found in the SRK Consulting ("SRK"), 2009 technical report entitled "Minto Phase IV Pre-Feasibility Technical Report."

15.2 Historical Testing

Details of previous test work programs and the metallurgical results are discussed in the SRK, 2009 technical report entitled "Minto Phase IV Pre-Feasibility Technical Report."

Direct quotes from the SRK report are presented in italic font.

Metallurgical test work on samples from Minto Main, Minto North, Minto South (southern portion of Minto Main pit), Ridgetop East, Area 2, and Area 118 deposits completed at G&T Metallurgical laboratory and SGS Lakefield were reviewed. The test work program consisted of flotation and comminution work and the samples used in the tests were composites of selected drill core intervals from each deposit. In addition variability flotation test work was completed on samples from Area 2 deposit. The results from the test work were used to develop the phase IV Minto flowsheet.

The criteria used in developing the flowsheet included an increase in throughput from 3,000 tpd to 3,775 tpd while meeting a predicted metal recovery and concentrate grade.

Comminution Test Work Conclusions

The design grind size selected for the Minto Phase IV study was 80% passing (P_{80}) 250 micron based on the flotation test work conclusions. Ausenco selected a BWI of 13 kWh/t for the comminution modelling based on the 75th percentile Bond ball work index data at the coarser closing screen size of 300 micron.

The Minto ores are of moderate competency and hardness, and amenable to grinding in a conventional SAG/ball milling circuit (SAB). Starkey and Associates completed surveys of the existing Minto milling circuit whilst treating Minto Main pit ore. These grinding surveys were used to adjust the Ausenco power based grinding models to allow future mill throughput predictions to be completed.

Flotation Test Work Conclusions

The mineralogy is relatively coarse grained and test work to date on Minto North, Area 2, Area 118 and Ridgetop indicated that a coarse primary grind size of 250 micron is feasible to achieve adequate liberation for flotation.

The latest test work campaigns conducted on Minto North, Ridgetop East and Area 118 in 2009 have indicated flotation performance consistent with the current main pit ore flotation characteristics. The test work has highlighted potential improvements to the existing flotation circuit that will be incorporated into the expansion of the plant. The major changes include:

- Inclusion of Regrind: The primary grind size will be increased from the current P80 grind size of 212 to 250 micron. A regrind stage with a target cleaner feed P80 grind size of 60 micron is required at this coarser primary grind.
- Increased Scavenger Efficiency: The expansion includes replacement of the Scavenger mechanisms (replace with external launders) which is expected to improve the efficiency in this portion of the flotation circuit.
- Increased Cleaner Stages: Three stages of cleaning provide improved circuit flexibility with regards to improving the final concentrate grade. The expansion will incorporate the increased cleaning stages and capacity.

Test Work Program

The metallurgical test work program used as the basis for this report consisted of flotation test work on Wildfire, Copper Keel, and Minto East deposits. The test work was performed by G&T under the management of Ausenco International Canada Inc ("Ausenco").

The test work programs were mainly completed on bulk composites designed to represent the complete orebody. No variability flotation test work was completed. The test work on the orebodies generally consists of:

- Ore characteristic test work including chemical content analysis, mineral content and distribution data, and bulk density data;
- Bench scale comminution testing, consisting of SAG Media Competency (SMC) test work, and Bond ball and rod mill work indices testing; and
- Bench scale flotation testing consisting of rougher kinetic flotation, cleaner flotation and locked-cycle tests.

Test Work Results

The following section is a summary of the test work undertaken by G&T and reported on in the G&T test work report, 2010 entitled "Preliminary Metallurgical Testing Wildfire, Copper Keel, & Minto East Zones; Minto Mine; KM2751".

Sample Origin

Samples for the new Minto deposits totalled 106 with a combined weight of 314 kg. The distribution of the samples for each new deposit was:

- Copper Keel, 43 samples with a combined weight of 114.0 kg;
- Minto East, 40 samples with a combined weight of 137.90 kg; and
- Wildfire, 23 samples with a combined weight of 61.60 kg.

The samples were placed in their designated composites then comminution and flotation samples split for testing.

Chemical Composition

Representative replicate sub-samples were collected and assayed for elements of interest. Results are presented in Table 15.1.

Table 15.1: Chemical Content Data

Sample	Assay – percent or g/t							
	Cu	Fe	Au	S	C	Cu(ox)	Cu(CN)	Ag
Copper Keel	1.79	5.27	1.03	1.04	0.38	0.06	1.25	10.0
Minto East	2.44	7.25	1.29	2.63	0.24	0.06	1.04	6.9
Wildfire	1.91	7.90	0.78	0.69	1.11	0.10	1.52	6.3

Note: All assays are averages of multiple cuts. Au and Ag reported in g/t, all others in percent.

Mineral Content and Distribution Data

- Mineral content and distribution for each composite was generated using the Particle Mineral Analysis ("PMA") function within QEMSCAN. The results from the PMA are:
- The sulphide mineral content in the three composites ranged between 3.5 and 5.5%. Pyrite is a small component of the overall feed weight, ranging between 0.1 and 0.3%;
- The deportment of copper, across the three samples, is highly variable. About 68% of the copper in the Minto East composite is present as chalcopyrite. Only 21 and 9% are present in this form, in Copper Keel and Wildfire composites. For Wildfire 63% of the copper feed is present as secondary copper minerals and 74% for Copper Keel.
- At a primary grind size of 200 micron the two-dimensional copper sulphide liberation is 70 and 74% for Copper Keel and Minto East. In comparison to only 45% for the Wildfire composite. The majority of the unliberated copper sulphide is present in binary form with non-sulphide gangue mineralization.

Bulk Density Data

Sixty individual core pieces were analysed. The data developed can be summarised as:

- The bulk density ranged in value from 2.6 to 3.2 g/cm³. The average bulk density was 2.8 g/cm³ with a standard deviation of ± 0.1 g/cm³.
- The bulk density distribution and average value was similar to other data sets collected under previous Minto Mine test programs.

Comminution Test Work Results

The SMC tests developed by JKTech Pty Ltd were performed at G&T, Kamloops.

The SMC test directly measures the strength of the rock when broken under impact conditions. The measure of this strength is reported as the drop-weight-index (DWi) and has the units kWh/m³. The DWi is directly related to the JK rock breakage parameters A and b.

The DWi, A and b parameters, and particle SG are reported in Table 15.2. The product of A*b is also a measure of resistance to breakage and along with the DWi were compared with the JKTech DW database. From this the Copper Keel, Minto East and Wildfire composites were categorized as having soft to medium competency.

Table 15.2: SMC Test Results

Sample	S.G. (g/cm³)	DWT Parameters		
		A x b	DWI	ta
Copper Keel	2.69	60.7	4.44	0.58
Minto East	2.80	54.1	5.16	0.50
Wildfire	2.77	67.5	4.09	0.63

The Bond rod mill work index (RWI) and ball mill work index (BWI) are used to determine the power draw required in the ball milling stage. The results are summarized in Table 15.3.

Table 15.3: RWI and BWI Test Results

Sample	Rod Grindability Test			Ball Grindability Test		
	RWI (kWh/t)	Sieve Size (micron)	Product Size (micron)	BWI (kWh/t)	Sieve Size (micron)	Product Size (micron)
Copper Keel	10.3	1180	890	16.0	106	83
Minto East	10.4	1180	901	17.1	106	84
Wildfire	9.7	1180	921	14.3	106	81

The hardness of the three samples varied from soft (Wildfire) to medium (Copper Keel) to hard (Minto East).

Previous test work on ore samples from other Minto deposits have shown a strong correlation between energy required for breakage and grind size. In the case of the three new samples it is expected that a primary grind of 250 micron will be required, this is significantly coarser than the closing size used for the BWI tests. If, Minto East behaves in the same manner as Minto Main then a BWI of 17.1 kWh/t is considered excessive.

The following figure comes from the SRK, 2009 technical report entitled "Minto Phase IV Pre-Feasibility Technical Report". Figure 15.1 shows a strong correlation between the BWI and the closing screen, or final grind size.

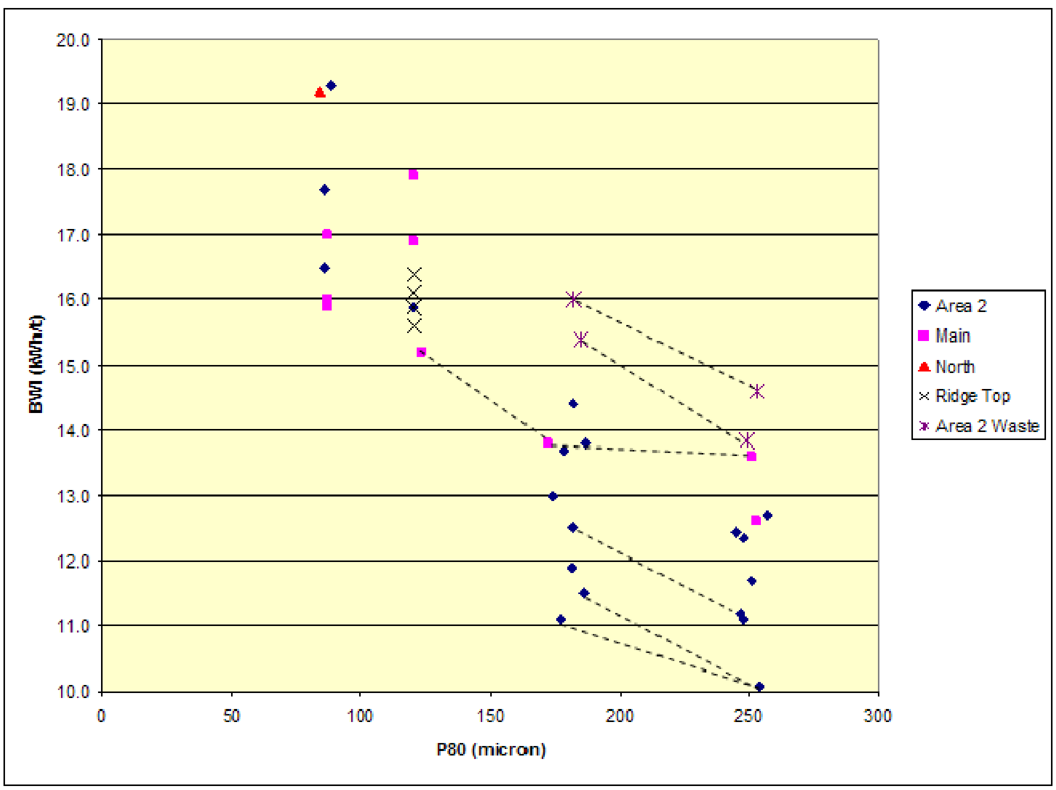


Figure 15.1: Bond Ball Mill Work Indices at varying closing sieve sizes

Flotation Test Work Results

Flotation test work for the Minto Phase V study was completed by G&T Metallurgical Laboratory. All of the test work focussed on bulk sulphide flotation in accordance with the existing Minto plant to produce a copper concentrate. Analysis of the test work was used to develop the plant process design criteria and estimate the concentrate grade, and copper and gold recovery.

Phase V Study Flotation Test Work

The flotation test work programs completed were primarily based on master composite samples designed to represent either the complete orebody or a zone within a particular orebody. The test work comprises:

- Rougher flotation kinetics to evaluate the effect of primary grind size between P80 150 – 250 micron;
- Open circuit cleaner flotation to evaluate the effect of regrind sizing on the cleaner circuit performance; and
- Locked cycle flotation to determine the effect of second and third cleaner tail recirculation on overall metallurgical performance.

Copper Keel

Rougher kinetic tests were conducted for the Copper Keel ore in G&T test work program KM2751, with P80 ranging from 129 micron to 250 micron. Open circuit cleaner tests were conducted with regrind P80 ranging from 66 micron to 189 micron. A locked cycle test was conducted at P80 of 271 micron and 73 micron regrind on the composite ore sample. The results are summarized below:

- Copper and gold recovery to the rougher concentrate was not adversely impacted by the primary grind size in the range of 150 – 250 micron;
- For the Copper Keel composite, 86.0% of the feed copper was recovered to the final product at a regrind of 87 micron. At this regrind the copper grade in the final product was 43%. The copper recovery increased to 96.4% with final product copper grade of 35% at the coarser copper rougher concentrate size of 189 micron. At the finest regrind size of 66 micron the copper recovery was 90.8% with final concentrate grade of 52%.
- Locked cycle tests with a regrind to P80 of 73 micron resulted in a final concentrate copper grade of 44.6% with 95% copper recovery. The concentrate gold grade was 23.2g/t with 84% gold recovery.

Minto East

Rougher kinetic tests were conducted for the Minto East ore in G&T test work program KM2751, with P80 ranging from 157 micron to 298 micron. Open circuit cleaner tests were conducted with regrind P80 ranging from 59 micron to 284 micron. A locked cycle test was conducted at P80 of 271 micron and 73 micron regrind on the composite ore sample. The results are summarized below:

- Copper and gold recovery to the rougher concentrate was not adversely impacted by the primary grind size in the range of 150 – 250 micron;
- For Minto East composite 92.9% of the feed copper was recovered to the final product at a regrind of 85 micron. At this regrind the copper grade in the final product was 41.8%. The copper recovery increased to 97.3% with final product copper grade of 34.4% at the coarser copper rougher concentrate size of 284 micron. At the finest regrind size of 59 micron the copper recovery was 86.0% with final concentrate grade of 43.0%.
- Locked cycle tests with a regrind to P80 of 73 micron resulted in a final concentrate copper grade of 40.4% with 94% copper recovery. The concentrate gold grade was 20.5g/t with 85% gold recovery.

Wildfire

Rougher kinetic tests were conducted for the Wildfire ore in G&T test work program KM2751, with P80 ranging from 154 micron to 229 micron. Open circuit cleaner tests were conducted with regrind P80 ranging from 61 micron to 132 micron. A locked cycle test was conducted at P80 of 231 micron and 95 micron regrind on the composite ore sample. The results are summarized below:

- Copper and gold recovery to the rougher concentrate was not adversely impacted by the primary grind size in the range of 150 – 250 micron;
- For Wildfire composite 92.3% of the feed copper was recovered to the final product at a regrind of 83 micron. At this regrind the copper grade in the final product was 46.2%. The copper recovery increased to 94.9% with final product copper grade of 36.4% at the coarser copper rougher concentrate size of 132 micron. At the finest regrind size of 61 micron the copper recovery was 93.2% with final concentrate grade of 52.1%.
- Locked cycle tests with a regrind to P_{80} of 95 micron resulted in a final concentrate copper grade of 38.7% with 93.9% copper recovery. The concentrate gold grade was 12.9g/t with 74% gold recovery.

Phase IV Study Flotation Test Work

The following summary of test work results is an extract from the SRK, 2009 technical report entitled "Minto Phase IV Pre-Feasibility Technical Report." This section covers test work performed on deposits included in the latest mine plan.

The flotation test work programs completed were primarily based on master composite samples designed to represent either the complete orebody or a zone within a particular orebody. The test work comprises:

- Rougher flotation kinetics;
- Open circuit cleaner flotation;
- Locked cycle flotation to determine the effect of second and third cleaner tail recirculation on overall metallurgical performance; and
- Mineralogical composition and fragmentation analyses by optical point counting methods and QEM*SCAN (Quantitative Mineralogy by Scanning Electron Microscopy).

Minto North

Rougher kinetic tests were conducted for the North Minto ore in G&T test work program KM2420, with P80 ranging from 156 micron to 273 micron. A locked cycle test was conducted at P80 of 200 micron and 65 micron regrind on the composite ore sample. The results are summarized below:

- 80% of the copper in the Minto North ore composite tested occurred as bornite. The amount of copper occurring as bornite is typically 50% in other Minto orebodies;
- The ore contained 5% sulphide minerals as bornite, chalcopyrite, chalcocite and pyrite (in their respective order of abundance);
- Two dimensional copper sulphide liberation was around 60% at a primary grind size of 250 micron;
- Copper and gold recovery to the rougher concentrate was not adversely impacted by the primary grind size in the range of 150 – 270 micron; and
- A regrind to P_{80} of 65 micron was required to achieve maximum final concentrate copper grade of 50% copper with 97% copper recovery.

Ridgetop East (RTE) and Area 118

The upper and lower zones were tested for both RTE and Area 118 in G&T test work program KM 2351. This consisted of a composite for the upper and lower portions of each zone as well as variability test work for each zone. The results are summarized below:

- Chalcopyrite was the dominant copper sulphide mineral in both Area 118 upper and RTE lower samples. Area 118 lower composite contained equal amounts of chalcopyrite and bornite. About half of the copper sulphide occurred in the form of chalcocite in the RTE upper composites;
- Copper recovery was affected by the higher than normal portions of non-sulphide copper minerals in the RTE upper sample (12% of the copper occurred as non-sulphides, mainly cuprite and native copper). Around 30% of the sulphide minerals were liberated at a primary grind size of 200 micron for the RTE upper composite, with unliberated copper mainly associated with non-sulphide gangue (NSG);
- At a primary grind size of 200 µm two dimensional copper sulphide liberation was 55 – 65% for Area 118 and RTE lower composites;
- Gold content of the four composites ranged from 0.2 – 1.0 g/tonne with the lower grades found in the upper portions of both zones;
- Based on the locked cycle test data, there was no sensitivity to primary grind size between P_{80} of 150 and 250 micron except for RTE upper composite which was not sensitive to P_{80} in range 150 to 200 µm;
- Locked cycle tests on RTE lower and Area 118 yielded overall copper recoveries of 93 – 97% with final concentrate grades of 32 – 44%. Average gold recovery was 77%; and
- Locked cycle tests on the RTE upper composite yielded an overall copper recovery of 85% and gold recovery of 47% (lower due to reasons discussed above).

Area 2

Ores from K, L, M, N, O, P& Q zones were tested. Variability tests were completed at approx P80 of 130 to 150 micron. Copper was mainly present as bornite and chalcopyrite.

Locked cycle tests on composite samples were at primary grind sizes (P80) of 150 and 270 micron with regrind of the rougher/scavenger concentrate to 100 micron followed by 2 stages of cleaning. In general, the copper recovery was unaffected by primary grind however gold recovery was approximately 10% lower for most of the composite samples tested. A summary of the test work by zone is shown in Table 15.4.

Table 15.4: KM 1966 Test work Summary by Zone

Composite	***Rougher Performance as a Function of P80***
L and M composites	*P80 300 micron primary grind is theoretically sufficient based on copper mineralogy.* *Locked cycle test indicated copper recovery similar at both 150 and 270 micron grinds but Au recovery reduces by 10 to 20% at the coarser grind.*
N composite	*Copper recovery is relatively insensitive to the grind sizes tested however further test work is required to confirm. Gold recovery was 9% lower for N zone at the coarser grind.*
O composite	*Copper and gold recovery were insensitive to the primary grind sizes tested.*
P Zone	*2% lower copper and 13% lower gold recoveries at the coarser 270 micron grind.*
Q Zone	*No difference in copper and about 8% lower gold recovery at the coarser 270 micron grind.*

Locked cycle test work on the L, M, N and O zones indicated that overall copper recoveries of 92 - 94% with 35 – 40% copper concentrate grades were achievable. The locked cycle tests on P and Q zones showed lower copper recoveries of 90%. The P zone ore is sensitive to primary grind size.

Locked cycle tests were completed on the L, M and N zone composites without the regrind stage on the rougher/scavenger concentrate to determine the effect of regrinding. The results indicated a drop in the copper concentrate grade of around 3% for the same overall recovery as the locked cycle tests with the regrind stage.

Minto South Primary Ore

Report KM 2024 contains test work on two composite samples from the South Pit that are less oxidized than the samples tested under the KM 1937 campaign. The test work completed locked cycle tests at P_{80} of 150 and 250 micron with regrind to P_{80} of 100 micron. Copper recoveries decreased above P_{80} of 200 micron (20% worse).

Locked cycle test work for composite 2 indicated a decrease in copper and gold recoveries of 5 – 10% at P_{80} 250 micron compared with P_{80} of 150 micron (Figure 15.2).

The flotation response to the increase in feed size from P_{80} of 150 micron to 250 micron was considerably more variable than indicated by main pit ore test work.

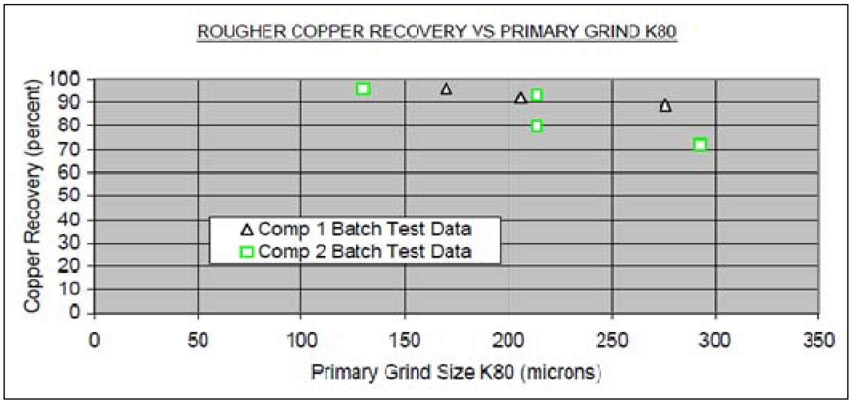


Figure 15.2: KM 2024 Batch Rougher Test Work Results

Minto South Partially Oxidized Ore

The ore used for the KM 1937 test work campaign contained 20% non-sulphide copper as compared to 8% for the South Pit ore used for the KM 2024 test work campaign.

Locked cycle test work for KM 1937 indicated a decrease in copper recovery of 4% with a primary grind size above P_{80} of 150 micron and a further 4% above P_{80} of 240 micron to 279 micron. The gold recovery loss is around 3% as the primary grind is increased above P_{80} of 240 micron. Report KM1937 presents a range of data on the impact of P_{80} on final tailings copper and gold grades as shown in Figure 15.3.

By inspection, it appears that only data outliers at the P_{80} of 150 and 350 micron indicate any significant impact on tailings grade, with the finer P_{80} of 150 micron potentially decreasing the copper tailings grade by 0.07% and the coarser P_{80} of 350 micron increasing both copper and gold tailings grades significantly compared with a P_{80} of 300 micron.

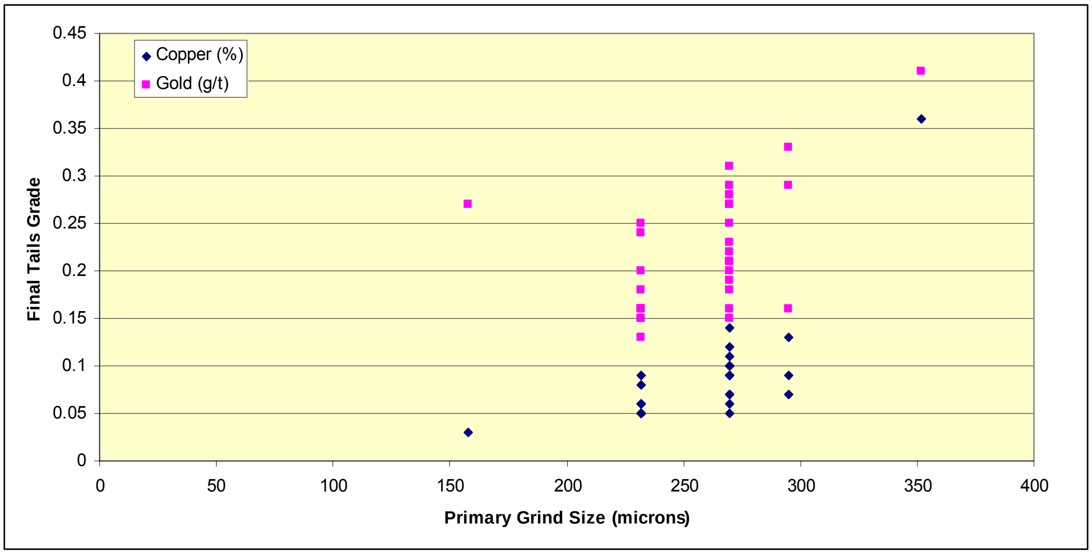


Figure 15.3: KM 1937 Primary Grind Size vs. Tails Grade

Addition of a sulphidizing agent (sodium hydrosulphide) as an activator improved the recovery of non-sulphide copper by around 30% or 2 - 4% in overall copper recovery.

During 2010 Minto Mine has made good progress with improving the recovery of oxide and partially oxide material with the use of AM28, Aerofloat and FLOMIN C7931 flotation reagents.

Comminution Test Work Conclusions

The design primary grind size selected for the Minto Phase IV study was 80% passing (P_{80}) 250 micron based on the flotation test work conclusions. The Minto Phase V test work program confirmed this grind size is appropriate when treating Copper Keel, Minto East and Wildfire ore types. Following discussions with the Capstone project group the three new deposits are expected to show similar grinding characteristics to ore types currently being processed. With this in mind Ausenco selected a BWI of 13 kWh/t for the comminution modeling at the coarser closing screen size of 300 micron.

The three new Minto ores are of moderate competency and hardness, and amenable to grinding in a conventional SAG/ball milling circuit (SAB).

Flotation Test Work Conclusions

The mineralogy is relatively coarse grained and test work to date indicated that a coarse primary grind size of 250 micron is feasible to achieve adequate liberation for flotation.

The latest test work campaigns conducted on Copper Keel, Minto East, and Wildfire in 2010 have indicated flotation performance consistent with the current main pit ore flotation characteristics. G&T made the following conclusions:

- The samples tested had copper feed grades ranging from about 1.9 to 2.4%. The gold content in the feed ranged from about 0.8 to 1.3 g/tonne.
- The copper deportment in these samples, with the exception of the Minto East composite, was somewhat atypical of previously tested Minto ores. For the Copper Keel sample, about 74 percent of the copper was present in bornite. The Wildfire composite had almost 88 percent of the feed copper present in secondary copper minerals chalcocite and covellite. The two-dimensional copper sulphide liberation was over 70 percent of the Copper Keel and Minto East composites. For the Wildfire composite, the two-dimensional copper sulphide liberation was much lower at about 42 percent.
- A series of rougher kinetic, open circuit batch cleaner and locked cycle tests were carried out on each ore type. Results from the kinetic rougher tests indicate that acceptable copper recoveries could be achieved at up to P_{80} 250 micron primary grind sizing. Open circuit batch cleaner tests were carried out at a primary grind target sizing of P_{80} 250 micron. The results indicated that a regrind sizing of about P_{80} 80 micron produced the best compromise between copper grade and recovery.
- In the open batch cleaner tests both the Wildfire and Copper Keel composites produced final copper concentrate grades of nearly 50% or higher. The higher copper concentrate grades reflect the higher proportion of bornite and secondary minerals, chalcocite and covellite.
- A single locked cycle test was carried out on each composite at a primary and regrind discharge sizing of 250 micron and 80 micron respectively. For all three composites, under these conditions, about 95% of the feed copper was recovered to the final copper concentrate. The copper grades in the final concentrate ranged between 39% and 45%.
- Final concentrates, produced in locked cycle testing, were analyzed for the presence of deleterious minor elements and all the elements that typically attract penalties were well below threshold. The concentrates all contain payable levels of silver and gold. The minor element data should be reviewed by a concentrate marketing specialist to confirm any concentrate salability issues.
- Based on the results of testing on these composites, it should be possible to process material from these new zones at a coarse grind sizing with good metallurgical performance. Consideration should be given to carrying out some variability testing, in particular on the Wildfire Zone, which is notably different mineralogy than typical Minto ores.
- There is good potential to increase recovery for Oxide and Partially Oxide ore by continuing to test reagents such as AM28, Aerofloat and FLOMIN C7931.
- There is good potential to increase recovery by increasing Cleaning Circuit capacity and Scavenger Cell efficiency.

15.3 Process Plant Design

General

Ore from the new deposits will be processed through a modified Minto process plant.

Process Plant Design Basis

The key criteria selected for the plant design are:

- Treatment of an average 3,442 dry metric tonnes per day for 2011, increasing to 3,750 dry metric tonnes per day for 2012 and beyond;
- Material from Minto Main, Minto North, Minto South, Ridgetop East, Area 2, and Area 118 will be processed through the Minto plant.
- Wildfire and Copper Keel deposits are not included in the latest mine plan and have not been considered further;
- Design availability of 91.30%, being 7,997 operating hours per year, with standby equipment in critical areas; and
- Sufficient plant design flexibility for treatment of all ore types as per test work completed at design throughput.

The selection of these parameters is discussed in detail below.

Throughput and Availability

An overall plant availability of 91.3% or 7,997 h/y was nominated. Benchmarking indicates that similar well operated plants with moderately abrasive ore have consistently achieved 91 to 92% overall plant availability.

The existing Minto process plant availability is below 91.3%. Through monitoring of equipment and record keeping, operations personnel have identified the cause of the lower availability and have commenced a program of preventative maintenance and equipment duplication (installing stand-by equipment). It is expected once the program is complete an availability of 91.3% will be achievable.

Major causes for reduced availability include:

- Excessive failure of the installed flotation mechanisms. These have been replaced with a new supplier and replacement frequency and costs are expected to reduce;
- Poor availability of the tailings treatment facility, particularly the filter circuit;
- Original pipe work around the milling area was not rubber lined. Pipe work was replaced with rubber lined pipes which will reduce the frequency of change-outs;
- Various pumps have been upgraded and standby tailings pumps installed under operating cost budgets.

The throughput selected is mainly a function of modifications planed to the crushing circuit and the existing Minto plant grinding circuit. From the review of Minto Phase IV study and recent test work data a plant throughput of 171 dry metric tonnes per hour based on 80% of the SAG feed material being finer than 25 mm is achievable. With a 91.3% availability and 25 mm top feed size an average of 3,750 tonnes per day can be processed.

Processing Strategy

The process design is based on treating ore with similar hardness to the current Minto Main ore being processed or similar to that tested by DJB Consultants in October 2007. Inputs for the Ausenco power based comminution model were based on a review of the Minto Phase IV study outcomes and test work for the new ore bodies as well as general plant observations by Minto operations personnel, Starkey & Associates, and DJB Consultants.

Head Grade

The plant is designed to treat various tonnages of primary ore with a sustained maximum head grade of 2.5% Cu and 1.5 g/t Au.

15.4 Process Description

Unit Process Selection

The unit operations used to model the plant throughput and metallurgical performance are well proven in the sulphide flotation industry. The flow sheet incorporates both new and existing unit process operations:

- Ore from the open pit is crushed using the existing primary jaw crusher to a crushed product size of nominally 80% passing (P_{80}) 115 mm. Jaw crusher product is then screened, (at a new portable screening facility as selected and installed by MintoEx), oversize material is crushed in the existing secondary crusher to a nominal 80% passing 25 mm. Undersize from the screen is combined with secondary crusher product and fed onto the existing stockpile stacking conveyor. Provision to by-pass some of the secondary crusher feed and send it to the existing stockpile will be accommodated in order to optimize SAG Mill performance;
- Conical stockpile with the existing single reclaim apron feeder;
- Existing 670 kW SAG mill, 5.03 m diameter with 1.52 m EGL;
- Existing twin 670 kW ball mills each 3.20 m diameter with 3.66 m EGL, in closed circuit with hydrocyclones, grinding to a product size of nominally 80% passing (P_{80}) 250 micron;
- Bulk rougher/scavenger flotation consisting of the existing three 40 m^3 forced air tank flotation cells and the existing four 15 m^3 cells retrofitted of new tank cell 20 mechanisms;

- Rougher/scavenger concentrate regrinding in a new 220 kW vertical stirred mill, grinding to a product size of nominally 80% passing (P_{80}) 80 micron. Regrind circuit will be added in from the third quarter of 2012;
- Cleaner 1 flotation consisting of the existing four 10 m^3 forced air tank flotation cells;
- Cleaner 2 flotation consisting of three new 10 m^3 tank cells;
- Cleaner 3 flotation consisting of the existing six 3 m^3 trough shaped flotation cells to provide a total of 25 minutes retention time. Cleaner 3 flotation will come on-line from the third quarter of 2012;
- Final cleaner 3 concentrate thickening in the existing 6 m diameter high rate thickener;
- Concentrate thickened slurry filtration in the existing Ceramic disk filter;
- Flotation tailings thickening in the existing 9.1 m diameter high rate thickener to an underflow density of 50% solids;
- After completion of ore extraction, utilization Minto Main pit for tailings deposition directly from the flotation tailings thickener underflow pumps;
- Plant reagents preparation and distribution systems as per the current Minto unit operations;
- Raw process plant water supply from the existing site water storage facility reticulated throughout the plant as required. (Harvesting and storage of raw water sufficient to allow continued water supply throughout the year is excluded from the Ausenco scope of work for this study);
- Process water dam and distribution system for reticulation of process water throughout the plant as required per the existing facilities. Process water is supplied from water reclaimed from tailings deposition in the Minto Main pit, from process operations and site run-off with raw water used as make-up water as required;
- Potable water as per the existing supply is distributed to the plant, and for miscellaneous purposes around the site; and
- Plant, instrument and flotation air services and associated infrastructure as per the existing facilities.

Infrastructure

Upgrades to the site infrastructure are described below.

Minto Mine Power

The current Yukon Energy Corporation's (YEC) Power Purchase Agreement with Minto Mine allows for a total capacity of 4400kVA with a total annual consumption limitation of 32.5GWh. Minto Mine currently utilizes approximately 3800kVA of its allotted capacity of 4400kVA.

The additional loads required by the underground mining operations and revisions to the milling activities will require an additional 1360kVA of capacity for a total average capacity of approximately 5160kVA. In order to accommodate the additional capacity a request to amend the YEC's Power Purchase Agreement has been submitted. The requested amendment is to allow Minto to utilize a capacity of 5500kVA and a total revised annual consumption limit of 45GWh.

Minto Distribution

The current mine's power is distributed from a YEC owned substation at 5kV. The current system configuration utilizes a main switch room in the mill with two smaller switch rooms in the tailings building.

New Distribution

In order to accommodate the new loads required for the underground mining operations a 300A 5kV feed will be routed from the spare contactor in the main mill switch room and will be routed to the portal site via a combination of direct burial and overhead power line that will be dependant of surface conditions. The new 5kV feed will be terminated in a switch room near the portal site. The switch room will contain a line-up of 5kV disconnect switches that will feed transformation for local surface loads and will feed the underground portion of the mine with a single 5kV feed.

Vent fan sites will be fed from an additional switch in the tailings switch room and will be fed either by direct buried cable or via overhead power line. Each vent fan building will contain the required 5kV switchgear, transformation and distribution required for the fans.

16 Mineral Resource and Mineral Reserve Estimates

16.1 Introduction

A primary objective of SRK's work was to produce a revised independent mineral resource evaluation for the Area 2/118 and for the Ridgetop deposits based on the studies from February, 2010 (Ridgetop) and June, 2010 (Area 2/118). The Minto North Zone was evaluated by Kirkham Geosystems Ltd (Kirkham Geosystems) in 2009, and has not been revised. Kirkham Geosystems have modelled a new deposit in June 2010 named Minto East. The Minto Main deposit was updated by Lions Gate Geological Consulting Inc., and independently verified by SRK. The methodology for the Minto Main deposit estimation is outlined in the Area 2 Pre-feasibility Study Minto Mine technical report (2007) and is not discussed further here.

The mineral resource evaluation reported herein supersedes earlier resource estimates presented in the 2009 Phase IV PFS Technical Report.

The resource estimate in the Area 2/118 and Ridgetop deposits was completed by Dr. Wayne Barnett, Ph.D., Pr.Sci.Nat., an independent qualified person as defined in National Instrument 43-101. The effective date of the resource estimate both in Area 2/118 and in Ridgetop is August 30, 2010. Marek Nowak, P.Eng., analyzed the data, reviewed and validated the mineral resource estimates. The Minto North deposit and Minto East deposit resource estimates were completed by Garth Kirkham, P.Geo., of Kirkham Geosystems.

This section describes the work undertaken by SRK and Kirkham Geosystems, including key assumptions and parameters used to prepare the mineral resource models for Area 2/118, Ridgetop, Minto North and Minto East deposits together with appropriate commentary regarding the merits and possible limitations of such assumptions. The following discussion concentrates on Cu grades, the most valuable commodity in the Minto deposits.

In the opinion of SRK, the block model resource estimate and resource classification reported herein are a reasonable representation of the global mineral resources at Area 2/118, Ridgetop, Minto Main. Minto North and Minto East deposits at the current level of sampling. The mineral resources presented herein have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines and are reported in accordance with Canadian Securities Administrators' National Instrument 43-101. Mineral resources are not mineral reserves and do not have demonstrated economic viability. The estimated mineral resources have been used in the preliminary feasibility study described in this report.

16.2 Resource Database

The database used to estimate the Area 2/118 and Ridgetop deposits was prepared by MintoEx personnel and verified by SRK. The Minto North and Minto East database was also prepared by MintoEx, and verified by Kirkham Geosystems. The mineralized domains of the deposits were modelled using Gemcom software based on lithological and structural interpretations.

SRK is of the opinion that the current exploration and structural information is sufficiently reliable to confidently interpret the mineralized boundaries and that the assay data are sufficiently reliable to support the estimation of mineral resources.

Table 16.1 provides a summary of the samples included in the Area 2/118, Ridgetop, and Minto North deposits database. Note that the actual number of samples within the modelled geology domains was lower.

Table 16.1: Exploration Data within the Modelled Deposits

Project	Year	DD Drill holes	Number of Drill Samples	Drill Total (m)
Area 2/118	2010	22	2,662	3,868
	2009	22	2,650	3,675
	2008	48	5,459	6,719
	2007	40	5,695	7,522
	2006	80	9,364	14,752
	Historical	23	622	1,735
	TOTAL	**235**	**26,452**	**38,270**
Ridgetop	2009	71	5,067	7,330
	2008	46	4,177	5,604
	2007	25	1,993	2,704
	Historical	21	840	2,068
	TOTAL	**163**	**12,077**	**17,706**
Minto East	2007	1	168	380
	2008	1	103	385
	2009	4	575	1,410
	2010	27	3,772	8,892
	Total	**33**	**4,618**	**11,067**
Minto North	2009	87*	4,651	11,263
	Total	**87***	**4,651**	**11,263**

** Note that out of the total of 87 holes drilled in 2009, 31 were completed prior to June 2009 and the remaining 56 holes were drilled from June through September, 2009.*

16.3 Area 2/118 Deposit

Geology Model

The Area 2 and Area 118 deposits are discussed together in this report since they are not spatially separate, but form part of the same system of mineralization; the Area 2/118 deposit. Area 118 is recognized to be structurally more complex and the boundary between the two deposits is defined in this study to be a fault dipping at 50^0 towards the northeast. The copper, gold and silver mineralization in the Area 2/118 deposit is associated with foliated granodiorite lithological units. The background non-mineralized rock is an unfoliated granodiorite. To constrain the interpolation during grade estimation, SRK built three dimensional solids of the foliated granodiorite units. They are modelled to be generally shallow dipping (19 to 30^0) towards the northeast.

The geological origin of the foliated zones is still under investigation. They are presumably ductile shear zones, but the established geometry of the zones is unusual. They may originally have been some sort of sill-like intrusive with a composition more amenable to strain focusing.

The continuity has been established by multiple intersections of the zones showing that the zones in a particular deposit to be traceable over the entire deposit.

The foliated zones have mineralogical, geochemical, grade and textural signatures that can be picked up in the logs and assays data, and can be used to identify zones and show continuity at least over several hundred metres. The style of mineralization also appears identical for all the other deposits in the area. In particular, the Main Minto deposit is currently being mined and the continuity of mineralization can be established without question.

There are number of aspects that complicate the resource continuity:

- The zones bifurcate, which means that a mineralized zone can contain a significant amount of waste, or that thinner ore zones can merge with larger zones. A bifurcating geometry complicates geological modelling and may expect to increase internal dilution.
- The width and dip of mineralized zones are locally variable. The zones therefore appear to pinch-and-swell. The change in thickness might be as much as an order of magnitude over less than 30 m in horizontal distance.
- At least some of the irregularity in the geometry and thickness of the mineralized zones is due to small-scale and large-scale structural displacements. No detailed structural model has been completed for either deposit, but at least two faults appear to be present in Area 2, and three possible faults displace the modelled zones in Area 118. Similar structures may be present throughout the deposit, each with displacements of a few metres or less.

The debate over the original nature of the foliated and mineralized zones means that the understanding of known geological processes cannot be utilized to define the resource geometry.

On the other hand, the Minto Main deposit pit exposures and the large number of drilling intersections define the range of possible geometries fairly well, and reduce the risk of incorrect geological interpolation away from known data. In addition, the understanding of the local geometries has been a successful factor in local exploration.

The updated Area 2/118 resource model was created using a commercial three-dimensional block modelling and mine planning software, GEMS version 6.2 (Gemcom®). The models were created in metric units using the mines local co-ordinate system (UTM NAD83 zone 8). The mineralized zone solids were considered hard boundaries where grades were not allowed into blocks outside of these solids.

The mineralized zone solids were built using top and bottom Laplacian grid surfaces that pass through the vertices representing the top and bottom drill hole intersecting contacts. The interpretation was initially done using vertical sectional interpretations provided by MintoEx geologists as references. These sections are spaced on 25 m intervals. SRK reviewed, adjusted and resolved the interpretations where necessary.

The contacts for a specific contact surface are made active by snapping polylines to the drill hole vertices, such that the polyline vertices are then used by GEMS as controls on the surface gridding. The grid triangulation vertices are then exported and re-imported as points. The final contact surface is then created from the imported grid points and the original polyline vertices using a regular surface creation technique.

This final surface has the surface triangulation vertices snapped precisely to both the grid points and the polyline vertices. The result is a contact surface that looks like a smoothed Laplacian grid but actually snaps to the drill hole intersections. The surfaces are then used to clip out or “carve-off” the mineralized zone domains and waste domains from an original solid wireframe representing the entire resource extents.

Up to 9 primary mineralized zones were assigned the following domain codes historically used by MintoEx geologists; J, K, L, M, N , O, P, Q, and R. Table 16.2 includes a list of the domain coding assigned to the drill data and the block model. Note that additional zones were modelled as bifurcations of the primary zones (noted in Table 16.2). These bifurcations are closely associated with the primary zones and for the purpose of the interpolation were considered part of the primary zone. Figure 16.1 is a 3-dimensional view of the zone solids, showing their block model codes (or Zone-ID).

Table 16.2: Modelled Domain Names and Block Model Codes

Domain Name	Block Model Code		Comments
	Area 2	Area 118	
J	20	21	
K	30	31	
L	40	41	Includes zone L2
M	50	51	Primary grade bearing domain. Includes zone M2
N	60	61	Very thick domain in Area 2. Appears to become thinner with weaker foliated texture in Area 118.
O	70	71	Includes zone O2
P	80	81	
Q	90	91	Includes zone Q2
R		101	Located below modelled resource in Area 2.
Overburden (OB)	500		
Air	0		
Waste	200		

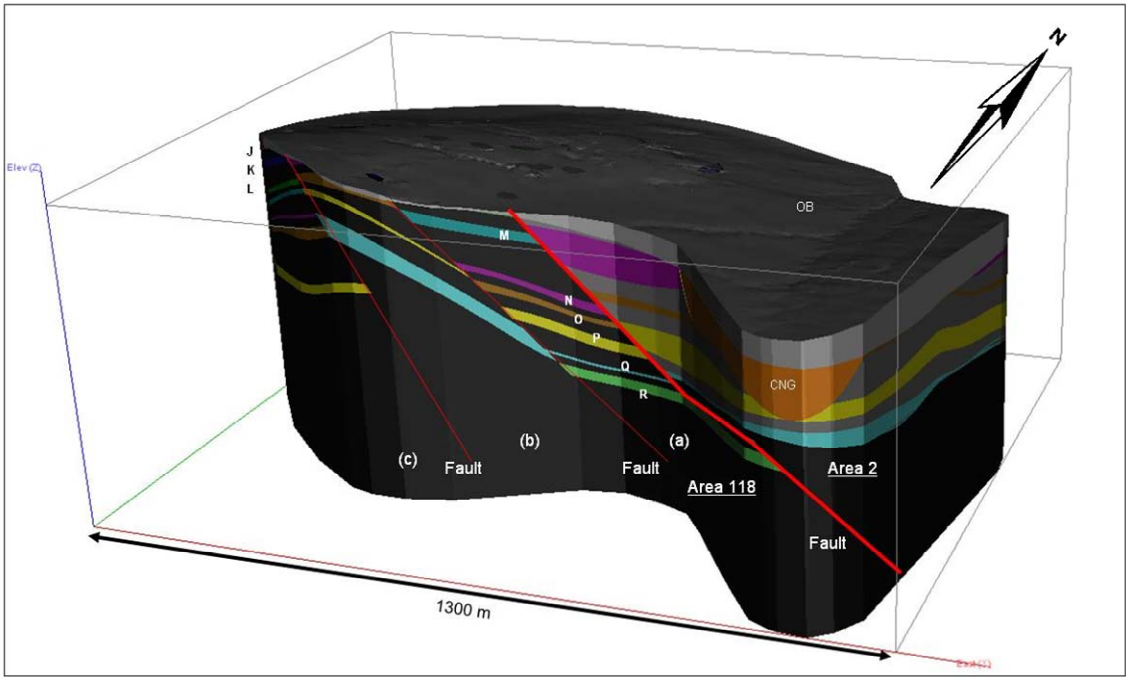


Figure 16.1: Isoclinal View Northwards of the Area 2 and Area 118 Mineralization Domain Solids

The boundary between Area 2 and Area 118 zones has been modelled as a fault. The drill hole intersections are of sufficient density to show the position of the fault accurately. Two additional faults have been modelled in order to explain intersection positions in Area 118, and these faults divide the Area 118 resource into three domains (labeled a, b and c in Figure 16.2). No study has been done on the drill core in order to define the characteristics of the faults.

The basic geometry indicates that the faults post-date the formation of the foliated zones, and that the dominant shear sense may be reverse. Faulting also presumed to post-date mineralization because of observations of displaced mineralization, but this has not been confirmed by any detailed study.

The position of the faults was confirmed as best as possible by three separate approaches. Firstly, lineaments were drawn onto the topographic surface. Secondly, the logged structural data was reviewed and structural zones were connected up to define possible faults. Thirdly, the possible position of faults was identified by irregularities or displacements in the geometry of the foliated zones. In the case of the modelled structures, all three approaches supported the position of the modelled fault surfaces.

The solids were then used to assign the domain and block model codes to the drill hole data (assays and composites) and the block model cells. Blocks above the topography surface were tagged as Air and the blocks outside of the zone solids were tagged as Waste.

There is unconsolidated material near surface, which is included in the model as Overburden. SRK reviewed the tagged assay, composite and block data on sections and visually in three dimensions, as well as in exported text files using external customized software, thereby ensuring that the process had worked properly.

To assess how well the modelled solids differentiate between lower and higher grade mineralization, grades on either sides of the modelled contacts were queried and listed. Any anomalous assay values were checked visually in three dimensions to determine whether the problems were errors or not.

The foliated granodiorite typically has a sharp boundary with unfoliated rock. In these cases the grade boundary is also sharp and coincident with the textural change. There are situations where the foliations become progressively weaker over a gradational contact zone. Logging observations indicate that grade is generally more weakly developed in poorly foliated rock, but only disappears once the foliations are completely gone. The geological logging does make a specific effort of noting the existence of foliated textures. These geological observations indicate the necessity of hard domain boundaries when estimating the resource in each mineralized domain.

Anomalous grade outside of foliated rock was reinvestigated, but on investigation was shown to be one of the following:

- Anomalous grade spikes associated with veins. This style of mineralization is considered subordinate and volumetrically insignificant compared to the foliation-hosted mineralization. It was not considered as part of the estimation process and assays outside of the geological foliation domains did not contribute to the estimation.
- Zones incorrectly logged as unfoliated in historical data logs. Where possible these logs were corrected with the help of the MintoEx geologists, in order to demonstrate the continuity of the foliated zones.

- Intervals incorrectly logged as unfoliated on the shoulder of foliated zones. This is a geological logging accuracy issue, where the contacts of the foliated rock were inaccurately positioned or where the foliation textures are gradational. Where possible these logs were corrected with the help of the MintoEx geologists.
- Thinner foliated zones separate from the larger zones, but too small to be included in the resource. These zones would typically be uneconomical because of the associated waste to ore rock ratio.

Data

At total of 15,157 grade measurements have been used in the design of mineralized domains from holes drilled roughly at 30 to 40 m spacing. More than 50% of the samples within the modelled domains were collected from 1.5 m intervals (Figure 16.2). All assays were composited to 1.5 m lengths.

Choice of the shorter composite length was guided by a small proportion (approx 20%) of relatively narrow, less than 4.5 m, mineralized zones. Shorter composite lengths ensured that most relevant, undiluted assays were included in the resource assessment.

Within the mineralized domains 14,590 composite assays were produced from 235 holes. The average thickness of highly mineralized horizons is 17 m (L, M, O, P) and 23 m in lower grade horizons.

Statistics of polygonally declustered 1.5 m Cu composites within each mineralized zone are presented in Figures 16.3 and 16.4. Statistics of the 1.5 m Au and Ag composites within each mineralized zone are given in an Appendix A.

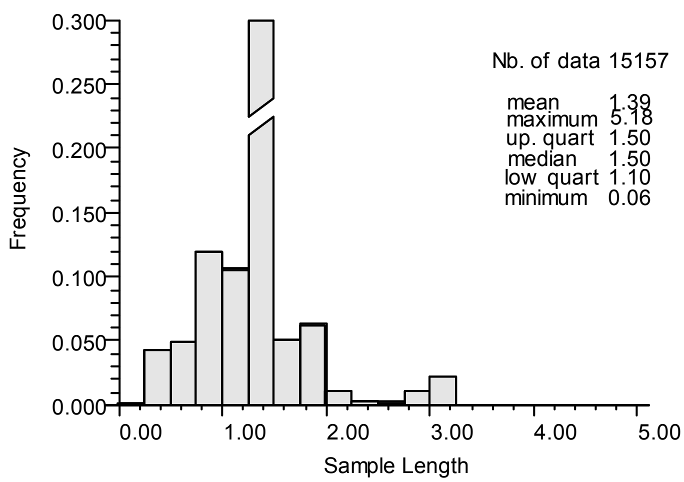
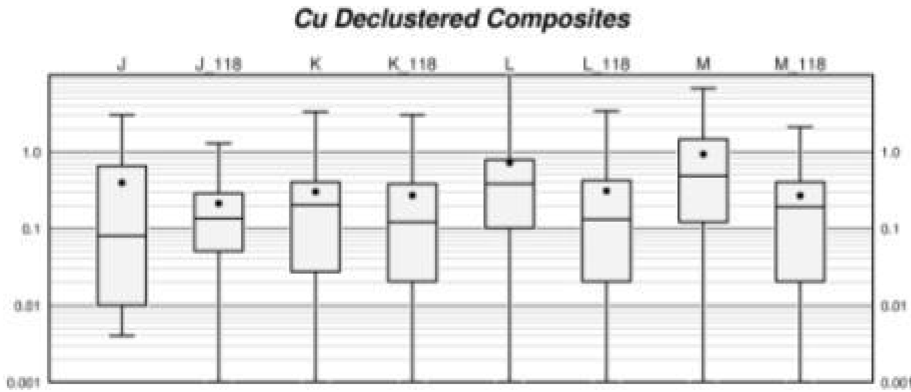
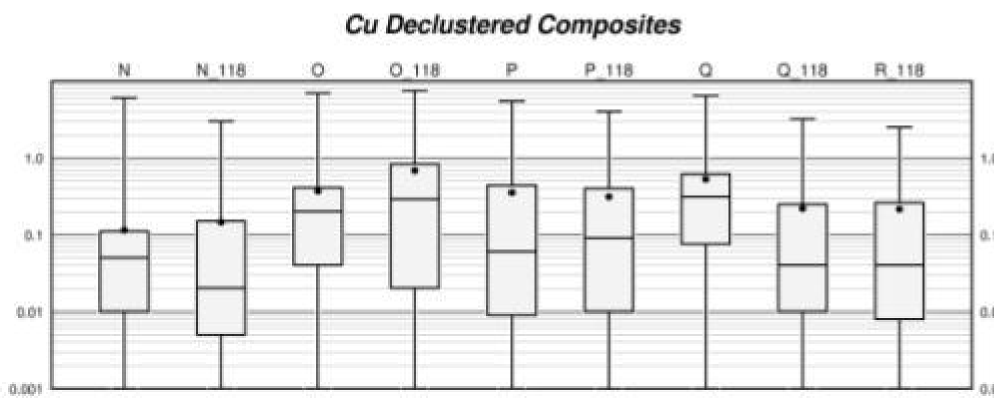


Figure 16.2: Area 2/118 - Histogram of Sample Lengths



	J	J_118	K	K_118	L	L_118	M	M_118	
Number of data	224	140	585	342	1218	373	1891	463	Number of data
Mean	0.391	0.21	0.299	0.268	0.711	0.306	0.924	0.266	Mean
Std. Dev.	0.56	0.239	0.398	0.387	1.148	0.444	1.058	0.313	Std. Dev.
Coef. of Var.	1.434	1.14	1.334	1.443	1.614	1.451	1.145	1.177	Coef. of Var.
Maximum	3.0	1.28	3.3	3.0	12.0	3.39	6.67	2.09	Maximum
Upper quartile	0.65	0.285	0.4	0.38	0.78	0.42	1.45	0.4	Upper quartile
Median	0.08	0.135	0.2	0.12	0.38	0.13	0.48	0.19	Median
Lower quartile	0.01	0.05	0.027	0.02	0.1	0.02	0.12	0.02	Lower quartile
Minimum	0.004	0.001	0.001	0.001	0.001	0.001	0.001	0.001	Minimum

Figure 16.3: Area 2/118 - Basic Statistics of Declustered Cu Composite Grades, for Domains J to M



	N	N_118	O	O_118	P	P_118	Q	Q_118	R_118	
Number of data	4032	546	943	666	1328	374	908	207	319	Number of data
Mean	0.114	0.145	0.369	0.682	0.353	0.311	0.526	0.218	0.214	Mean
Std. Dev.	0.286	0.293	0.608	1.095	0.687	0.551	0.769	0.46	0.381	Std. Dev.
Coef. of Var.	2.513	2.014	1.645	1.606	1.945	1.773	1.462	2.109	1.787	Coef. of Var.
Maximum	6.0	3.0	6.94	7.51	5.5	4.0	6.46	3.2	2.5	Maximum
Upper quartile	0.11	0.15	0.41	0.83	0.44	0.4	0.615	0.248	0.26	Upper quartile
Median	0.05	0.02	0.2	0.29	0.06	0.09	0.31	0.04	0.04	Median
Lower quartile	0.01	0.005	0.04	0.02	0.009	0.01	0.075	0.01	0.008	Lower quartile
Minimum	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	Minimum

Figure 16.4: Area 2/118 - Basic Statistics of Declustered Cu Composite Grades, for Domains N to R

Figure 16.5 shows bivariate statistics of the Cu and Au assays. Note: very good correlation, indicated by a regression curve (white thick line) showing a general tendency of increased Cu assays for higher Au assays. This high correlation lead to a design of variogram models along identical major directions of continuity for both Cu and Au grades.

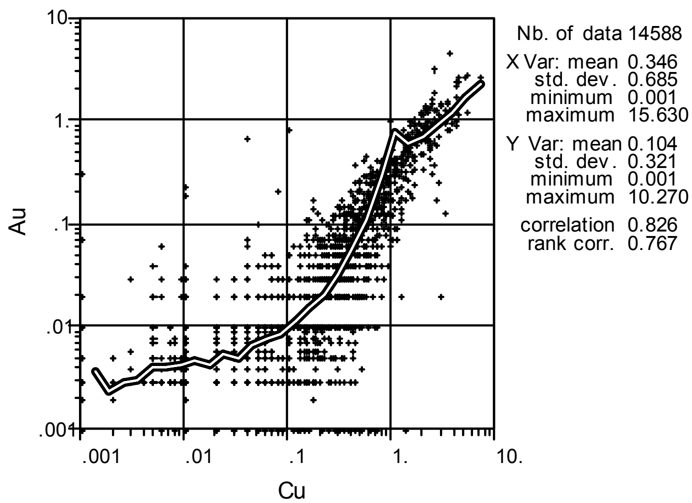


Figure 16.5: Area 2/118 - Bivariate Statistics of Cu and Au Assays

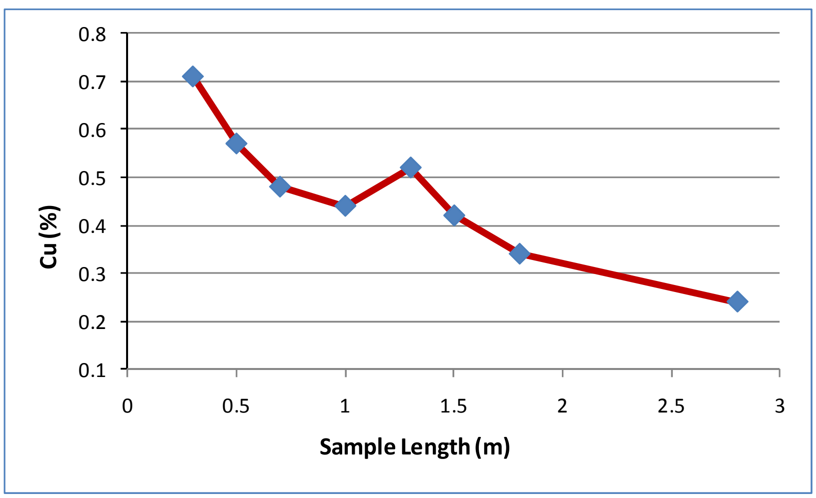


Figure 16.6: Area 2/118 - Grade Variation with the Sample Length

Evaluation of Extreme Assay Values

Block grade estimates may be unduly affected by very high grade assays. Therefore, the assay data were evaluated for the high grades outliers. An analysis of the high grade assays indicates negative correlation between the assay data and the sample lengths (Figure 16.6).

This suggests that sampling was based on visual indications of mineralization. In view of the above, no capping was done before assay compositing to 1.5 m lengths.

Variogram Analysis

Experimental variograms and variogram models in the form of correlograms were generated for Cu and Au grades in both in Area 2 and 118. The nugget effect values (i.e., metal variability at very close distance) were established from down hole variograms. The nugget values for Cu range from 5 to 35 percent of the total sill. Note that the sill represents the grade variability at a distance beyond which there is no correlation in grade. Variogram models used for Cu grade estimation are summarised in Tables 16.3 and 16.4. Note that no variogram models were designed for Ag grades. The Ag was estimated by the inverse distance squared method.

Table 16.3: Area 2 Cu Exponential Variogram Models

Zone	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
J	0.05	0.55	-60	-15	0	70	30	12
		0.4				150	50	15
K	0.2	0.6	45	15	0	70	70	10
		0.2				240	100	20
L	0.1	0.6	45	15	0	50	70	17
		0.3				650	200	19
M	0.15	0.6	100	18	-37	150	100	30
		0.25				500	160	60
N	0.15	0.6	45	15	0	30	40	15
		0.25				110	200	55
O	0.2	0.6	45	15	0	50	90	17
		0.2				100	160	30
P	0.05	0.45	45	15	0	20	20	20
		0.5				1000	1000	60
Q	0.3	0.5	0	0	0	80	80	20
		0.2				110	75	60

Table 16.4: Area 118 Cu Exponential Variogram Models

Zone	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
J118	0.25	0.5	45	20	0	60	40	20
		0.25				100	70	60
K118	0.25	0.5	45	30	0	60	40	20
		0.25				100	70	60
L118	0.15	0.7	60	30	0	50	90	15
		0.15				250	450	40
M118	0.15	0.7	60	30	0	50	90	15
		0.15				250	450	40
N118	0.35	0.4	-30	75	-15	15	50	25
		0.25				30	200	80
O118	0.1	0.4	90	30	0	120	80	30
		0.5				300	150	60
P118	0.05	0.4	90	30	0	40	30	5
		0.55				250	120	10
Q118	0.05	0.4	0	-60	30	5	40	30
		0.55				10	250	120
R118	0.05	0.4	30	15	0	30	40	5
		0.55				120	250	10

Resource Estimation Methodology

The geometrical parameters of the block model are summarised in Table 16.5.

Table 16.5: Specifications for the Area 2/118 Block Model

Description	Easting (Xm)	Northing (Ym)	Elevation (Zm)
Block Model Origin (Lower left corner)	384,200	6,943,700	325
Parent Block Dimension	10	10	3
Number of Blocks	147	126	225
Rotation	0	0	0

All 1.5 m composite assays were coded by modelled mineralized domains. Blocks in a mineralized domain were estimated only from the assays within that domain. Ordinary Kriging was used to estimate Cu and Au grades and inverse squared distance weighting to estimate Ag grades.

Treatment of High Grade Composite Grades

Instead of capping the composites for high grade assays, SRK elected to limit the influence of the high grade intersections during the estimation process. Continuity of the high grade assays was studied with a technique called “p-gram”.

Figure 16.7 shows the continuity of high grade assays at different thresholds. High grade continuities can be indicated up to a distance where plotted curves roughly level off. For example, at 4% threshold maximum distance at which the continuity could be shown is roughly 40 to 60 m.

For grade estimation in all mineralized zones high grade assays were only used if they were found within search ellipsoid of 40 x 30 x 15 m size. In both Area 2 and Area 118 high grade thresholds were defined from statistical analysis, separately for each domain. The direction of the search ellipsoid was aligned with the overall direction of grade continuity in each zone.

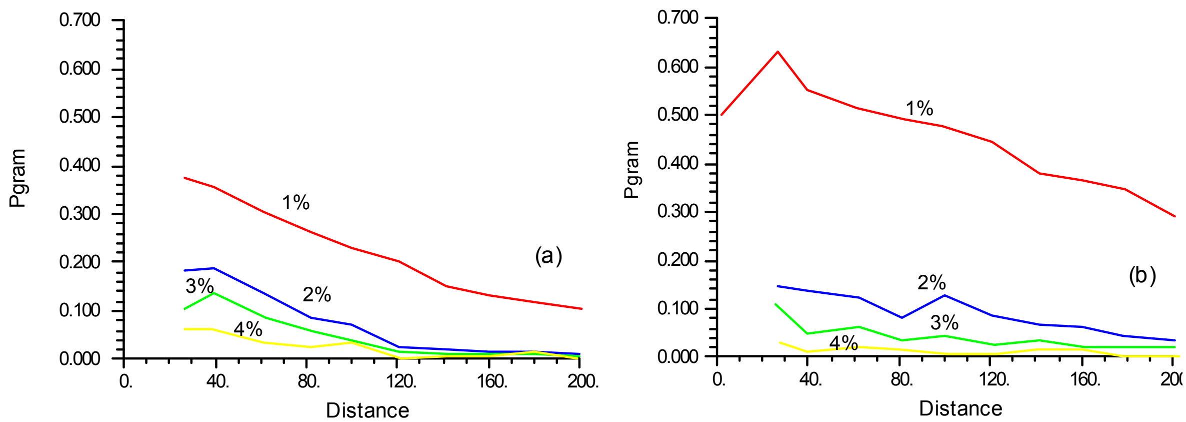


Figure 16.7: Area 2/118 - Continuity of High Grade Assays at Different Thresholds: (left) Zone L, (right) Zone M

Estimation Parameters

The selection of the search radii was guided by modelled ranges from variograms and was established to estimate a large portion of the blocks within the modelled area with limited extrapolation. The parameters were established by conducting repeated test resource estimates and reviewing the results as a series of plan views and sections (see Tables 16.6 and 16.7). As mentioned in the previous section, high grade assays were only used during the estimation process if they were found within a much smaller high grade ellipsoid of 40 x 30 x 15 m size.

Table 16.6: Area 2 - Estimation Parameters

Parameters	J	K	L	M	N	O	P	Q
Rotated Search X (m)	90	90	100	100	70	70	90	100
Rotated Search Y (m)	50	65	70	50	100	100	90	70
Rotated Search Z (m)	30	30	30	30	30	30	30	30
Min data	4	4	4	4	4	4	4	4
Max data	16	16	16	16	16	16	16	16
Max number of samples per dh	4	4	4	4	4	4	4	4
Minimum number of octants	1	1	1	1	1	1	1	1
Minimum number of holes	1	1	1	1	1	1	1	1

Table 16.7: Area 118 - Estimation Parameters

Parameters	J118	K118	L118	M118	N118	O118	P118	Q118	R118
Rotated Search X (m)	100	90	70	70	30	100	100	30	70
Rotated Search Y (m)	70	60	100	90	100	70	70	100	100
Rotated Search Z (m)	30	30	30	30	70	30	30	70	30
Min data	4	4	4	4	4	4	4	4	4
Max data	16	16	16	16	16	16	16	16	16
Max number of samples per dh	4	4	4	4	4	4	4	4	4
Minimum number of octants	1	1	1	1	1	1	1	1	1
Minimum number of holes	1	1	1	1	1	1	1	1	1

Specific Gravity Estimation

There is sufficient variation in specific gravity data (Figure 16.8) to warrant estimating specific gravity into the block model. For the estimation, all specific gravity ("SG") values lower than 2.4 were adjusted to 2.4 and all very high values were capped at 3.2. Block specific gravity values were estimated by the inverse squared distance method. At least eight samples within a 200 x 200 x 50 m radius were needed to estimate a block.

All un-estimated blocks in mineralized domains were assigned average SG values within those domains.

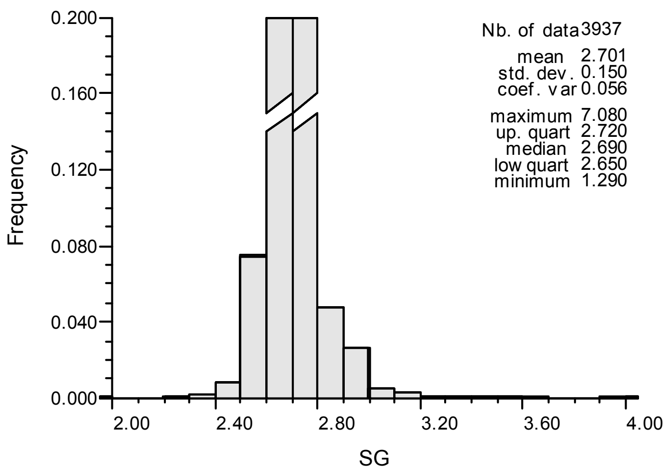


Figure 16.8: Ridgetop – Continuity of High Grade Assays at Different Thresholds: (a) in Zone R100, (b) Zone R140

Resource Validation

Most of the dollar value of the Area 2/118 deposit is in copper (approx 85%). Therefore, the validation was limited to the Cu block estimates. The deposits were validated by completing a series of visual inspections and by:

- Comparison of local "well-informed" block grades with composites contained within those blocks;
- Comparison of average assay grades with average block estimates along different directions – swath plots.

Figure 16.9 shows a comparison of estimated Cu block grades with drill hole assay composite data contained within those blocks in the Area 2 L and M zones. On average, the estimated blocks are very similar to the composite data, with high correlation between the estimates and the assays.

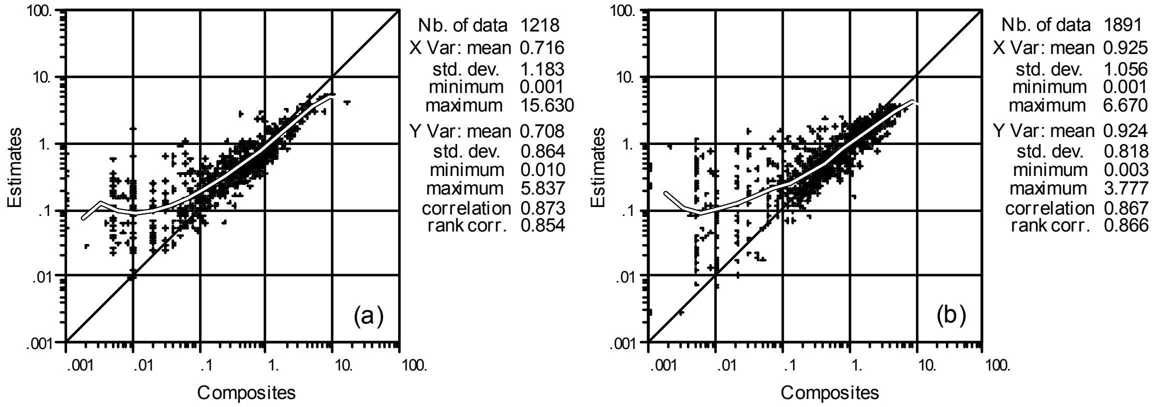


Figure 16.9: Area 2/118 - Comparison of Cu Block Estimates with Composite Assay Data Contained Within the Blocks in (a) L zone (b) M zone

As a final check, average composite grades and average block estimates were compared along different directions. This involved calculating de-clustered average composite grades and comparing them with average block estimates along east-west, north-south and horizontal swaths.

Figure 16.10 shows the swath plots from the Area 2 M zone. Here, and similarly in other zones, the average Cu composite grades and the average Cu estimated block grades are quite similar in all directions. Overall, the validation shows that current resource estimates are excellent reflection of drill hole assay data.

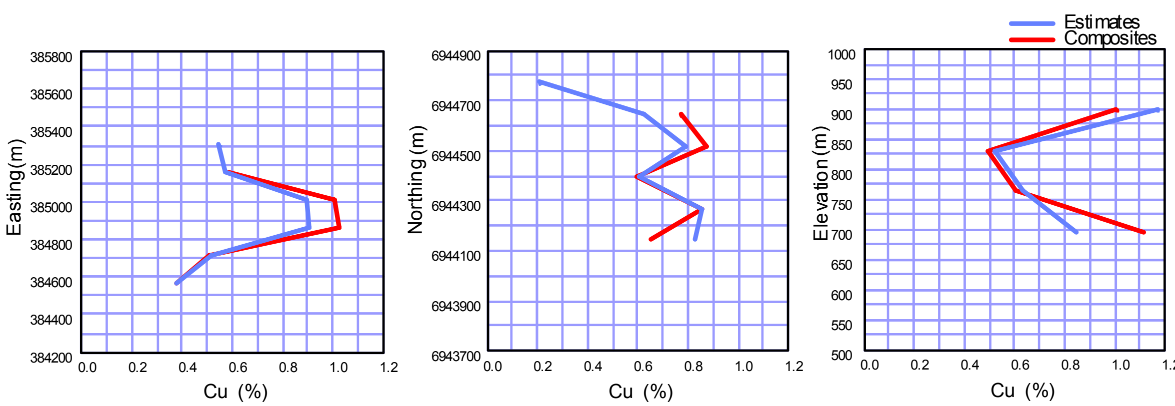


Figure 16.10: Area 2/118 - Declustered Average Cu Composite Grades Compared to Cu Block Estimates in the M zone

Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. There is insufficient information in this early stage of study to assess the extent to which the mineral resources will be affected by these factors that are more suitably assessed in a conceptual study.

Mineral Resources for the Area 2/118 deposit was classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by Dr. Wayne Barnett, Ph.D., Pr.Sci.Nat., an “independent competent person” as defined by National Instrument 43-101.

Drill hole spacing in Area 2/118 is sufficient for geostatistical analysis and evaluating spatial grade variability. SRK is therefore of the opinion that the amount of sample data is adequate to demonstrate very good confidence of the grade estimates in both deposits.

The estimated blocks were classified according to:

- Confidence in interpretation of the mineralized zones;
- Continuity of Cu grades defined from variogram models;
- Number of data used to estimate a block; and
- Average distance to the composites used to estimate a block.

In order to classify mineralization as Measured Mineral Resource, “quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters”. To satisfy this requirement, the following procedure was used to classify blocks as Measured:

- Blocks were flagged as measured if informed from more than 12 composites from more than three separate drill holes and an average distance to the data used to estimate the grade was less than 35 m

In order to classify mineralization as an Indicated Mineral Resource, “the nature, quality, quantity and distribution of data” must be “such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization.” (CIM Definition Standards on Mineral Resources and Mineral Reserves, December 2005) To satisfy this requirement, the following procedure was used to classify blocks as Indicated:

- Blocks were flagged as indicated if informed from more than 8 composites from three or more separate drill holes and if an average distance to the data used to estimate the grade was more than 35 m and less than 60 m
- Final broad areas of measured and indicated resources were designed from classification envelopes encompassing blocks flagged for the measured and indicated categories. This approach ensured consistent definition of the areas assigned to measured and indicated

categories, thereby removing small, discontinuous clusters of blocks assigned to those categories. All estimated block grades not assigned to either measured or indicated category were given an inferred resource category.

Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 16.8 shows global quantities and grade in the Area 2/118 deposit at different Cu cut-off grades. Resource tabulation is limited to a Whittle shell with slope angles of 50 degrees using 10x10x3 m block model. The reader is cautioned that these values should not be misconstrued as a mineral resource. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade. Grade tonnage curves for different resource categories are presented in Figure 16.11 and Figure 16.12.

Table 16.8: Area 2/118 - Sensitivity Analysis of Global Tonnage and Grades Deposit at Various Cu Cut-off Grades

Classification	Cut-Off (Cu%)	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Cu (000's lbs)*	Contained Gold (000's oz)*	Contained Ag (000's oz)*
Measured (M)	>2.0	1,127	2.60	1.17	9.58	64,466	42.3	347
	>1.5	2,232	2.18	0.93	7.76	107,010	66.8	557
	>1.0	3,684	1.80	0.74	6.32	146,524	87.5	749
	>0.5	7,043	1.28	0.49	4.40	198,344	110.0	996
	>0.4	8,394	1.14	0.42	3.91	211,706	114.5	1,056
	>0.3	10,103	1.01	0.36	3.42	224,803	118.4	1,111
	>0.2	12,811	0.85	0.30	2.85	239,560	122.5	1,175
	>0.1	17,500	0.66	0.22	2.20	254,459	125.9	1,238
Indicated (I)	>2.0	771	2.66	0.98	10.67	45,200	24.4	264
	>1.5	2,065	2.07	0.74	7.99	94,127	49.4	530
	>1.0	5,284	1.54	0.55	5.87	179,582	93.7	996
	>0.5	19,411	0.92	0.30	3.32	393,939	185.5	2,071
	>0.4	26,946	0.79	0.24	2.82	468,165	211.7	2,444
	>0.3	37,108	0.67	0.20	2.38	546,269	237.4	2,843
	>0.2	50,160	0.56	0.16	1.99	617,201	258.1	3,216
	>0.1	70,984	0.44	0.12	1.58	683,436	276.6	3,596
Total (M+I)**	>2.0	1,898	2.62	1.09	10.02	109,666	66.7	611
	>1.5	4,296	2.12	0.84	7.87	201,137	116.2	1,087
	>1.0	8,968	1.65	0.63	6.05	326,106	181.2	1,745
	>0.5	26,454	1.02	0.35	3.61	592,283	295.5	3,066
	>0.4	35,340	0.87	0.29	3.08	679,870	326.1	3,500
	>0.3	47,211	0.74	0.23	2.60	771,073	355.8	3,954
	>0.2	62,971	0.62	0.19	2.17	856,761	380.6	4,391
	>0.1	88,484	0.48	0.14	1.70	937,895	402.5	4,834
Inferred	>2.0	17	2.20	0.89	9.03	833	0.5	5
	>1.5	421	1.68	0.69	6.70	15,642	9.4	91
	>1.0	1,263	1.41	0.58	5.48	39,181	23.7	222
	>0.5	5,573	0.83	0.26	2.89	101,519	46.6	518
	>0.4	8,347	0.70	0.20	2.43	128,837	54.4	653
	>0.3	12,591	0.58	0.15	1.98	161,555	61.6	802
	>0.2	17,024	0.49	0.12	1.69	185,723	67.9	927
	>0.1	23,500	0.40	0.10	1.38	206,424	72.6	1,045

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

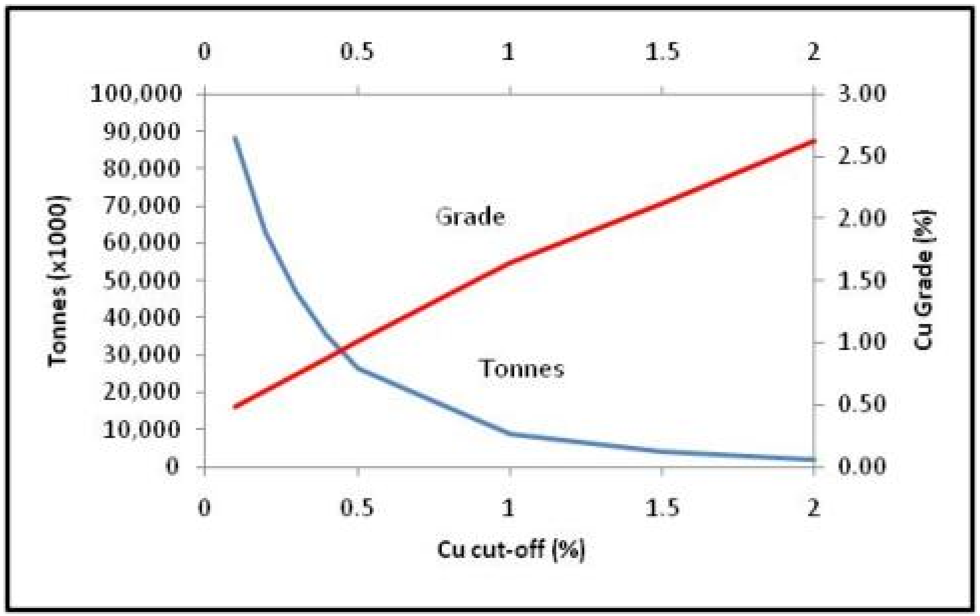


Figure 16.11: Area 2/118 - Cu Grade Tonnage Curve for Measured and Indicated Resources

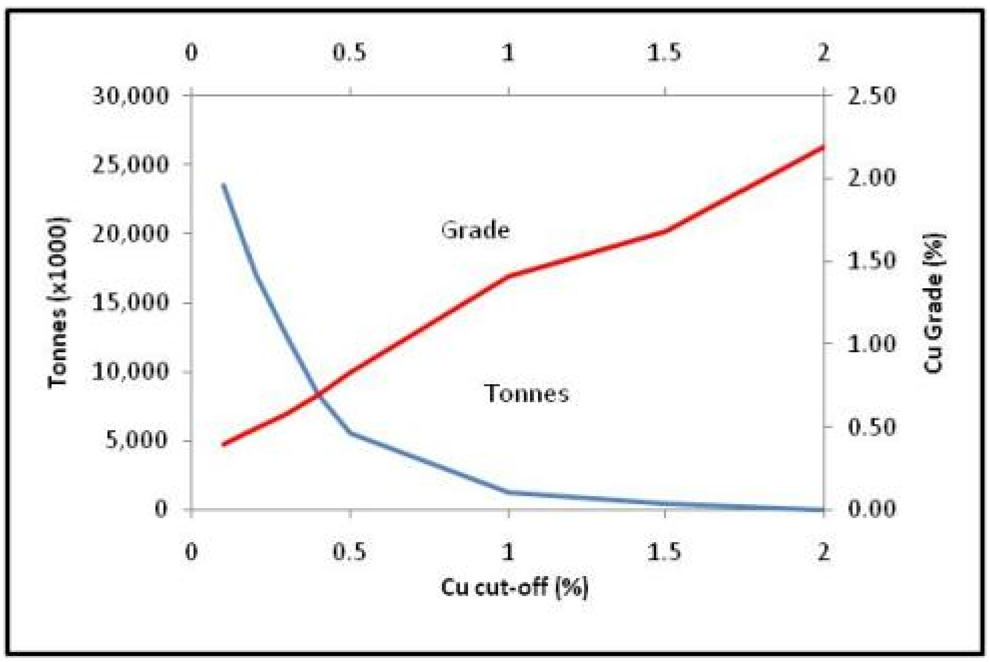


Figure 16.12: Area 2/118 - Cu Grade Tonnage Curve for Inferred Resources

Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

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

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account the likely extraction scenarios and process metal recoveries.

In order to meet this requirement, SRK considers that the Area 2/118 deposit is amenable for open pit extraction.

The open pit mineral resources are reported at a cut-off value of 0.5% Cu per tonne, based on a combined processing and G&A cost of C$5.00 per tonne of material processed and metal prices of US$2.85 per pound for copper, US$900 per ounce gold, and US$12 per ounce silver. The open pit resource is constrained by an optimized Whittle shell based on the NSR model, overall slope angles of 50 degrees and the site operating costs listed above.

Table 16.9 presents the mineral resource statement for the Area 2/118 deposit. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 16.9: Mineral Resource Statement at 0.5% Cu Cut-off for the Area 2/118 Deposit, SRK Consulting August 30, 2010

Classification	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Cu (000's lbs)*	Contained Gold (000's oz)*	Contained Ag (000's oz)*
Measured (M)	7,043	1.28	0.49	4.40	198,344	110	996
Indicated (I)	19,411	0.92	0.30	3.32	393,939	186	2,071
Sub-total (M+I)**	26,454	1.02	0.35	3.61	592,283	296	3,066
Inferred	5,573	0.83	0.26	2.89	101,519	47	518

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

16.4 Ridgetop Deposit

Geology Model

The Ridgetop deposit consists of seven mineralized foliated granodiorite zones. As in the case of Area 2/118 deposit, these zones are generally shallow dipping, on average 24^0 towards the northeast. The same geometrical characteristics are evident for this deposit as for Area 2/118, and the same geological understanding applies. However, the zones have undergone gentle folding along N-S trending axes. At least one synformal and one antiformal axis can be identified from the wireframe interpolation. In addition, the zones in this deposit get progressively steeper towards the north, apparently reaching a dip of 70^o within 15 m from the northeastern boundary limit of the modelled deposit. It is believed that the northeastern boundary is controlled by a northwest striking fault, and the ore zones are dragged downwards towards this fault zone. The exact position, orientation and properties of the fault zone have not been identified yet.

The Ridgetop resource model was created using a commercial three-dimensional block modelling and mine planning software, GEMS version 6.2 (Gemcom®). The model was created in metric units using the mine's local co-ordinate system (UTM NAD83 zone 8). The solids were considered hard boundaries where grades were not allowed into blocks outside of these solids.

The seven mineralized horizons were assigned the following domain codes based on those codes historically used by mine geologists; 80, 90, 100, 110, 120, 140 and 160. The process of identifying and naming the zones was done by importing and reviewing the sectional interpretation provided by MintoEx geologists. Minor modifications to contacts and zone orientations allowed simplification and enhanced continuity of the zones in places. There are therefore fewer interpreted and modelled zones than identified during the exploration program. Table 16.10 includes a list of the domain coding assigned to the drill data and the block model. Figure 16.13 is a three dimensional view of the zone solids, showing their domain codes.

Table 16.10: Ridgetop Modelled Domain Names and Block Model Codes

Domain Name	Block Model Code	Comments
R80	82	Chalcocite partial oxidation
R90	93	Chalcocite partial oxidation
R100	103	Chalcocite dominant zone. Primary ore-bearing zone.
R110	110	Chalcopyrite dominant zone. Primary ore-bearing zone.
R120	120	
R140	140	
R160	160	
Waste	200	Non-mineralized granodiorite
Conglomerate (Cng)	300	Cretaceous aged erosion surface, removing ore zones
Overburden (OB)	500	Unconsolidated waste material
Air	0	

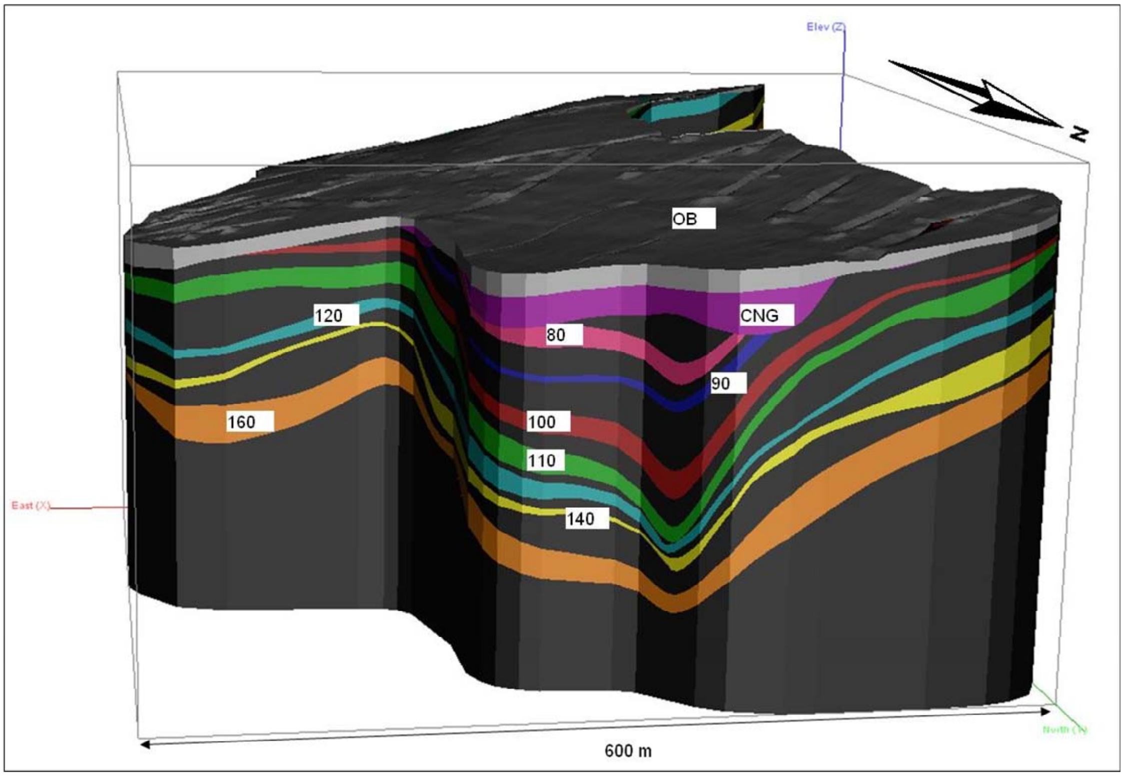


Figure 16.13: View South of the Modelled Ridgetop Mineralized and Waste Domains

The solids were then used to assign the domain and block model codes to the drill hole data (assays and composites) and the block model cells. Blocks above the topography surface were tagged as Air and the blocks outside of the zone solids were tagged as Waste. A Cretaceous conglomerate is developed on the northeastern side of the deposit. It gets rapidly thicker towards the northeast and is presumably strongly influenced by the bounding fault zone. A small amount of conglomerate was included in the model. There is also unconsolidated material near surface, which is included in the model as Overburden.

SRK reviewed the tagged assay, composite and block data on sections and in 3D, and in exported text files using external customized software to ensure the process had worked properly.

To assess how well the modelled solids differentiate between lower and higher grade mineralization, grades on either sides of the modelled contacts were queried and listed. Any anomalous assay values were checked in 3D to determine whether the problems are errors or not. There were far fewer of such anomalous assay values than for Area 2, primarily because the holes are more recent and logged to consistent standards.

Data

A total of 6,193 grade measurements have been used in the design of mineralized domains from holes drilled roughly at 20 m spacing in the North-West and 40 m spacing in the South-East portions of the deposit. Approximately 27% of the samples within the modelled domains were collected from 1.5 m intervals (Figure 16.14). Similarly to the Area 2/118 deposit, the assays were composited to 1.5 m lengths. Shorter composite lengths ensured that most relevant, undiluted assays were included in the resource assessment. Within the mineralized domains 5,833 composite assays were produced from 163 holes. The average thickness of the high grade mineralized horizons is 12 m and in lower grade mineralized horizons is 10 m.

Statistics of polygonally declustered 1.5 m Cu composites within each mineralized zone are presented in Figure 16.15. Statistics of the 1.5 m Au and Ag composites within each mineralized zone are given in an Appendix A.

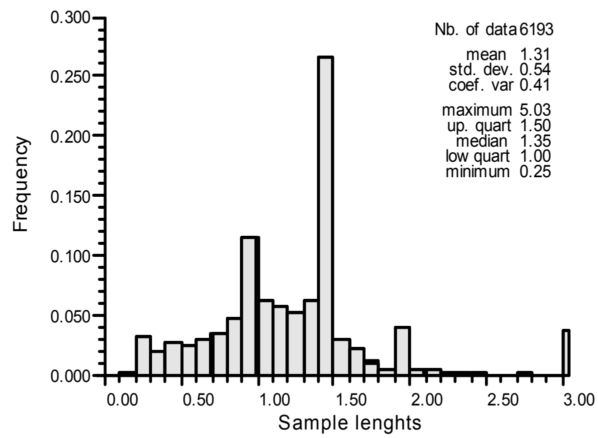
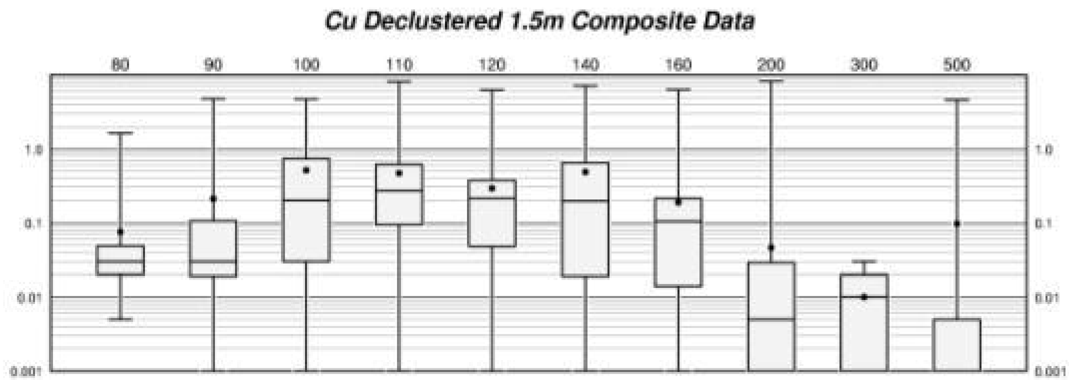


Figure 16.14: Ridgetop - Histogram of Sample Lengths



	80	90	100	110	120	140	160	200	300	500	
Number of data	169	262	837	1301	1048	1112	1219	7626	133	689	Number of data
Mean	0.075	0.211	0.516	0.47	0.294	0.487	0.189	0.046	0.01	0.097	Mean
Std. Dev.	0.177	0.53	0.742	0.598	0.388	0.755	0.318	0.141	0.009	0.346	Std. Dev.
Coef. of Var.	2.37	2.511	1.438	1.271	1.319	1.55	1.68	3.098	0.893	3.587	Coef. of Var.
Maximum	1.636	4.797	4.72	8.132	6.23	7.128	6.282	8.284	0.03	4.61	Maximum
Upper quartile	0.049	0.106	0.738	0.617	0.38	0.65	0.214	0.029	0.02	0.005	Upper quartile
Median	0.03	0.03	0.2	0.272	0.213	0.196	0.104	0.005	0.01	0.001	Median
Lower quartile	0.02	0.019	0.03	0.094	0.048	0.019	0.014	0.001	0.001	0.001	Lower quartile
Minimum	0.005	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	Minimum

Figure 16.15: Ridgetop - Basic Statistics of Declustered Cu Composite Grades

Figure 16.16 shows bivariate statistics of the Cu and Au assays in two higher grade domains. Note moderate degree of correlation in the 100 domain and high degree of rank correlation in the 110 domain, indicated by a regression curve (white thick line) showing a general tendency of increased Cu assays for higher Au assays. This positive correlation lead to a design of variogram models along identical major directions of continuity for both Cu and Au grades.

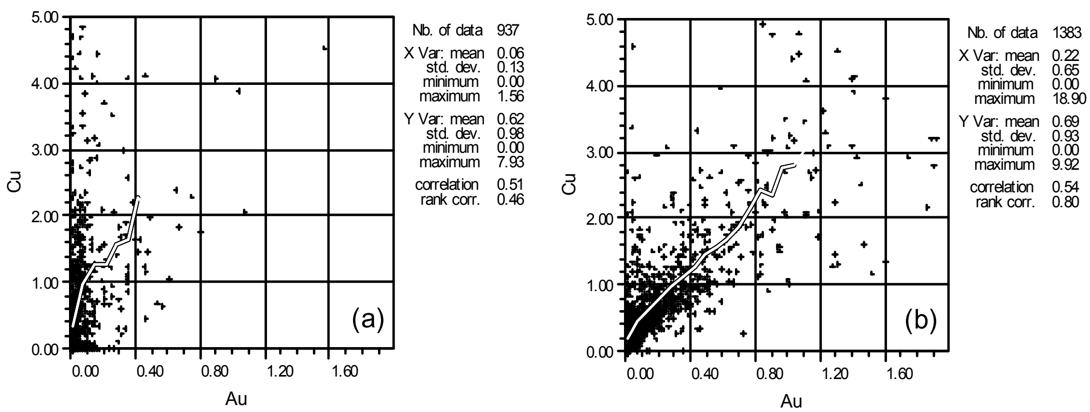


Figure 16.16: Ridgetop - Bivariate Statistics of Cu and Au Assays in (a) 100 domain, (b) 110 domain

Evaluation of Extreme Assay Values

Block grade estimates may be unduly affected by very high grade assays. Therefore, the assay data were evaluated for the high grades outliers. An analysis of the high grade assays indicates relatively strong negative correlation between the assay data and the sample lengths (see Figure 16.17). This suggests that sampling was based on visual indications of mineralization. In view of the above, as in Area 2/118, no capping was done before assay compositing to 1.5 m lengths.

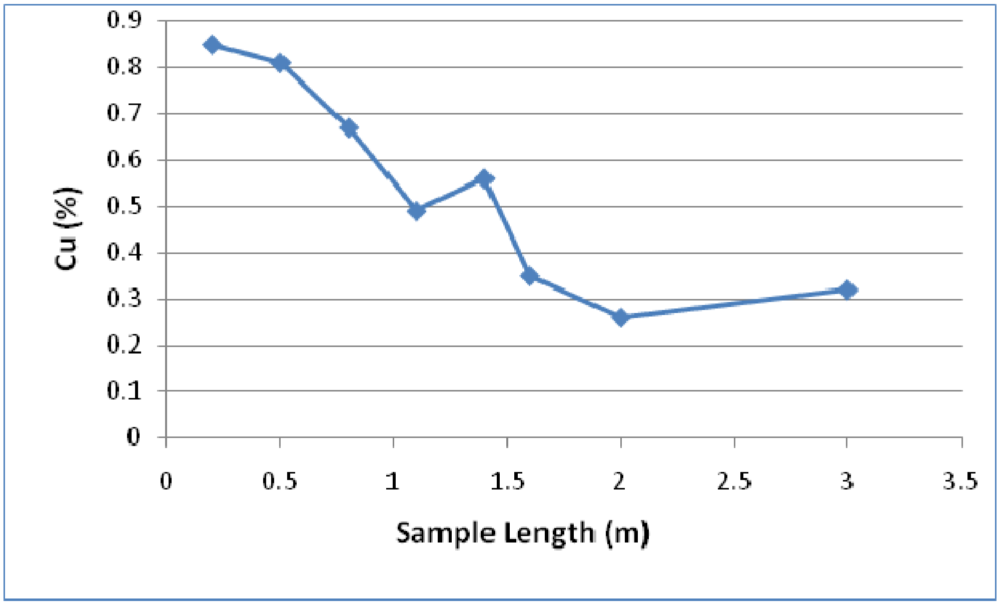


Figure 16.17: Ridgetop - Grade Variation with the Sample Length

Variogram Analysis

Experimental variograms and variogram models in the form of correlograms were generated for Cu and Au grades. The nugget effect values (i.e., metal variability at very close distance) were established from down hole variograms. The nugget values range from 5 to 25 percent of the total sill. Cu variogram models used for grade estimation are summarised in Table 16.11. Note that no variogram models were designed for Ag grades. The Ag was estimated by the inversed distance squared method.

Table 16.11: Ridgetop Cu Exponential Variogram Models

Domain	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
80*	0.05	0.75	50	24	-48	25	40	15
		0.20				300	80	20
90*	0.05	0.75	50	24	-48	25	40	15
		0.20				300	80	20
100	0.05	0.75	50	24	-48	25	40	15
		0.20				300	80	20
110	0.05	0.60	50	24	-45	30	35	16
		0.35				180	500	35
120	0.20	0.50	50	24	-48	50	20	10
		0.30				200	80	12
140	0.05	0.75	50	24	-45	55	65	15
		0.20				85	550	20
160	0.10	0.55	50	24	-48	15	75	15
		0.35				220	150	40

** Variogram models assigned from Domain 100*

Exponential variogram models have been used with practical ranges of continuity

Resource Estimation Methodology

The geometrical parameters of the block model are summarised in Table 16.12.

Table 16.12: Specifications for the Ridgetop Block Model

Description	Easting (Xm)	Northing (Ym)	Elevation (Zm)
Block Model Origin (Lower left corner)	384,650	6,943,200	595
Parent Block Dimension	10	10	3
Number of Blocks	90	90	135
Rotation	0	0	0

All 1.5 m composite assays were coded by modelled mineralized domains. Blocks in a mineralized domain were estimated only from the assays within that domain. Ordinary kriging was used to estimate Cu and Au grades and Inverse Squared Distance weighting to estimate Ag grades.

Treatment of High Grade Composite Grades

As in Area 2/118, instead of capping the composites for high grade assays, SRK elected to limit the influence of the high grade intersections during the estimation process. Figure 16.18 shows the continuity of high grade assays at different thresholds. High grade continuities can be indicated up to a distance where plotted curves roughly level off. For example, at 4% threshold maximum distance at which the continuity could be shown is roughly 40 m.

For grade estimation in all mineralized zones high grade assays were only used if they were found within search ellipsoid of 40 x 30 x 15 m size. High grade thresholds were defined from statistical analysis, separately for each domain. The direction of the high grade search ellipsoid was aligned with the overall direction of grade continuity in each zone.

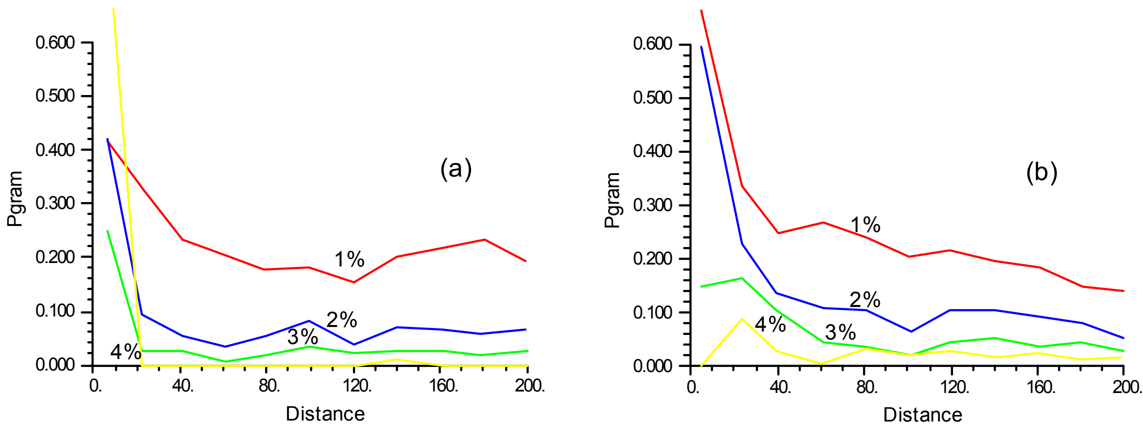


Figure 16.18: Ridgetop – Continuity of High Grade Assays at Different Thresholds: (left) in Zone R100, (right) Zone R140

Estimation Parameters

The selection of the search radii was guided by modelled ranges from variograms and was established to estimate a large portion of the blocks within the modelled area with limited extrapolation. The parameters were established by conducting repeated test resource estimates and reviewing the results as a series of plan views and sections (see Table 16.13).

Table 16.13: Ridgetop Estimation Parameters

Parameters	80	90	100	110	120	140 Step 1	140 Step 2	160
Rotated X (m)	60	60	70	35	70	35	50	60
Rotated Y (m)	35	35	40	60	40	60	80	60
Rotated Z (m)	20	20	25	20	25	20	30	20
Min data	4	4	3	4	3	3	3	4
Max data	16	16	16	16	16	16	16	16
Max number of samples per dh	4	4	4	4	4	4	4	4
Minimum number of octants	1	1	1	1	1	1	1	1
Minimum number of holes	1	1	1	1	1	1	1	1

Specific Gravity Estimation

There is sufficient variation in specific gravity data (Figure 16.19) to warrant estimating specific gravity into the block model. For the estimation, 11 very high SG values were capped. Block specific gravity values were estimated by the ID2 method. At least eight samples within a 120 x 120 x 40 m radius were needed to estimate a block.

All un-estimated blocks in mineralized domains were assigned average SG values within those domains.

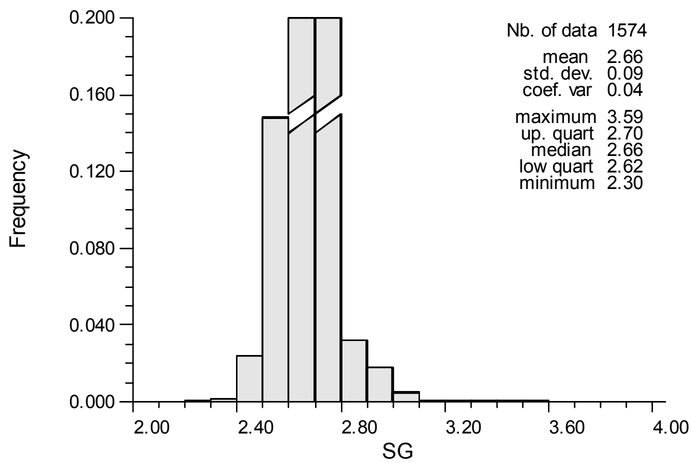


Figure 16.19: Ridgetop - Distribution of SG Values in the Mineralized Domains

Resource Validation

Most of the dollar value in the Ridgetop deposit is in copper (approx 90%). Therefore, the validation was limited to the Cu block estimates. The deposits were validated by completing a series of visual inspections and by:

- Comparison of local “well-informed” block grades with composites contained within those blocks;
- Comparison of average assay grades with average block estimates along different directions – swath plots.

Figure 16.20 shows a comparison of estimated Cu block grades with drill hole assay composite data contained within those blocks. On average, the estimated blocks are similar to the composite data, with good correlation between the estimates and the assays.

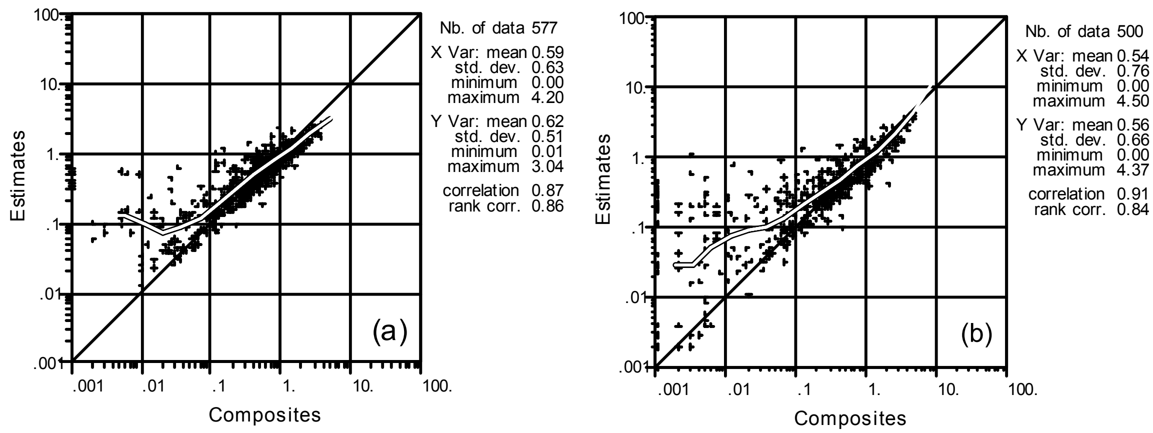


Figure 16.20: Ridgetop - Comparison of Cu Block Estimates with Composite Assay Data Contained Within the Blocks: (a) 110 domain, (b) 140 domain

As a final check, average composite grades and average block estimates were compared along different directions. This involved calculating de-clustered average composite grades and comparing them with average block estimates along east-west, north-south and horizontal swaths

Figure 16.21 shows the swath plots from the 140 zone. Here, and similarly in other zones, the average Cu composite grades and the average Cu estimated block grades are quite similar in all directions. Overall, the validation shows that current resource estimates are very good reflection of drill hole assay data.

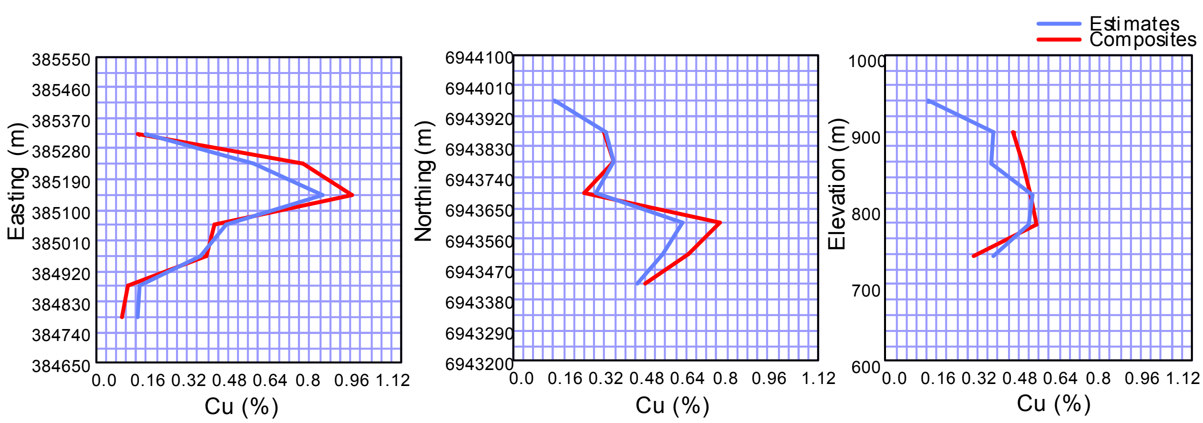


Figure 16.21: Ridgetop - Declustered Average Cu Composite Grades Compared to Cu Block Estimates in the 140 domain

Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. There is insufficient information in this early stage of study to assess the extent to which the mineral resources will be affected by these factors that are more suitably assessed in a conceptual study.

Mineral Resources for the Ridgetop deposit were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by Dr. Wayne Barnett, Ph.D., Pr.Sci.Nat., an "independent competent person" as defined by National Instrument 43-101.

Drill hole spacing at Ridgetop is sufficient for geostatistical analysis and evaluating spatial grade variability. SRK is therefore of the opinion that the amount of sample data is adequate to demonstrate good confidence of the grade estimates in the deposit.

The estimated blocks were classified according to:

- Confidence in interpretation of the mineralized zones;
- Continuity of Cu grades defined from variogram models;
- Number of data used to estimate a block; and
- Average distance to the composites used to estimate a block.

In order to classify mineralization as Measured Mineral Resource, "quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters". To satisfy this requirement, the following procedure was used to classify blocks as Measured:

- Blocks were flagged as measured if informed from more than 12 composites from more than three separate drill holes and an average distance to the data used to estimate the grade was less than 35 m

In order to classify mineralization as an Indicated Mineral Resource, "the nature, quality, quantity and distribution of data" must be "such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization." (CIM Definition Standards on Mineral Resources and Mineral Reserves, December 2005) To satisfy this requirement, the following procedure was used to classify blocks as Indicated:

- Blocks were flagged as indicated if informed from more than 8 composites from three or more separate drill holes and if an average distance to the data used to estimate the grade was more than 35 m and less than 60 m

Final broad areas of measured and indicated resources were designed from classification envelopes encompassing blocks flagged for the measured and indicated categories. This approach ensured consistent definition of the areas assigned to measured and indicated categories, thereby removing small, discontinuous clusters of blocks assigned to those categories. All estimated block grades not assigned to either measured or indicated category were given an inferred resource category.

Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 16.14 shows global quantities and grade in the Ridgetop deposit at different Cu cut-off grades. Resource tabulation is limited to a Whittle shell with slope angles of 50 degrees using 10x10x3 m block model. The reader is cautioned that these values should not be misconstrued as a mineral resource. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade. Grade tonnage curves for different resource categories are presented in Figure 16.22 and Figure 16.23.

Table 16.14: Ridgetop Sensitivity Analysis of Global Tonnage and Grades in the Ridgetop Deposit at Various Cu Cut-off Grades

Classification	Cut-Off (Cu%)	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Cu (000's lbs)*	Contained Gold (000's oz)*	Contained Ag (000's oz)*
Measured (M)	>2.0	55	2.36	0.69	4.40	2,874	1.2	8
	>1.5	187	1.90	0.54	3.66	7,816	3.3	22
	>1.0	555	1.45	0.39	2.91	17,731	7.0	52
	>0.5	1,531	0.98	0.25	2.14	33,204	12.3	105
	>0.4	1,821	0.90	0.22	1.97	36,083	13.1	116
	>0.3	2,150	0.81	0.20	1.80	38,607	13.7	124
	>0.2	2,472	0.74	0.18	1.66	40,381	14.2	132
	>0.1	2,836	0.67	0.16	1.51	41,588	14.6	138
Indicated (I)	>2.0	114	2.53	1.66	12.35	6,380	6.1	45
	>1.5	324	1.99	1.07	8.41	14,235	11.2	88
	>1.0	841	1.51	0.68	5.70	28,055	18.5	154
	>0.5	3,534	0.87	0.30	2.87	67,901	33.8	326
	>0.4	4,965	0.75	0.24	2.41	82,039	38.4	384
	>0.3	6,905	0.64	0.19	2.00	96,892	42.7	445
	>0.2	9,896	0.52	0.15	1.62	113,142	46.9	514
	>0.1	13,015	0.43	0.12	1.34	123,595	49.9	562
Sub-total (M+I)**	>2.0	170	2.48	1.35	9.76	9,254	7.3	53
	>1.5	511	1.96	0.88	6.67	22,051	14.4	110
	>1.0	1,396	1.49	0.57	4.59	45,786	25.5	206
	>0.5	5,064	0.91	0.28	2.65	101,104	46.2	431
	>0.4	6,786	0.79	0.24	2.29	118,122	51.6	500
	>0.3	9,054	0.68	0.19	1.96	135,499	56.4	569
	>0.2	12,369	0.56	0.15	1.62	153,523	61.0	646
	>0.1	15,851	0.47	0.13	1.37	165,183	64.5	700
Inferred	>2.0	0	0.00	0.00	0.00	0	0.0	0
	>1.5	4	1.60	0.17	3.59	130	0.0	0
	>1.0	50	1.22	0.07	1.00	1,338	0.1	2
	>0.5	318	0.75	0.13	1.57	5,250	1.3	16
	>0.4	446	0.66	0.12	1.45	6,523	1.7	21
	>0.3	597	0.58	0.10	1.29	7,695	2.0	25
	>0.2	798	0.50	0.09	1.12	8,769	2.2	29
	>0.1	1,057	0.42	0.07	0.95	9,667	2.4	32

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

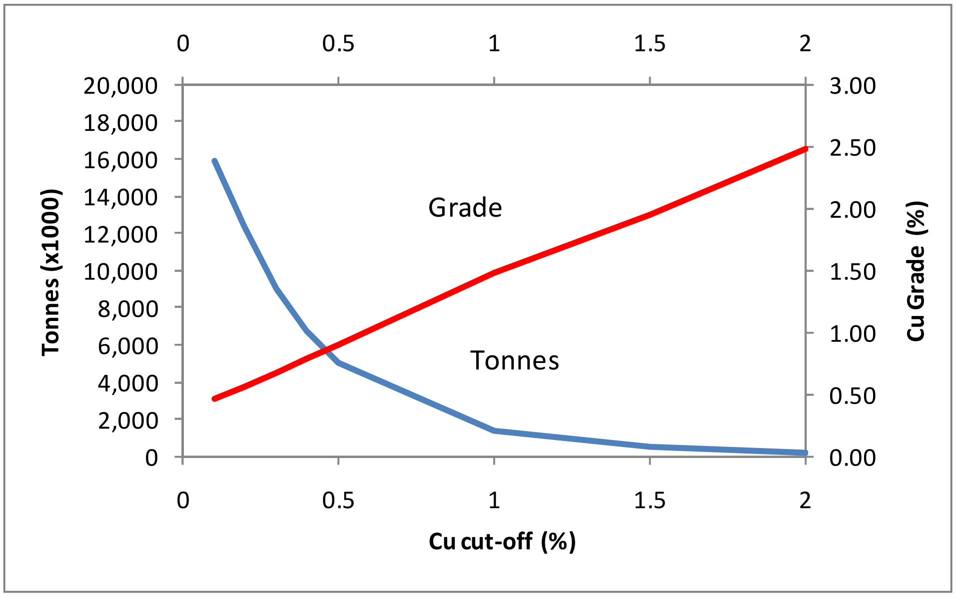


Figure 16.22: Ridgetop - Cu Grade Tonnage Curve for Measured and Indicated Resources

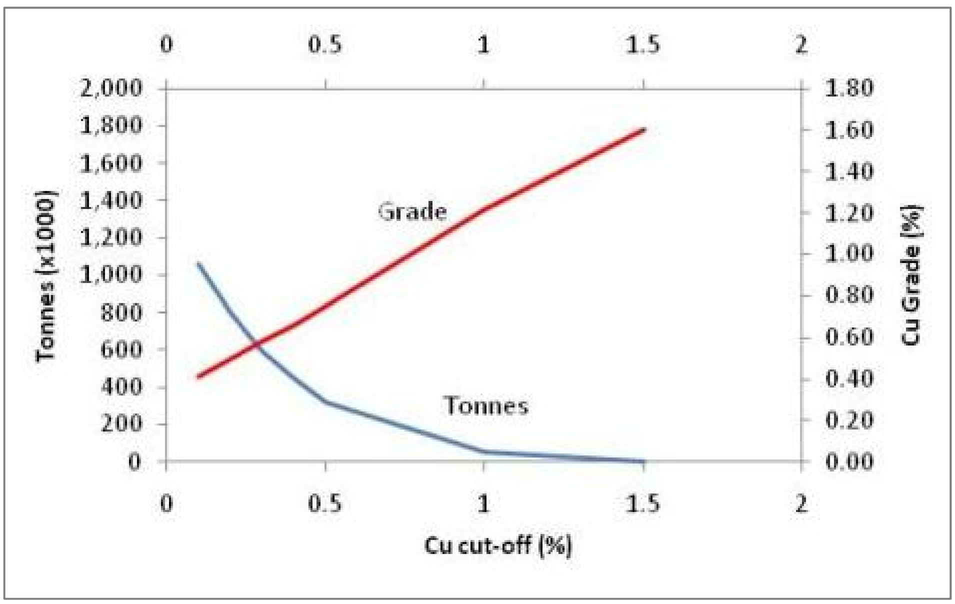


Figure 16.23: Ridgetop - Cu Grade Tonnage Curve for Inferred Resources

Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

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

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account the likely extraction scenarios and process metal recoveries.

In order to meet this requirement, SRK considers that the Ridgetop deposit is amenable for open pit extraction.

The open pit mineral resources are reported at a cut-off value of 0.5% Cu per tonne, based on a combined processing and G&A cost of C$5.00 per tonne of material processed and metal prices of US$2.85 per pound for copper, US$900 per ounce gold, and US$12 per ounce silver. The open pit resource is constrained by an optimized Whittle shell based on the NSR model, overall slope angles of 50 degrees and the site operating costs listed above.

Table 16.15 presents the mineral resource statement for the Ridgetop deposits.

Table 16.15: Mineral Resource Statement at 0.5% Cu Cut-off for the Ridgetop Deposit, SRK Consulting August 30, 2010

Classification	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Cu (000's lbs)*	Contained Gold (000's oz)*	Contained Ag (000's oz)*
Measured (M)	1,531	0.98	0.25	2.14	33,204	12.3	105
Indicated (I)	3,534	0.87	0.30	2.87	67,901	33.8	326
Sub-total (M+I)**	5,064	0.91	0.28	2.65	101,104	46.2	431
Inferred	318	0.75	0.13	1.57	5,250	1.3	16

Rounded to nearest thousand* *****Totals may not add exactly due to rounding*

16.5 Minto North Deposit

The Minto North deposit is a new discovery made in early 2009 and comprises near surface, higher grade copper-gold mineralization. In June 2009, the first mineral resource estimate for the Minto North deposit, using a 0.5% copper cut-off, was estimated (Table 16.16) and presented in the Capstone Press Release dated June 9, 2009. The June resource was based on 31 drill holes. Solids were created based on mineralized intersections and used to constrain the interpolation of grades.

Subsequently, additional 56 drill holes were drilled from June through September 2009 as part of an in-fill and delineation program. The goal of this program was to better define the ore boundaries and constraining solids and upgrade indicated and inferred resources to measured and indicated. The resultant resource estimate is detailed and reported in the following sections.

Table 16.16: Tonnage & Grade Estimates of the Minto North Deposit Reported in June 2009

Classification	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Cu (000's lbs)*	Contained Gold (000's oz)*	Contained Silver (000's oz)*
Measured (M)	-	-	-	-	-	-	-
Indicated (I)	1,237	2.49	1.86	9.7	67,853	74	385
Sub-total (M+I)**	1,237	2.49	1.86	9.7	67,853	74	385
Additional Inferred	634	1.88	1.03	6.4	26,318	21	130

Geology Model

A solid model of the 115, 120 and 130 ore zones within the Minto North Deposit was created from sections and based on a combination of lithology, copper grades and site knowledge (see Figure 16.24). It is important to note that the 2009 drilling resulted in new insights into the mineralization and grade distribution which greatly assisted in the creation of the solids. The ore zone solids were used for constraining the interpolation procedure. In addition, a large cross-cutting dyke that transects the deposit and the zones was also modelled using sectional interpretations and subsequently utilized to mask out the estimated tonnage related to this barren unit.

Every intersection was inspected and the solids were then manually adjusted to match exactly the interval intercepts. Once the solids models were created, they were used to code the drill hole assays and composites for subsequent geostatistical analysis. For the purpose of the resource model, the solid zone was utilized to constrain the block model by matching assays to those within the zones in a process called geologic matching so that only composites that lie within a particular zone are used to only interpolate the blocks within that zone. The orientation and ranges (distances) utilized for search ellipsoids used in the estimation process were derived from strike and dip of the mineralized zone, site knowledge and on-site observations by MintoEx's geological staff.

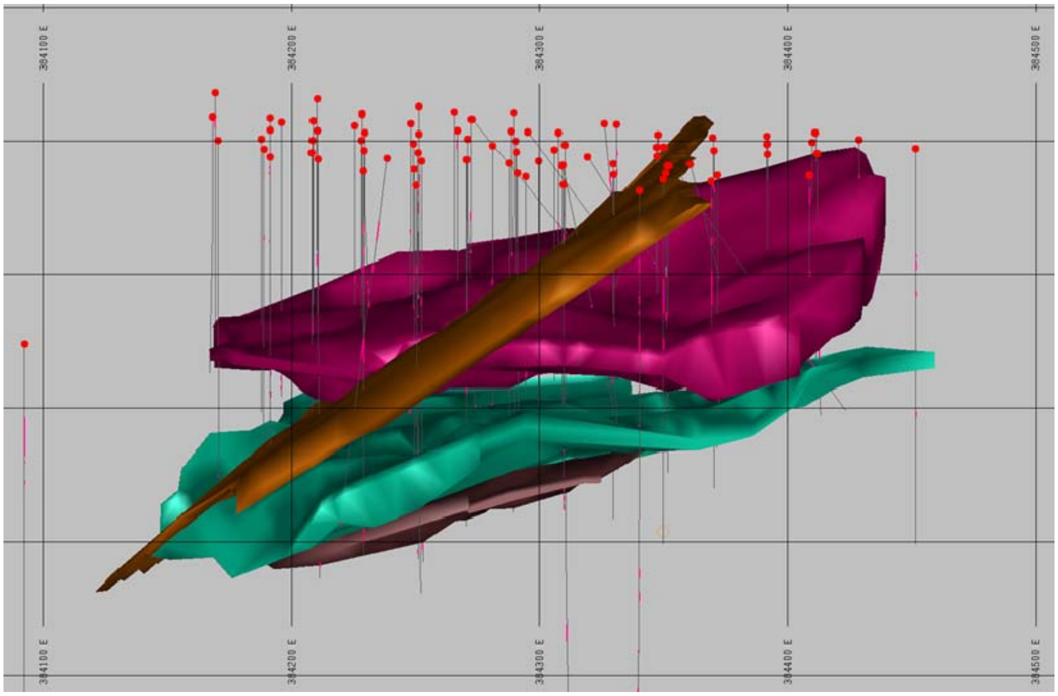


Figure 16.24: View from the North of the Modelled Minto North Mineralized Domains

Data

The drill hole database was supplied in electronic format by MintoEx. This included collars, down hole surveys, lithology data and assay data (i.e. Au g/t, Cu%, Ag g/t, SG with down hole from and to intervals in metric units. The database was numerically coded by mineralized zone solid; 115 Zone Ore = 115, 120 Zone Ore = 120, 130 Zone Ore = 130 and Waste = 8. The database was then manually adjusted drill hole by drill hole to insure accuracy of zonal intercepts.

Table 16.17 and Figure 16.25 show statistics of copper assays weighted by assay intervals. Statistics of gold and silver assays have been given in Appendix A. The highest by far average Cu, Au, and Ag grades are found in zone 115 (2.12%, 1.15 g/t, 7.62 g/t respectively). Note that the overall average grades from all three mineralized domains are higher than in Area 2/118 and at Ridgetop deposits.

Table 16.17: Minto North – Statistics for Copper Assays Weighted by Assay Interval

CU	Length	Min	Max	Mean	1st Quartile	Median	3rd Quartile	SD	CV
115	1,637.0	0.00	39.60	2.12	0.69	1.37	2.56	2.79	1.32
120	651.8	0.00	13.85	0.33	0.06	0.14	0.34	0.86	2.62
130	124.6	0.00	2.07	0.26	0.02	0.14	0.30	0.37	1.43
Total	2,413.4	0.00	39.60	1.54	0.22	0.89	1.92	2.49	1.62
All	**4,943.5**	**0.00**	**39.60**	**0.77**	**0.02**	**0.02**	**0.81**	**1.90**	**2.48**

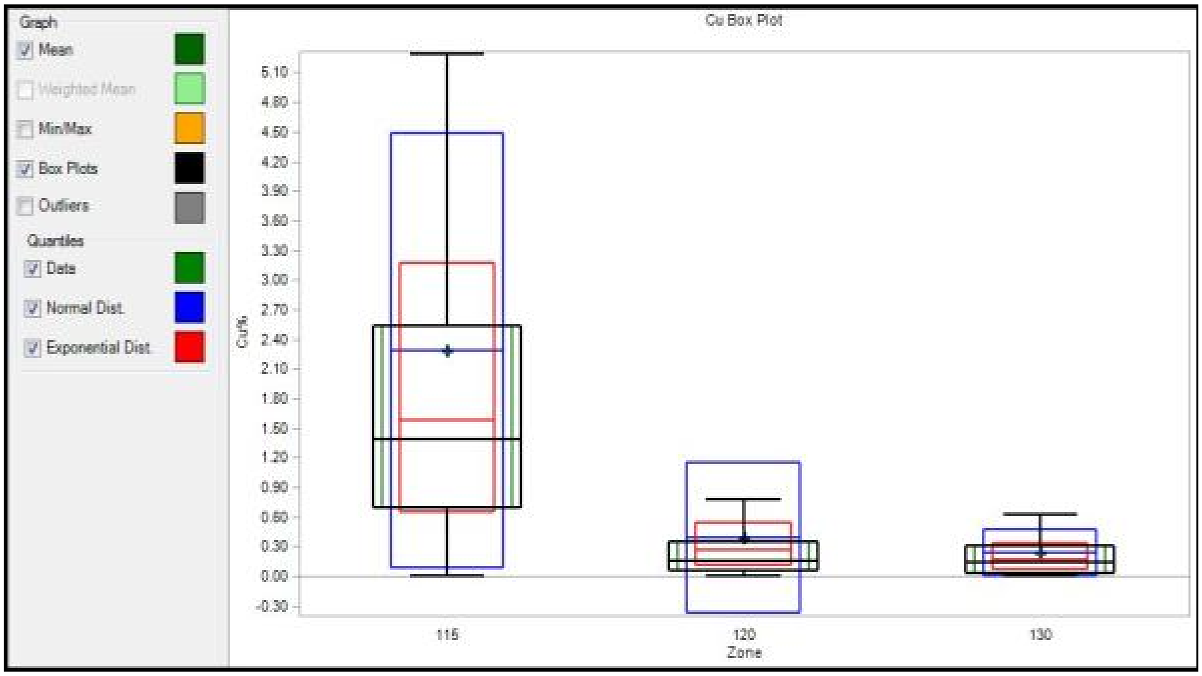


Figure 16.25: Minto North - Basic statistics of Cu assay grades in the mineralized zones

Composites

It was determined that the 1.5 m composite lengths offered the best balance between supplying common support for samples and minimizing the smoothing of the grades in addition to reducing the undue influence of very high grades. Table 16.18 and Figure 16.26 show the basic statistics for the 1.5 m Cu composite grades within the mineralized domains. Statistics of the Au and Ag composites are presented in Appendix A.

Table 16.18: Minto North - Composite Statistics Weighted by Length

CU	Length	Min	Max	Mean	1st Quartile	Median	3rd Quartile	SD	COV
115	1,637.0	0.00	27.41	2.12	0.84	1.39	2.40	2.40	1.13
120	651.8	0.00	7.83	0.33	0.07	0.15	0.32	0.69	2.12
130	124.6	0.00	1.56	0.26	0.07	0.15	0.32	0.32	1.26
Total	**2,413.4**	**0.00**	**27.41**	**1.54**	**0.26**	**0.92**	**1.85**	**2.18**	**1.41**
All	**4,943.5**	**0.00**	**27.41**	**0.77**	**0.01**	**0.04**	**0.84**	**1.70**	**2.22**

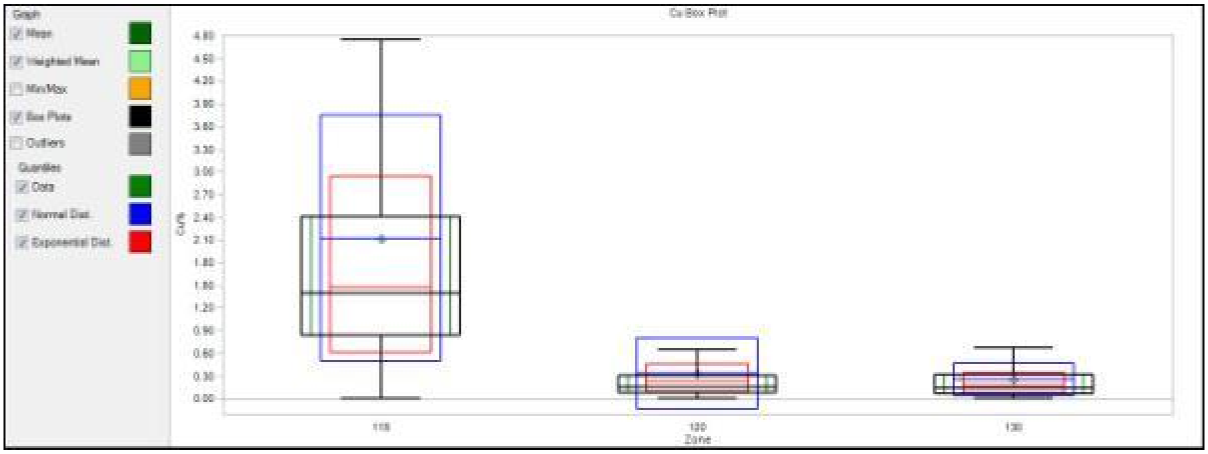


Figure 16.26: Minto North – Basic Statistics of Cu Composite grades in the mineralized zones

Evaluation of Extreme Assay Values

During the estimation process in Zone 115 influence of assays higher than 11% Cu, 50 g/t Ag, and 5 g/t Au has been quite limited. Similarly, in Zone 120 the same restriction was applied to assays higher than 1.2% Cu, 15 g/t Ag, and 2 g/t Au. There are no very high grades in the 130 Zone, therefore, during the estimation process there was no restriction on high grade influence in that zone. The range at which to limit grades greater than the high grade assay cutoff was chosen to be 40 x 30 x 7 m oriented at 165 degrees in the major axis and 0 degrees dip. In other words, composite grades greater than the threshold amounts would not be used in the estimation of blocks if those high grade composites are outside the respective distance from that block. It is important to emphasize that the method employed for this study was not to cut the high grade outliers but to limit their influence.

Specific Gravity Estimation

A total of 2,711 bulk specific gravity (SG) measurements were provided by MintoEx of which 1,422 are within the mineralized solids. The SG's in the mineralized solids ranged from a low of 2.07 to a high of 4.56 with a mean value of 2.71, standard deviation of 0.14 and CV of 0.05 illustrating a very tight distribution. The SG values were interpolated into the blocks using the inverse distance to the second power interpolator. At least 4 samples within a 100 x 100 x 25 m radius were needed to estimate a block. Values greater than 3.3 were limited to a 20 m radius in influence.

Variography

Experimental variograms and variogram models in the form of correlograms were generated for Cu, Au and Ag grades. The nugget effect values (i.e., metal variability at very close distance) were established from down hole variograms. The nugget values range from 15 to 22 percent of the total sill. Cu, Au and Ag variogram models used for grade estimation are summarised in Table 16.19. Note that the rotations of the angles are given according to the GSLIB convention used by MineSight™ Compass.

Table 16.19: Minto North - 115 Zone Variogram Model

Parameter	Cu			Au			Ag		
Nugget (C0)	0.15			0.22			0.14		
C1	0.85			0.78			0.86		
	Range	Rotation	Angle	Range	Rotation	Angle	Range	Rotation	Angle
Major	70	R1	166	60	R1	37	80	R1	115
Minor	60	R2	-1	30	R2	-11	60	R2	20
Vertical	7	R3	-28	37	R3	12	10	R3	-16

Note; R1 is the rotation around the Z axis, R2 is the rotation around the X axis with counter-clock wise being positive and R3 is the rotation around the Y axis with clock-wise being positive.

Block Model Definition

The Block Model used for calculating the resources was defined according to the limits specified in Table 16.20. The block model is orthogonal and non-rotated reflecting the orientation of the deposit. The block size chosen was 10 x 10 x 3 m, roughly reflecting drill hole spacing (i.e. 1 – 2 blocks between drill holes) which are at approximately 15 to 20 m centers and a proposed 3 m bench height.

Table 16.20: Specifications for the Minto North Block Model

Description	Easting (Xm)	Northing (Ym)	Elevation (Zm)
Block Model Origin	384,000	6,945,750	750
Block Dimension	10	10	3
Number of Blocks	60	50	80
Rotation	0	0	0

Resource Estimation Methodology

The estimation plan includes the following items:

- Mineralized zone code and percentage of modelled mineralization in each block;
- Estimated bulk specific gravity based on an inverse distance squared method;
- Estimated block Cu, Au, and Ag grades by ordinary kriging, using a two pass estimation strategy for all mineralized zones. The two estimation passes enabled better description of local metal grades.

For the 115 Zone, major direction of continuity of the Cu grades was the ellipsoid direction chosen for the estimation process was chosen to be 165 degrees azimuth and 0 degrees dip for the major axis, 285 degrees azimuth and 0 degrees dip for the minor axis and 0 degrees azimuth and 90 degrees dip for the vertical axis. This direction follows the general orientation of the modelled 115 Zone. For the 120 and 130 Zones, the ellipsoid direction chosen for the estimation process was same as for the 115 Zone. Table 16.21 summarizes the search ellipse dimensions for the estimation passes.

Table 16.21: Minto North Search Ellipse Parameters for 115, 120 and 130 Zones

Pass	Major Axis	Semi-Major Axis	Minor Axis	1st Rotation Angle Azimuth	2nd Rotation Angle Dip	3rd Rotation Angle	Min. No. Of Comps	Max. No. Of Comps	Max. Samples per Drill hole
1	70	60	10	165	0	0	4	16	4
2	40	30	7	165	0	0	4	16	4

Resource Validation

A graphical validation was done on the block model. This graphical validation serves several purposes:

- Checks the reasonableness of the estimated grades, based on the estimation plan and the nearby composites;
- Checks that the general drift and the local grade trends compared to the drift and local grade trends of the composites;
- Ensures that all blocks in the core of the deposit have been estimated;
- Checks that topography has been properly accounted for;
- Checks against manual approximate estimates of tonnage to determine reasonableness; and
- Inspection and explanation for potentially high grade block estimates in the neighbourhood of the extremely high assays.

A full set of cross sections, long sections and plans were used to check the block model on the computer screen, showing the block grades and the composites. No evidence of any block being wrongly estimated was found; it appears that every block grade could be explained as a function of the surrounding composites, the variogram model used, and the estimation plan applied.

These validation techniques included the following:

- Visual inspections on a section-by-section and plan-by-plan basis;
- The use of Grade Tonnage Curves;
- Swath Plots comparing kriged estimated block grades with inverse distance and nearest neighbour estimates;
- An inspection of histograms of distance of closest samples to the estimated blocks, average distance to blocks for all composites used in the estimation which gives a quantitative measure

of confidence that blocks are adequately informed in addition to assisting in the classification of resources; and

- Analysis of Relative Variability Index, which quantifies variability and relative error on a block-by block basis within the deposit in addition to assisting with the classification of resources.

Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. There is insufficient information in this early stage of study to assess the extent to which the mineral resources will be affected by these factors that are more suitably assessed in a conceptual study.

Mineral Resources for the Minto North deposit were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by Garth Kirkham, P.Geo., an "independent competent person" as defined by National Instrument 43-101.

Drill hole spacing in Minto North deposit is sufficient for geostatistical analysis and evaluating spatial grade variability. Kirkham Geosystems is therefore of the opinion that the amount of sample data is adequate to demonstrate very good confidence of the grade estimates in the deposit.

The estimated blocks were classified according to:

- Confidence in interpretation of the mineralized zones;
- Continuity of Cu grades defined from variogram models;
- Number of data used to estimate a block;
- Number of composites allowed per drill hole;
- Distance to nearest composite used to estimate a block;
- Average distance to the composites used to estimate a block; and
- An evaluation of relative error on a block by block basis.

The classification of resources was based primarily upon distance to nearest composite however all of the quantitative measures, as listed above were inspected and taken into consideration. In addition, the classification of resources for each zone was considered individually by virtue of their relative depth from surface and the ability to derive meaningful geostatistical results.

For the 115 Zone, measured blocks were determined to have a block to nearest composite of 30 meters. In addition, the blocks were inspected for average distance to composite which was less than 40 meters, minimum number of drill holes which was 3 however in cases where the minimum number of drill holes was less than 3 then the distance to composite, average distance to composite, number of composites and error were evaluated to insure that confidence in the categorization of resources was warranted. Indicated blocks were determined to have a distance to composite greater than 30 meter however there were no blocks that exceeded 50 meters.

In addition, the number of drill holes, average distance to block from composite and the number of composites used along with relative error, were evaluated to ensure confidence.

For the 120 zone, the same criteria was employed however resources categorized for the indicated category were determined to have a block to nearest composite of 30 meters. In addition, the blocks were inspected for average distance to composite which was less than 40 meters, minimum number of drill holes was in most cases 2 however in cases where the minimum number of drill holes was less than 2 then the distance to composite, average distance to composite, number of composites and error were evaluated to insure that confidence in the categorization of resources was upheld. Inferred blocks were determined to be have a distance to composite greater than 30 meter however there were no block that exceeded 50 meters. In addition, the number of drill holes, average distance to block from composite, number of composites used along with relative error was evaluated.

For the 130 Zone, although the zone has demonstrated geological continuity, it does not have demonstrated geostatistical continuity by virtue of the relatively low number of data points available and the relatively small footprint of the zone. Therefore, the 130 zone is categorized as inferred at this time.

Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 16.22 and 16.20 shows global quantities and grade in the Ridgetop deposit at different Cu cut-off grades. The reader is cautioned that these values should not be misconstrued as a mineral resource. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade. Cu grade tonnage curves for different resource categories are presented in Figure 16.27 and Figure 16.28.

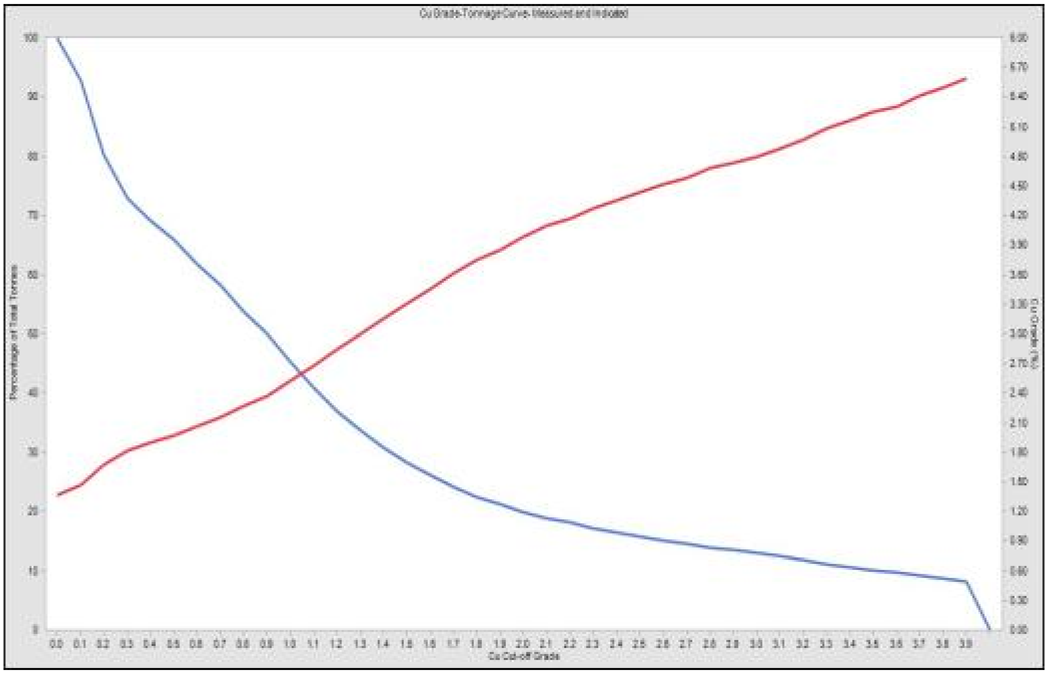


Figure 16.27: Minto North - Cu Grade Tonnage Curve for Measured and Indicated Resources

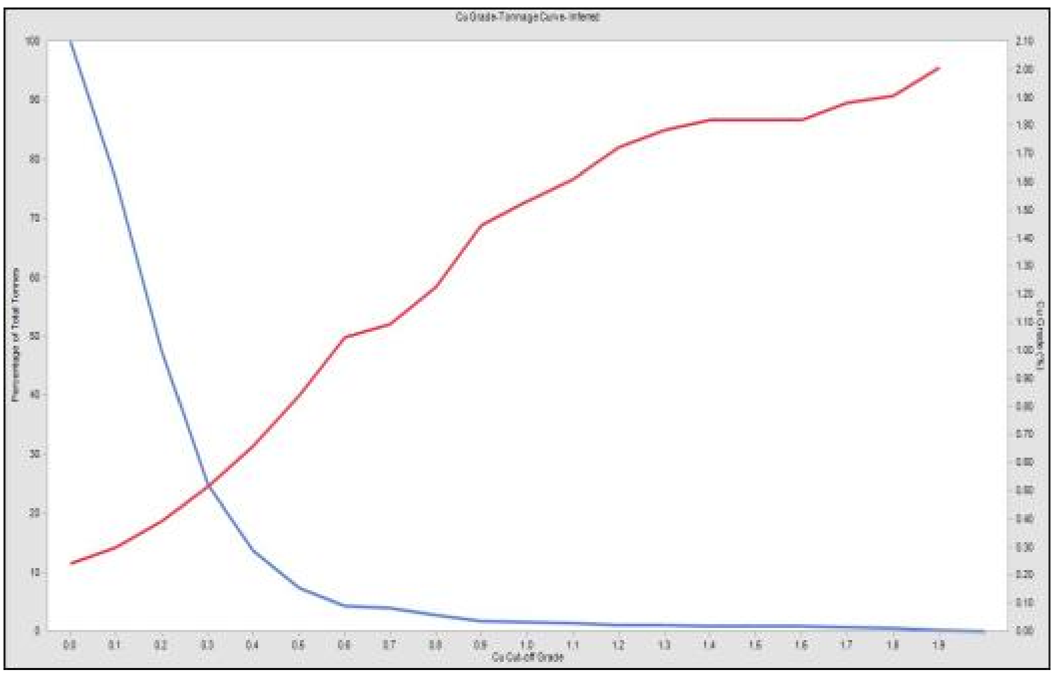


Figure 16.28: Minto North - Cu Grade Tonnage Curve for Inferred Resources

Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

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

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account the likely extraction scenarios and process metal recoveries. It is the opinion of the Qualified Person that the Minto North Deposit, as classified, has a reasonable expectation of economic extraction.

Table 16.22 presents the mineral resource statement for the Minto North deposit.

Table 16.22: Mineral Resource Statement at 0.5% Cu Cut-off for the Minto North Deposit, Kirkham Geosystems December 1, 2009

Classification	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lbs)*	Contained Gold (K oz)*	Contained Silver (K oz)*
Measured (M)	1,844	2.15	1.11	7.7	87,530	66	456
Indicated (I)	264	1.04	0.6	5.76	6,055	5	49
Sub-total (M+I)**	**2,108**	**2.01**	**1.04**	**7.46**	**93,585**	**71**	**505**
Additional Inferred	**25**	**0.84**	**0.40**	**4.4**	**457**	**0**	**3**

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

16.6 Minto East Deposit

The Minto East deposit is also a new discovery made in late 2009 and comprises relatively near surface (i.e. approximately 280 metres below surface), higher grade copper-gold mineralization. This is the first mineral resource estimate for the Minto East deposit and it is reported using a 0.5% and a 1.5% copper cut-off, as tabulated below (Table 16.23; see Capstone Press Release dated June 23, 2010). This resource estimate was based on a total of 33 holes within a very tightly constrained area. A solid was created based on mineralized intersections and used to constrain the interpolation of grades.

Table 16.23: Tonnage & Grade Estimates of the Minto East Deposit Reported in June 2010

Classification	Tonnes	Copper	Gold	Silver	Contained Cu	Contained Gold	Contained Silver
	(000's)*	(%)	(g/t)	(g/t)	(000's lbs)*	(000's oz)*	(000's oz)*
For a 0.5% Cu Cut-off							
Measured (M)	-	-	-	-	-	-	-
Indicated (I)	736	2.35	0.99	6.2	38,132	23	147
Additional Inferred	588	1.73	0.81	5.6	22,426	15	106
For a 1.5% Cu Cut-off							
Measured (M)	-	-	-	-	-	-	-
Indicated (I)	541	2.88	1.16	7.1	34,349	20	123
Additional Inferred	284	2.55	0.95	6.4	15,966	9	58

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

This resource estimate was updated based on the addition of 17 drill holes from the 2010 exploration campaign that were focussed at upgrading the resources for the Minto East Deposit. These new resources are the subject of this section.

Geology Model

A solid model of the 700 ore zone within the Minto East Deposit (Figure 16.29) was created from sections and based on a combination of lithology, copper grades and site knowledge. Although there are a number of mineralized layers intersected within the drill holes, the grades are not sufficient at this time to warrant estimation of the upper zones and the estimation was limited to the 700 zone only. The ore zone solid was used to constrain grade interpolation.

Every intersection was inspected and the solid was then manually adjusted to match the drill intercepts. Once the solid model was created, it was used to code the drill hole assays and composites for subsequent geostatistical analysis. The solid zone was utilized to constrain the block model by matching assays to those within the zones. The orientation and ranges (distances) utilized for search ellipsoids used in the estimation process were derived from strike and dip of the mineralized zone and site knowledge and on-site observations by MintoEx’s geological staff.

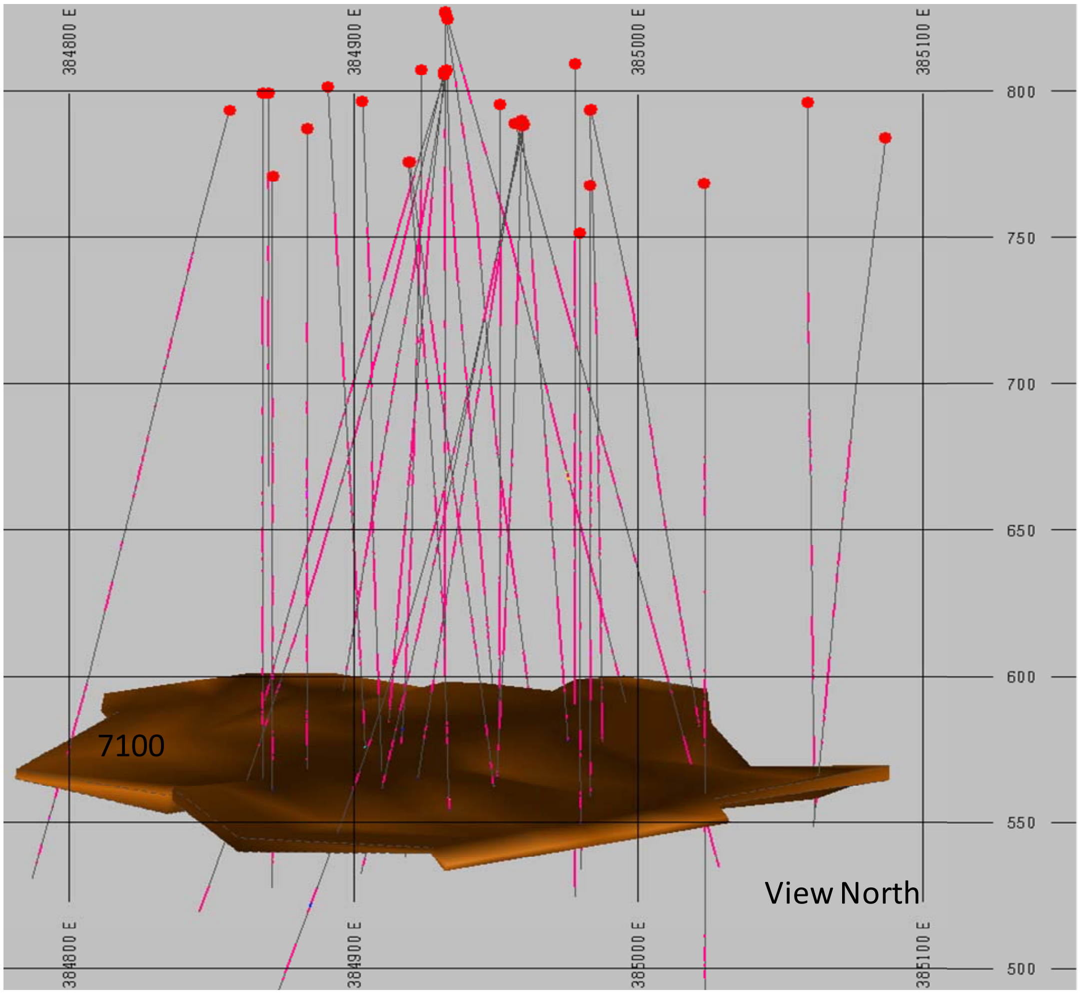


Figure 16.29: View North of the Modelled Minto East Mineralized Domain

Data

The drill hole database was supplied in electronic format by MintoEx. This included collars, down hole surveys, lithology data and assay data (i.e. Au g/t, Cu %, Ag g/t, SG with down hole from and to intervals in metric units). The database was numerically coded by mineralized zone solid; 700 Zone Ore. The database was then manually adjusted drill hole by drill hole to insure accuracy of zonal intercepts.

Table 16.24 shows statistics of copper, gold and silver assays. The average Cu, Au, and Ag grades for the 700 zone are 1.88%, 1.05 g/t, 5.58 g/t, respectively. Note that the overall average grades from the 700 zone is relatively similar to the Minto North (120 zone) deposit which are both higher than in Area 2/118 and the Ridgetop deposits.

Table 16.24: Statistics for Unweighted Copper, Gold and Silver Assays for the Minto East Deposit

700 Zone	#	Min	Max	Mean	1st Quartile	Median	3rd Quartile	SD	CV
Cu	232	0	7.07	1.88	0.09	1.77	2.91	1.78	0.95
Au	232	0	31	1.05	0.02	0.7	1.13	2.96	2.8
Ag	232	0.1	48.8	5.59	0.51	4.6	8.99	5.85	1.05

All Minto East Assays	#	Min	Max	Mean	1st Quartile	Median	3rd Quartile	SD	CV
Cu	4,448	0	7.07	0.18	0	0.03	0.11	0.59	3.34
Au	4,448	0	31	0.07	0.02	0.02	0.02	0.72	9.95
Ag	4,448	0.1	48.8	0.64	0.12	0.22	0.51	1.86	2.9

Note that the table above shows those intervals that intersect the high grade 700 zone only in addition to all of the intervals for those holes. This illustrates the significance of the 700 zone in addition to the high variability outside the zone (in the waste).

Composites

It was determined that the 1.5 m composite lengths offered the best balance between supplying common support for samples and minimizing the smoothing of the grades in addition to reducing the undue influence of very high grades. Distribution of sample lengths within the mineralized domains is presented in Figure 16.30. Table 16.25 shows the basic statistics for the 1.5 m Cu composite grades within the mineralized domains.

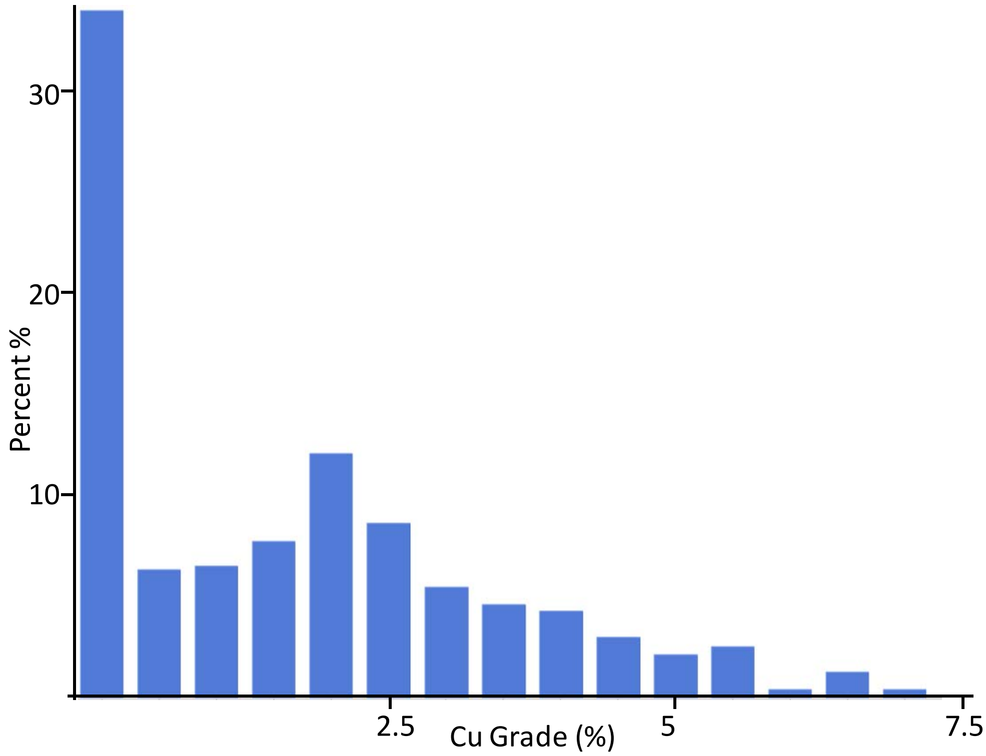


Figure 16.30: Minto East – Histogram of Cu Composite grades in the mineralized zones

Table 16.25: Composite Statistics Weighted by Length

700 ZONE	Length	Min	Max	Mean	1st Quartile	Median	3rd Quartile	SD	CV
700 Zone									
Cu	347.4	0.00	7.07	1.88	0.09	1.77	2.91	1.78	0.94
Au	347.4	0.00	31.00	1.05	0.02	0.70	1.13	2.94	2.81
Ag	347.4	0.10	48.80	5.58	0.51	4.60	8.99	5.83	1.04
All Minto East Assays									
Cu	6,441.9	0.00	7.07	0.18	0.00	0.03	0.11	0.60	3.28
Au	6,441.9	0.00	31.00	0.07	0.02	0.02	0.02	0.72	9.81
Ag	6,441.9	0.10	48.80	0.66	0.12	0.22	0.51	1.89	2.86

Note that the table above shows those intervals that intersect the high grade 700 zone only in addition to all of the intervals for those holes. This illustrates the significance of the 700 zone in addition to the high variability outside the zone (in the waste).

Evaluation of Extreme Assay Values

During the estimation process the influence of composites greater than 5.2% Cu, 13 g/t Ag, and 3 g/t Au have been limited. In the case of the Cu composites, values higher than 5.2% were cut whilst in the case of Ag and Au, grades greater than 13 g/t and 3 g/t, were restricted to a 15 m range. In other words, composite grades greater than the threshold amounts were not used in the estimation of blocks if those high grade composites are outside the respective distance from that block.

Specific Gravity Estimation

A total of 615 in-situ dry bulk density (ISBD) measurements were provided by MintoEx of which 133 are within the mineralized solids. The SG's in the mineralized solids ranged from a low of 2.47 to a high of 3.34 with a mean value of 2.78, standard deviation of 0.15 and CV of 0.05 illustrating a very tight distribution. The SG values were interpolated into the blocks using the inverse distance to the second power. At least 2 samples within a 150 x 150 x 50 m radius were needed to estimate a block.

Variography

Experimental variograms and variogram models in the form of correlograms were generated for Cu, Au and Ag grades. The nugget effect values (i.e., metal variability at very close distance) were established from down hole variograms. The nugget values range from less than 1% to 32% of the total sill. The Cu, Au and Ag variogram models are summarised in Table 16.26. The block model derived from ordinary kriging was useful for comparison purposes but appeared overly smoothed so it was judged by the author that the inverse distance model was a better representation of the grade distribution within the Minto East deposit.

Table 16.26: 700 Zone Variogram Model

	CU			AU			AG		
Nugget (C0)	0.05			0.002			0.321		
C1	0.718			0.91			0.41		
C2	0.232			0.088			0.289		
First Structure									
	Range	**Azim**	**Dip**	**Range**	**Azim**	**Dip**	**Range**	**Azim**	**Dip**
Maximum	26	357	-7	23	281	-1	17	335	-11
Intermediate	16	131	-80	25	11	20	32	49	55
Minimum	10	86	7	3	194	70	3	252	33
Second Structure									
	Range	**Azim**	**Dip**	**Range**	**Azim**	**Dip**	**Range**	**Azim**	**Dip**
Maximum	274	291	54	132	153	72	197	340	-15
Intermediate	78	172	19	27	23	12	20	60	31
Minimum	36	72	29	130	110	-14	50	271	54

Block Model Definition

The Block Model used for estimating the resources was defined according to the limits specified in Table 16.27. The block model is orthogonal and non-rotated reflecting the orientation of the deposit. The block size chosen was 10 m by 10 m by 3 m, roughly reflecting drill hole spacing (i.e. 3 – 4 blocks between drill holes) which are at approximately 30 to 40 m centers and a proposed 3 m bench height.

Table 16.27: Specifications for the Minto East Block Model

Description	Easting (X)	Northing (Y)	Elevation (Z)
Block Model Origin	384,700	6,944,750	3
Block Dimension	10	10	3
Number of Blocks	40	40	171
Rotation	0	0	0

Resource Estimation Methodology

The estimation plan includes the following items:

- Mineralized zone code and percentage of modelled mineralization in each block;
- Estimated bulk specific gravity based on an inverse distance squared method;
- Estimated block Cu, Au, and Ag grades by inverse distance to the second power, using a three estimation pass strategy for the mineralized zone. The three estimation passes enabled better description of local metal grades.

For the 700 Zone, the search ellipsoid was oriented along the major direction of continuity of the Cu grades, 340° azimuth and -15° dip for the major axis, 50° azimuth and 0° dip for the minor axis and 340° azimuth and 75° dip for the vertical axis. Table 16.28 summarizes the search ellipse dimensions for the estimation passes.

Table 16.28: Search Ellipse Parameters for the 700 Zone

Pass	Major Axis	Semi-Major Axis	Minor Axis	1st Rotation Angle Azimuth	2nd Rotation Angle Dip	3rd Rotation Angle	Min. No. Of Comps	Max. No. Of Comps	Max. Samples per Drill hole
1	30	30	10	340	-15	0	6	20	5
2	60	60	20	340	-15	0	6	20	5
3	150	150	50	340	-15	0	2	20	5

Resource Validation

A graphical validation was done on the block model. This graphical validation serves several purposes:

- Checks the reasonableness of the estimated grades, based on the estimation plan and the nearby composites;
- Checks that the general drift and the local grade trends compared to the drift and local grade trends of the composites;
- Ensures that all blocks in the core of the deposit have been estimated;
- Checks that topography has been properly accounted for;
- Checks against manual approximate estimates of tonnage to determine reasonableness; and
- Inspection and explanation for potentially high grade block estimates in the neighbourhood of the extremely high assays.

A full set of cross sections, long sections and plans were used to check the block model on the computer screen, showing the block grades and the composites. No evidence of any block being wrongly estimated was found; it appears that every block grade could be explained as a function of the surrounding composites, the variogram model, and the estimation plan applied.

These validation techniques included the following:

- Visual inspections on a section-by-section and plan-by-plan basis;
- The use of Grade Tonnage Curves;
- Swath Plots comparing Kriged estimated block grades with Inverse Distance and Nearest Neighbour estimates;

- An inspection of histograms of distance of first composite to nearest block, average distance to blocks for all composites used which gives a quantitative measure of confidence that blocks are adequately informed in addition to assisting in the classification of resources; and

Analysis of Relative Variability Index, which quantifies variability and relative error on a block-by block basis within the deposit in addition to assisting with the classification of resources.

Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. There is insufficient information in this early stage of study to assess the extent to which the mineral resources will be affected by these factors that are more suitably assessed in a conceptual study.

Mineral Resources for the Minto East deposit were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by Garth Kirkham, P.Geo., an "independent qualified person" as defined by National Instrument 43-101.

Drill hole spacing in Minto East deposit is sufficient for preliminary geostatistical analysis and evaluating spatial grade variability. Kirkham Geosystems is therefore of the opinion that the amount of sample data is adequate to demonstrate very good confidence of the grade estimates in the deposit.

The estimated blocks were classified according to:

- Confidence in interpretation of the mineralized zones;
- Continuity of Cu grades defined from variogram models;
- Number of data used to estimate a block;
- Number of composites allowed per drill hole;
- Distance to nearest composite used to estimate a block;
- Average distance to the composites used to estimate a block; and
- An evaluation of relative error on a block by block basis.

The classification of resources was based primarily upon distance to nearest composite however all of the quantitative measures, as listed above were inspected and taken into consideration. In addition, the classification of resources for each zone was considered individually by virtue of their relative depth from surface and the ability to derive meaningful geostatistical results.

For the 700 Zone, blocks were classified as measured if they were within 20 m of a composite, had an average distance of all composite used less than 40 m and were interpolated with a minimum of two drill holes. Blocks were classified as indicated if the nearest composite greater than 20 metre but less than 40 metres away. In addition, the number of drill holes used for the estimate, the average distance of all composites used and the number of composites and the relative error were evaluated to ensure confidence. The remaining blocks were classified as inferred, however, all of these blocks were within a maximum of 60 m of the nearest composite and had at least 2 drill holes contributing to the estimate of that block. There were a small percentage of inferred block that were interpolated with one drill hole however these blocks are below cut-off and not reported.

Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 16.29 shows global quantities and grade in the Minto East deposit at different Cu cut-off grades. The reader is cautioned that these values should not be misconstrued as a mineral reserve. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade. Cu grade tonnage curves for different resource categories are presented in Figure 16.31 and 16.32.

Table 16.29: Minto East – Sensitivity analyses of Global Tonnage and Grades Deposit at Various Cu Cut-off Grades

Classification	Cut-Off (Cu%)	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (000's lbs)*	Contained Gold (000's oz)*	Contained Silver (000's oz)*
Measured (M)	2	425	3.03	1.42	7.8	28,382	19	106
	1.5	487	2.87	1.34	7.5	30,766	21	118
	1	562	2.65	1.24	7.1	32,803	22	128
	0.5	688	2.30	1.07	6.3	34,842	24	139
	0.4	713	2.23	1.04	6.1	35,092	24	140
	0.3	725	2.20	1.03	6.0	35,188	24	141
	0.2	732	2.18	1.02	6.0	35,224	24	141
	0.1	745	2.15	1.00	5.9	35,266	24	141
Indicated (I)	2	197	2.69	1.01	6.5	11,663	6	41
	1.5	268	2.44	0.95	6.1	14,401	8	52
	1	344	2.17	0.86	5.5	16,437	10	61
	0.5	490	1.74	0.70	4.6	18,805	11	72
	0.4	526	1.65	0.67	4.4	19,167	11	74
	0.3	568	1.56	0.62	4.1	19,483	11	75
	0.2	627	1.43	0.57	3.8	19,804	12	77
	0.1	671	1.35	0.54	3.6	19,946	12	78
Sub-total (M+I)	2	622	2.92	1.29	7.37	40,045	26	147
	1.5	755	2.71	1.20	7.00	45,167	29	170
	1	906	2.47	1.09	6.47	49,240	32	189
	0.5	1,177	2.07	0.92	5.57	53,647	35	211
	0.4	1,239	1.99	0.88	5.37	54,259	35	214
	0.3	1,293	1.92	0.85	5.19	54,671	35	216
	0.2	1,359	1.84	0.81	4.99	55,028	36	218
	0.1	1,416	1.77	0.78	4.81	55,212	36	219
Additional Inferred	2	-	0	0	0	-	-	-
	1.5	4	1.76	0.78	4.9	151	0	1
	1	5	1.66	0.72	4.6	191	0	1
	0.5	14	1.03	0.45	2.8	316	0	1
	0.4	16	0.96	0.41	2.6	336	0	1
	0.3	21	0.81	0.34	2.2	373	0	1
	0.2	23	0.75	0.31	2.0	385	0	2
	0.1	24	0.74	0.31	2.0	387	0	2

**Rounded to nearest thousand*

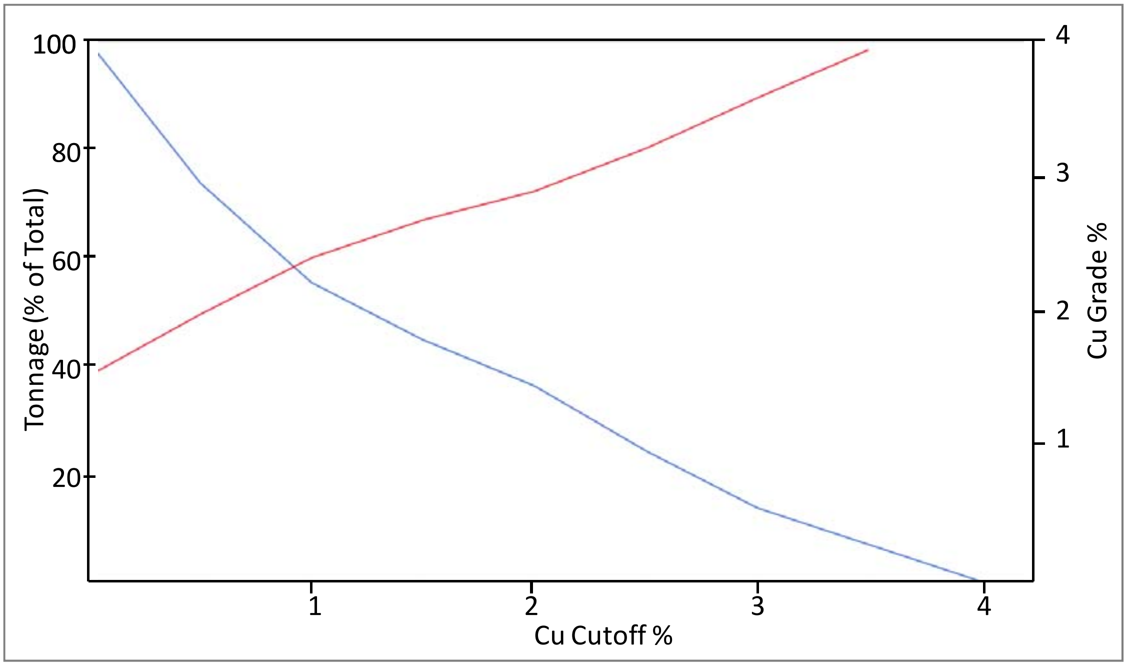


Figure 16.31: Minto East - Cu Grade Tonnage Curve for Measured and Indicated Resources (red- Grade, blue- Tonnes)

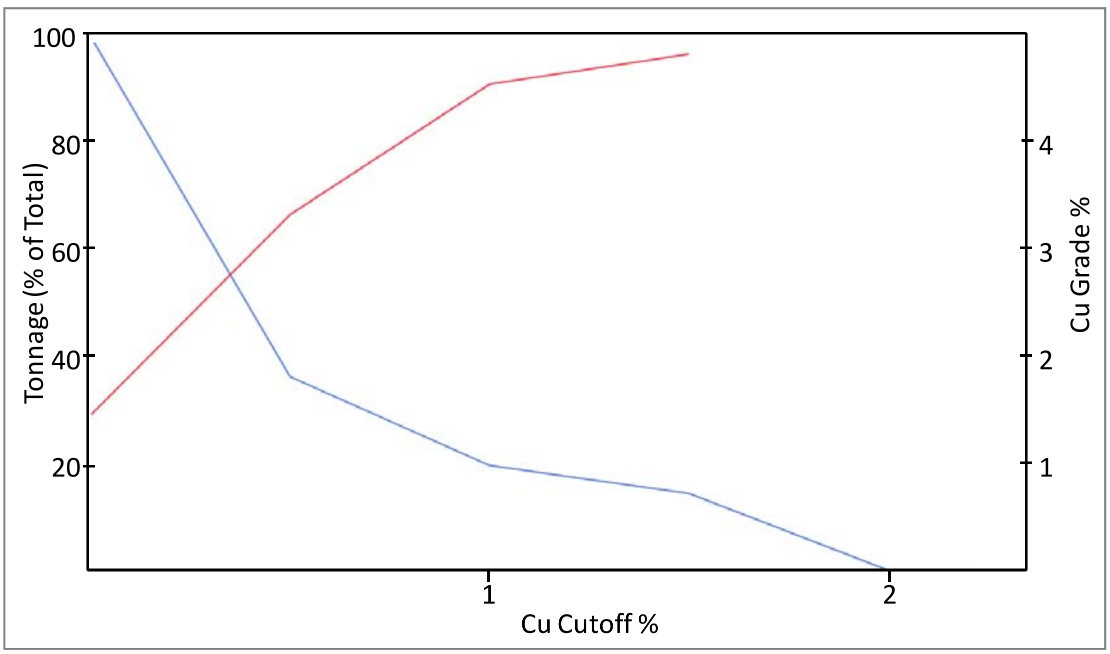


Figure 16.32: Minto East - Cu Grade Tonnage Curve for Inferred Resources (red- Grade, blue- Tonnes)

Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

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

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account the likely extraction scenarios and process metal recoveries. It is the opinion of the Qualified Person that the Minto East Deposit, as classified, has a reasonable expectation of economic extraction.

Table 16.30 presents the mineral resource statement for the Minto East deposit.

Table 16.30: Tonnage & Grade Estimates of the Minto East Deposit Reported by Class in October 2010 (at a 0.5% Cu cut-off)

Classification	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lbs)*	Contained Gold (K oz)*	Contained Silver (K oz)*
Measured (M)	688	2.30	1.07	6.3	34,842	24	139
Indicated (I)	489	1.74	0.70	4.6	18,805	11	72
Sub-total (M+I)**	1,177	2.07	0.92	5.57	53,647	35	211
Additional Inferred	14	1.03	0.45	2.8	316	0	1

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

Table 16.31 presents combined mineral resource at a 0.5% Cu cut-off for Area 2/118, Ridgetop, Minto Main, Minto North and Minto East Deposits. The Minto Main deposit resource has been appropriately reduced to account for all material removed by mining up until December 31, 2010.

Table 16.31: Combined Mineral Resource Statement at 0.5% Cu Cut-off for Area 2/118, Ridgetop, Minto Main, Minto North and Minto East Deposits, December 31, 2010

Classification	Tonnes (Kt)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lbs)*	Contained Gold (K oz)*	Contained Silver (K oz)*
Measured (M)	13,136	1.40	0.57	4.71	406,624	239	1,989
Indicated (I)	24,341	0.93	0.31	3.33	498,917	240	2,604
Sub-total (M+I)**	**37,476**	**1.10**	**0.40**	**3.82**	**905,540**	**479**	**4,592**
Additional Inferred	**5,955**	**0.83**	**0.25**	**2.82**	**107,879**	**48**	**540**

**Rounded to nearest thousand* ***Totals may not add exactly due to rounding*

16.7 Mineral Reserves - Summary

Table 16.32 shows a summary of the Minto reserve estimation as of January 1, 2011.

Table 16.32: Reserve Estimation

Deposit	Reserve Class	Tonnes (Mt)	Cut-off Grade (%Cu equiv.)	Diluted grade			Contained Metal		
				Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Au (koz)	Ag (koz)
Main*	Proven Probable	2.25	0.62	1.35	0.46	4.79	67	33	346
	Sub-total	2.25	0.62	1.35	0.46	4.79	67	33	346
North	Proven	1.52	0.52	2.36	1.28	8.55	79	63	419
	Probable	0.005	0.52	2.25	0.81	9.38	0	0	2
	Sub-total	1,53	0.52	2.36	1.27	8.56	79	63	421
Ridgetop	Proven	0.63	0.54	1.10	0.25	2.05	15	5	41
	Probable	0.71	0.54	1.11	0.37	3.55	17	9	81
	Sub-total	1.34	0.54	1.11	0.32	2.85	33	14	122
Area 2	Proven	3.37	0.54	1.41	0.53	4.94	105	58	536
	Probable	1.45	0.54	1.08	0.31	3.59	35	15	167
	Sub-total	4.82	0.54	1.32	0.47	4.53	140	72	703
118	Proven								
	Probable	0.49	0.54	1.29	0.09	1.73	14	1	27
	Sub-total	0.49	0.54	1.29	0.09	1.73	14	1	27
Under-ground	Proven								
	Probable	2.44	1.20	1.90	0.82	6.71	102	64	527
	Sub-total	2.44	1.20	1.90	0.82	6.71	102	64	527
Total	Proven	7.77	0.56	1.56	0.63	5.37	266	158	1,343
	Probable	5.09	0.86	1.50	0.54	4.91	169	89	804
	Total	12.87	0.68	1.53	0.60	5.19	435	247	2,146

**includes stockpile balance of 1,631 kt at beginning of 2011 for Main pit*

16.8 Mineral Reserves –Open Pit

Net Smelter Model

The 3-D resource models were used as the basis for deriving the economic pit limit for the Phase V pits. These models included the Minto North model, as provided by Kirkham Geosystems, as well as SRK's Area 2, 118 and Ridgetop models, along with remaining ore and stockpiles from the Minto Main deposit, provided by MintoEx based on a forecast of production as of the start of 2011. A number of calculations were performed on the model in order to determine the net smelter return ("NSR") of each individual block. The parameters used in the calculations are summarized in Table 16.33 below.

Table 16.33: NSR Parameters

Metal Prices				Comments
Metal prices (US$)	US	$2.25	/lb Cu	C$2.62/lb
	US	$300	/oz Au	as per Silver Wheaton agreement
	US	$3.90	/oz Ag	as per Silver Wheaton agreement
Exchange Rate				**Comments**
US Dollars/Canadian Dollars		0.86		Estimate
Grade Factor				**Comments**
Dilution		5.00%	% waste rock in mill feed	Minto North
		8.00%		Area 2/118/Ridgetop
Grade of waste rock		0.00%	% Cu	
		0.00%	Au g/t	
		0.00%	Ag g/t	
Mill Recovery				**Comments**
Mill Recovery	Cu	92%		
	Au	70%		
	Ag	80%		
Concentrate Produced				**Comments**
Moisture Content in Concentrates		8.00%		
Contained Metal in Concentrate	Cu	40.00%		
	Au	variable	varies with Au and Cu head grade	
	Ag	variable	varies with Ag and Cu head grade	
Payable Metal in Concentrate				**Comments**
Payable metal terms were used as per the				
Treatment and Refining				**Comments**
Cu conc. treatment	US	$40.00	/dmt	
Cu refining	US	$0.04	/lb Cu	
Au refining	US	$5.00	/oz Au	
Ag refining	US	$0.40	/oz Ag	
Freight and Marketing				**Comments**
Freight & Marketing (all inclusive)	US	$149.41	/wmt	includes trucking; shipping; port charges; insurance
Freight & Marketing (all inclusive)	US	$162.40	/dmt	
Royalty				**Comments**
Royalty		1.00%	of Net Value	Payable to Selkirk First Nations

The NSR calculations allow for the accounting of:

- Ore grades (Cu, Au, and Ag) thus taking into account the variability in the precious metal content of the deposit (on a whole block basis);
- Ore mill recoveries;
- Contained metal in concentrate;
- Deductions and payable metal value as per MRI Trading contract;
- Metal prices;
- Freight costs (both shipping and trucking);
- Smelting and refining charges, and;
- Royalty charges.

Economic Pit Limit

The ultimate economic pit limits are based on Whittle™ pit optimization evaluations of the resources in the NSR models. This evaluation included the aforementioned NSR calculations as well as geotechnical parameters, mining dilution and recoveries, and mining/milling/G&A costs. The economic pit limits have been constrained to only consider measured and indicated reserve class material. For Area 2, an OP/UG cross-over optimization was conducted in order to account for the UG mining potential of this deposit.

Optimization Parameters and Results

The geotechnical parameters, dilution/recovery, mining, milling and G&A costs (based on an assumed maximum mill throughput of 1.46 mtpa) are summarized in Table 16.34. The estimated projected topography at the completion of mining in the Main Pit was used as the starting surface for the pit optimization and was based on the 2011 Budget schedule compiled by MintoEx in November 2010. The external mining dilution is based on a calculation of the number of waste blocks that are adjacent to an "ore" block in the mineral inventory model, along with an assumed dilution applied to each "waste" edge. The internal (or mill) cut-off grade incorporates all operating costs except mining. This internal cut-off is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the Whittle optimization. The various mill cut-offs were applied to all of the mineral resource estimates that follow.

A series of Whittle™ pit shells were generated based on varying revenue factors and the results analyzed with pit shells chosen as the basis for further design work and preliminary phase designs for each of the deposits of Phase V.

Table 16.34: Pit Optimization Parameters

Item	Unit	Value		
		North	Ridgetop	Area 2/118
Operating Costs				
Waste mining Cost	C$/waste tonne	2.20		
Ore Mining Cost	C$/ore tonne	2.20		
Processing and G&A Cost	C$/milled tonne	24.80		
UG Mining Cost	C$/ore tonne	34.00		
Pit Slope Angles				
Overburden	inter-ramp °	30°	30°	30°
Rock	inter-ramp °	52°	47°west, 53°east	47°west, 53°east
Dilution	%	5.0%	8.0%	8.0%
Mining recovery	%	100%	100%	100%
Strip ratio (est.)	t:t	7.0	6.0	6.0
Internal NSR cut-off	C$/t	26.04	26.78	26.78
External NSR cut-off (est.)	C$/t	43.64	42.18	42.18
Processing rate	t/day milled	4,000		
Processing rate	t/yr milled	1,460,000		

The reserves within the various pit shells were generated from the following 3-D block model items:

- Block centroid coordinates;
- Copper grade;
- Gold grade;
- Silver grade;
- Class (measured, indicated only);
- Topography percentage;
- Overburden tag; and
- Specific gravity.

The results of the Whittle™ pit optimization evaluation for varying revenue factors are summarized in Tables 16.35 through 16.37, as well as Figures 16.33 through to 16.35, for measured and indicated resources only. The selected Whittle shell (based on an evaluation of the results) used as the basis for the detailed pit designs is highlighted in each of the tables.

Table 16.35: North Pit Optimization Results

Final	Revenue	Mine	Ore Diluted	Diluted Grades				Contained Metal			Waste	Strip	Total	Total CF	NPV Best	NPV Worst
Pit	Factor	Life	(ktonnes)	Cu (%)	Au (g/t)	Ag (g/t)	NSR1(C$/t)	Cu (Mlbs)	Au (koz)	Ag (koz)	(ktonnes)	Ratio	(ktonnes)	(C$x1000)	(C$x1000)	(C$x1000)
10	0.52	0.7	1,009	2.99	1.75	11.20	134.78	66	57	363	9,635	9.55	10,643	87,508	83,243	83,243
11	0.54	0.7	1,036	2.95	1.72	11.04	133.01	67	57	368	9,760	9.42	10,796	88,379	83,957	83,957
12	0.56	0.7	1,065	2.91	1.69	10.85	131.10	68	58	372	9,872	9.27	10,937	89,162	84,580	84,580
13	0.58	0.7	1,073	2.90	1.69	10.83	130.84	69	58	374	9,978	9.30	11,052	89,503	84,869	84,869
14	0.60	0.7	1,075	2.90	1.68	10.82	130.69	69	58	374	9,982	9.28	11,057	89,544	84,899	84,899
15	0.62	0.7	1,084	2.88	1.67	10.76	130.04	69	58	375	9,991	9.22	11,075	89,696	85,007	85,007
16	0.64	0.8	1,101	2.86	1.65	10.64	128.78	69	59	377	10,036	9.11	11,137	90,014	85,234	85,234
17	0.66	0.8	1,195	2.72	1.55	10.05	122.37	72	60	386	10,234	8.56	11,429	91,473	86,215	86,215
18	0.68	0.8	1,222	2.68	1.52	9.89	120.64	72	60	389	10,273	8.40	11,495	91,864	86,467	86,467
19	0.70	0.9	1,314	2.57	1.43	9.42	115.48	74	61	398	10,525	8.01	11,839	93,115	87,247	87,247
20	0.72	1.0	1,394	2.49	1.37	9.05	111.60	76	61	406	10,812	7.75	12,207	94,180	87,894	87,894
21	0.74	1.0	1,452	2.44	1.33	8.84	109.25	78	62	413	11,123	7.66	12,575	94,931	88,345	88,345
22	0.76	1.0	1,468	2.42	1.32	8.78	108.54	78	62	414	11,174	7.61	12,641	95,095	88,461	88,461
23	0.78	1.0	1,485	2.40	1.30	8.71	107.75	79	62	416	11,228	7.56	12,714	95,253	88,607	88,605
24	0.80	1.0	1,507	2.39	1.29	8.65	106.97	79	63	419	11,385	7.55	12,892	95,495	88,831	88,823
25	0.82	1.0	1,513	2.38	1.29	8.63	106.73	79	63	420	11,399	7.54	12,912	95,534	88,867	88,857
26	0.84	1.0	1,524	2.37	1.28	8.59	106.26	80	63	421	11,435	7.50	12,959	95,611	88,938	88,922
27	0.86	1.1	1,571	2.33	1.25	8.44	104.47	81	63	426	11,701	7.45	13,271	95,928	89,230	89,173
28	0.88	1.1	1,591	2.31	1.24	8.36	103.61	81	63	428	11,765	7.39	13,357	96,024	89,318	89,233
29	0.90	1.1	1,604	2.30	1.23	8.31	103.11	81	63	429	11,814	7.37	13,418	96,074	89,363	89,259
30	0.92	1.1	1,616	2.29	1.22	8.28	102.67	82	64	430	11,895	7.36	13,511	96,119	89,405	89,279
31	0.94	1.1	1,620	2.29	1.22	8.27	102.54	82	64	431	11,914	7.36	13,534	96,129	89,414	89,281
32	0.96	1.1	1,646	2.27	1.21	8.19	101.52	82	64	433	12,044	7.32	13,690	96,184	89,463	89,272
33	**0.98**	**1.1**	**1,651**	**2.26**	**1.20**	**8.18**	**101.34**	**82**	**64**	**434**	**12,061**	**7.31**	**13,712**	**96,190**	**89,468**	**89,266**
34	1.00	1.1	1,657	2.26	1.20	8.19	101.23	83	64	436	12,176	7.35	13,833	96,194	89,472	89,254
35	1.02	1.2	1,716	2.22	1.17	8.00	99.09	84	64	442	12,527	7.30	14,243	96,176	89,451	89,056
36	1.04	1.2	1,717	2.22	1.17	8.00	99.08	84	64	442	12,527	7.30	14,244	96,176	89,451	89,055
37	1.06	1.2	1,750	2.19	1.15	7.89	97.88	85	65	444	12,712	7.26	14,462	96,112	89,390	88,874
38	1.08	1.2	1,752	2.19	1.15	7.89	97.83	85	65	444	12,718	7.26	14,469	96,108	89,387	88,865
39	1.10	1.2	1,753	2.19	1.15	7.89	97.79	85	65	444	12,724	7.26	14,477	96,104	89,383	88,858
40	1.12	1.2	1,788	2.16	1.13	7.78	96.66	85	65	447	13,008	7.27	14,796	95,952	89,242	88,575
41	1.16	1.2	1,792	2.16	1.13	7.77	96.54	85	65	448	13,040	7.28	14,832	95,931	89,223	88,539
42	1.18	1.2	1,794	2.16	1.12	7.77	96.47	85	65	448	13,051	7.27	14,845	95,921	89,214	88,522
43	1.22	1.2	1,796	2.16	1.12	7.76	96.41	85	65	448	13,073	7.28	14,869	95,904	89,198	88,497
44	1.24	1.2	1,796	2.16	1.12	7.76	96.41	85	65	448	13,074	7.28	14,870	95,904	89,198	88,496
45	1.26	1.2	1,818	2.14	1.11	7.70	95.68	86	65	450	13,262	7.29	15,080	95,686	88,998	88,196
46	1.30	1.2	1,818	2.14	1.11	7.70	95.68	86	65	450	13,264	7.30	15,082	95,685	88,996	88,194
47	1.32	1.2	1,824	2.14	1.11	7.69	95.54	86	65	451	13,359	7.33	15,183	95,600	88,919	88,104
48	1.34	1.2	1,824	2.14	1.11	7.69	95.54	86	65	451	13,360	7.33	15,184	95,598	88,917	88,102
49	1.36	1.3	1,830	2.13	1.11	7.67	95.32	86	65	451	13,412	7.33	15,242	95,525	88,850	88,006
50	1.40	1.3	1,831	2.13	1.11	7.67	95.28	86	65	451	13,428	7.33	15,259	95,506	88,833	87,983

Table 16.36: Area 2/118 Pit Optimization Results

Final	Revenue	Mine	Ore Diluted	Diluted Grades				Contained Metal			Waste	Strip	Total	Total CF	NPV Worst
Pit	Factor	Life	(ktonnes)	Cu (%)	Au (g/t)	Ag (g/t)	NSR1(C$/t)	Cu (Mlbs)	Au (koz)	Ag (koz)	(ktonnes)	Ratio	(ktonnes)	(C$x1000)	(C$x1000)
10	0.58	0.1	154	1.77	0.15	2.72	73.72	6	1	13	554	3.60	707	5,965	5,920
11	0.60	0.1	163	1.76	0.15	2.75	73.21	6	1	14	603	3.69	766	6,219	6,169
12	0.62	0.1	172	1.75	0.16	2.95	72.73	7	1	16	653	3.79	825	6,435	6,380
13	0.64	0.2	324	1.52	0.11	2.11	62.86	11	1	22	1,192	3.68	1,517	9,009	8,866
14	0.66	0.2	340	1.50	0.11	2.16	62.18	11	1	24	1,233	3.63	1,573	9,251	9,096
15	0.68	0.2	351	1.49	0.11	2.16	61.93	12	1	24	1,277	3.64	1,628	9,442	9,280
16	0.70	0.2	362	1.48	0.11	2.15	61.48	12	1	25	1,315	3.63	1,677	9,602	9,431
17	0.72	0.3	368	1.48	0.11	2.14	61.25	12	1	25	1,330	3.61	1,698	9,675	9,501
18	0.74	2.2	3,249	1.45	0.50	4.83	62.95	104	52	504	20,637	6.35	23,885	71,388	61,583
19	0.76	2.3	3,343	1.44	0.49	4.79	62.54	106	53	515	21,036	6.29	24,379	72,532	62,290
20	0.78	3.0	4,315	1.39	0.47	4.56	60.24	132	65	633	26,476	6.14	30,791	85,171	69,506
21	0.80	3.0	4,407	1.38	0.47	4.52	59.95	134	66	641	26,857	6.09	31,264	86,102	69,987
22	0.82	3.1	4,567	1.37	0.45	4.44	59.27	138	67	652	27,319	5.98	31,886	87,289	70,687
23	0.84	3.2	4,708	1.37	0.46	4.44	59.35	142	69	672	28,395	6.03	33,102	89,838	72,368
24	0.86	3.3	4,794	1.36	0.45	4.42	59.06	144	70	682	28,771	6.00	33,566	90,411	72,581
25	0.88	3.3	4,852	1.36	0.45	4.41	58.89	145	70	688	29,089	6.00	33,941	90,739	72,649
26	0.90	3.4	4,922	1.35	0.45	4.38	58.64	147	71	693	29,392	5.97	34,314	91,076	72,669
27	0.92	3.4	4,982	1.35	0.45	4.37	58.49	148	71	700	29,752	5.97	34,734	91,423	72,723
28	0.94	3.5	5,054	1.35	0.45	4.36	58.40	150	72	709	30,232	5.98	35,287	92,178	73,029
29	0.96	3.6	5,191	1.34	0.44	4.34	58.03	153	74	724	31,082	5.99	36,273	92,668	72,845
30	0.98	4.0	5,771	1.30	0.42	4.22	56.46	166	79	784	34,558	5.99	40,329	93,956	71,096
31	**1.00**	**4.0**	**5,855**	**1.30**	**0.42**	**4.22**	**56.35**	**168**	**80**	**795**	**35,171**	**6.01**	**41,026**	**94,476**	**71,108**
32	1.02	4.0	5,884	1.30	0.42	4.21	56.23	168	80	797	35,233	5.99	41,117	94,466	71,017
33	1.04	4.1	5,955	1.29	0.42	4.20	56.03	170	81	804	35,649	5.99	41,604	94,469	70,793
34	1.06	4.4	6,467	1.30	0.43	4.23	56.26	185	89	879	40,720	6.30	47,187	99,646	72,383
35	1.08	4.4	6,497	1.30	0.43	4.21	56.18	186	90	880	40,902	6.30	47,399	99,620	72,233
36	1.10	4.5	6,538	1.30	0.43	4.21	56.16	187	90	885	41,285	6.31	47,823	99,837	72,180
37	1.12	4.6	6,658	1.29	0.43	4.18	55.81	189	91	896	42,001	6.31	48,659	99,449	71,456
38	1.14	4.6	6,741	1.29	0.43	4.18	55.77	191	93	906	42,738	6.34	49,479	99,885	71,352
39	1.16	4.6	6,759	1.28	0.43	4.17	55.72	191	93	907	42,874	6.34	49,633	99,776	71,163
40	1.18	4.7	6,856	1.29	0.43	4.18	55.76	194	94	922	43,846	6.40	50,702	100,691	71,301
41	1.20	4.7	6,895	1.28	0.43	4.17	55.66	195	95	924	44,135	6.40	51,029	100,530	70,999
42	1.22	4.7	6,927	1.28	0.43	4.16	55.57	196	95	926	44,369	6.41	51,296	100,315	70,653
43	1.24	4.8	6,959	1.28	0.43	4.15	55.50	196	95	929	44,657	6.42	51,617	100,116	70,326
44	1.26	4.8	6,978	1.28	0.42	4.15	55.44	197	95	930	44,761	6.42	51,739	99,941	70,105
45	1.28	4.9	7,086	1.28	0.43	4.15	55.50	200	97	945	46,033	6.50	53,119	100,680	69,942
46	1.30	4.9	7,150	1.28	0.43	4.15	55.51	202	98	955	46,731	6.54	53,881	101,063	69,843
47	1.32	4.9	7,164	1.28	0.43	4.15	55.47	202	98	955	46,861	6.54	54,025	100,872	69,614
48	1.34	5.1	7,375	1.26	0.42	4.09	54.80	205	99	971	48,550	6.58	55,925	98,218	66,879
49	1.36	5.1	7,400	1.26	0.42	4.09	54.72	206	100	972	48,771	6.59	56,171	97,857	66,520
50	1.38	5.1	7,438	1.26	0.42	4.09	54.76	207	100	977	49,250	6.62	56,688	98,120	66,549
51	1.40	5.1	7,466	1.26	0.42	4.08	54.69	208	101	979	49,570	6.64	57,036	97,692	66,138

Table 16.37: Ridgetop Pit Optimization Results

Final	Revenue	Mine	Ore Diluted	Diluted Grades				Contained Metal			Waste	Strip	Total	Total CF	NPV Best	NPV Worst
Pit	Factor	Life	(ktonnes)	Cu (%)	Au (g/t)	Ag (g/t)	NSR1(C$/t)	Cu (Mlbs)	Au (koz)	Ag (koz)	(ktonnes)	Ratio	(ktonnes)	(C$x1000)	(C$x1000)	(C$x1000)
10	0.58	0.0	52	1.37	0.04	1.93	56.38	2	0	3	91	1.76	143	1,323	1,319	1,319
11	0.60	0.0	55	1.36	0.04	1.93	55.90	2	0	3	94	1.72	148	1,370	1,366	1,366
12	0.62	0.0	57	1.35	0.04	1.95	55.66	2	0	4	102	1.78	159	1,417	1,413	1,413
13	0.64	0.0	64	1.32	0.05	1.93	54.49	2	0	4	109	1.70	173	1,523	1,518	1,518
14	0.66	0.1	73	1.28	0.05	1.96	52.87	2	0	5	120	1.64	193	1,630	1,625	1,625
15	0.68	0.1	216	1.43	0.67	5.95	63.31	7	5	41	1,470	6.81	1,686	4,604	4,555	4,555
16	0.70	0.2	247	1.41	0.64	5.81	62.60	8	5	46	1,664	6.75	1,911	5,118	5,056	5,056
17	0.72	0.2	280	1.38	0.63	5.69	61.11	9	6	51	1,802	6.45	2,082	5,573	5,496	5,496
18	0.74	0.2	298	1.38	0.63	5.72	61.02	9	6	55	1,944	6.52	2,243	5,865	5,779	5,779
19	0.76	0.2	303	1.38	0.63	5.72	60.96	9	6	56	1,982	6.54	2,286	5,936	5,848	5,848
20	0.78	0.2	348	1.33	0.60	5.42	58.99	10	7	61	2,157	6.20	2,505	6,377	6,268	6,268
21	0.80	0.2	362	1.32	0.58	5.29	58.26	11	7	61	2,188	6.05	2,549	6,488	6,372	6,372
22	0.82	0.3	382	1.30	0.57	5.19	57.54	11	7	64	2,285	5.98	2,668	6,655	6,530	6,530
23	0.84	0.3	401	1.29	0.56	5.12	56.90	11	7	66	2,362	5.89	2,763	6,784	6,651	6,651
24	0.86	0.3	406	1.28	0.55	5.07	56.62	12	7	66	2,374	5.84	2,780	6,816	6,681	6,681
25	0.88	0.3	415	1.27	0.55	5.01	56.17	12	7	67	2,385	5.75	2,800	6,855	6,716	6,716
26	0.90	0.3	429	1.26	0.53	4.90	55.66	12	7	68	2,444	5.69	2,873	6,920	6,775	6,775
27	0.92	0.3	449	1.25	0.52	4.81	55.00	12	8	69	2,536	5.65	2,985	6,992	6,838	6,838
28	0.94	0.3	465	1.23	0.51	4.69	54.24	13	8	70	2,566	5.51	3,031	7,029	6,869	6,869
29	0.96	0.9	1,325	1.10	0.33	2.93	47.39	32	14	125	8,319	6.28	9,644	8,715	8,161	8,161
30	0.98	0.9	1,373	1.10	0.33	2.89	47.31	33	14	128	8,675	6.32	10,047	8,788	8,210	8,210
31	**1.00**	**1.0**	**1,463**	**1.09**	**0.32**	**2.83**	**46.74**	**35**	**15**	**133**	**9,106**	**6.22**	**10,569**	**8,840**	**8,223**	**8,223**
32	1.02	1.0	1,494	1.08	0.32	2.82	46.51	36	15	135	9,230	6.18	10,724	8,829	8,213	8,212
33	1.04	1.2	1,749	1.06	0.31	2.74	45.70	41	17	154	11,002	6.29	12,751	8,505	7,915	7,823
34	1.06	1.2	1,779	1.06	0.31	2.73	45.55	42	18	156	11,162	6.27	12,941	8,448	7,863	7,753
35	1.08	1.2	1,793	1.06	0.31	2.73	45.50	42	18	157	11,255	6.28	13,048	8,409	7,827	7,708
36	1.10	1.3	1,845	1.05	0.31	2.71	45.26	43	18	161	11,573	6.27	13,418	8,226	7,661	7,507
37	1.12	1.3	1,870	1.05	0.31	2.71	45.17	43	18	163	11,753	6.29	13,623	8,106	7,553	7,380
38	1.14	1.3	1,936	1.04	0.30	2.68	44.88	45	19	167	12,199	6.30	14,135	7,761	7,241	7,015
39	1.16	1.4	2,065	1.03	0.29	2.65	44.19	47	20	176	12,957	6.27	15,023	7,002	6,562	6,211
40	1.18	1.5	2,159	1.02	0.29	2.63	43.82	49	20	182	13,592	6.30	15,751	6,418	6,044	5,589
41	1.20	1.5	2,174	1.02	0.29	2.62	43.75	49	20	183	13,690	6.30	15,864	6,306	5,946	5,473
42	1.22	1.5	2,199	1.01	0.29	2.61	43.60	49	20	185	13,811	6.28	16,010	6,125	5,786	5,283
43	1.24	1.5	2,217	1.01	0.29	2.61	43.48	49	20	186	13,882	6.26	16,099	5,981	5,658	5,135
44	1.26	1.6	2,288	1.01	0.29	2.60	43.45	51	21	191	14,781	6.46	17,069	5,110	4,890	4,260
45	1.28	1.6	2,292	1.01	0.29	2.60	43.42	51	21	192	14,820	6.47	17,112	5,046	4,834	4,198

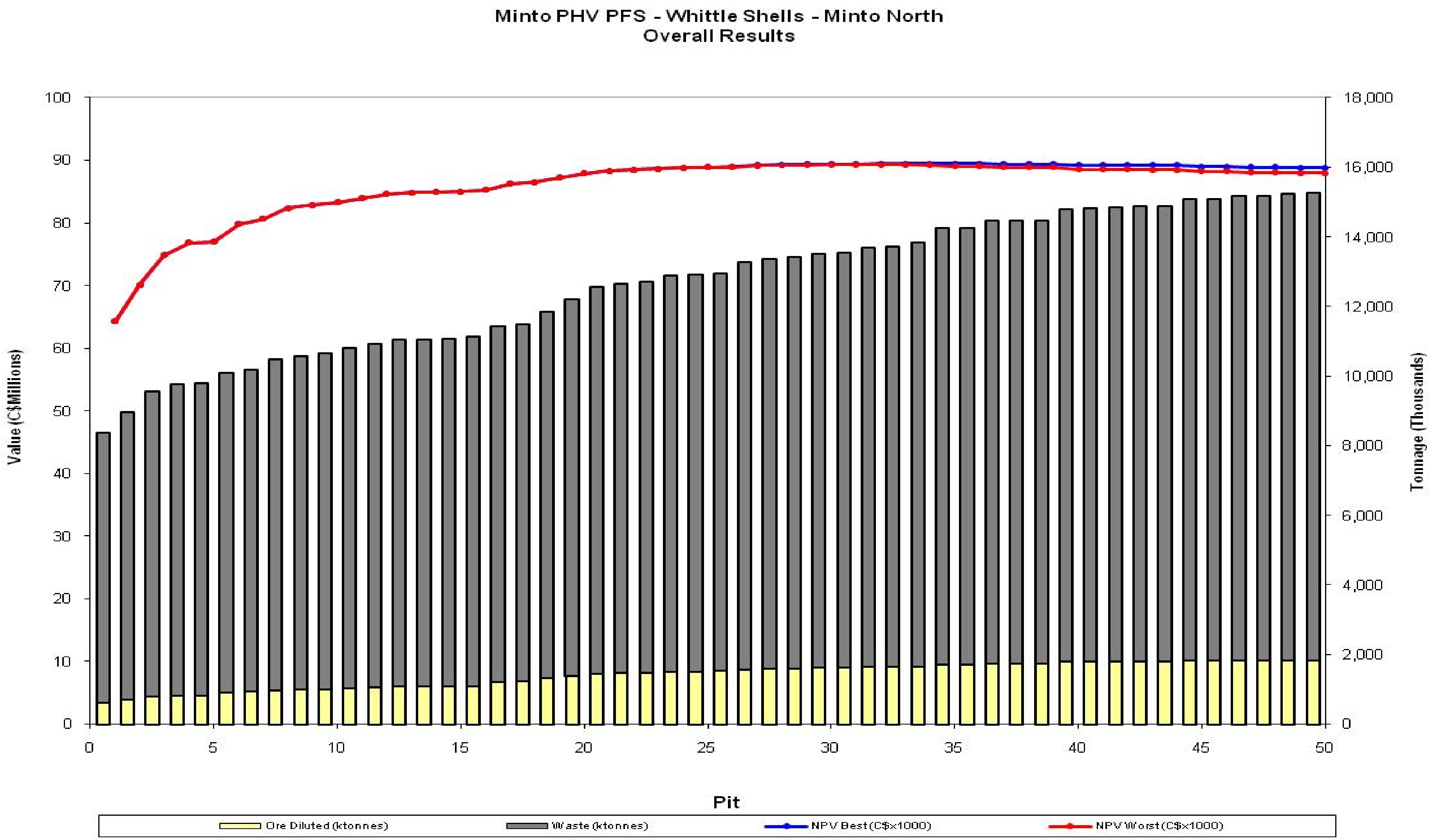


Figure 16.33: North Pit Optimization Results

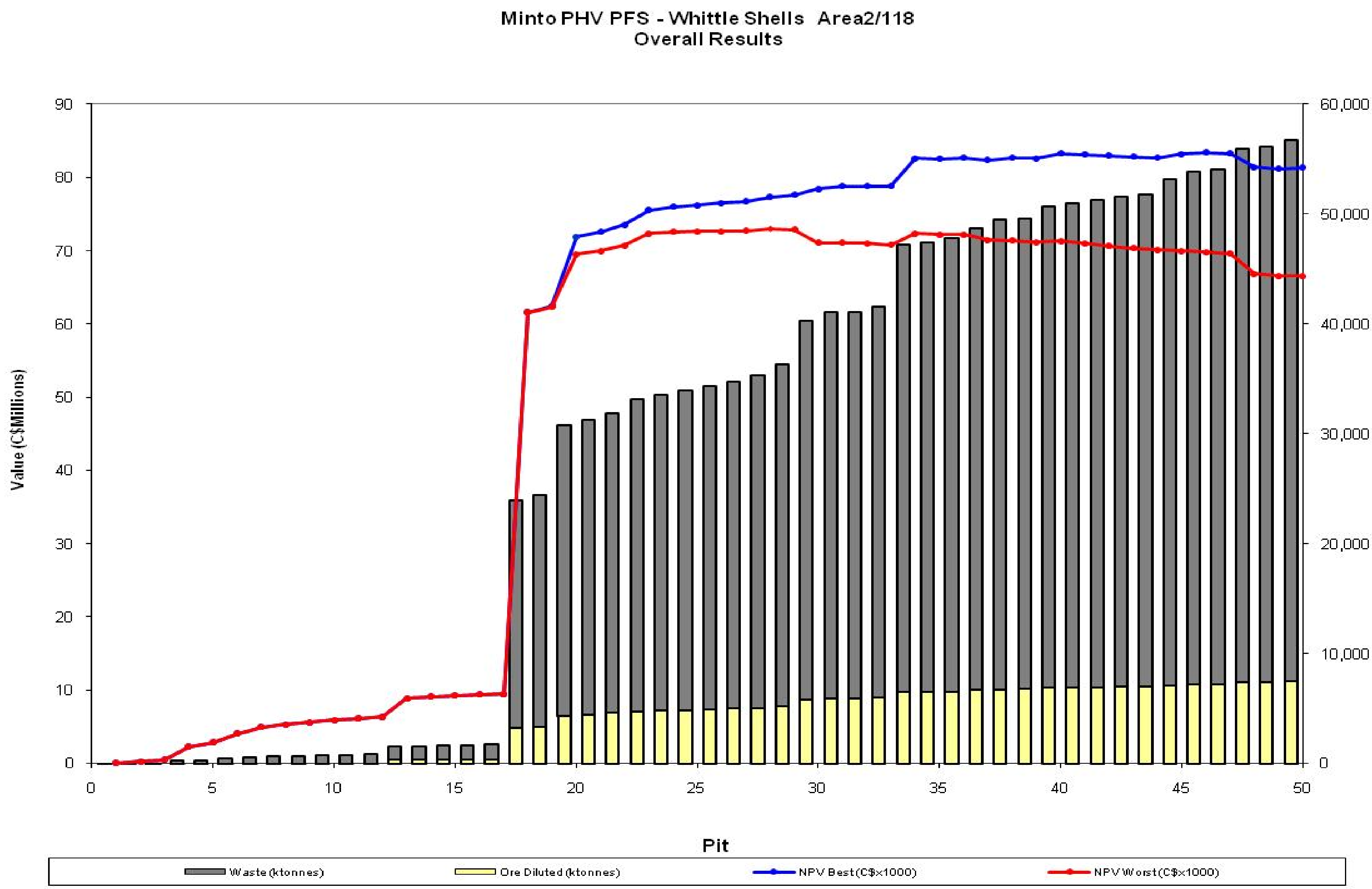


Figure 16.34: Area 2/118 Optimization Results

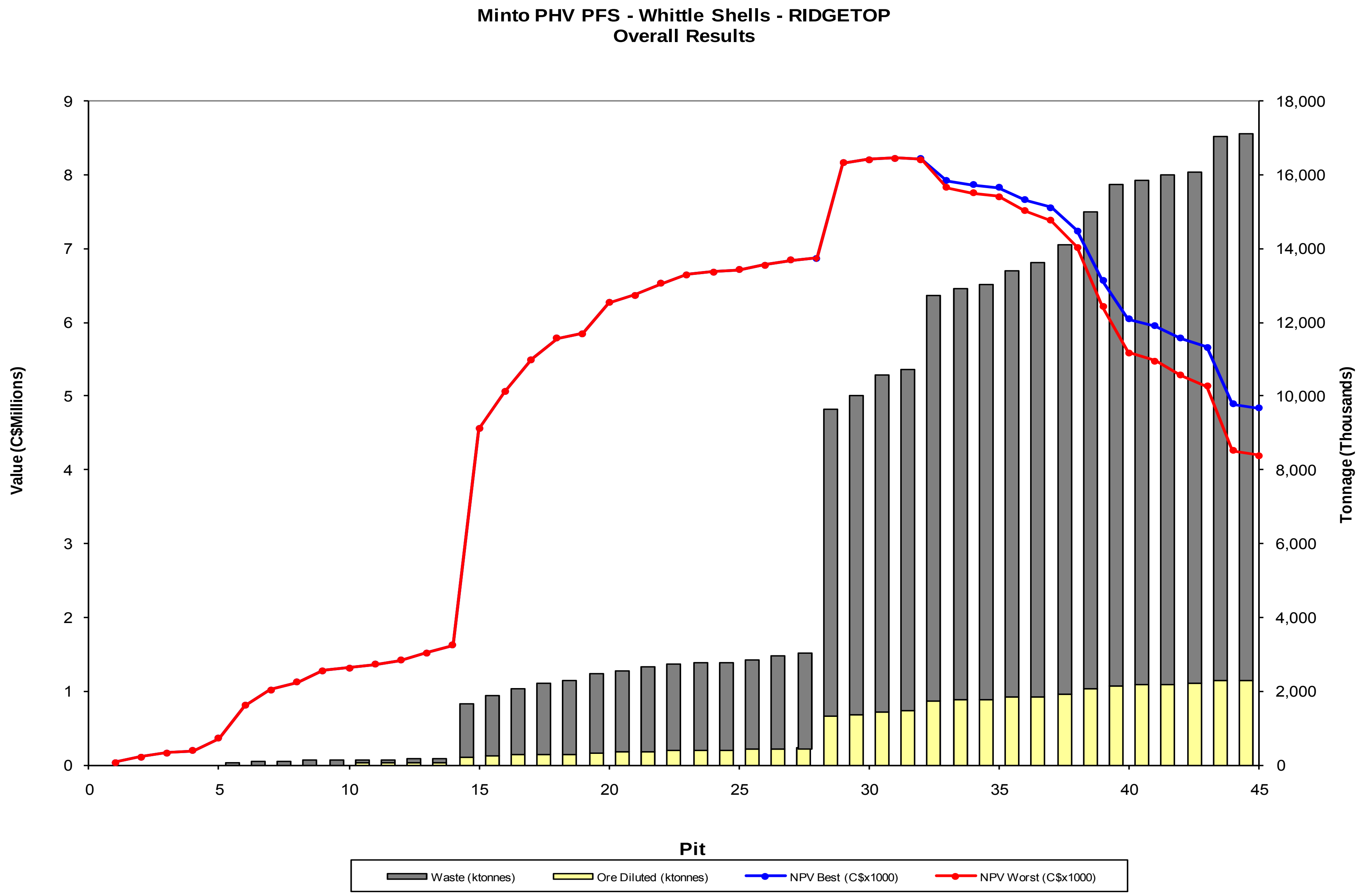


Figure 16.35: Ridgetop Optimization Results

Based on the thorough analysis of the above results the chosen Whittle shell was used as the basis for the detailed pit designs created for each of the Phase V pits. These detailed pit designs take into consideration, minimum mining widths, access ramps, and detailed bench configurations as summarized in Table 16.38 below.

Table 16.38: Detailed Pit Design Parameters

Design Parameter	Unit	North	Area 2/118	Ridgetop
Overburden angle	°	30	30	30
Inter-ramp angle	°	52	47 west, 53 east	47 west, 53 east
Ramp width	m	25	25	25
Ramp grade	%	10	10	10
Bench height	m	9	9	9
Bench face angle	°	72	74	64
Bench configuration	single/double	Double	Double	Double
Berm width	m	8	8	8/4.8

Sub-out maximum depth 6.0 m
Single lane ramp width 15 m @10%

Open Pit Mineral Reserves

The mineral reserves estimate for the detailed open pit designs are summarized in Table 16.39 below for both proven and probable reserve classification. The mineral reserve for Main Pit includes the ore stockpile balance predicted for the beginning of 2010 as well as proposed mining from 2011 going forward (as provided by Minto personnel). The various estimated copper cut-off grades used within the planned pits are as noted.

Table 16.39: PFS Mineral Reserve Estimates

Deposit	Reserve Class	Tonnes ('000s)	Cut-off Grade (%Cu equiv.)	Diluted grade			Contained Metal		
				(%Cu)	(g/t Au)	(g/t Ag)	Cu (Mlb)	Au (koz)	Ag (koz)
Main*	Proven	2,249	0.62	1.35	0.46	4.79	67	33	346
	Probable								
	Sub-total	2,249	0.62	1.35	0.46	4.79	67	33	346
North	Proven	1,523	0.52	2.36	1.28	8.55	79	63	419
	Probable	5	0.52	2.25	0.81	9.38	0	0	2
	Sub-total	1,529	0.52	2.36	1.27	8.56	79	63	421
Ridgetop	Proven	627	0.54	1.10	0.25	2.05	15	5	41
	Probable	710	0.54	1.11	0.37	3.55	17	9	81
	Sub-total	1,337	0.54	1.11	0.32	2.85	33	14	122
Area 2	Proven	3,373	0.54	1.41	0.53	4.94	105	58	536
	Probable	1,447	0.54	1.08	0.31	3.59	35	15	167
	Sub-total	4,820	0.54	1.32	0.47	4.53	140	72	703
118	Proven								
	Probable	491	0.54	1.29	0.09	1.73	14	1	27
	Sub-total	491	0.54	1.29	0.09	1.73	14	1	27
Total OP only	Proven	7,773	0.56	1.56	0.63	5.37	266	158	1,343
	Probable	2,653	0.54	1.13	0.29	3.24	66	25	277
	Total	10,426	0.55	1.45	0.55	4.83	333	183	1,619

**includes stockpile balance of 1,631 kt at beginning of 2011 for Main pit but excludes approximately 0.25Mt of partially oxidized material from stockpile. Numbers may not add up due to rounding.*

Within these detailed pit designs there a total of 42 kt of inferred mineral resources at a copper grade of 1.36%. These inferred tonnes were not included in the LOM production plan. There is no certainty that these inferred mineral resources will be converted to the measured and indicated categories through further drilling, or into mineral reserves, once economic considerations are applied.

Cut-off Grade Calculation

Table 16.40 summarizes the cut-off grade calculations for the various deposits in Phase V. These copper cut-off grades are estimates only, since the actual modelling and optimization work was conducted with the NSR model previously described in the report.

Table 16.40: Copper Cut-off Grade Estimate

Parameter	Unit	2010 PFS			
		Area 2/118/Ridgetop		Minto North	
Revenue, smelting & refining		Res.COG	Incr. COG	Res.COG	Incr. COG
Cu price	US$/lb Cu	2.25	2.25	2.25	2.25
Exchange rate	C$/US$	1.16	1.16	1.16	1.16
Cu price	C$/lb Cu	2.62	2.62	2.62	2.62
Payable metal	% Cu	96.75%	96.75%	96.75%	96.75%
TC/RC/Transport	C$/lb Cu payable	0.30	0.30	0.30	0.30
NSR (Cu only)	C$/lb Cu payable	2.23	2.23	2.23	2.23
Opex estimates					
Mining cost	C$/t mined	2.20	0.00	2.20	0.00
Strip Ratio	t:t	6.00	0.00	7.00	0.00
Mining Cost	C$/t milled	15.40	0.00	17.60	0.00
Processing and G&A cost	C$/t milled	24.80	24.80	24.80	24.80
Site Cost	C$/tonne milled	40.20	24.80	42.40	24.80
Recovery and Dilution					
Recovered Cu grade	%Cu	0.82	0.50	0.86	0.50
Process Recovery	average %	92%	92%	92%	92%
Plant feed Cu grade	diluted %Cu	0.89	0.55	0.94	0.55
Dilution	%	8.0%	8.0%	5.0%	5.0%
Cut-off Grade					
In-situ cut-off Cu grade (Cu only)	%Cu	0.97	0.60	0.99	0.58
By-product contribution (est.)	% of Cu value	10%	10%	10%	10%
In-situ cut-off Cu grade (inc. by-product value)	%Cu	0.88	0.54	0.90	0.52

16.9 Mineral Reserves - Underground

Cut-off Grade

Preliminary estimated on-site costs, which included mining operating costs of $33.80 per tonne of ore, $12.90 processing and $11.90 G&A costs were used to determine a cut-off grade.

NSR calculations were performed to estimate ore value in the block model so reserve optimization could be conducted. The mining and economic parameters used in the NSR and cut-off grade calculation are summarized in Table 16.41 below.

Table 16.41: Underground COG Estimate

Parameter	Unit	UG COG
Revenue, smelting & refining		
Cu price	US$/lb Cu	2.25
Au price	US$/oz Au	300.00
Ag price	US$/oz Ag	3.90
Exchange rate	C$/US$	1.16
Recovery to Cu Concentrate		
Cu	%	92
Au	%	70
Ag	%	80
Cu Concentrate Grade		
Cu	%	40
Au	g/t	variable
Ag	g/t	variable
Moisture content	%	8
Smelter Payables		
Cu in Cu conc	%	96.75
Au in all cons	%	per MRI contract
Au deduction in all cons	g/t in conc	0
Ag in all cons	%	100
Ag deduction in all cons	g/t in conc	30
Off site costs		
Cu conc treatment	US$/dmt conc	40.00
Cu refining charge	US$/lb pay Cu	0.04
Au refining charge	US$/oz pay Au	5.00
Ag refining charge	US$/oz pay Ag	0.40
Transport, marketing, ins., etc.		
Ocean freight to Japan	US$/wmt conc	60.00
Truck freight to Skagway	US$/wmt conc	57.33
Skagway port charges & maint.	US$/wmt conc	12.50
Port user fee	US$/wmt conc	11.50
Wharfage	US$/wmt conc	2.00
Customs Fee	US$/wmt conc	1.85
Survey, assay, umpire	US$/wmt conc	3.15
Port property taxes	US$/wmt conc	1.08
Marine/transportation insurance	%	variable w/conc grade
Transport, marketing, ins, etc.	US$/wmt conc	149.41
Transport, marketing, ins, etc.	US$/dmt conc	162.40
Royalties	%	1.00
Operating Costs		
UG mining cost	C$/ore tonne	33.80
Processing cost	C$/milled tonne	12.90
G&A cost	C$/milled tonne	11.90
Total Site Costs	**C$/milled tonne**	**58.60**
Dilution	%	10
Cut-off Grade		
Plant feed (diluted) cut-off CuEq grade	$\%Cu_{equiv}$	1.24
In-situ (undiluted) cut-off CuEq grade	$\%Cu_{equiv}$	1.36

All of the above parameters were used to calculate factors for input to the block model for each contributing metal, as follows:

NSR (\$/t) = \$7.39 x Au (g/t) + \$0.08 x Ag (g/t) + \$44.43 x Cu (%).

Those factors were incorporated into geological block model to estimate value of each ore block.

Capstone management decided to use cut-off grade of 1.2 % Cu_{equiv} (copper equivalent) for design of stope shapes.

Mineable Reserves Optimiser (“MRO”) software application developed by Datamine was used to produce “mineable envelopes” and estimate tonnes and grade within those envelopes. First, the Gemcom block model, which had uniform block sizes and specified percentage of the block containing mineralisation, was converted into a subcell model, which contains blocks of varying sizes that follow the interpreted mineralised wireframes and thus contain 100% mineralisation. Only Measured and Indicated Resources not extracted by Minto open pit operation were evaluated by MRO.

MRO was run using the minimum mining height of 3 m and a cut-off grade of 1.2% Cu. The “floating stope” was moved through the block models at intervals of 5 m in both lateral directions and 0.5 m vertically; thus, this minimum height and grade criteria were evaluated on 5m centres across the deposit. As it moves toward the margins of the mineralisation, the grade of each subsequent 0.5 m increment may drop to below the cut-off or may be waste: these increments would not be part of this “minimum envelope” despite the fact that the resulting overall envelope may still be economic. A “maximum mineable envelope” would include this dilution up to the point where the overall “mineable envelope” ceases to be economic; the actual stope extents would be somewhere in between the minimum and maximum envelopes.

Once the “mineable envelopes” had been generated by MRO, they were visually evaluated for practicality: whether the lateral extents of a “mineable envelope” were sufficient in order for there to be a stope and whether access would be possible and if there is a sufficient pillar between it and nearby “mineable envelopes”. Only envelopes deemed to have a potential for being mined as stopes were selected. An additional evaluation of the access development required versus the value contained was conducted later to define if particular stope would be economical to be mined.

Dilution and Recovery

Dilution and recovery factors were estimated for each individual mining shape based on the various mining thicknesses, orebody dip and mining method to be used. Dilution is defined as the ratio of waste to ore. Two sources of dilution would be expected in the mine: internal and external dilution.

Internal dilution derives from material with grades that are less than a cut-off grade that falls within a designed stope boundary (i.e. it will be drilled and blasted within the stope during mining).

The internal dilution is incorporated into the reserve when estimating the ore contained within the designed stope. In the room and pillar, in many cases where less than cut-off grade zones are encountered, they may be left as pillars due to the highly flexible nature of room and pillar mining.

External dilution derives from low or zero grade material from beyond the stope design boundaries due to blasting overbreak, adverse geological structure, failure within zones of weak rock, and when mucking on the top of backfill material. External dilution is almost always generated and an allowance is always made for it during the reserve estimation process.

External dilution for RAP and PPCF mining has been estimated for the stopes based on various mining thickness and dip by adding 0.25 m overbreak to stope dimensions in waste rock (roof and rock floor) of zero grade, and 0.2 m backfill dilution when mucking on a backfill floor.

An extraction ratio was estimated for each stope to account for losses in the permanent pillars, in the corners of the footwall and hangingwall adjacent to the designed stopes.

An additional mining operational recovery was applied as not all ore could be recovered from the designed stopes due to a number of causes such as:

- Underbreak – the ore is not blasted and remains on the stope walls;
- Ore loss within stope – the blasted ore is left in the stope due to poor access for the loader, buried by falls of waste rock from walls, left on the floor, blasted but does not fall from flatter lying walls, or gets mixed into the backfill floor and is left behind.

The contribution of each of these losses varies depending on the particular mine. For the purposes of this study, an average mining operational recovery of 95% (in addition to the extraction) was assumed for both the RAP and PPCF stoping methods.

Table 16.42: Measured and Indicated Resource at 1.2% Cu Cut-off for the Minto Underground Operation

Area	Stope	Resources (Kt)	Copper (%)	Gold (g/t)	Silver (g/t)	NSR ($/t)	Ave. Thickness (m)	Recovery (%)	Dilution (%)
Area 118	101	815	2.04	0.77	8.00	97	8.8	72	10.7
	102	36	1.44	0.31	3.71	67	6.3	81	15.0
	103	63	1.56	0.52	6.31	74	12.2	72	9.7
	104	19	1.49	0.97	12.20	74	4.9	69	15.0
	105	11	1.47	1.02	9.26	73	4.8	65	15.0
	106	40	1.76	1.03	7.06	86	6.5	65	15.0
	107	16	1.73	0.97	8.23	85	5.1	65	15.0
	108	30	1.48	0.59	6.10	70	5.1	81	15.0
Area 2	201	456	2.07	0.93	6.82	99	13.0	75	10.1
	202	20	1.31	0.22	3.94	60	4.0	81	15.0
	203	355	1.86	0.82	9.04	89	6.8	69	9.5
	204	173	1.73	0.67	6.06	82	10.8	71	10.3
	205	53	1.55	0.58	6.37	74	6.8	69	8.0
	206	14	1.72	0.55	6.32	81	4.6	75	8.0
	207	27	1.70	0.62	5.39	80	7.6	75	8.0
	208	25	1.96	1.22	8.58	97	6.2	75	8.0
	209	27	1.47	0.81	8.57	72	4.7	78	10.0
	210	23	1.50	0.69	6.32	72	5.1	78	10.0
	211	61	1.94	0.79	8.41	93	8.9	78	10.0
Minto East	301	863	2.49	1.14	6.73	120	12.9	75	9.5
Total		**3,127**	**2.07**	**0.89**	**7.33**	**99**	**10.2**	**73**	**10.1**

Dilution and recovery factors were then applied to estimate the reserve.

Stopes 102, 108, and 202 were excluded from mineral reserve estimate due to the remote location making extraction uneconomic.

Table 16.43: Minto Mineral Reserve Estimates for Underground Mine

Deposit	Reserve Class	Tonnes ('000s)	Cut-off Grade (%Cu_{equiv})	Diluted grade			Contained Metal		
				Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Au (Koz)	Ag (Koz)
Area 118	Probable	764	1.2	1.78	0.69	7.17	30.0	17.0	176.1
Area 2	Probable	967	1.2	1.73	0.75	6.77	36.8	23.5	210.6
Minto East	Probable	709	1.2	2.28	1.04	6.15	35.6	23.8	140.2
Total	**Probable**	**2,441**	**1.2**	**1.90**	**0.82**	**6.71**	**102.4**	**64.3**	**526.8**

Totals may not be exact due to rounding.

17 Other Relevant Data and Information

17.1 Geotechnical Pit Slope Design

SRK (2009) has carried out a pre-feasibility level geotechnical rock mechanics evaluation for the Area 2, Area 118, Ridgetop and North deposit areas. The Copper Keel deposit was not targeted for geotechnical data collection at the time of the field investigation. In 2010, SRK conducted a pre-feasibility level geotechnical evaluation to support underground mine planning for underground zones in Area 2, 118, and East Minto. The following sections are intended to summarize the above evaluations.

The following comprised the principle stages of the geotechnical evaluation:

- Discontinuity orientation and geotechnical logging of core;
- Geomechanical laboratory strength testing and geologic materials characterization;
- Development of geotechnical models to provide bases for excavation stability analyses; and
- Recommendation of optimal pit slope angles and pit architecture for mine design purposes.
- Recommendation of room and pillar dimensions as well as ground support requirements for the alternative underground development of Areas 2, 118, and East.

As commissioned, the work reported was performed at a pre-feasibility design level.

Geotechnical Data Collection

A geotechnical core logging program was developed to yield information pertinent to modeling of pit slope stability, such as geologic contacts, profiles of rock strength, and characteristics and frequency of discontinuities.

Geotechnical logging, field point load testing and discontinuity orientation of core recovered from a total of eight drill holes were conducted for this investigation. In addition to the eight geotechnical core holes drilled for this investigation, data from three additional geotechnical core holes drilled in 2007 as part of the previous SRK (2007) Area 2 Pre-feasibility Pit Slope Evaluation were also considered in the analyses. Collar locations and average drill hole plunges and azimuths of the geotechnical drill holes are summarized in Table 17.1.

Table 17.1: Summary of Drill holes Oriented and Logged for Geotechnical Data

SRK Hole ID	Minto Hole ID	Collar Coordinates			Azimuth (deg)	Inclination (deg)	Length (m)
		Northing	Easting	Elev.			
C09-01	09SWC424	6944462.5	384615.2	876.8	236	-57	325.0
C09-02	09SWC422	6944276.4	384751.3	893.9	239	-58	280.5
C09-03	09SWC420	6944390.8	384933.1	861.4	213	-61	376.5
C09-04	09SWC427	6943813.0	384955.7	890.1	245	-60	175.5
C09-05	09SWC429	6943654.8	384933.1	916.9	58	-59	199.5
C09-06	09SWC431	6943632.3	385112.7	889.2	238	-60	150.0
C09-07	09SWC495	6945925.0	384238.0	951.4	196	-60	153.0
C09-08	09SWC497	6945953.0	384320.0	940.7	47	-55	141.0
C07-06	07SWC206	6944784.8	384609.5	822.6	223	-61	155.1
C07-07	07SWC201	6944506.4	384808.9	861.0	211	-57	243.5
C07-08	07SWC196	6944640.7	384876.9	832.9	070	-60	249.6

Laboratory Strength Testing

Geomechanical testing was conducted at The University of Arizona Rock Mechanics Laboratory in Tucson, Arizona, to determine strength characteristics of the in-situ materials. The overall laboratory program consisted of direct shear, uniaxial and triaxial compressive strength, and direct tensile strength testing and measurement of unit weight and elastic properties. A total of 51 laboratory tests were conducted on samples selected to represent the range of the rock conditions observed in the eight 2009 geotechnical borings.

Laboratory uniaxial axial compressive strength (UCS) testing was conducted on 30 samples, producing the following:

- UCS ranging from 48.9 to 172.3 MPa, with a mean value of 116.0 MPa;
- Young's Moduli ranging from 14.9 to 66.5 GPa, with a mean value of 47.8 GPa; and,
- Poisson's Ratios ranging from 0.084 to 0.302, with a mean value of 0.229.

Laboratory UCS and elastic properties are summarized by geotechnical domain in Table 17.2.

Table 17.2: Summary of Laboratory Testing by Geotechnical Domain

Domain	Hole ID	Depth (m)	UCS (MPa)	E (GPa)	ν	BTS (MPa)
A118 Weathered	C09-01	32.10	88.21	50.5	0.22	
A118 Fresh	C09-01	89.50	119.56			
A118 Fresh	C09-01	187.00	150.39			
A118 Fresh	C09-01	220.30	164.68	66.5	0.30	
A118 Fresh	C09-01	293.16	156.10			
A118 Fresh	C09-02	122.67	71.69	49.2	0.21	
A118 Fresh	C09-02	150.10				10.8
A118 Fresh	C09-02	179.54	128.30			
A118 Fresh	C09-02	271.90	149.87			9.36
Area 2 Weathered	C09-03	38.00	48.94	14.9	0.08	
Area 2 Weathered	C09-03	77.33	72.30			
Area 2 Fresh	C09-03	130.84	66.03			
Area 2 Fresh	C09-03	161.03	104.39	47.3	0.23	7.63
Area 2 Fresh	C09-03	282.10	102.63			
Area 2 Fresh	C09-03	361.70	149.58			
Ridgetop Weathered	C09-04	30.40	63.15			
Ridgetop Weathered	C09-05	33.00	70.92			
Ridgetop Weathered	C09-06	37.20	121.20			8.9
Ridgetop Weathered	C09-06	71.22	131.32	52.5	0.29	
Ridgetop Fresh	C09-04	91.10	140.72			
Ridgetop Fresh	C09-04	150.25	153.42			
Ridgetop Fresh	C09-05	92.70	74.34			
Ridgetop Fresh	C09-05	150.11	86.71	53.9	0.26	7.2
Ridgetop Fresh	C09-06	108.35	122.78			
Ridgetop Fresh	C09-06	138.00	100.70			
Minto North	C09-07	29.32	172.29			
Minto North	C09-07	86.34	139.69			
Minto North	C09-07	124.57	124.68			
Minto North	C09-08	47.53	157.71			
Minto North	C09-08	89.15	94.31			
Minto North	C09-08	129.40	153.60			

Triaxial compressive strength (TCS) testing was conducted on six samples of core, under confining pressures (σ_3) ranging between 6.9 and 20.7 MPa.

Hoek-Brown and Mohr-Coulomb failure envelopes from the traixial and uniaxial data were calculated and are shown in Figure 17.1.

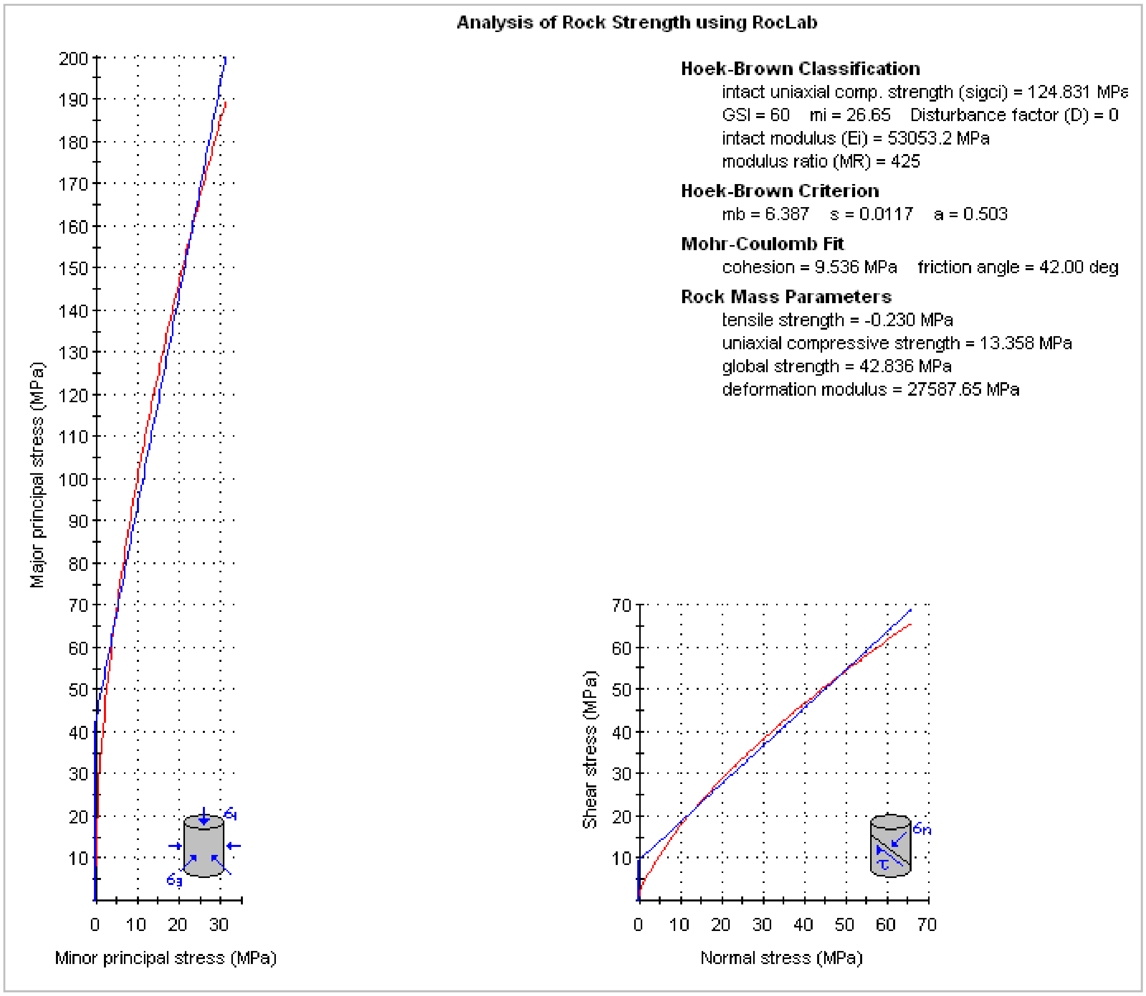


Figure 17.1: Analysis of Lab Data

Ten samples of naturally-occurring discontinuities encountered in the core were tested using four-point, small-scale direct shear tests to obtain discontinuity shear strength data, resulting in:

- Calculated friction angles (Φ) ranged from 33° to 46°, with a mean of 36°**;** and,
- Apparent cohesion values ranging from 1 to 22 kPa, with a mean of 10 kPa.

Brazilian tensile strength (BTS) testing was conducted on five samples producing intact tensile strengths ranging from 7.2 to 10.8 MPa, with a mean value of 8.8 MPa.

Prior to actual testing of UCS and TCS core samples, sample dimensions and weights were measured and used to calculate total unit weights for each sample. The combined data set included 36 unit weight measurements ranging from 24.9 to 26.7 kN/m^3 with a 26.2 kN/m^3 mean.

Geotechnical Model

For each area under study, a geotechnical model was developed to provide a framework for slope stability modeling by mathematically simulating site geotechnical conditions and then calculating the anticipated response to stress changes resulting from the proposed open pit excavations. A typical geotechnical model is composed of individual regions (domains), each of which is comprised of materials exhibiting internally similar geomechanical properties. Pertinent geotechnical parameters are assigned to each domain defined, based on engineering properties that are determined during field data collection and laboratory testing programs.

17.1.1 Surface Mining Geotechnical Domains

To initiate the geotechnical modeling, the basic geotechnical parameters recorded for each core run were applied to the Laubscher (1990) In-situ Rock Mass Rating (IRMR) system, thereby creating a profile of IRMR with depth for each of the eight geotechnical holes drilled for this investigation. Based upon the IRMR as well as upon its individual components, available site geology information and laboratory test results, drill cores were divided into geotechnical intervals or domains that are expected to behave uniformly when exposed to open pit excavation-induced stresses, for each of the deposit areas. Given the relatively consistent nature of geologic materials at Minto, the materials were divided into two basic domains at Area 2, Area 118 and Ridgetop, i.e., weathered and fresh rock. As explained later, the Minto North rock was classified into a single domain.

The weathered rock domain is typically characterized by relatively higher fracture frequencies, consistently lower intact rock strengths and zones of heavy alteration and oxidation as a result of moderate to heavy surface weathering and is typified by core that also typically shows consistently lower RQD and IRMR values. Consequentially, the weathered bedrock is of significantly lower geomechanical quality than is the fresh rock which underlies it.

In general, the fresh rock is consistently a much more competent rock mass than is the weathered bedrock, possessing relatively lower fracture frequencies and higher intact rock strengths. The fresh rock encountered is relatively massive and exhibits fewer signs of alteration and weathering when compared to the weathered rock and, consequently, possesses higher overall RQD and IRMR values.

The fresh rock domains contain intermittent zones of weaker material which typically correspond to intervals of increased fracturing, weathering and/or alteration, including minor fault zones and surface weathering. However, such intermittent weaker rock zones represent a relatively small portion of the overall fresh rock domain and are not anticipated to adversely impact the performance of the fresh rock mass.

Several zones of foliated granodiorite were encountered in the fresh rock, but those zones exhibited similar intact rock strengths and rock mass properties as did samples of non-foliated granodiorite collected from the same coreholes. The foliated zones are judged to be discontinuous and are not expected to impact overall pit slope stability differently than will the non-foliated zones.

Therefore, the foliated and non-foliated rock was grouped together into their respective weathered or fresh domains.

A summary of IRMR values per domain is presented in Table 17.3.

Table 17.3: Summary of In-situ Rock Mass Rating Distributions

Deposit	Domain	Distribution	Sample No.	Mean	Std. Dev.	Min	Max
Area 2	Weathered	Normal	162	46.4	8.6	18	68
Area 2	Fresh	Normal	409	59.8	9.7	29	82
Ridgetop	Weathered	Normal	225	51.8	12.3	18	84
Ridgetop	Fresh	Logistic	99	51.0	10.1	18	76
North	-	Logistic	172	50.5	10.0	19	82
Area 118	Weathered	Logistic	59	50.8	9.2	21	72
Area 118	Fresh	Logistic	334	58.3	10.8	22	81

17.1.2 Area 2

A relatively deep soil overburden deposit exists under the northeast portion of the proposed Area 2 Pit, consisting primarily of transported silt and fine sand with occasional lenses of clay and coarse sand to gravel. The soil is high in organic content and is known to contain permafrost. It appears that the soil has filled a relatively deep erosional feature on the order of 60 to 90 m deep with an invert located between Area 2 and the Main Pit to the north. Previous geotechnical work done by SRK and others have indicated that the material contains permafrost down to near the bedrock contact at its deepest portions and is most likely frozen down to the bedrock contact in shallower portions. Ubiquitously, the upper 1 m is “active”, i.e., seasonally freezing and thawing.

Based on available information from resource and geotechnical drilling, Area 2 is covered with soil overburden ranging from about 5 to 15 m in depth in the southwest portion, with up 20 to 45 m along much of the north and east walls, and reaching a maximum depth of 70 m at the far north.

While it is possible that the frozen overburden may extend farther south, available information suggests that the overburden at the south and west ends of the proposed Area 2 Pit consists of a thin veneer of organic soil underlain by approximately 5 m to 15 m of completely weathered, in-situ bedrock (granular soil) or residuum.

Based on geotechnical drill hole data, the Area 2 weathered domain is adjudged to extend to depths of approximately 50 to 100 m below the current ground surface.

17.1.3 Copper Keel

The Copper Keel deposit is a southeasterly extension of Area 2 with slopes ranging between approximately 75 and 125m in height. The soil overburden deepens significantly between southern Area 2 and Ridge Top North. The upper approximately 50m of slopes at Copper Keel are expected to consist of overburden soils.

17.1.4 Area 118

The majority of the proposed Area 118 open pit footprint is covered with up to approximately 5 m of overburden, except in its southwestern portion, where the soil locally deepens to approximately 16 m. The depth of bedrock weathering at Area 118 is generally to about 30 to 60 m below the current ground surface.

17.1.5 Ridgetop

The western regions of the proposed Ridgetop pits are anticipated to contain 1 to 5 m of soil overburden, deepening to the east to from 5 to 15 m on the east side and with a maximum depth of 21 m at the northeast portion of Ridgetop North and the east portion of Ridgetop South.

The bedrock at Ridgetop is generally weathered to a depth of approximately 45 to 70 m below current ground surface.

17.1.6 Minto North

Due to the relatively shallow depth of the Minto North pit and the presence of multiple structures and weaker zones, there was a less significant distinction between the weathered and fresh rock materials and, consequentially, materials at Minto North were combined together into a single domain for modeling.

17.1.7 Model Methodology

Evaluation of the results of the field and laboratory data collection programs indicates a high degree of variation in rock strength and geologic structure at Minto. This natural variability in rock strength and structure suggests that a probability-based method of analyses is most appropriate, yielding less conservative slope angles than would the selection of a unique, potentially over-conservative value, as is typical to strictly deterministic analyses. As such, for this work, model parameters were characterized by statistical distributions of values having a central tendency and some variation around that central tendency, rather than by a single, unique value.

A rock mass shear strength/normal stress relationship was developed for each domain using the Generalized Hoek-Brown strength model (Hoek, et. al. 2002). Probability density functions (PDF) were selected to represent distributions of Geological Strength Index (GSI), material constant (mi) and disturbance factor (D). The distributions selected were based on the results of field and laboratory testing as well as on SRK's experience.

Interramp/Overall Slope Stability Analysis

The mathematical geotechnical model was input into the commercially available slope stability modeling software package Slide 5.039, developed by Rocscience, Inc. (2003). Slide is a two-dimensional, limit equilibrium slope stability analysis program that analyzes slope stability by various methods of slices, from which Spencer's method was chosen for this evaluation due to its consideration of both force and moment equilibrium.

Results of slope stability modeling generally indicated probabilities of failure (PoF) ranging from near zero to approximately 5%. It should be noted that while a near zero percent probability of failure does demonstrate a very low likelihood of slope instability; it does not imply that slope instability is impossible; rather, a reported zero probability simply indicates that, for the potential failure surfaces characterized by one of 300 samples drawn from the strength distributions defined, no surfaces had a Factor of Safety (FoS) less than 1.0. See Table 17.4 for modelling results.

Table 17.4: Interramp Slope Stability Modelling Results

Deposit	Sector	Height (m)	Mean FoS	PoF (%)
Area 2	Northeast	130	2.5	1
Area 2	Southwest	214	2.1	3
North	-	130	2.3	0
Ridgetop		130	2.2	2

Given the small size of the proposed Area 118 pit as well as its close proximity and geotechnical similarities to Area 2, additional interramp slope stability modelling was not deemed necessary for Area 118 at the current pre-feasibility level.

Geologic Discontinuity Analysis

Geologic discontinuities were analyzed at both the pit wall and bench scales. The term discontinuity refers to any break or fracture, ranging from faults at the upper limit to joints at the lower limit, having negligible tensile strength. Discontinuities are formed by a wide range of geological processes and can collectively include most types of joints, faults, fissures, fractures, veins, bedding planes, foliation, shear zones, dikes and contacts.

Major Structures

Major geologic structures are those features, such as faults, dikes, shear zones, and contacts that have dimensions on the same order of magnitude as the area being characterized. These structures are treated as individual elements for design purposes, as opposed to joints, which are handled statistically.

Typically, high angle structures do not adversely impact pit slopes on the overall scale and as such, were not specifically targeted for this pre-feasibility level evaluation. As such, geotechnical drilling at the pre-feasibility evaluation level is targeted to obtain data representative of overall rock mass conditions and, secondarily, to individual structures such as those previously mentioned.

Several faults or shear zones have been identified in resource and geotechnical drilling at all subject Minto sites. Most of these structures are not, however, anticipated to significantly impact pit slope stability due to their apparent lack of persistence and to the generally limited degree of rock degradation, e.g., highly plastic gouge development, associated with them. However, the potential for one or more major structures to adversely impact stability of the Area 2 west wall has been identified and, as discussed in the SRK recommendations, should be further investigated as the project advances.

Specifically, both resource and geotechnical drilling in south-western Area 2 suggest the presence of a major fault or faults, potentially striking sub-parallel to the Area 2 Pit west wall, with a moderate to steep northeast dip similar to faults suggested by resource geology in adjacent Area 118. In particular, exploration holes 06SWC082 and 06SWC106 encountered deep brittle structure(s) approximately 279 m and 243 m, respectively, down hole. Similar indications of fault intercepts were not observed in adjacent holes, thereby suggesting a high dip angle for the structure or structures.

Geotechnical drill holes C09-03 and C07-07 also encountered zones of major rock disturbance at shallower depths that would be consistent with the potential structure(s) and would coincide with the western Area 2 ultimate pit wall.

Rock Fabric

Minor discontinuities such as joints, foliation and bedding planes, represent an infinite population for practical purposes and, due to sampling limitations, are best modelled with stochastic (probabilistic) techniques. A discontinuity set denotes a grouping of discontinuities that are expected to have similar impact upon the proposed design. In open pit design, this criterion is usually modified so that all discontinuities in a similar range of orientations (dip direction and dip) are designated as a single discontinuity set.

Slope angles within an open pit mine are influenced not only by geologic structure, rock mass strength and pore water pressures, but also by pit wall orientation and other operational considerations. The ultimate pits were evaluated for such regions of similar structural characteristics and pit slope orientation called “design sectors” which are expected to exhibit similar response to pit development.

Both the weathered and fresh rock domains at Minto are characterized by relatively strong intact rock strengths and by very similar discontinuity orientations. As such, pit slope design sectors were delineated based primarily on variations in structural (discontinuity) systems relative to mean pit wall orientations.

Field discontinuity measurements were converted into in-situ orientations and the combined data set of discontinuities was divided into categories of which, given significant persistency, had the potential to create structurally controlled failures. Plane shear and wedge type failures were evaluated for pit sectors assuming an average orientation of the pit walls in each sector.

Preliminary kinematic analyses indicated that the south and west sectors of Area 2, Area 118 and Ridgetop had potential for bench scale instabilities; consequentially, additional, back break analyses were carried out for those sectors. SRK's backbreak analyses use stochastic simulations of discontinuity properties (such as orientation, spacing, persistence, and shear strength) to analyze the likelihood for plane shear and wedge type failures to occur in a given bench configuration and orientation. The analyses yield a distribution of achievable bench face angles and catch bench widths. The interramp/overall and bench stability analyses together yield an optimized pit slope angle, providing of sufficient rock fall containment.

Results indicated that, based on the existing data, achievable mean bench face angles of approximately 64 degrees should be expected for the south and west sectors of Area 2 and Area 118. Due to the flatter discontinuity dips at Ridgetop relative to the anticipated shear strength of the discontinuities, steeper achievable bench face angles, on the order of 73 degrees, are expected for both Ridgetop pits.

While discontinuity analyses indicate that there is a slight potential for bench scale instability in the southwest section of the Minto North pit, the relatively low probability and the relatively small size of the pit, recommendations for Minto North are based on interramp slope angles alone.

Pit Slope Design Recommendations

Based on SRK's experience, interramp/overall slope angles that yield probabilities of failure of up to 30% for slopes with low failure consequences and approximately 5% to 10 % for high failure consequences are appropriate for most open pit mines. Slopes of high failure consequence are generally those slopes that are critical to mine operations, such as those on which major haul roads are established, those providing ingress or egress points to the pit, or those underlying infrastructure such as processing facilities or structures.

For certain geologic environments, the combination of the average anticipated bench face angle and the preferred interramp angle, based on global stability considerations, alone, do not provide a sufficiently wide average catch bench width to efficaciously control rockfall and/or overbank slough accumulation. In such instances, recommended interramp angles are flattened sufficiently to provide adequately wide average catch benches.

Based on the criteria described above, pre-feasibility pit slope design recommendations for each of the Minto areas are summarized in Table 17.5.

Table 17.5: Summary of Pit Slope Design Recommendations

Deposit Area	Sector(s)	Max. Slope Height (m)	Max Inter-ramp Angle (°)	Bench Face Angle (°)	Bench Height (m)	Berm Width (m)	Step-out Width* (m)
Area 2	Soil Overburden	50	30	30	-	-	15
Area 2	Rock – Northwest and Northeast	170	53	73	18	8	-
Area 2	Rock – South and West	210	47	64	18	8	
Area 118	Soil Overburden	18	30	30	-	-	15
Area 118	Rock - Northeast	35	53	73	18	8	-
Area 118	Rock - Southwest	36	47	64	18	8	-
Minto North	Soil Overburden	14	30	30	-	-	15
Minto North	Rock	125	52	72	18	8	-
Ridgetop - North	Soil Overburden	13	30	30	-	-	15
Ridgetop - North	Rock	132	53	73	18	8	-
Ridgetop - South	Soil Overburden	19	30	30	-	-	15
Ridgetop - South	Rock	78	53	73	18	8	-

* *Where soil overburden depths are anticipated to exceed 7 m, a 15 m offset or stepout should be incorporated at, or vertically near, the contact between the overburden and the bedrock.*

17.2 Underground Mining Geotechnical Assessment

Underground mining is now being proposed for deeper zones in Areas 2, 118 and Minto East.

Bieniawski Rock Mass Rating (RMR_{89}) and Barton Q values were evaluated for the underground zones. An average RMR_{89} of 65 and Q of 10 were estimated. Mining guidelines were then developed from empirical, analytical, numerical models and practical experience. Room and pillar mining designs with 10 m spans and 5 m by 5 m pillars yielding an extraction of 89% are recommended. Specifics of the recommendations are as follows:

- Stope Openings:
  - Span limit 10 m;
  - Pillars:
    - 5 x 5 up to 7 m height;
    - >7 m to 10 m height, shotcrete 2” and bolt to floor;
    - >10 m fill required.
- Support:
  - Minimum Standard (80% of stoping and pillars estimated) - Pattern Bolting (back, and to within 2 m from floor);
    - 8’ - ¾” fully grouted rebar, 1.2 m c-c;
    - #7 gage weld wire screen 4”by 4”;
  - Pillars >7 m (20% of pillars estimated):

    - Bolts to floor plus addition of 2" plain shotcrete.
  - Bad Ground Conditions or Spans > 10 m (20% of stoping estimated):
    - Standard 1. Plus addition of 12' Super Swellex or 5 m single cables on 2.0 m c-c, or
    - Standard 1. Plus shotcrete posts to reduce spans to 7 m.

- Development Openings:
  - Span limit 5 m, arch back;
  - Minimize 4-way intersections.

- Support:
  - Minimum Standard (80% of development estimated) - Pattern Bolting (back and ~1.5 m from floor);
    - 6' - ¾" fully grouted rebar, 1.2 m c-c:
    - #7 gage weld wire screen 4" x 4".
  - Intersections > 7 m inscribed circle - Pattern Bolting (back and ~1.5 m from floor):
    - 8' ¾" fully grouted rebar, 1.2 m c-c;
    - #7 gage weld wire screen 4" x 4".
  - Bad Ground Conditions or Spans > 10m (20% of development estimated):
    - 1 plus addition of 12' Super Swellex or 5 m cables.

- Raises:
  - Alimak 3 m by 5 m:
    - 6'- ¾" Rebar; 14 bolts per ring alternating pattern, 2 rings per 6-8' round.
  - Raisebore 4 m diameter:
    - 6' - ¾" Rebar; 10 bolts per ring alternating pattern, 1.3 m ring spacing.

Development mining standoff guidelines were also developed in consideration of long term extraction.

- Infrastructure Standoff Recommendations:
  - Recommendation based on 85-93% extraction;
    - 20 m above stope areas for the central ½;
    - 10 m above in outer ¼;
    - Abutment areas not an issue.

18 Mining Operations

18.1 Underground Mine

18.1.1 Introduction

The known deposits that have the grade, continuity and volume to be considered potentially mineable by underground operation are Area 118, Area 2, and Minto East.

An OP/UG cross-over study was done to determine the pit limit as described in Section 16.8 of this report. A 20 m crown pillar was assumed as a barrier between the pit walls, pit bottom and the proposed underground mine.

18.1.2 Underground Mining Context

There are well over 20 individual mineral deposits in general Area 2, 118 and East zones. The deposits range in size from hundreds of tonnes to hundreds of thousands of tonnes. The context or physical characteristics of each mineral deposit determine the appropriate mining method(s) that can be applied. The general characteristics of the UG Minto deposits are shown in Table 18.1.

Table 18.1: Deposit Context

Parameters	Unit	Value	Comment
Depth below surface	m	150-320	
Dip	deg.	10-30	
Thickness	m	3-25	10 m average
Size (aerial)	m	100x150	Average size
Production Capacity	t/vm	10,000	Approximate tonnes per vertical metre
Mineral Value	$/t NSR	90	Average value
Mineralization	Mineralized zones are visually and geochemically obvious due to density of visible sulphides and the degree of foliation.		
Continuity	The zones appear to be continuous over tens of metres.		
Regularity	The deposits appear to be well defined zones that are thick in the middle and thin toward the edges with sharp hangingwall and footwall contacts.		
Geotechnical	Generally very favourable rock conditions with strong granitic rock in deposit and in FW and HW. Some faulting but generally not seen to be a significant issue. No anticipated concerns with seismic activity created by mine excavations		
Hydrogeology	Not well defined, but tightness of the rock infers that there will likely not be hydrogeological issues.		
Constraints	There are no known constraints such as heat, radiation, groundwater or rock stress.		

18.1.3 Underground Mining Method Selection

The choice of mining method was determined after taking into consideration all of the known contextual factors of the Minto deposits. The main factors for determining an appropriate mining method were:

- The irregular geometry of the mineralization, varying thicknesses with 10 m average, and a 20° average dip angle, that makes the caving and sub-level open stoping mining methods not suitable for the deposit;
- The value of the ore, in most deposits cannot support the economics of a drift and fill method with cemented backfill;
- The pillar height over 10 m would require rockfill to provide pillar stability.

It was considered that the most suitable mining method for the Minto deposit would be a room and pillar (RAP) mining method. The method is simple and has numerous examples of success in low-dipping, moderately thick, shallow deposits with favourable rock conditions. The method allows excellent production capacity potential and relatively low cost while still providing mining flexibility and low dilution.

Productivity from room and pillar mines is normally very high due to multiple mining faces available, and has a simple, repetitive mining sequence. That fact that the method does not use backfill means that there is no time lost with a backfilling sequence temporarily constraining mining areas. Mining mobile equipment for RAP is the same as used in development mining, therefore, specialty equipment is not required.

The strong, massive nature of the Minto rock and shallow depth of the deposits mean that fairly high extraction ratios (plus 75%) would reasonably be expected.

For deposits with a vertical thickness of over 10 m, a hybrid post pillar cut and fill (“PPCF”) method is proposed. These thicker zones would require more than two 5 m high cuts making RAP undesirable due to the ore left behind in large pillars.

The PPCF method allows thick zones to be mined using minimal pillar widths by backfilling around the pillars after each lift is completed. The backfill provides confining forces around the pillars so their exposed height is kept to a minimum, thus minimizing their required volume.

Fill for the PPCF method normally comes from 2 sources; cycloned hydraulic tailings or waste rock. Waste rock was selected as the preferred type of fill, because it would be available from the mine development and from open pit operation, and would be brought into the mine on the ore truck backhaul.

18.1.4 Description of Room and Pillar Mining

RAP mining is an open stoping method that utilizes un-mined rock as pillars to support a series of rooms or small stopes around the pillars. The method normally is designed with pillars in a checkerboard pattern. See Figure 18.1. The pillars can be under survey control or done in a more random manner depending on the geotechnical needs. It is usually advantageous to leave lower grade rock in pillars so higher grade material can be mined. Pillars can sometimes be mined on retreat to help improve the overall mining extraction.

The RAP method is normally quite productive, flexible, and requires minimal access development before production starts (Figure 18.1).

At Minto, many of the mineralized zones are thicker than can be mined in a single pass. In these areas, a hanging wall (HW) cut will be made first, the back supported and then the bottom cut or bench taken out. This sequence enables the back to only be supported (rock bolted) once and would help the overall productivity. A two-boom development jumbo drill would be used for drilling both the initial HW drift and the bench. Based on the thickness of the mineralised zones, an estimate of the percent volume of each deposit that could be potentially benched, as opposed to drift mined, was calculated. Benching is more efficient than drifting and thus has a lower mining cost per tonne.

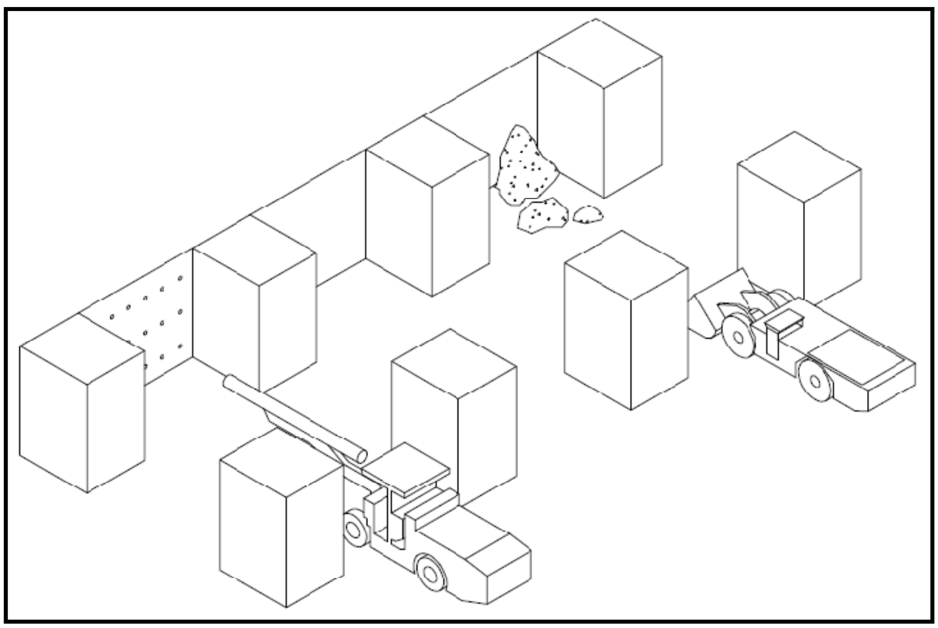


Figure 18.1: Simplified RAP Mining Method Illustration

18.1.5 Description of Post-Pillar Cut and Fill Mining

Post-pillar cut and fill ("PPCF") is a variation of cut and fill and has the advantage of being able to be used in thicker (plus 10 m), irregular-shaped deposits while keeping dilution and pillar sizes to a minimum. See Figure 18.2.

PPCF stoping is done using development jumbos or jackhammers. It is typically started by driving a -17% gradient decline access cross-cut from the main access decline on the footwall side of the mineralized zone, down to the bottom of the mining block.

The lowest two levels of each sub-level are mined in a room and pillar fashion with 15-20% of the ore remaining behind in regular, checker-board patterned pillars. An average typical face height (lift) is proposed to be 5 m.

Once mining of the two lifts is complete, the mined-out area would be filled with waste rock backfill obtained from the open pit or underground development.

To start the next lift, the access ramp would be slashed (breasted) at an appropriate gradient, up to plus 17%, to gain required elevation. The breasted waste rock would be left in place and is as a ramp. Once the ramp is re-established, room and pillar mining would begin again working off of the waste rock backfill.

Once established with sufficient headings, PPCF can be a very productive, repetitive mining method. Maintaining satisfactory production rates is based upon developing and following an efficient mining cycle of ground support, drilling, blasting, mucking, hauling and filling.

Once these cycle elements are understood by employees they can be optimized to achieve consistent mining rates with superior mining flexibility.

In order to maintain strength and continuity, the pillars of each lift must be surveyed and positioned exactly over the pillars from the previous cut. The main disadvantage is that it is development-style mining and an entry method requiring rock support on every cut.

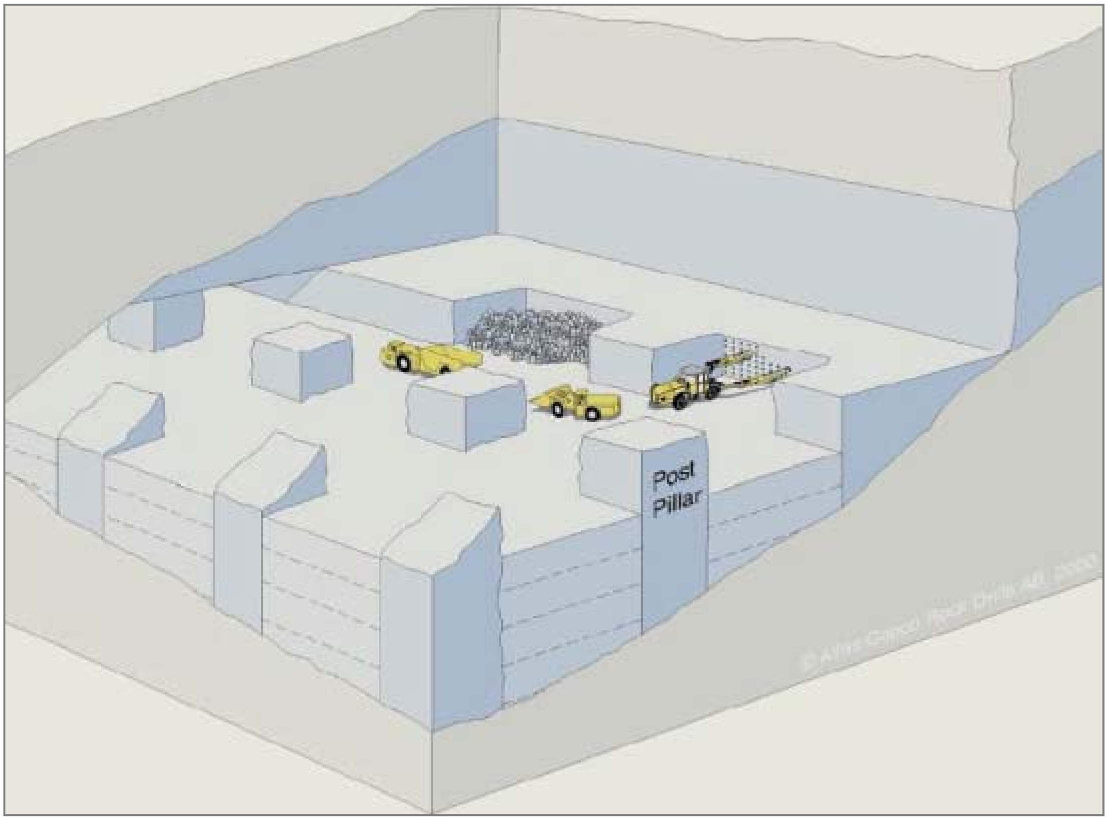


Figure 18.2: Simplified PPCF Mining Method Illustration (from Atlas Copco)

18.1.6 Mine Access

The main access to the ore body is proposed to be via a single decline developed at a -15% gradient. It will be used for ore and waste haulage, access for personnel, equipment, materials, and services. It would also be utilized as an exhaust airway.

A trackless decline is considered as the most appropriate mine access method for Minto ore body geometry, location of the production stopes, selected mining method, and mine life. It would provide early access to ore and the ability to start production at earlier stage, reduce initial capital cost, and provide potential access to the Copper Keel prospect.

The decline is proposed to be driven on the footwall side of the deposit and would provide multiple accesses to the orebody through the cross-cuts. See Figures 18.4 and 18.5.

The location of the decline portal was chosen in an area of minimal overburden, 40 m away from the proposed open pit and near the access road on the south of the Area 2 open pit.

The size of the decline was selected according to the mobile equipment size, required clearances, and ventilation requirements during development and production. It was estimated that a 5.0 m wide by 5.0 m high decline would be satisfactory for a 40 t truck (and 50 t trucks in the future, if desired) and ventilation requirements for 2,000 t/d production rate. See Figure 18.3. A 25 m ramp curve radius was assumed for convenience to drive a mobile drill jumbo.

Re-muck bays were planned to be developed every 150 m along the decline to allow efficient use of the drilling equipment and would hold two rounds of development muck. The re-muck bays would be of a similar size as the decline and would be typically 15 m long. After they are no longer used for development, the bays would be used for equipment storage, pump stations, drill bays, refuge chambers, etc.

Installation of 1.8 m fully grouted resin rebar bolts on the back and the walls of the ramp on 1.2 m x 1.2 m pattern, 100% mesh coverage and an allowance of 50 mm of shotcrete for 5% of the total length of the ramp was assumed for ground support.

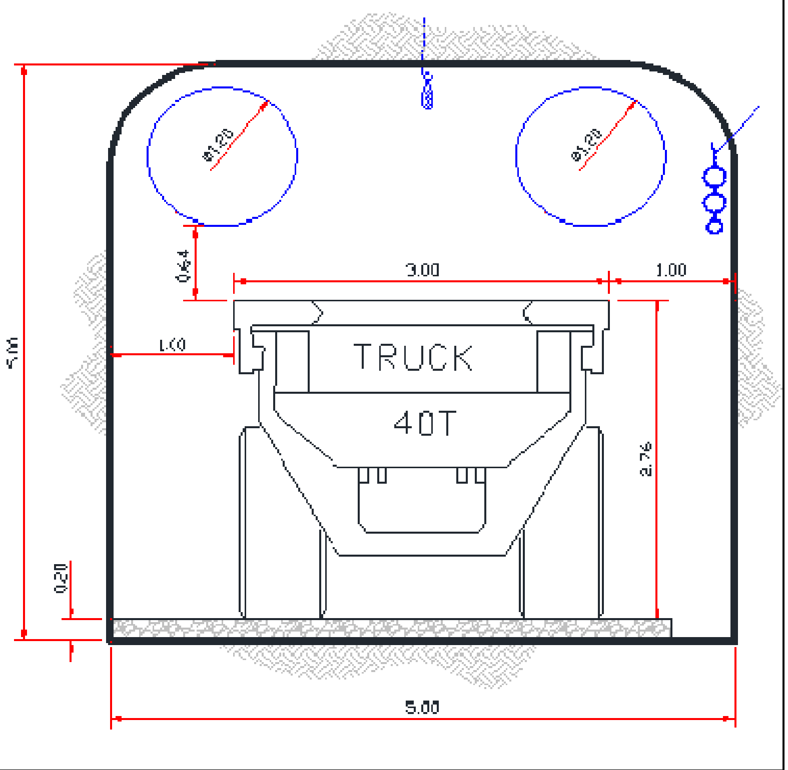


Figure 18.3: Proposed Decline Cross Section

Two ventilation raises with exits to surface were planned at strategic locations off of the main ramp and would provide ventilation circuit. They would have a man-way equipped with ladders and platforms to provide an auxiliary exit from the mine in case of emergency. The raises are planned to be rectangular and a size of 3.0 m x 5.0 m, which is defined by ventilation requirements and includes a manway of 1.0 m x 3.0 m.

An intake raise would be developed on the west side of the Area 2 pit with the raise collar in an area of minimal overburden. The top 145 m of the ventilation raise from the ramp elevation of 776 m to surface is planned to be developed using an Alimak. It is planned to extend the intake raise down with the main ramp to the bottom of the 101 stope, which will be in production in year 2012. Below the 776 m ramp elevation, the raise will be mined in four stages between sub-levels. The lower stages are short (from 15 m to 30 m long) and would be developed using drop raises technique or conventional method of raise development.

Ventilation access drifts would be developed to connect the level development and ramp to the ventilation raises. Those drifts would be 15 m to 40 m long and could be developed at -15% gradients to reduce length of the raise.

Two internal ventilation raises will be developed between the stopes in Area 2 to provide flow-through ventilation. All raises will be developed at 70^o inclination.

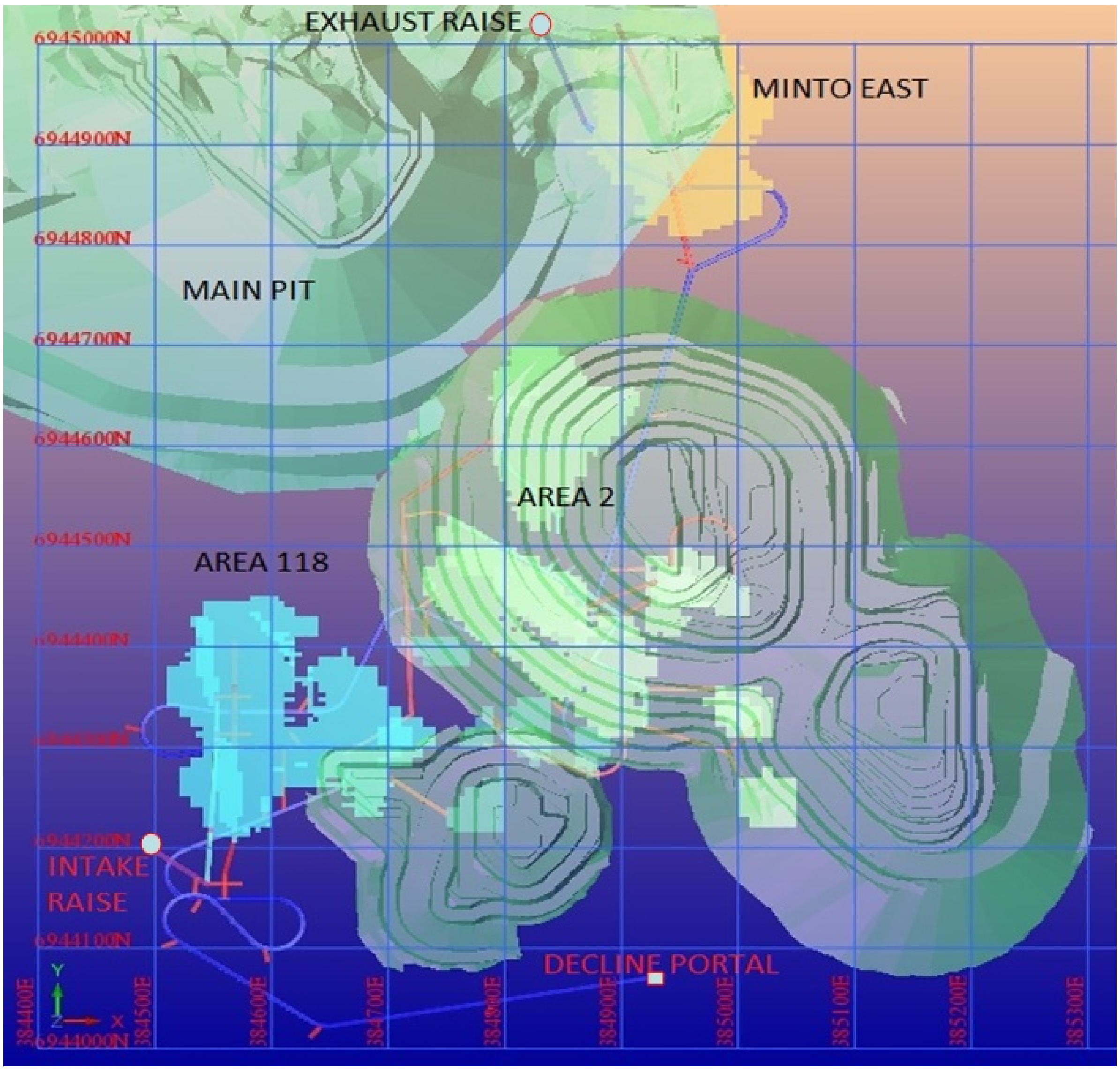


Figure 18.4: Mine Access Plan View

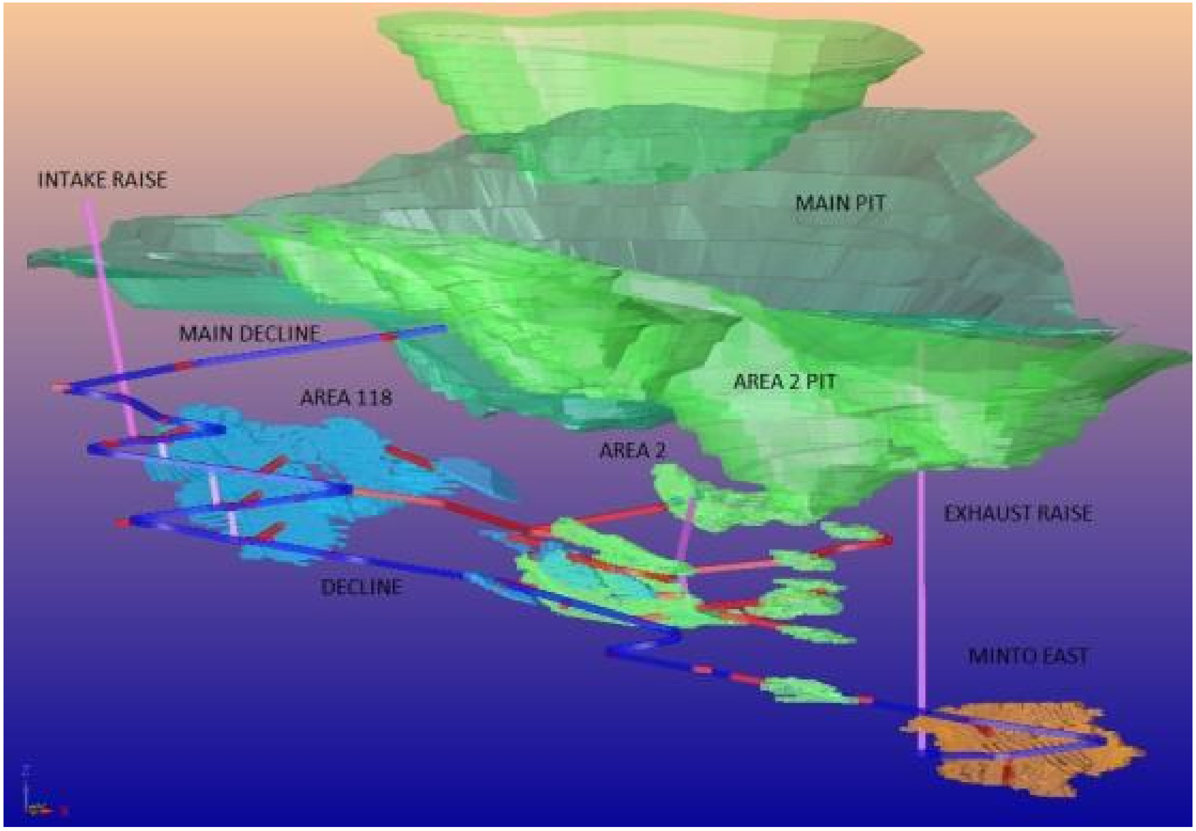


Figure 18.5: Mine Access Isometric View

Stoping

All mining is planned to be done with electric-hydraulic two-boom jumbos. Ground support would be done using mechanized rockbolters, and in some cases manually from the blasted rock muck pile, or from a scissor lift platform using jacklegs and stopers. Mucking was planned to be done using diesel LHDs loading underground trucks. The trucks would be loaded near the mining face. Mined-out rooms and cross-cuts were planned to be used as re-muck bays to store ore as needed to keep face advance moving without delay. The loaded haul trucks would transport the ore up to the 15% main access ramp out of the mine to the main mine stockpile area.

Ore and Waste Haulage

A combination of 5.4 m^3 (10 t) LHD and 40 t truck with 22 m^3 box was selected as the most economical option for ore and waste haulage at Minto underground mine. The equipment cycle times, productivities, mucking and trucking requirements, equipment operating and capital costs were considered to select an optimum equipment combination.

The waste rock from the development headings would be mucked by LHDs directly to the trucks or to remuck bays located up to 150 m from the face. The waste rock would then be hauled by the trucks to the waste dump on surface during the pre-production period.

When underground mine production commences, it would be possible to use mine waste rock from development as stope backfill along with the waste rock from the surface.

The broken ore from the stopes was planned to be mucked by stope LHDs to remuck bays, or loaded directly onto 40 t underground trucks. The trucks would be used to carry ore from the mine directly to the plant, or to a stockpile.

Table 18.2 describes the parameters were used for mucking and haulage productivities and fleet requirements estimates.

Table 18.2: Input Parameters for Mucking and Haulage Estimate

Description	Unit	LHD	Truck
Rock Swell Factor	%	50	50
Capacity	t	10	40
Actual Capacity	t	8.9	35.8
Bucket / Box Size	m^3	5.4	22
Fill Factor	%	90	90
Load / Dump / Manoeuvre	min	1.25	10.2
Speed Loaded (0%)	km/hr	5	15
Speed Empty (0%)	km/hr	8	15
Speed Loaded (+15%)	km/hr	5	6
Speed Empty (+15%)	km/hr	10	10
Speed Loaded (-15%)	km/hr	5	7.5
Speed Empty (-15%)	km/hr	15	15
Availability	%	80	80

The average truck haulage distances from the stope to the decline portal were estimated using the mine design model to estimate trucking productivities and cost. It was assumed that the trucks would travel approximately 700 m on surface from the portal to the ore stockpile.

The same trucks were scheduled to bring backfill material from the surface to mined-out stopes. Coordination of the trucks would be important to ensure safe and efficient haulage. The volume of waste rock from the potential underground excavations is insignificant when compared to the waste rock volumes generated by the open pits (see Table 18.3).

Table 18.3: Annual Material Movement and Haulage Fleet Requirements

Description	Units	2011	2012	2013	2014	2015
Ore Production	tonnes	12,780	630,000	730,000	730,000	337,794
Waste Production	tonnes	87,730	222,630	36,342	29,109	29,177
Total Material Mined	tonnes	100,510	852,630	766,342	759,109	366,971
Average Surface Haulage Distance	m	700	700	700	700	700
Average Ramp Haulage Distance	m	1,020	1,020	1,770	2,000	1,830
LHDs required	each	1	2	2	2	2
Trucks required	each	2	4	4	4	4

Mine Services

Ventilation

The design basis of the ventilation system at Minto underground operation was to adequately dilute exhaust gases produced by underground diesel equipment. Air volume was calculated on a factor of 0.06 m^3/s per installed kW of diesel engine power (100 cfm per installed hp). The kW rating of each piece of underground equipment was determined and then utilization factors, representing the diesel equipment in use at any time, applied to estimate the amount of air required.

Ventilation losses were included at 20% of the total ventilation requirements. Table 18.4 lists the air requirements for full production with the total of 140 m^3/s (296,000 cfm) air volume required.

Table 18.4: Ventilation Requirements at Full Production

Description	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m^3/s)
Jumbo (2-boom)	2	74	10	15	0.89
Rockbolter	2	55	20	22	1.32
LHD, 5.4 m3	2	220	80	352	21.12
Truck, 40 t	4	354	80	1,133	68.01
Grader	1	149	30	45	2.68
Explosives Truck	1	95	20	19	1.15
ANFO Loader	2	95	30	57	3.44
Cassette Carrier	2	112	50	112	6.71
Mechanics Truck	1	112	25	28	1.68
Scissor Lift	2	112	25	56	3.36
Supervisor Vehicle	3	95	20	57	3.44
Electrician Vehicle	1	95	30	29	1.72
Forklift/Tractor	1	63	20	13	0.76
Sub-Total				**1,938**	**117**
Losses	**20%**				**23**
Total Air Requirements					**140**

Air velocity in the main ramp was restricted to a range of 0.25 m/s to 6 m/s. This range was used to determine the size of development. The main intake fan would be installed on surface at the collar of the Area 118 intake ventilation raise and the main exhaust fan at the collar of the Minto East exhaust ventilation raise.

In the first year of production, fresh air was designed to be downcast through the main intake ventilation raise, and exhaust up-cast through the decline. This improves the quality of the ventilation since viceated air from the active stopes would then proceed past the haulage trucks on the main decline and is then exhausted to surface. Once the Minto East exhaust ventilation raise is developed and equipped with an exhaust fan, about 70% of total air would be exhausted through that raise and the remaining 30% will be viceated air that proceeds up the decline. No ventilation doors or regulators would be installed in the main decline as the exhaust fan will provide an appropriate air distribution between the mining areas.

Air movement to the stopes would be controlled by directing air flow with ventilation curtains and using the auxiliary ventilation fans. Ventilation regulators, doors, and bulkheads would also be used to control airflow in the mine.

The ventilation system design was modelled using VentSim Mine Ventilation Simulation Software (VentSim). This software allows input parameters including resistance, k-factor (friction factor), length, area, perimeter, and fixed quantities (volume) of air. The ventilation circuits during the initial production and full production are presented in Figure 18.6 and Figure 18.7, respectively.

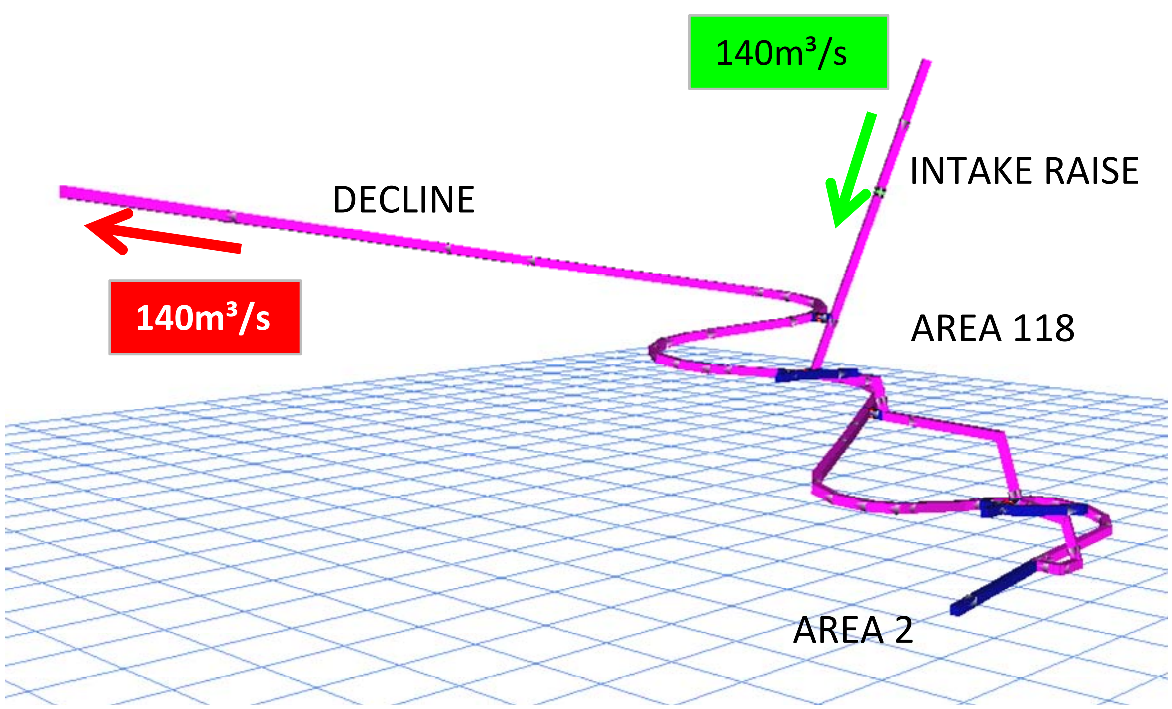


Figure 18.6: Ventilation Circuit at Start Production

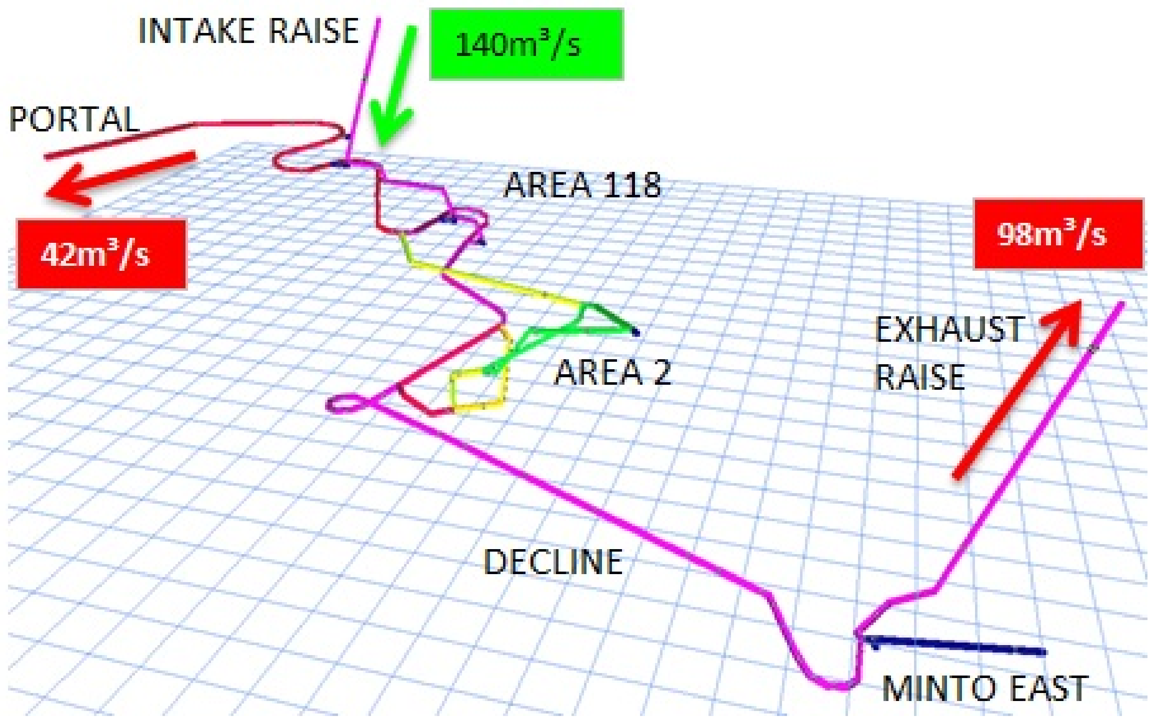


Figure 18.7: Ventilation Circuit at Full Production

Ventilation of Headings During Development

At least 35 m^3/s (75,000 cfm) of air would be required to dilute and remove exhaust from a 40 t truck, a 5.4 m^3 LHD, and a double-boom jumbo working in development heading as outlined in Table 18.5.

Table 18.5: Ventilation Requirements for Development Heading

Description	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m3/s)
LHD, 5.4 m3	1	220	100	220	13
Truck, 40 t	1	354	100	354	21
Jumbo, two-boom	1	111	10	11	1
Total				**585**	**35**

The requirements for auxiliary ventilation were estimated for the 1,300 m long development heading, the longest decline development distance. The auxiliary ventilation fans and ventilation ducts would be used to provide required amount of air at the development face. Only duct resistance was considered to calculate the pressure loss and power requirements as the resistance of the heading is negligible by comparison. Using Atkinson's equation for air flow in ducts, two 75 kW auxiliary fans with two separate 1.2 m diameter ventilation ducts were selected for the longest distance of decline development (Table 18.6).

Table 18.6: Atkinson Equation for Air Flow in Ventilation Ducts

Duct Diameter (m)	Duct Area (m^2)	Duct Perimeter (m)	Air Volume (m^3/s)	Duct Air Velocity (m/s)	Friction Factor (kg/m^3)	Duct Length (m)	Pressure Loss (kPa)	Power Required (kW)	Fan Power (kW)
Two Fans and Ducts									
1.0	0.79	3.14	18	22.4	0.003	1,300	7.7	135	181
1.2	1.13	3.77	18	15.5	0.003	1,300	3.1	54	73
1.3	1.33	4.08	18	13.2	0.003	1,300	2.1	36	49
1.4	1.54	4.40	18	11.4	0.003	1,300	1.4	25	34
1.5	1.77	4.71	18	9.9	0.003	1,300	1.0	18	24
Single Fan and Duct									
1.0	0.79	3.14	35	44.7	0.003	1,300	30.9	1,084	1,445
1.2	1.13	3.77	35	31.1	0.003	1,300	12.4	435	581
1.3	1.33	4.08	35	26.5	0.003	1,300	8.3	292	389
1.4	1.54	4.40	35	22.8	0.003	1,300	5.7	201	269
1.5	1.77	4.71	35	19.9	0.003	1,300	4.1	143	190

The 50 kW and 40 kW fans and 1.2 m diameter ducts would be used to provide auxiliary ventilation in other development headings and production stopes.

Mine Air Heating

At the request of Minto management, the mine will not be heated but will instead rely on a brine system to keep drilling water from freezing during the winter.

Preliminary investigations were performed by SRK to determine economic viability of using a brine drilling system instead of heating the underground mine. These investigations included discussions with mine equipment, instrumentation and chemical suppliers, as well as personnel who have worked with brine systems in the past. SRK estimates the operating expenditure associated with propane heating of the ventilated air is $1.97 annually versus brine at $90,000 annually.

The freezing point of calcium chloride brine can be lowered to -51°C if the concentration of calcium chloride ($CaCl_2$) is correct. Brine is typically made on surface and transported underground. It would be prepared using batch mixing in a tank with an agitator or using a brine saturator to create brine from bulk salt. It is recommended to keep all $CaCl_2$ brine at an average concentration of 20% for the following two reasons:

- To resist sudden temperature drops. If the brine is designed for a warmer temperature and the temperature unexpectedly drops, even for a short period of time, pipes can be blocked or damaged. Therefore, in the colder months the concentrate would be increased above 20%
- To resist corrosion. $CaCl_2$ brine is most corrosive to metal at concentrations between 2% to 6% and is much less corrosive at higher concentrations

Once mixed, the brine would be stored in a surface storage tank, allowing for large batches to be made at one time. The brine could then be distributed underground as required using trucks or through steel pipes.

All mobile equipment would be fitted with heated, enclosed cabs to help protect workers from exposure to low temperatures. These operating conditions are similar to those that have been used in other underground mines in Canada.

The operating cost of using the brine system was estimated from the first principles and was adjusted from brine consumption estimates obtained from similar operations. About $1.8M per annum would be saved in propane costs if the brine system would be used. Also, about $0.3M would be saved in capital costs due to eliminating the need for propane tanks and mine air heaters. To manage corrosion, increased operating costs are expected for corrosion management. Some mobile equipment may also be modified with corrosion resistant components.

In terms of health and safety, many companies operating in the arctic have used brine systems in the past including the Raglan Mine which has been using brine since 1997. The Quebec Ministry of Labour completed studies regarding vapours and other aspects with minimal concerns. This information will be requested by the site in 2011 in order to prepare training programs and ensure proper systems and personal protective equipment is in place before the use of brine commences.

Underground Electrical Power Distribution System

The major electrical power consumption in the mine would be from the following:

- Main and auxiliary ventilation fans;
- Drilling equipment;
- Mine dewatering pumps;
- Air compressors; and
- Maintenance shop

High voltage cable would enter the mine via the decline and be distributed to electrical sub-stations located near production stopes. The power cables would be suspended from the back of development headings. All equipment and cables would be fully protected to prevent electrical hazards to personnel.

High voltage power would be delivered at 4.16 kV and reduced to 600 V at electrical sub-stations. All power would be three-phase. Lighting and convenience receptacles would be single phase 120 V power.

Table 18.7 lists equipment power usage for underground mine.

Table 18.7: Power System Requirements for Underground Mine

Description	Quantity	Unit (kW)	Load Factor (%)	Utilization (%)	Power Consumption (kW/yr)
Surface					
Shop Equipment	1	50	80	20	70,080
Air Compressor	1	100	80	10	70,080
Lighting	1	10	80	60	42,048
Office	1	10	80	80	56,064
Parking Lot	1	10	80	30	21,024
Main Ventilation Fan	1	160	80	100	1,121,280
Pumps	1	10	85	67	49,888
Heat Trace	1	25	80	60	105,120
Underground					
Jumbo, two-boom	2	135	95	60	1,348,164
Rockbolter	2	70	95	60	699,048
Exploration Drill	1	75	95	50	312,075
Portable Compressor	2	100	80	30	420,480
Portable Welder	2	34	80	10	47,654
Auxiliary Fan, 75 kW	2	75	80	90	946,080
Auxiliary Fan, 50 kW	2	50	80	90	630,720
Auxiliary Fan, 40 kW	2	40	80	90	504,576
Refuge Chamber	2	5	80	100	70,080
Main Dewatering Pump	1	100	85	67	498,882
Portable Pump	3	15	85	50	167,535
Sub-total					7,180,879
Miscellaneous Power Allowance	10%				718,088
Total Power					**7,900,000**

Underground Communication System

A leaky feeder communication system would be used as the communication system for mine and surface operations. Telephones will be located at key infrastructure locations such as the electrical sub-stations, refuge stations, and main sump.

Key personnel (such as mobile mechanics, crew leaders, and shift bosses) and mobile equipment operators (such as loader, truck, and utility vehicle operators) would be supplied with an underground radio for contact with the leaky feeder network.

Explosives Storage and Handling

Explosives would be stored on surface in permanent magazines. Detonation supplies (NONEL, electrical caps, detonating cords, etc.) would be stored in a separate magazine.

Underground powder and cap magazines would be prepared near Area 2 production stopes. Day boxes would be used as temporary storage for daily explosive consumption.

Ammonium nitrate (AN) and fuel oil (FO) would be used as the major explosive for mine development and production. Packaged emulsion would be used as a primer and for loading lifter holes in the development headings. Smooth blasting techniques may be used as required main access development headings, with the use of trim powder for loading the perimeter holes.

During the decline development, blasting in the development headings would be done at any time during the shift when the face is loaded and ready for blast. All personnel underground would be required to be in a designated Safe Work Area during blasting. During production period, a central blast system would be used to initiate blasts for all loaded development headings and production stopes at the end of the shift. All blasting in the mine would be development-style blasting. No large scale blasts would be undertaken.

Fuel Storage and Distribution

An average fuel consumption rate of approximately 4,900 l/d is estimated for the period of full production as shown in Table 18.8.

Table 18.8: Underground Mining Fuel Consumption

Description	Quantity	Consumption (l/hr)	Load Factor (%)	Utilization (%)	Total Fuel (l/day)
LHD, 5.4 m3	2	57.5	75	80	1,179
Truck, 40 t	4	68.9	70	80	2,637
Jumbo, two-boom	2	22	50	10	38
Rockbolter	2	18	50	20	62
Grader	1	36	75	30	138
Explosives Truck	1	27	50	20	46
ANFO Loader	2	22	50	30	113
Cassette Carrier	2	27	70	50	323
Mechanics truck	1	22	50	25	47
Scissor Lift	2	27	50	25	115
Supervisor Vehicle	3	22	50	20	113
Electrician Vehicle	1	22	50	30	56
Forklift	1	16	60	20	33
Total					**4,900**

Haulage trucks, LHDs, and all auxiliary vehicles would be fuelled at fuel stations on surface. The fuel/lube cassette will be used for the fuelling/lubing of drills and rock bolters.

Compressed Air

The underground mobile drilling equipment such as jumbos, rockbolters and ANFO loaders would be equipped with their own compressors. No reticulated compressed air system was envisioned to be required underground.

Two portable compressors would be required to satisfy compressed air consumption for miscellaneous underground operations, such as: jackleg and stoper drilling, Alimak raise development and pumping with pneumatic pumps.

Water Supply

The major drilling equipment such as jumbos, rockbolters and exploration drills would use brine system in winter months as described in section "Mine Air Heating".

Mine Dewatering

Water volumes from underground are expected to be small due to the tight nature of the rock. Brine used for drilling should be recalculated as much as possible.

Development of the main sump is proposed at the bottom of the mine near the Minto East ventilation raise. It would be a typical two-bay sump to allow suspended solids to settle out of the water before pumping. Coarse material settled out in the main sump will be removed periodically by LHD and disposed. Old remuck bays would be utilized as temporary sumps during main access ramp development.

Water was planned to be pumped from the main sump by a high-pressure pump through a 150 mm diameter steel pipe located in the ventilation raise to the final tailing pump box on surface. The sump would be equipped with two high-head submersible pumps – one for operation and one on standby.

Transportation of Personnel and Materials Underground

All mine supplies and personnel would access the underground via the main access decline.

Two personnel vehicles would be used to shuttle employees from surface to the underground workings and back during shift changes. Supervisors, engineers, geologists, and surveyors would use diesel-powered trucks as transportation underground. Mechanics and electricians would use the mechanics' truck and maintenance service vehicles.

A boom truck with a 10 t crane would be used to move supplies, drill parts, and other consumables from surface to active underground workings.

Equipment Maintenance

Mobile underground equipment was envisioned to be maintained in a mechanical shop located on the surface. Some small maintenance and emergency repairs would be performed in a service bay underground. A mechanics truck would be used to perform emergency repairs underground.

A maintenance supervisor would provide a daily maintenance work schedule, ensure the availability of spare parts and supplies, and provide management and supervision to maintenance crews. The supervisor would also provide training for the maintenance workforce.

A maintenance planner would schedule maintenance and repair work, as well as provide statistics of equipment availability, utilization and life cycle. A computerized maintenance system is recommended to facilitate planning.

The equipment operators would provide equipment inspection at the beginning of the shift and perform small maintenance and repairs as required.

Mine Safety

The portable refuge stations would be provided in the main underground work areas. The refuge chambers are designed to be equipped with compressed air, potable water, and first aid equipment; they will also be supplied with a fixed telephone line and emergency lighting. The refuge chambers would be capable of being sealed to prevent the entry of gases. The portable refuge chambers would be move to the new locations as the working areas advance, eliminating the need to construct permanent refuge stations.

Fire extinguishers would be provided and maintained in accordance with regulations and best practices at the underground electrical installations, pump stations, fuelling stations, and other strategic areas. Every vehicle would carry at least one fire extinguisher of adequate size and proper type. It is recommended that underground heavy equipment would be equipped with automatic fire suppression systems.

A mine-wide stench gas warning system would be installed at the main intake raise to alert underground workers in the event of an emergency.

The main access decline would provide primary access and the ventilation raises with dedicated manway would be equipped with ladders and platforms providing the secondary exit in case of emergency.

Mine Equipment

Criteria used in the selection of underground mining equipment include:

- Mining method;
- Orebody geometry and dimensions;
- Mine production rate;
- Ventilation requirements;
- Operating and capital cost.

Table 18.9 lists underground mobile equipment selected for the 2,000 t/d mine production rate.

Table 18.9: Underground Mobile Equipment List

Equipment	Quantity
Drilling Equipment	
Development / Production Jumbo (2 boom)	2
Rockbolter	2
Loading & Hauling Equipment	
Production / Development LHD, 5.4 m^3 (10 t)	2
Haulage Truck, 40 t	4
Service Vehicles	
Grader	1
Explosive Truck	1
ANFO Loader	2
Cassette Carrier	2
Personnel Cassette	2
Boom Cassette	1
Fuel / Lube Cassette	1
Mechanics Truck	1
Scissor Lift	1
Supervisor/Engineering Vehicle	3
Electrician Vehicle - Scissor Lift	1
Shotcrete Sprayer	1
Transmixer	1
Forklift	1

The equipment list was developed based on the scheduled quantities of work and estimated from first principle cycle times and productivities (83% operational efficiency was used accounting for 50 min of usable time in one operating hour). Some other efficiency factors such as: 80% efficiency for the second boom on the drill jumbo, fill factors for LHD and trucks, additional time for travel, setup and teardown were used in cycle time estimations. The number of operating units was calculated based on 85% shift efficiency (shift change, lunch break, and equipment inspection time were excluded from the shift hours) and then converted to a fleet size by accounting for 80% equipment mechanical availability.

Stationary equipment was selected and would be installed and used for the following:

- Primary and auxiliary ventilation;
- Compressed air;
- Mine water management;
- Underground electrical;
- Communication;
- Mine safety;

- Explosives storage;
- Engineering equipment;
- Miscellaneous.

Personnel

The mining employees at the Minto underground operation were divided into two categories: salaried personnel, and hourly labour.

The personnel requirement estimates were based on the following:

- A 2,000 t/d production rate; and
- A crew rotation of two 12-h shifts per day with two crews working on site and two crews off.

A mining contractor would be used for the raise development. Contractor labour and supervisory staff is not included in this section.

The labour and personnel requirements described in this section were estimated for the production stage of the mine life.

Salaried personnel requirements, including engineering, technical, and supervisory staff, are listed in Table 18.10.

Table 18.10: Technical and Supervisory Staff

	Quantity
Staff Mine Operation	
Mine Superintendent	1
Senior Mining Engineer	1
Mine Ventilation/Project Engineer	1
Geotechnical Engineer	1
Geologist	1
Geological Technician	1
Mine Rescue / Safety / Training Officer	2
Surveyor	2
Mine Technical	1
Mine Captain	1
Mine Supervisor / Shift Boss	4
Total Operating Staff	**16**
Staff Mine Maintenance	
Maintenance Superintendent	1
Maintenance Planner	2
Mechanical / Electrical Foreman	1
Maintenance Supervisor/Shift Boss	2
Total Mine Maintenance Staff	**6**
Total Mining Staff	**22**

Hourly personnel were estimated based on production and development rates, operation productivities, and maintenance requirements. Personnel productivities were estimated for all main activities by developing cycle times for each operation.

Hourly labour requirements at full production are listed in Table 18.11.

Table 18.11: Hourly Labour

Labour Description	Personnel per Shift	Personnel per Day	Total Payroll
HOURLY MINE LABOUR			
Production / Development			
Jumbo Operator	2	4	8
Ground Support	2	4	8
Blaster	1	2	4
Haulage			
Scoop-Loader Operator	2	4	8
Truck Drivers	3	6	12
Mine Services & Safety			
General Labourer / Service Crew	1	2	4
Grader Operator	1	1	2
Utility Vehicle Operator/Nipper	1	2	4
General Helper	1	2	4
Sub-total Mine Operating	**14**	**27**	**54**
MINE MAINTENANCE			
Lead Mechanic / Electric	1	1	2
HD Mechanic, mobile	1	2	4
Mechanic, stationary	1	1	2
Electrician	1	2	4
Welder	1	1	2
Tireman / Instrument Man	1	1	2
Mechanic Apprentice	1	1	2
Dry / Lampman / Bitman	1	1	2
Sub-total Mine Maintenance	**8**	**10**	**20**
Total Mine Operating	**22**	**37**	**74**

18.2 Open Pit Mine Plan

Mine planning for the Phase V open pit deposits was conducted using a combination of Mintec Inc. MineSight® software and Gemcom GEMS™ and Whittle™ software. The 3-D mineral inventory model for Minto North was produced by Kirkham Geosystems Ltd., while the Area 2, 118 and Ridgetop models were created by SRK. Further NSR modelling was conducted by SRK using GEMS™. The detailed pit designs and production scheduling was undertaken with the use of MineSight®.

The 2011 Main Pit Budget, along with the ultimate Main Pit configuration (as compiled by MintoEx), was used to determine the starting point and remaining tonnages for the Main Pit portion of this pre-feasibility study. Based on the thorough analysis of the Whittle™ pit shells and preliminary schedules (discussed in Mineral Reserve section of the report), base case pit shells were chosen for the various Phase V deposits and used as the basis for the detailed ultimate pit designs for Area 2, 118, North and Ridgetop, along with associated pit staging. Waste dump were then designed to account for the material produced in each mining stage.

Table 18.12 below summarizes the detailed pit design tonnages and grades for each of the deposits (using the internal cut-off grade and dilution calculated above). Table 18.13 further summarizes the Minto pits by material types.

Table 18.12: Open Pit Design

Deposit	Diluted Ore (Kt)	Waste (Kt)	Total material (Kt)	Strip ratio (t:t)	Ore grade			Contained Metal		
					Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Au (koz)	Ag (koz)
Main Ore Stockpile	1,631	NA	1,631	NA	1.24	0.35	4.18	44	18	219
Main	618	1,631	2,249	2.6	1.64	0.75	6.40	22	15	127
Minto North	1,529	11,712	13,240	7.7	2.36	1.27	8.56	79	63	421
Ridgetop	1,337	8,403	9,740	6.3	1.11	0.32	2.85	33	14	122
118	491	2,257	2,748	4.6	1.29	0.09	1.73	14	1	27
Area2	4,820	34,179	39,000	7.1	1.32	0.47	4.53	140	72	703
Subtotal O/P	**10,426**	**58,182**	**68,608**	**6.6**	**1.45**	**0.55**	**4.83**	**333**	**183**	**1,619**

Table 18.13: Material by Type

Pit	Rock (kt)	Overburden (kt)	Sulphide Ore (kt)	Total Material (kt)
Main Ore Stockpile	NA	NA	1,631	1,631
Main	496	1,134	618	2,249
Minto North	9,649	2,063	1,529	13,240
Ridgetop	6,968	1,435	1,337	9,740
118	1,841	416	491	2,748
Area 2	25,127	9,052	4,820	39,000
Total	**44,081**	**14,101**	**10,426**	**68,608**

The open pit mining activities for the Minto pits were assumed to transition from the current contract mining to an owner-operator mine for this pre-feasibility study. This transition to an owner-operated mine has been assumed to commence in 2012 and correlates with the completion of mining in the Main Pit and the first stage of the Area 2 pit. The owner-operator mining cost unit rate used in the Whittle optimization was $2.20 per tonne of material for pit and dump operations, road maintenance and mine supervision. Technical services and senior management costs were incorporated into the G&A costs. The mining unit rate was calculated based on equipment required to achieve a processing rate of 1.46 Mtpa. Mining costs were developed from first principles for similar sized operations, along with labour, fuel and power costs supplied by MintoEx.

18.2.1 Mine Equipment

The major mining equipment requirements are indicated in Table 18.14 and are based on similar sized operations, as well as current practices at Minto. The proposed plant processing rate of 1.4 Mtpa was used to estimate the mining equipment fleet required. The fleet has an estimated maximum capacity of 25,000 tpd total material, which will be sufficient for the proposed LOM plan.

Table 18.14: Mine Equipment

No. of units	Equipment Type
1	Hitachi EX1900 Front Shovel
6	Cat 777F Haul Truck
1	Cat 992G Loader
1	Cat 365CL Excavator
3	Cat D9T Dozer
2	Cat 16 m Grader
1	Cat 824H Rubber-tired Dozer
1	Atlas Copco PV235 Drill
1	Atlas Copco D9-11 Drill
1	Cat 777C Water Truck
1	Cat 777B w/trailer

18.2.2 Unit Operations

The AC PV235 drill will perform the majority of the waste production drilling in the mine, with the smaller AC D9 drill used for secondary blasting requirements and may be used on the tighter spaced patterns required for pit development blasts. The main loading and haulage fleet consists of Cat 777F-100 ton haul trucks, which are loaded primarily with the diesel Hitachi EX1900 front shovel or the Cat 992G wheel loader, depending on pit conditions. As pit conditions dictate, the Cat D9 dozers are used to rip and push material to the excavators, as well as maintaining the waste dumps.

The portion of the equipment listed in Table 18.14 will be used to maintain and build access roads, and to meet various site facility requirements, (including coarse mill feed stockpile maintenance and further exploration development).

The work schedule is based on two 12 hour shifts, seven days a week, 365 days per year.

18.2.3 Grade Control

In order to minimize ore dilution, maximize ore recovery, and thereby improve the operation's overall economics, grade control will play an important role throughout the mining process.

Grade control begins with the proper identification of the ore/waste zones and contacts in the field through;

- Information obtained from up-to-date 3-D resource model;
- Blast hole sampling;
- Driller reports;
- Face sampling (includes mapping, visual inspections, sampling); and
- Trenching (as required, to provide better definition of ore/waste contacts, sampling).

Once the above information has been gathered and compiled it will be communicated to operational personnel through;

- Daily/weekly production meetings;
- Detailed "dig" maps – outlining ore zones, waste contacts, faults; and
- Field surveying and layout of dig limits, ore contacts, trenching required.

In order to maintain the effectiveness of the grade control process; regular field inspections will be undertaken by engineering/geology personnel; and clear lines of communication will be maintained with operational personnel, including equipment operators and front line supervisors.

As part of the grade control process, variable bench heights may be necessary in order to maximize the ore recovery. These include: variable bench heights in waste in order to target the top of the ore zone; and a varying bench height within the ore zones (reduce height at the periphery of the zone). Drill and blast control will also play an important role in order to minimize dilution of the ore zones during the blasting process (e.g. minimize heave in the ore zone)

18.3 Production Schedule

18.3.1 Mine Sequence and Phasing – Open Pit and Underground

The detailed pit designs for the various deposits for Minto were divided into various stages for the mine plan development to maximize grade in the early part of the schedule, reduce pre-stripping requirements, incorporate underground production, while providing the required mill feed production per period. The overall Phase V site plan final configuration is illustrated in Figure 18.8 below.

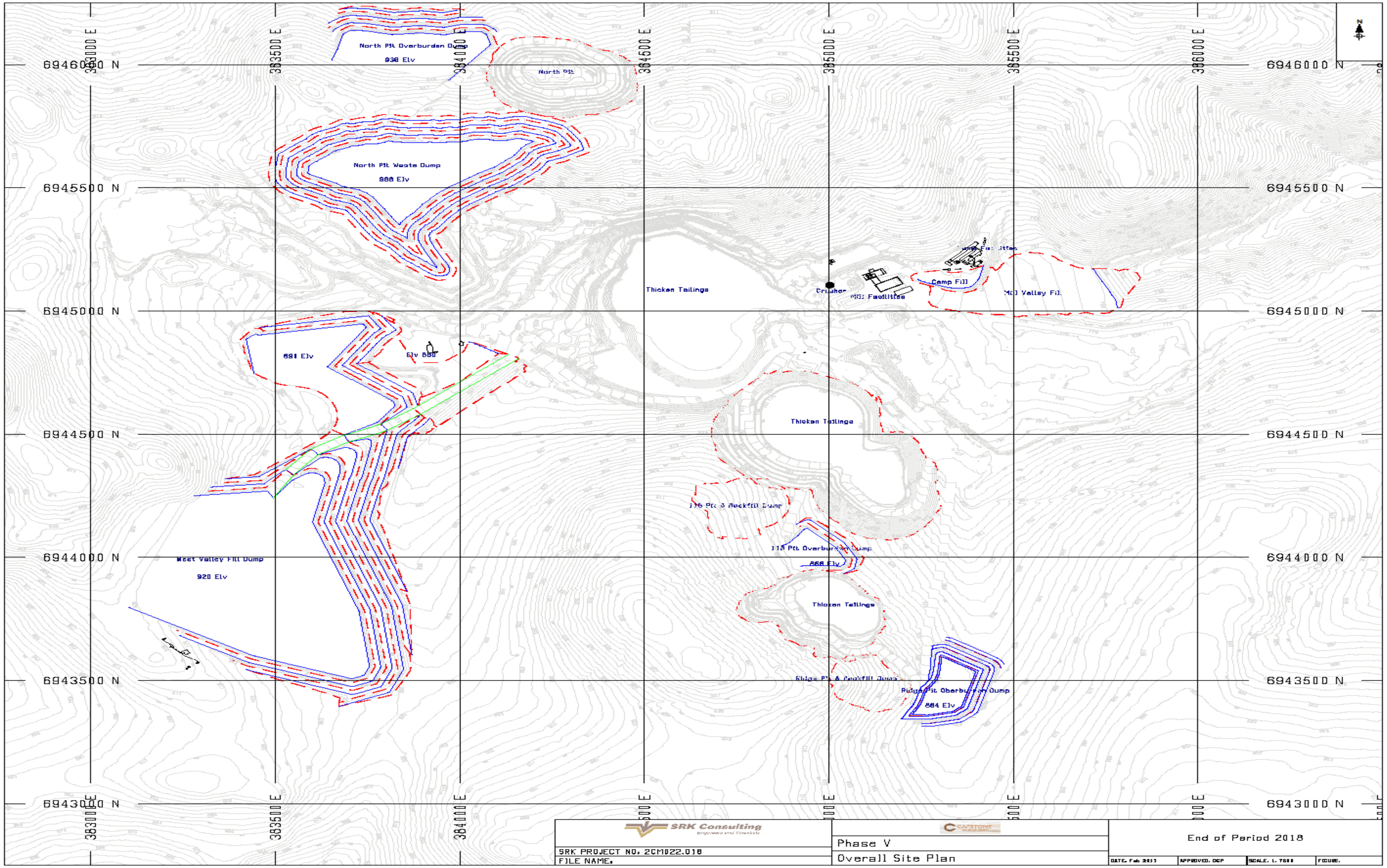


Figure 18.8: Overall Site Plan Final Configuration

The mining sequence, which mines higher grade material early on in the schedule, begins with completion of the Main Pit. This will allow processed tailings from the Phase V pits to be backfilled into the Main Pit, thereby, eliminating the current need of drying the tailings and significantly reduce overall costs. The construction of the Main Pit buttress (required for slope stability) is required prior to the placing of tailings in Main Pit.

Mining of the Minto Main Pit will be followed by mining of first two stages of Area 2, Minto North, 118, Ridgetop, and ends with the final stages of the Area 2 deposit. Underground production supplements the open pit mine ore feed.

During the initial pre-stripping of the Phase V pits, the mill feed will be supplemented with stockpiled ore from the Main Pit in order to attain the scheduled mill throughput, while maintaining highest possible copper head grades. Ridgetop has been split into two pits, North and South. Area 2 Pit has been divided into four stages. The underground production will come from Area 2, Area 118 and Minto East). The stage tonnages and associated grades are summarized in Table 18.15 for both OP and UG, while a breakdown of material types for the OP deposits only is summarized in Table 18.16.

Table 18.15: OP and UG Stage Tonnages and Grades

Stage	Stage Quantities									
	Diluted Ore (Kt)	Waste (Kt)	Total material (kt)	Strip ratio (t:t)	Ore grade			Contained Metal		
					Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Au (koz)	Ag (koz)
OPEN PIT										
Main Ore Stockpile	1,631	NA	1,631	NA	1.24	0.35	4.18	44	18	219
Main	618	1,631	2,249	2.6	1.64	0.75	6.40	22	15	127
Subtotal Main	**2,249**	**1,631**	**3,879**	**2.6**	**1.35**	**0.46**	**4.79**	**67**	**33**	**346**
Subtotal Minto North	**1,529**	**11,712**	**13,240**	**7.7**	**2.36**	**1.27**	**8.56**	**79**	**63**	**421**
Ridgetop South	270	1,913	2,183	7.1	1.33	0.78	6.69	8	7	58
Ridgetop North	1,067	6,490	7,557	6.1	1.05	0.20	1.88	25	7	64
Subtotal Ridgetop	**1,337**	**8,403**	**9,740**	**6.3**	**1.11**	**0.32**	**2.85**	**33**	**14**	**122**
Subtotal 118	**491**	**2,257**	**2,748**	**4.6**	**1.29**	**0.09**	**1.73**	**14**	**1**	**27**
Area2 - Stage 1	1,330	11,815	13,145	8.9	1.43	0.53	5.23	42	23	223
Area2 - Stage 2	2,220	13,376	15,595	6.0	1.38	0.50	4.52	67	36	323
Area2 - Stage 3	598	5,025	5,623	8.4	1.22	0.41	4.47	16	8	86
Area2 - Stage 4	673	3,963	4,636	5.9	0.98	0.27	3.28	15	6	71
Subtotal Area 2	**4,820**	**34,179**	**39,000**	**7.1**	**1.32**	**0.47**	**4.53**	**140**	**72**	**703**
SUBTOTAL O/P	**10,426**	**58,182**	**68,608**	**6.6**	**1.45**	**0.55**	**4.83**	**333**	**183**	**1,619**
UNDERGROUND										
Area 118	764	338	1,102	NA	1.78	0.69	7.17	30	17	176
Area 2	967	NA	967	NA	1.73	0.75	6.77	37	23	211
Minto East	709	NA	709	NA	2.28	1.04	6.15	36	24	140
SUBTOTAL U/G	**2,441**	**338**	**2,778**	NA	**1.90**	**0.82**	**6.71**	**102**	**64**	**527**
GRANDTOTAL O/P and U/G	**12,866**	**58,520**	**71,386**	NA	**1.53**	**0.60**	**5.19**	**435**	**247**	**2,146**

**strip ratio does not include Main Ore Stockpile starting balance*

Proven and Probable reserves only (Inferred included in Waste)

Table 18.16: OP Material Types

Phase	Rock (kt)	Overburden (kt)	Sulphide ore (kt)	Total Material (kt)
Main*	496	1,134	2,249	3,879
Minto North	9,649	2,063	1,529	13,240
Ridgetop South	1,562	351	270	2,183
Ridgetop North	5,405	1,084	1,067	7,557
118	1,841	416	491	2,748
Area2 - Stage 1	9,296	2,519	1,330	13,145
Area2 - Stage 2	11,562	1,814	2,220	15,595
Area2 - Stage 3	1,463	3,563	598	5,623
Area2 - Stage 4	2,807	1,156	673	4,636
Grand total	**44,081**	**14,101**	**10,426**	**68,608**

Note: Main pit includes ore stockpile start balance

Figure 18.9 further summarizes the stage designs for each of the OP deposits illustrating mineralized rock and waste tonnages, and copper grade.

The pit stages were based on the detailed pit designs created. Phase V OP and UG mining will see the expansion of the existing Main Waste Dump, the Overburden dump, as well as the Valley Fill Dump. Backfilling of Ridgetop and Area2 pits with also be required and the creation of a Mill Valley Dump will also be required. All process plant feed rock will be hauled to the appropriate ROM ore stockpiles.

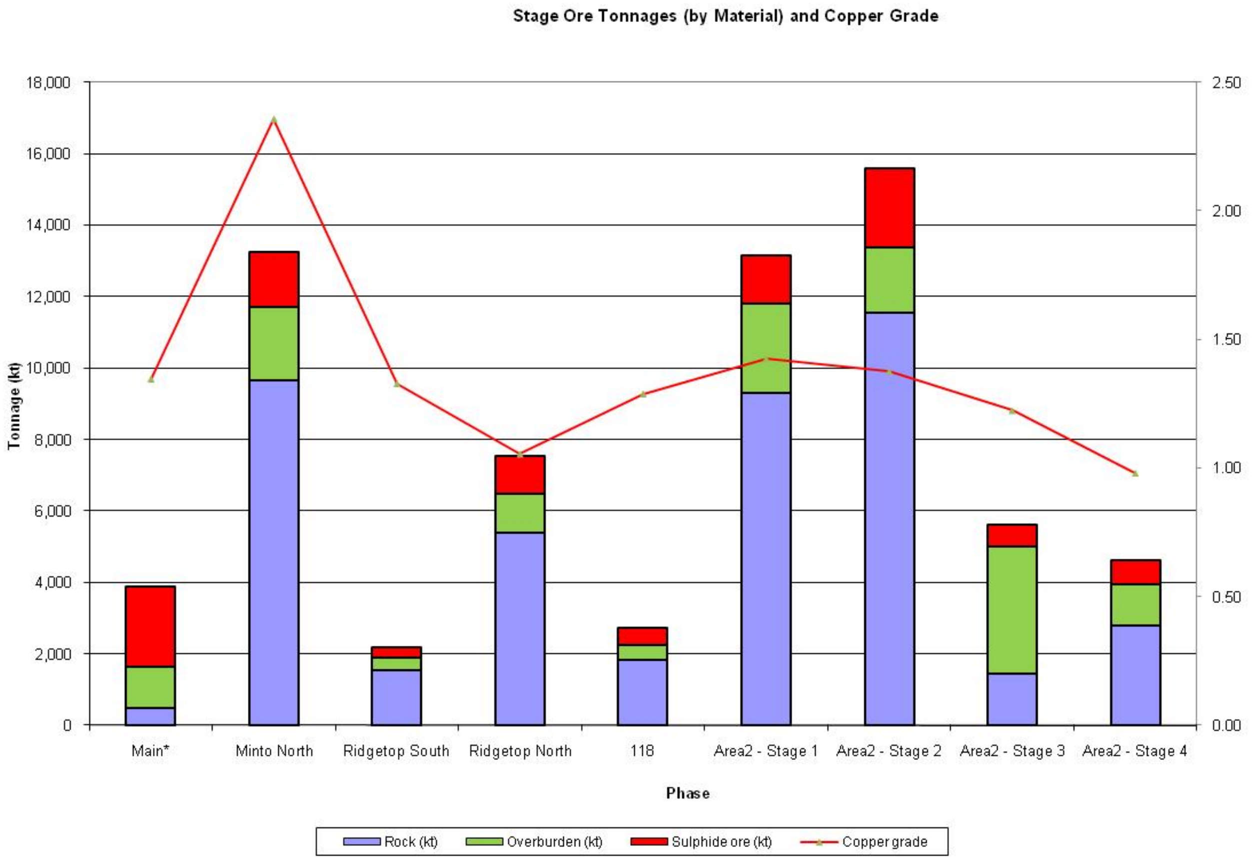


Figure 18.9: Stage Summary

18.3.2 Mine Production Schedule – Open Pit and Underground

The production schedule for the Minto deposits was developed with the aid of MineSight™ software, and incorporated the open pit deposits at Main, North, Ridgetop, 118 and Area 2, as well as the underground components of Area 2, 118 and Minto East, mentioned above.

The planned ramp up in plant throughput is as follows:

- 2011 at an average of 3,442 tpd;
- 2012 and beyond at an average of 3,750 tpd.

Completion of Main Pit will be carried out first and is scheduled to be completed by early 2011. Main pit production will be followed by the Phase IV Area 2 pit mining concurrent with UG exploration decline development. Upon approval of the Phase V permit, assumed to be on October 1, 2012, production will commence from the Phase V pits starting with Minto North. All Area 2 Stage 1 and a portion of stage 2 mining is planned to be mined by Pelly Construction. The Minto open pit fleet will mine Minto North, Ridgetop, 118 and the remaining Area 2 stages.

The maximum production rate from the Phase V open pits is approximately 26,700 tpd. The average total open pit mining rate is planned to be 23,000 tpd. The maximum underground production rate is 2,000 tpd of ore. Only measured and indicated resources were used in the LOM plan.

Table 18.17 summarizes the total material movement by year for the mine production schedule, with Table 18.18 summarizing the process schedule.

Table 18.17: Mine Production Schedule

Parameter	Unit	Total	2011	2012	2013	2014	2015	2016	2017	2018
Total OP/UG										
Overburden	kt	**14.10**	**3.55**	**2.04**	**0.87**	**0.98**	**0.50**	**1.44**	**4.57**	**0.15**
Rock	kt	**44.42**	5.48	8.51	7.21	4.73	6.96	6.09	3.25	2.19
Total Waste	kt	**58.52**	9.03	10.55	8.08	5.72	7.46	7.53	7.82	2.34
ROM ore	kt	**11.24**	0.82	1.97	1.01	2.53	1.55	1.51	1.01	0.82
Cu Grade	%Cu	**1.58**	1.47	1.56	1.73	2.14	1.36	1.40	1.14	1.07
Au Grade	g/t	**0.63**	0.61	0.58	0.80	0.98	0.51	0.53	0.31	0.33
Ag Grade	g/t	**5.33**	5.50	5.78	5.23	7.21	4.81	4.58	2.74	4.05
Total Mined Cu	Mlbs Cu	**391**	27	68	38	120	47	47	25	19
Total Mined Au	koz Au	**229**	16	37	26	80	26	26	10	9
Total Mined Ag	koz Ag	**1927**	145	366	170	587	240	223	89	107
ROM ore	t/day	**3,848**	2,243	5,392	2,770	6,940	4,260	4,150	2,776	2,252
Total Material	t/day	**22,937**	26,701	32,071	22,834	20,595	23,664	24,768	24,196	8,669
Total OP PELLY FLEET only										
Overburden	kt	**4.40**	3.55	0.85	-	-	-	-	-	-
Rock	kt	**12.60**	5.39	7.21	-	-	-	-	-	-
Total Waste	kt	**17.00**	8.94	8.06	-	-	-	-	-	-
ROM ore	kt	**2.14**	0.81	1.34	-	-	-	-	-	-
Cu Grade	%Cu	**1.44**	1.47	1.42						
Au Grade	g/t	**0.56**	0.61	0.53						
Ag Grade	g/t	**5.24**	5.47	5.09						
Total Mined Cu	Mlbs Cu	**68**	26	42						
Total Mined Au	koz Au	**38**	16	23						
Total Mined Ag	koz Ag	**361**	142	219						
Strip ratio	t:t	**7.9**	11.1	6.0						
ROM ore	t/day	**2,937**	2,208	3,666						
Total Material	t/day	**26,227**	26,701	25,754						
Total OP MINTO FLEET only										
Overburden	kt	**9.70**	-	1.19	0.87	0.98	0.50	1.44	4.57	0.15
Rock	kt	**31.48**	-	1.11	7.18	4.73	6.92	6.09	3.25	2.19
Total Waste	kt	**41.18**	-	2.31	8.05	5.71	7.42	7.53	7.82	2.34
ROM ore	kt	**6.65**	-	-	0.28	1.80	1.22	1.51	1.01	0.82
Cu Grade	%Cu	**1.50**			1.02	2.22	1.27	1.40	1.14	1.07
Au Grade	g/t	**0.59**			0.56	1.02	0.45	0.53	0.31	0.33
Ag Grade	g/t	**4.86**			2.72	7.37	4.30	4.58	2.74	4.05
Total Mined Cu	Mlbs Cu	**220**			6	88	34	47	25	19
Total Mined Au	koz Au	**126**			5	59	17	26	10	9
Total Mined Ag	koz Ag	**1039**			25	427	168	223	89	107
Strip ratio	t:t	**6.2**			28.7	3.2	6.1	5.0	7.7	2.8
ROM ore	t/day	**3,490**		0	770	4,940	3,334	4,150	2,776	2,252
Total Material	t/day	**20,788**		6,317	22,834	20,595	23,664	24,768	24,196	8,669
Total UG only										
Rock	kt	**0.34**	0.09	0.18	0.02	0.00	0.04	-	-	-
Total Waste	kt	**0.34**	0.09	0.18	0.02	0.00	0.04	-	-	-
ROM ore	kt	**2.44**	0.01	0.63	0.73	0.73	0.34	-	-	-
Cu Grade	%Cu	**1.90**	1.84	1.84	2.00	1.95	1.71	0.00	0.00	0.00
Au Grade	g/t	**0.82**	0.69	0.69	0.90	0.89	0.75	0.00	0.00	0.00
Ag Grade	g/t	**6.71**	7.23	7.23	6.20	6.81	6.64	0.00	0.00	0.00
Total Mined Cu	Mlbs Cu	**102**	1	26	32	31	13	0	0	0
Total Mined Au	koz Au	**64**	0	14	21	21	8	0	0	0
Total Mined Ag	koz Ag	**527**	3	146	145	160	72	0	0	0
ROM ore	t/day	**836**	35	1,726	2,000	2,000	925	0	0	0
Total Material	t/day	**952**	272	2,233	2,067	2,009	1,032	0	0	0

Table 18.18: Processing Production Schedule – Minto Deposits

		2011-2020 Total	Y E A R									
Parameter	**UNIT**		**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**
Mill Feed Rate	dmt/day	**3,718**	3,442	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750
Mill Feed Total	Mt	**12.9**	1.256	1.373	1.369	1.369	1.369	1.373	1.369	1.369	1.369	0.653
Feed Grade	Cu %	**1.53**	1.60	1.86	1.70	2.86	1.62	1.68	1.11	0.96	0.78	0.78
	Au g/t	**0.6**	0.6	0.7	0.7	1.5	0.6	0.7	0.3	0.3	0.2	0.2
	Ag g/t	**5.2**	6.0	7.2	5.4	10.0	5.8	5.7	2.9	3.2	2.2	2.2
Recovery to Conc.	Cu	**92.0%**	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%
	Au	**70.0%**	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%
	Ag	**78.0%**	78.0%	78.0%	78.0%	78.0%	78.0%	78.0%	78.0%	78.0%	78.0%	78.0%
Conc. Grade	% Cu	**39%**	41.5%	38.0%	39.0%	38.0%	38.0%	38.0%	38.6%	38.6%	38.7%	38.7%
Conc. Production	dmt	**470,478**	44,633	61,728	54,846	94,937	53,519	55,956	36,066	31,201	25,449	12,134
Conc. Metal	Mlb Cu	**400.4**	40.8	51.7	47.2	79.5	44.8	46.9	30.7	26.6	21.7	10.4
	oz Au	**173,146**	16,807	22,531	22,259	45,118	19,488	20,538	9,375	8,246	5,948	2,836
	oz Ag	**1,673,940**	188,612	246,680	184,654	341,791	197,647	194,509	100,627	110,290	73,897	35,234

With an assumed schedule start date of January 2011, the Minto open pit and underground mine is planned to produce a further of 12.9 million tonnes (Mt) of mill feed (includes Main Pit stockpile balance at start of schedule) and 58.5 Mt of waste rock over an 7.5-year mine operating life (yielding an overall strip ratio of 6.6:1 (t:t). The mine schedule focuses on achieving the required plant feed production rate, mining of higher grade material early in schedule, incorporating underground production, while balancing grade and strip ratios.

Mill operations continue for an additional 2 years, processing the accumulated 2.0 Mt of ore stockpiled when mining ceases, for a total mill operating life of 9.5 years. The ROM stockpiles are used in the schedule in order to smooth out mill head grades and provide required mill feed as required. Figure 18.10 summarizes mined ore production tonnages and grade by period and area.

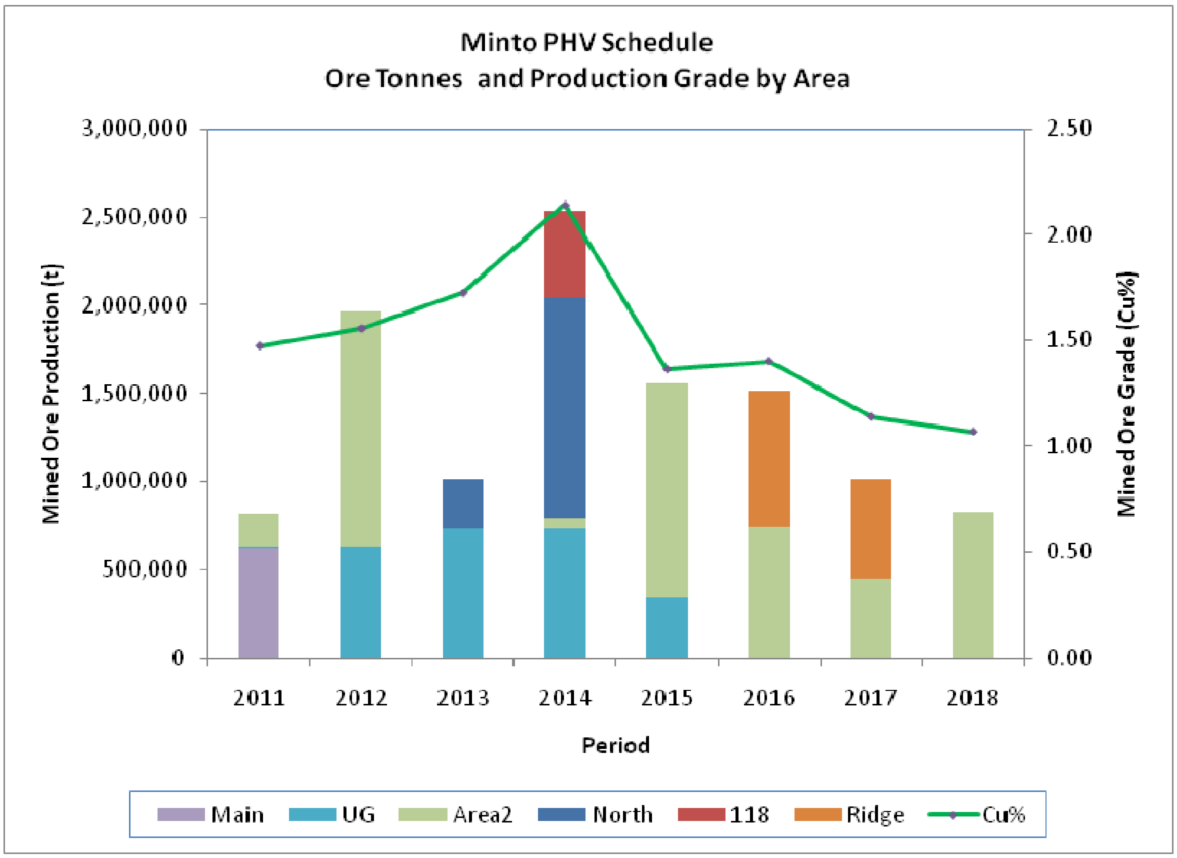


Figure 18.10: Period Ore Tonnage and Copper Grade

To further illustrate the progression of mining and processing of the Minto deposits, Figure 18.11 provides a timeline of the mine production tonnage and grade from each of the areas as well as mill feed tonnes and copper head grade. Figure 18.12 summarizes open pit annual mined benches from each stage.

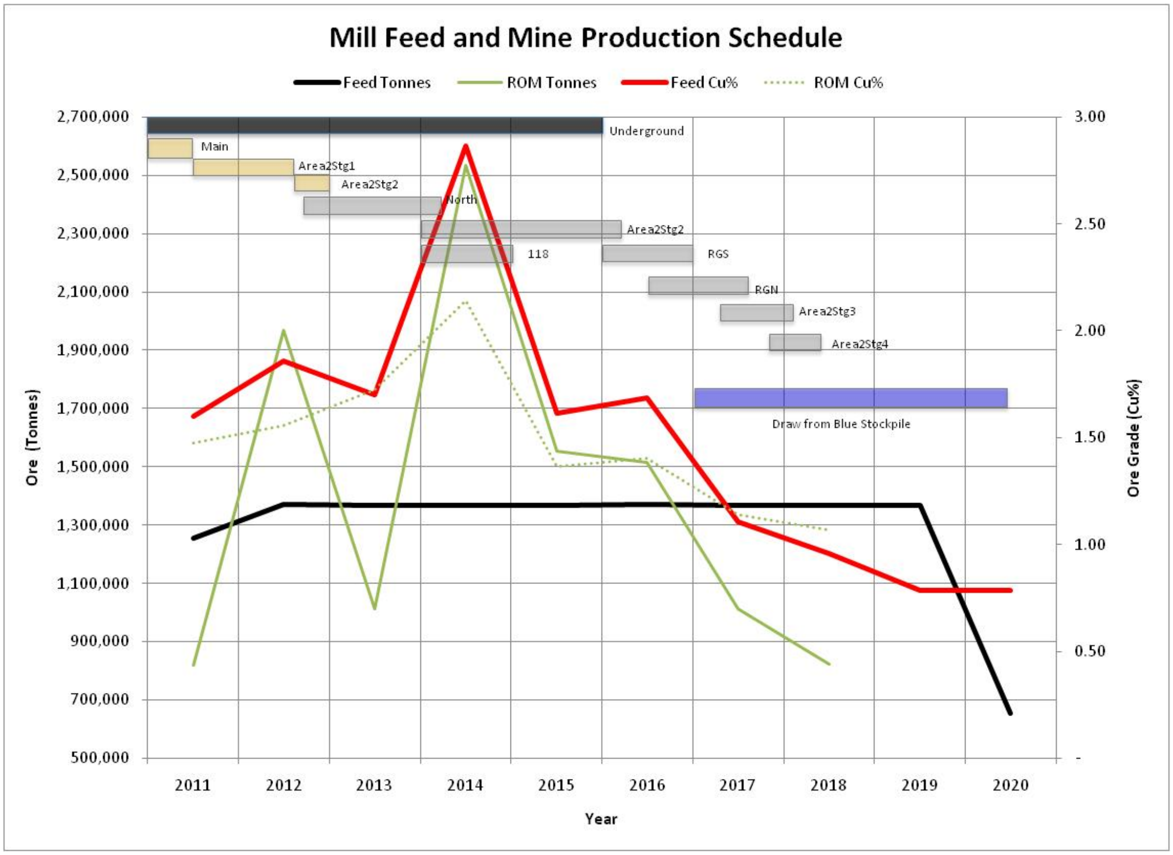


Figure 18.11: Phase V Production Timeline

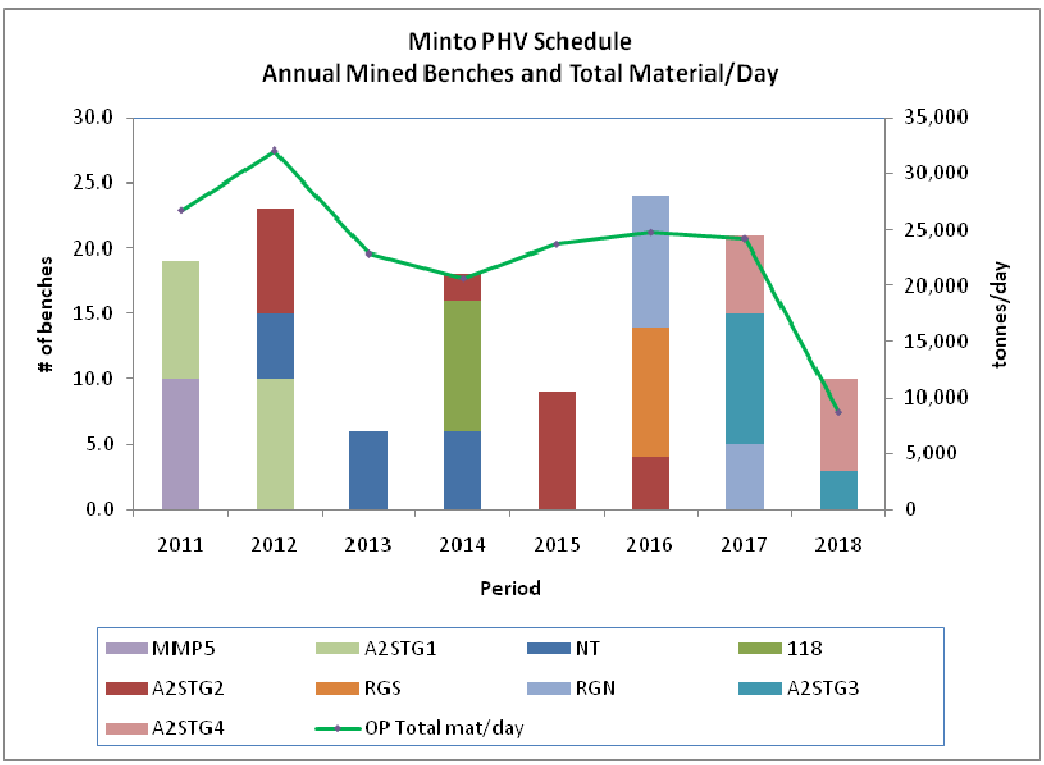


Figure 18.12: Annual Mined Benches

To further illustrate the progression of mining of the Minto deposit, Table 18.19 provides the OP and UG mined ore tonnages for each period by stage. Section 28 of this report contains end of period maps that provide the status of the pit configuration, dump advance, as well as the tailings management facility, at the end of years 2011 through to the end of mining in 2018, respectively.

Table 18.19: Mined Ore Tonnage by Stage and Period

Area	Unit	2011	2012	2013	2014	2015	2016	2017	2018
Minto Main	Kt	618							
UG	Kt	13	630	730	730	338			
Minto North	Kt			281	1,248				
Ridgetop	Kt						773	564	
A2 Stage 1/2	Kt	188	1,338		65	1,217	742		
118	Kt				491				
A2 Stage 3/4	Kt							449	822

Pit/Underground Development

2011: Mining of the Main Pit is completed by second quarter. Transition stage from the Main Pit to Phase V pits with Area 2 open pit mining commencing in the second quarter. A waste rock buttress is constructed on the south side of the Main Pit and tailings converted from dry-stack method to thickened tailings and deposition back into the Main Pit commences. Underground development commences and mining in Area 118. A total of 0.8 Mt of plant feed is mined in the year, with the balance of the mill feed required drawn from existing stockpile inventories. Mill head grade for the year averages 1.60% Cu. 8.9 Mt of waste is produced for a mined strip ratio of 11.1:1 (waste: ore).

2012: Mining of Stage 1 of Area 2 open pit completed and Stage 2 is started by Pelly. It is assumed that Phase V permit approval in granted by Oct 1, 2012. Minto North mining begins at this time using the Minto fleet. Underground production continues. Open pit ore production is planned to be 1.34Mt at a strip ratio of 7.7:1. Underground ore production 0.6 Mt at a mined grade of 1.84 %Cu. Processing rate averages 3,750 tpd for the year.

2013: Minto North pit continues production but is mainly waste removal. The open pit waste produced over the period totals 8.1 Mt for a 28:1 strip ratio. Underground production reaches steady state of 2,000 ore tpd. Average total mined grade is 1.73% Cu. Processing rate maintained at 3,750 tpd with a mill head grade of 1.70% Cu.

2014: Minto North finishes in the first part of the year and provides high-grade ore to the mill. Mining restarts at Area 2 with Stage 2 mining as well as 118 pit. Underground production at 2,000 ore tpd. Processing mill head copper grade is 2.86% Cu .

2015: Area 2 Stage 2 mining nears completion and underground production is completed by the end of the year. A total of 1.6 Mt of plant feed mined in the period at 1.36% Cu. Total waste tonnage is 7.4 Mt.

2016: Area 2 Stage 2 completed at beginning of the year, with Ridgetop South completed by year end. Mining commences in Ridgetop North. Mill feed head grade at 1.68% Cu. The OP strip ratio is 5:1 with 7.5 Mt of waste mined. In-pit tailings deposition commences in Area 2 Stage 1&2 pit.

2017: Ridgetop North completed and mining commences in final stages of Area 2. 1.0 Mt of ore mined and mill head grade decreases to 1.11% Cu with 0.8 Mt of low grade stockpile material fed to plant. In-pit tailings deposition in Ridgetop North commences.

2018: Mining completed for PH V ore bodies with 0.8 Mt of ore mined. Mill head grade of 0.96% Cu with 0.9 Mt of low grade stockpile ore processed. All remaining tailings produced deposited in Area 2 ultimate pit.

2019: 1.4 Mt of ore milled at head grade of 0.78% Cu, all fed from low grade stockpile.

2020: Final year of processing 0.7 Mt of low grade stockpile ore at 0.78% Cu.

18.3.3 Ore Stockpiles

Several ore stockpiles exist on the property that will remain active throughout the LOM plan. The stockpiles are defined in terms of estimated copper grade mined as shown in Table 18.20 and their locations are noted on the site plan in the report.

Table 18.20: Ore Stockpiles

Stockpile	Copper grade (%)
Low grade-Blue	cut-off to 1.0
Regular grade-Green	1.0-2.0
Medium grade-Yellow	2.0-4.0
High grade-Red	greater than 4.0

Table 18.21 and Figure 18.13 detail the various predicted ending stockpile balances on a yearly basis. The stockpiled ore will be used to supplement open pit ore throughout the schedule and allow for some increase in flexibility in the mine plan while providing the highest mill head grade possible.

The annual mill feed from the stockpiles is shown in Table 18.22. As illustrated by the lack of year end inventories, the higher grade ores are fed to the mill as they are mined, in order to maintain the ore production at the highest possible head grade while mining. The lower grade stockpiles are depleted gradually and are used to smooth the mill feed as required.

Of note is the large low grade stockpile which reaches a predicted maximum balance of 2.8 Mt. In order to accommodate this large volume of low grade ore, a new stockpile location will be required.

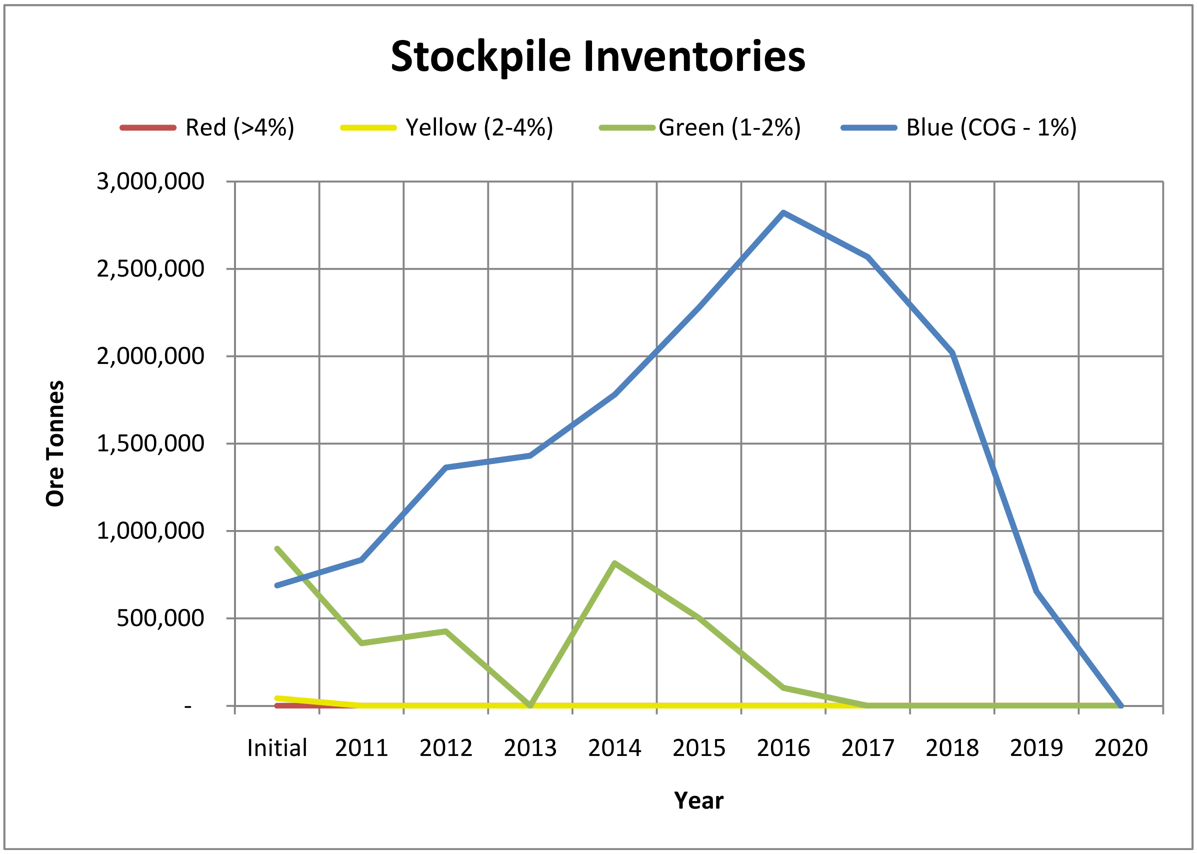


Figure 18.13: Stockpile Inventories

Table 18.21: Ore Stockpile Balance at Year End

Stockpile	Unit	Initial	2011	2012	2013	2014	2015	2016	2017	2018	2019
Red (>4%) Stockpile	Kt										
	Cu (%)										
	Au (g/t)										
	Ag (g/t)										
	Main Pit Ore										
Yellow (2-4%) Stockpile	Kt	43									
	Cu (%)	2.74									
	Au (g/t)	0.92									
	Ag (g/t)	12.74									
	Main Pit Ore	100%									
Green (1-2%) Stockpile	Kt	899	358	425		815	502	103			
	Cu (%)	1.47	1.47	1.50		1.51	1.51	1.51			
	Au (g/t)	0.41	0.41	0.45		0.54	0.54	0.54			
	Ag (g/t)	5.06	5.06	5.24		4.86	4.86	4.86			
	Main Pit Ore	100%	100%	84%		0%	0%	0%			
Blue (COG-1%) Stockpile	Kt	689	835	1,363	1,431	1,780	2,279	2,821	2,568	2,021	653
	Cu (%)	0.83	0.82	0.80	0.80	0.80	0.79	0.79	0.78	0.78	0.78
	Au (g/t)	0.22	0.20	0.20	0.20	0.18	0.19	0.19	0.19	0.19	0.19
	Ag (g/t)	2.48	2.38	2.28	2.28	2.20	2.21	2.18	2.11	2.15	2.15
	Main Pit Ore	100%	96%	59%	56%	45%	35%	28%	23%	19%	19%
Total All Stockpiles	Kt	1,631	1,193	1,789	1,431	2,596	2,782	2,924	2,568	2,021	653
	Cu (%)	1.24	1.01	0.96	0.80	1.02	0.92	0.81	0.78	0.78	0.78
	Au (g/t)	0.35	0.27	0.26	0.20	0.30	0.25	0.20	0.19	0.19	0.19
	Ag (g/t)	4.18	3.18	2.98	2.28	3.04	2.69	2.28	2.11	2.15	2.15
	Main Pit Ore	100%	97%	65%	56%	31%	29%	27%	23%	19%	19%

Table 18.22: Ore Stockpile Feed to Mill

Stockpile	Unit	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020
Red (>4%) Stockpile	Kt										
	Cu (%)										
	Au (g/t)										
	Ag (g/t)										
	Main Pit Ore										
Yellow (2-4%) Stockpile	Kt	43									
	Cu (%)	2.74									
	Au (g/t)	0.92									
	Ag (g/t)	12.74									
	Main Pit Ore	100%									
Green (1-2%) Stockpile	Kt	541		425		313	400	103			
	Cu (%)	1.47		1.50		1.51	1.51	1.51			
	Au (g/t)	0.41		0.45		0.54	0.54	0.54			
	Ag (g/t)	5.06		5.24		4.86	4.86	4.86			
	Main Pit Ore	100%		84%		0%	0%	0%			
Blue (COG-1%) Stockpile	Kt							719	946	1,369	653
	Cu (%)							0.79	0.78	0.78	0.78
	Au (g/t)							0.19	0.19	0.19	0.19
	Ag (g/t)							2.18	2.11	2.15	2.15
	Main Pit Ore							28%	23%	19%	19%
Total All Stockpiles	Kt	584		425		313	400	821	946	1,369	653
	Cu (%)	1.57		1.50		1.51	1.51	0.88	0.78	0.78	0.78
	Au (g/t)	0.45		0.45		0.54	0.54	0.23	0.19	0.19	0.19
	Ag (g/t)	5.64		5.24		4.86	4.86	2.52	2.11	2.15	2.15
	Main Pit Ore	100%		84%		0%	0%	25%	23%	19%	19%

18.3.4 Underground Development and Production Schedule

Mine Access Development Schedule

The objective of the mine schedule was to achieve early ore production from higher-grade areas. The mine development is divided into two periods: pre-production development (prior to mine production) and ongoing development (during production).

Pre-production development was scheduled to:

- Provide access for trackless equipment;
- Provide ventilation and emergency egress;
- Establish ore and waste handling systems;
- Install mining services.

It was assumed that the decline and lateral development would be performed by owner operator crews. The raise development would be done by contractor. The development schedule was planned based on estimated cycle times for jumbo and Alimak development and best practises of North American contractors (See Table 18.23).

Table 18.23: Development Cycle Times

Unit	Ramp	Raise	Raise Slash
Design Criteria			
Width (m)	5.0	3.0	3.0
Height (m)	5.0	3.0	5.0
Gradient (% / deg.)	15%	70^{o}	70^{o}
Summary Cycle Times			
Drilling (Hrs)	4.2	4.7	2.7
Blasting (Hrs)	1.8	2.0	1.7
Re-Entry (Hrs)	0.5	0.5	0.5
Mucking (Hrs)	2.0	1.3	0.7
Support (Hrs)	5.3	5.2	3.6
Services (Hrs)	1.0	1.2	0.0
Secondary Mucking (Hrs)	5.3	0.0	0.0
Trucking (Hrs)	5.9	1.9	1.4
Single Heading			
Critical Path Cycle Time (Hrs)	14.8	13.5	8.4
Advance Per Shift (m)	2.7	2.0	3.3
Advance Per Day (m)	5.3	4.0	6.5

The raise would be developed in two stages with initial development size of 3.0 m x 3.0 m and then slashed to full raise size of 3.0 m x 5.0 m. Then the raise would be equipped with ladders and platforms to provide a manway for secondary egress from the mine in case of emergency.

It was assumed that approximately one month would be required for mobilization of mining equipment and crews to the site and establish the required services. Another month would be required to develop a portal box-cut, the jumbo crew would then start developing decline from the portal. It was assumed that in first month the decline advance rate would be at 50% of average or 75 m.

It was assumed that the advance rate of the main decline development would be approximately 150 m/mo per single heading. When multiple headings are available, the advance rate per jumbo crew would increase to 220 m to 250 m per month.

It was planned that the main decline would be developed all the way to the bottom of the mine to the Minto East area at the maximum advance rate to provide an opportunity to start production of the high grade ore from Minto East as soon as possible.

The pre-production and main access development schedule is shown in the Table 18.24.

Table 18.24: Pre-production and Capital Development Schedule

Development	Length (m)	Advance (m/d)	Duration (days)	2011	2012	2013	2014	2015
Crews Mobilization			15	1				
Setup Services			15	1				
Portal Boxcut Excavation			30	1				
Decline Portal	25	2	25	25				
Main Access Decline	2,460	5.3	462	950	1,510			
Remuck Bays	240	4.3	56	90	150			
Ventilation Drift	280	4.3	66	15	265			
Crosscut	160	4.3	38	85	75			
Ore Development	180	5.3	34	180				
Alimak Nest	30	4.0	8	15	15			
Total Jumbo Development	**3,375**			**1,360**	**2,015**			
Vent Raise Collar			15					
Vent Raise 3 m x 3 m	453	4.0	112	145	120	188		
Raise Slash to 3 m x 5 m	453	6.5	70	145		308		
Internal Raise 3 m x 5 m	180	2.5	72	27	153			
Total Raise Development	**633**			**172**	**273**	**188**		

Totals may not add up due to rounding

Underground Production Schedule

The criteria used for scheduling of underground mine production at the Minto mine were as follows:

- Target mining blocks with higher grade ore in the early stages of mine life to improve project economics;
- Production sequence of the mining blocks would be from the top down;
- An average annual ore production rate of 730,000 t was scheduled, including ore from development and stopes;
- The mine will operate two 12-h shifts per day, 365 d/a;
- Provide enough production faces to support a daily mine production rate of 2,000 t/d.

Ore mining is planned to commence at the stope 101 in Area 118, which provides access to higher ore grades at the early stage of the mine life. The smaller mining blocks with low grades would be mined at the end of the mine life.

The stope cycle times and productivities were estimated from the first principles for each type of mining method per average production stope size of 8 m wide and 5 m high.

It was estimated that the average production from the single heading at RAP mining would be approximately 540 t/day. It would require 4 production faces working at any time to meet daily production requirements of 2,000 t/day. The number of mining blocks and flexibility of RAP and PPCF mining methods would allow as many stopes in production as is required to support the production rate.

The UG mine production schedule by stope is detailed in Table 18.25. Ore production by year is shown in Table 18.26. Figures 18.14 to 18.18 show the annual progression of the UG mine plan.

Underground Mine Production Rate

The following factors were considered in the estimation of the underground mine production rate:

- Reserve tonnage and grade;
- Geometry of the orebodies;
- Amount of the required development;
- Stopes productivities; and
- Sequence of the mining and stopes availability.

The underground mine production rate of 2,000 t/d was considered appropriate due to the high degree of mechanization, potential high productivities of selected stoping methods. Development scheduling was planned to have at least 2 production stopes, with multiple faces, available at all times.

The tonnage in an average production stope size of 8 m (W) x 5 m (H) with advance of 3.85 m per cycle would yield about 450 tonnes, therefore, 2,000 tpd of production would require approximately 4.5 full cycles per day. Based on the presence of several deposits and ability to have production from different sub-levels, SRK considers the UG production rate to be achievable.

Table 18.25: Underground Mine Production Schedule by Stope

						Year				
Area	**Stope**	**Reserves (Kt)**	**RAP Drifting (%)**	**RAP Benching (%)**	**PPCF (%)**	**2011 (Kt)**	**2012 (Kt)**	**2013 (Kt)**	**2014 (Kt)**	**2015 (Kt)**
118	101	649.8	37	20	43	12. 8	630.0	7.0		
	103	49.6	22	10	68					49.6
	104	15.0	100							15.0
	105	7.9	94	6					7.9	
	106	29.8	65	35					29.8	
	107	12.2	83	17					12.2	
2	201	376.3	17	6	77			376.3		
	203	268.1	68	19	12			21.0	240.3	6.7
	204	135.5	26	19	12			55.7	79.8	
	205	39.3	74	26						39.3
	206	11.0	93	7						11.0
	207	22.1	71	29						22.1
	208	20.4	76	24						20.4
	209	23.0	95	5						23.0
	210	19.4	81	19						19.4
	211	52.2	44	23	33					52.2
East	301	709.0	16	7	77			270.0	360.0	79.0
Total						**12.8**	**630.0**	**730.0**	**730.0**	**337.8**

Table 18.26: Underground Mine Production Schedule

Parameter	**Units**	Year					
		Total	**2011**	**2012**	**2013**	**2014**	**2015**
Mining							
Ore	Kt	**2,440.6**	12.8	630.0	730.0	730.0	337.8
Waste Rock	Kt	**405.0**	87.7	222.6	36.3	29.1	29.2
Total Material	Kt	**2,845.6**	100.5	852.6	766.3	759.1	367.0
Mined Cu grade	%	**1.9**	1.8	1.8	2.0	2.0	1.7
Mined Au grade	g/t	**0.8**	0.7	0.7	0.9	0.9	0.8
Mined Ag grade	g/t	**6.7**	7.2	7.2	6.2	6.8	6.6
NSR	$/t	**91.1**	87.6	87.6	95.9	93.9	81.6
Mined Contained Cu	Mlb	**102.4**	0.5	25.6	32.2	31.5	12.7
Mined Contained Au	Koz	**64.3**	0.3	14.1	21.0	20.8	8.1
Mined Contained Ag	Koz	**526.8**	3.0	146.5	145.4	159.8	72.2
RAP Drifting	Kt	**839.8**	12.8	235.4	138.6	278.7	174.4
RAP Benching	Kt	**331.4**	0.0	123.8	57.3	99.3	51.0
PPCF	Kt	**1,269.4**	0.0	270.8	534.1	352.0	112.5

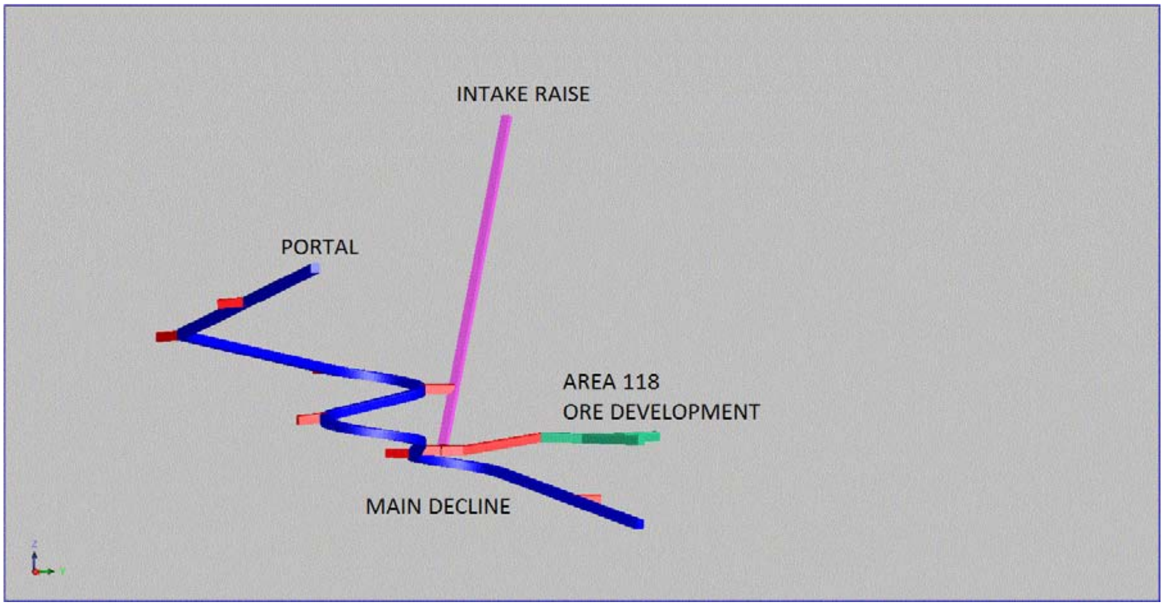


Figure 18.14: Pre-Production Development

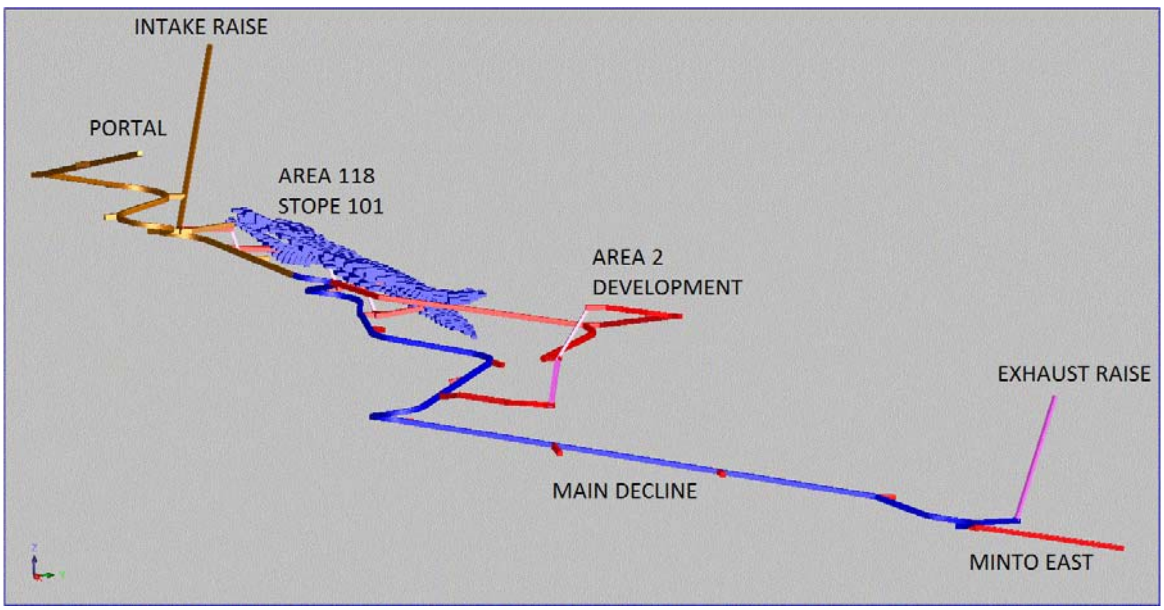


Figure 18.15: End of Year 2012

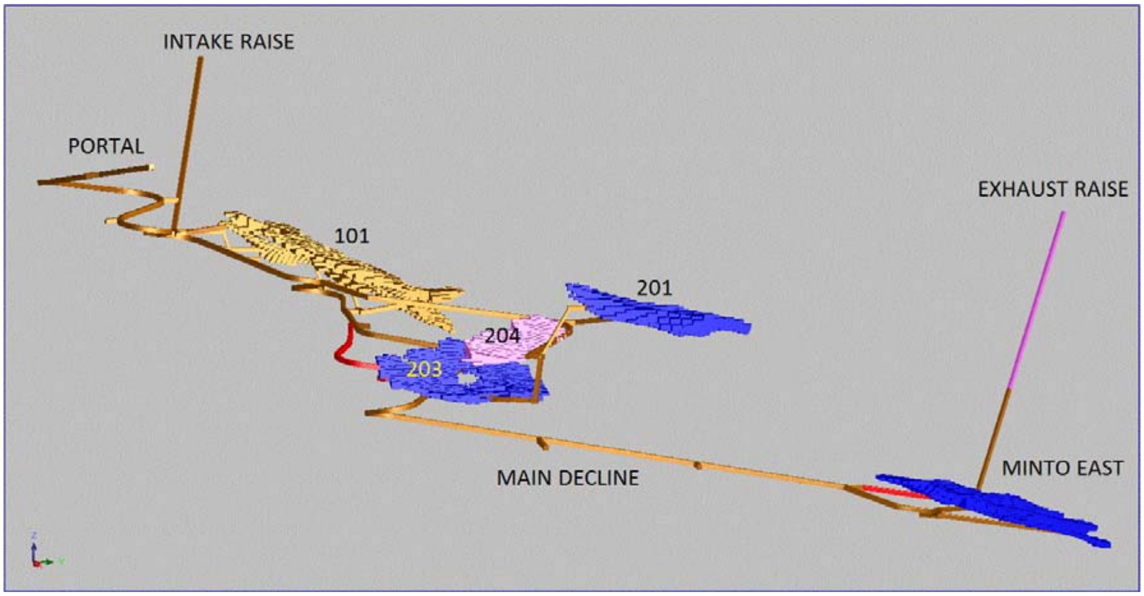


Figure 18.16: End of Year 2013

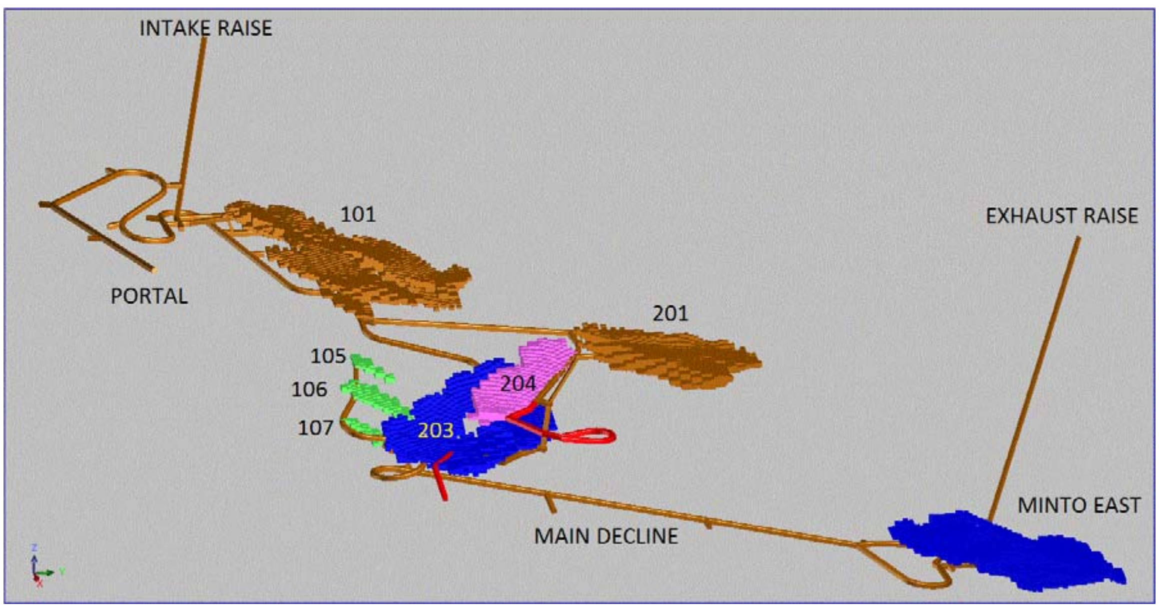


Figure 18.17: End of Year 2014

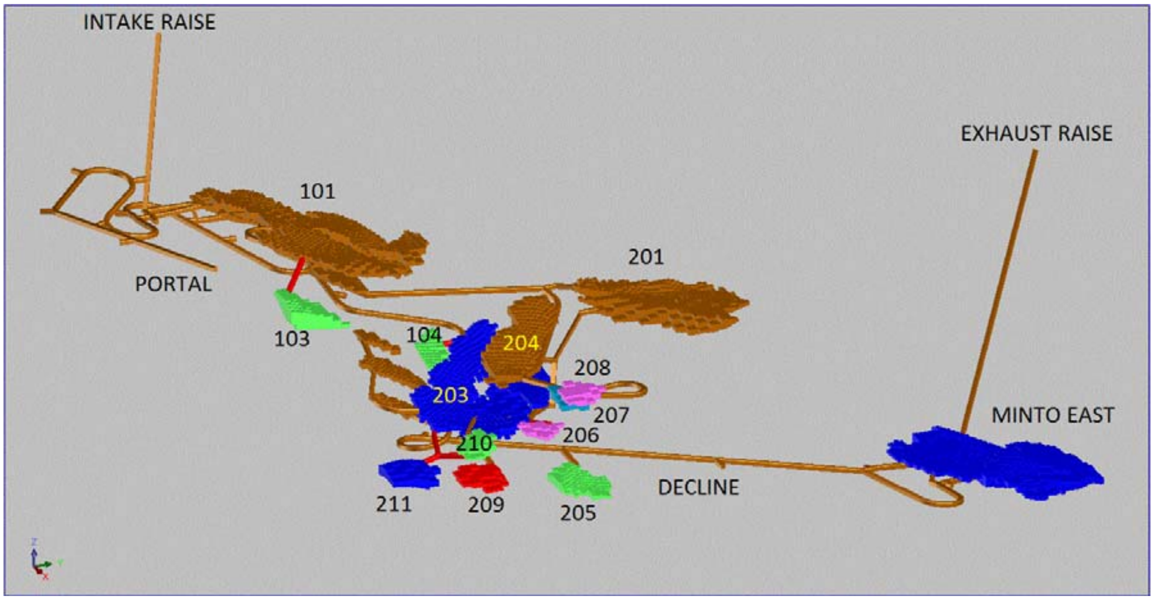


Figure 18.18: End of Year 2015

18.4 Waste Management

Tailings from the mill will be sent to the currently permitted existing dry-stack location for the life of the Main Pit (to mid 2011). Upon completion of mining in the Main Pit, thickened tailings generated from processing ores from other Phase V deposits will then be deposited into the Main Pit. The permit application for the deposition of tailings into the Main Pit was part of the Phase IV permit that was filed in August 2010 and is assumed to be approved in March 2011. Additional capacity required annually to store approximately 700,000 cubic metres of water associated with freshet flows, plus incremental storage to meet minimum and maximum operational requirements has been taken into consideration.

Further in-pit tailings storage capacity becomes available once Area 2 is mined and this Area 2 storage capacity will be required in order to hold a portion of the tailings to be produced from the Phase V LOM plan. Ridgetop North is also used as an in-pit storage facility until mining is completed in the final stages of Area 2.

This deposition of tailings into the Area 2 and Ridgetop North Pits will form part of the Phase V permit application to be submitted early in 2011 and is assumed to be approved before the 2nd quarter of 2012.

Although these tailings deposition plans are not yet permitted, they offer a potentially viable solution to tailings disposal that provides backfill material for the Main, Area 2 and Ridgetop North pits, reduces the amount disturbed land that would normally be required by mining of the Phase V deposits, and provides a significant cost savings over the current dry-stack method.

Waste rock from the current Main pit, as well as a significant portion of the Phase V deposits, will be deposited in an expansion of the existing permitted West Valley Fill waste dump located in the lower valley southwest of the Main Pit. In addition, waste rock from Minto North is proposed to be stacked onto the existing Main pit dump, while some waste material from the Phase V deposits will be deposited in a proposed Mill Valley dump to the west of the existing mill facilities. Waste rock material from Area 2 will also be placed in Main pit to act as a south wall buttress. Backfilling of Ridgetop South and 118 pits will also provide waste storage capacity and benefit the final reclamation plan. Overburden material will be placed in temporary dumps adjacent to the various deposits and used for final reclamation. Any excess overburden will be added to existing Overburden dump.

18.4.1 Waste Rock Dump Designs

West Valley Fill Dump

The waste rock material, and low grade oxide, generated from the Main Pit, as well as a significant portion of the waste rock from the Phase V deposits, will continue to be placed in the currently permitted West Valley Fill dump in the lower valley to the south west of the pit. The dump has an overall face slope angle of 24 degrees (toe-crest) with safety berms spaced at regular 10 m (vertical) intervals. The berms are designed to have a width of 15 m.

Overburden material will be placed in temporary dumps adjacent to the various deposits and used for final reclamation. Any excess overburden will be added to existing Overburden dump located to the north of the West Valley Dump.

Mill Valley Dump

The Mill Valley Dump is a smaller dump located east of the existing process facilities and north of the dry stack tailings facility. Waste rock material from Area 2 pit will be placed in this dump.

Minto North Dump

Minto North pit waste material is to be placed on top of the existing Main dump located on a south-facing slope west of the Main Pit and immediately south of the Minto North pit. The dump has an overall face slope angle of 25 degrees (toe-crest) with safety berms spaced at regular 12 m (vertical) intervals. The berms are designed to have a width of 12 m.

18.4.2 Backfill Dumps

Ridgetop South pit and the 118 pit will be backfilled with waste generated from subsequent mining. Overburden material will then be placed as a cap on the backfilled pits.

Capacities and Sequence

Table 18.27 below summarizes the waste quantities produced by each stage of this pre-feasibility report. Material is reported in terms of type as well as tonnage and cubic metres.

Table 18.27: Waste Quantities by Stage

Zone	Overburden	Rock	Total Waste	Overburden	Rock	Total Waste
	(kt)	(kt)	(kt)	m3 x 1,000	m^3 x 1,000	m^3 x 1,000
Main Pit	1,134	496	1,631	737	236	973
Minto North	2,063	9,649	11,712	1,169	4,647	5,816
Ridgetop	1,435	6,968	8,403	897	3,447	4,344
118	416	1,841	2,257	257	902	1,159
Area 2	9,052	25,127	34,179	5,790	12,315	18,105
Grand Total Waste	**14,101**	**44,081**	**58,182**	**8,851**	**21,546**	**30,397**

**Note 1.3 swell factor used (m^3/bcm)*

A summary of the metal content of the waste rock material to be generated from the PH V pits is shown in Table 18.28 below as well as in Figures 18.19 and 18.20. The waste rock material has been classified into four grade bins based on Cu%:

- Grade Bin 1: 0% Cu;
- Grade Bin 2: 0.0 to 0.10% Cu;
- Grade Bin 3: 0.10 to 0.20% Cu; and
- Grade Bin 4 0.20% Cu to 0.58% Cu (incremental cut-off grade).

Table 18.28: Mineralized Waste Rock Classification

Stage	Total Rock	Total	Grade Bin 0.20-0.58					Grade Bin 0.10-0.20					Grade Bin 0.00-0.10					Grade Bin 0				
	(kt)	Cu %	Waste (kt)	Cu %	Cu (Mlbs)	Au g/t	Ag g/t	Waste (kt)	Cu %	Cu (Mlbs)	Au g/t	Ag g/t	Waste (kt)	Cu %	Cu (Mlbs)	Au g/t	Ag g/t	Waste (kt)	Cu %	Cu (Mlbs)	Au g/t	Ag g/t
Area2 Stg1	9,313	0.07	1,530	0.34	11.5	0.05	0.92	435	0.15	1.5	0.03	0.50	341	0.05	0.4	0.01	0.27	7,007		0.0		
Area2 Stg2	11,577	0.06	1,698	0.38	14.2	0.06	1.00	405	0.15	1.4	0.03	0.48	294	0.05	0.3	0.02	0.26	9,179		0.0		
Area2 Stg3	1,470	0.06	132	0.63	1.8	0.19	2.37	0	0.19	0.0	0.04	0.89	0	0.07	0.0	0.01	0.13	1,338		0.0		
Area2 Stg4	2,816	0.09	577	0.40	5.1	0.08	1.08	72	0.16	0.3	0.03	0.56	24	0.07	0.0	0.01	0.49	2,144		0.0		
118	1,837	0.04	134	0.43	1.3	0.04	0.91	72	0.17	0.3	0.02	0.53	5	0.06	0.0	0.02	0.25	1,627		0.0		
North	9,649	0.00	85	0.47	0.9	0.11	1.51	1	0.14	0.0	0.07	0.64	0	0.00	0.0	0.00	0.00	9,562		0.0		
RGS	1,569	0.10	349	0.35	2.7	0.09	1.16	166	0.15	0.5	0.03	0.42	87	0.06	0.1	0.01	0.21	966		0.0		
RGN	5,429	0.09	1,013	0.39	8.6	0.06	0.91	316	0.15	1.0	0.03	0.45	712	0.04	0.7	0.03	0.22	3,388		0.0		
Total	**43,658**	0.00	**5,517**		**46.1**			**1,468**		**4.9**			**1,463**		**1.6**			**35,210**		**0.0**		

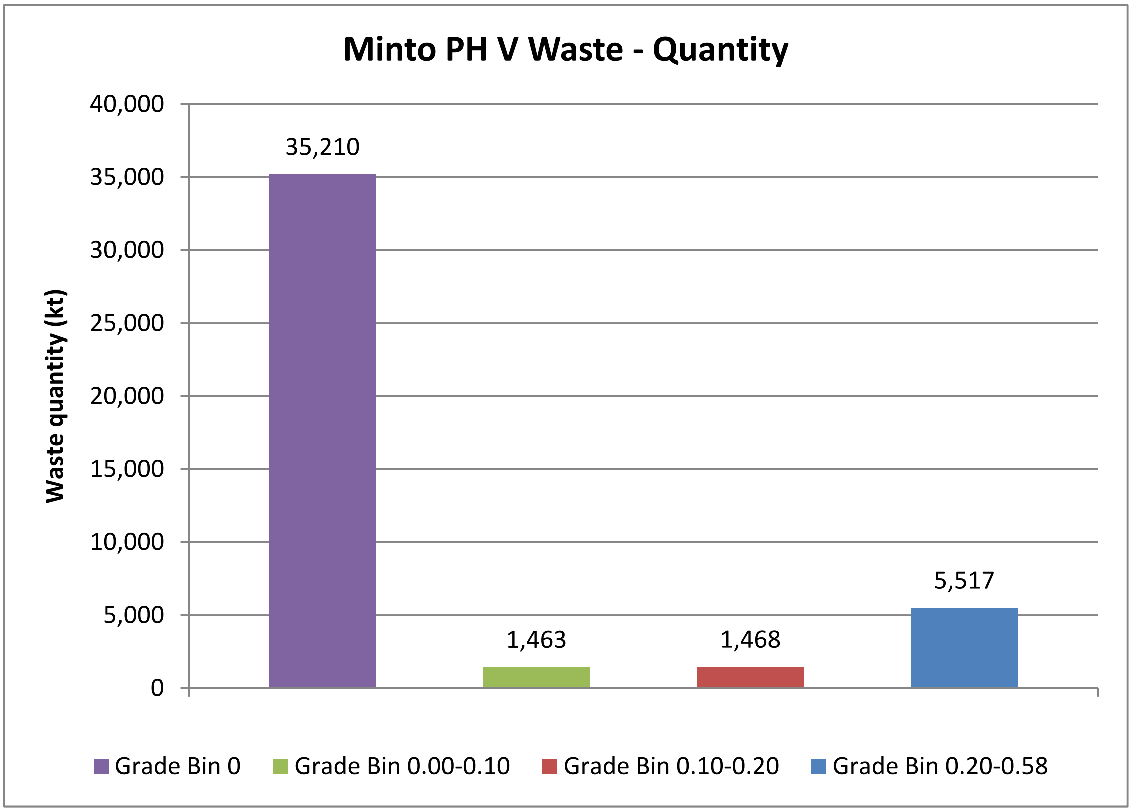


Figure 18.19: Mineralized Waste Quantity

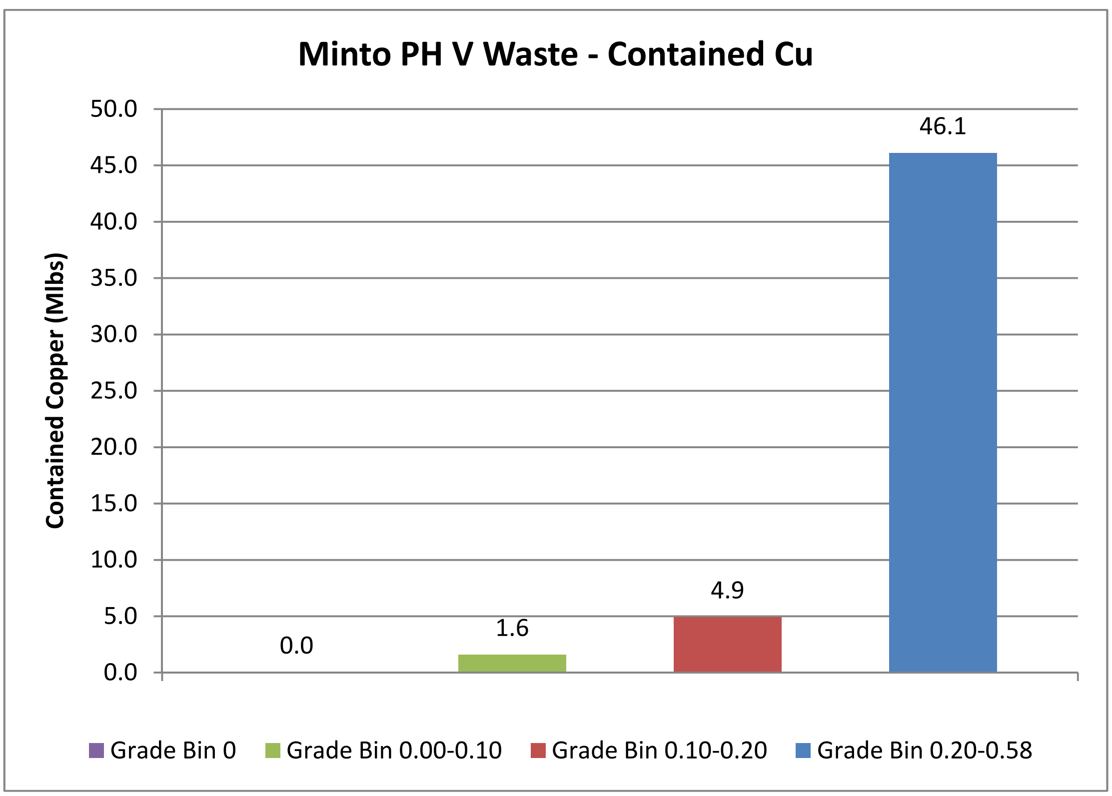


Figure 18.20: Mineralized Waste Contained Cu

Waste rock material containing higher copper grades will be placed into the mined out pits, either as the buttress material for Main pit or into Area 2 pit (given scheduling constraints of the LOM plan).

18.4.3 In-pit Tailings Disposal

In-pit tailings management will be an important part of the Phase V LOM plan. Based on preliminary laboratory testing on tailings from the thickener underflow, the average dry density of the deposited tailings is estimated to be 1.30 t/m^3 for the anticipated sub-aqueous deposition conditions (volume losses for ore concentrate were ignored). This value requires further verification due to the preliminary nature of the testing, experience at other similar copper tailing projects (values as low as 1.12 t/m^3 have been encountered) and the implications to the proposed sequence of tailings disposal in the event that the average in-place density of the tailings is significantly less than 1.30 t/m^3 (volume losses for ore concentrate were ignored).

For this PHV PFS it has been assumed that tailings from the mill will be sent to the currently permitted existing dry-stack location for the life of the Main Pit (to mid-2011). Upon completion of mining in the Main Pit thickened tailings generated from processing ores from other Phase V deposits will be then deposited into Main pit, Area 2 and Ridgetop North pits. A total of 9.4 Mm^3 of tailings storage capacity will be required for this Phase V LOM plan (includes the additional 1.6 Mt of low-grade ore stockpiled during the mine life that will be processed following the completion of mining of the PH V pits).

The construction of the Main Pit buttress (required for slope stability) is required prior to the placing of tailings in Main Pit. Additional capacity required annually to store approximately 700,000 cubic metres of water associated with freshet flows, plus incremental storage to meet minimum and maximum operational requirements has been taken into consideration.

Further in-pit tailings storage capacity becomes available once Area 2 is mined and, this Area 2 storage capacity will be required in order to hold a portion of the tailings to be produced from the Phase V LOM plan. Ridgetop North is also used as an in-pit storage facility until mining is completed in the final stages of Area 2.

Although these tailings deposition plans are not yet permitted, they offer a potentially viable solution to tailings disposal that provides backfill material for the Main, Area 2 and Ridgetop North pits, reduces the amount disturbed land that would normally be required by mining of the Phase V deposits, and provides a significant cost savings over the current dry-stack method.

Table 18.29 shows the potential LOM in-pit storage capacities of the Main, Area 2 and Ridgetop North open pits. The Main pit tailings storage capacity takes into account the volume taken up by the buttress design. Minto North, Ridgetop South and 118 pits are not considered for in-pit tailings backfill.

Table 18.29: Tailings Storage Capacity by Pit

Pit	Max. elev (m)	Capacity (Mm^3)
Main Pit*	786	5.1
Area 2	800	7.5
Ridgetop North	860	1.1
Total		**13.7**

**incorporating buttress design*

It was determined that the difference in total maximum in-pit storage capacity available of 13.7 Mm^3 (excluding Ridgetop South, 118 and Minto North pits) and the volume occupation of 12.2 Mt of tailings produced (~9.4 Mm^3) leaves 4.3 Mm^3 theoretical excess storage capacity for freshet and mineralized waste in Main/Area2/Ridgetop pits combined. A typical freshet runoff volume of 0.7 Mm^3 was assumed to be required.

Topography of the proposed Main Pit and Phase V pits is depicted in plan view in Figure 18.21 and in section view in Figures 18.22 to 18.25. The figures show the Main Pit wall and south wall buttress configurations; the currently planned Phase V pits; existing ground surface; and planned post-mining pit topography. In addition, the site topography shown was used to determine stage-area-capacity characteristics for evaluation of storage options.

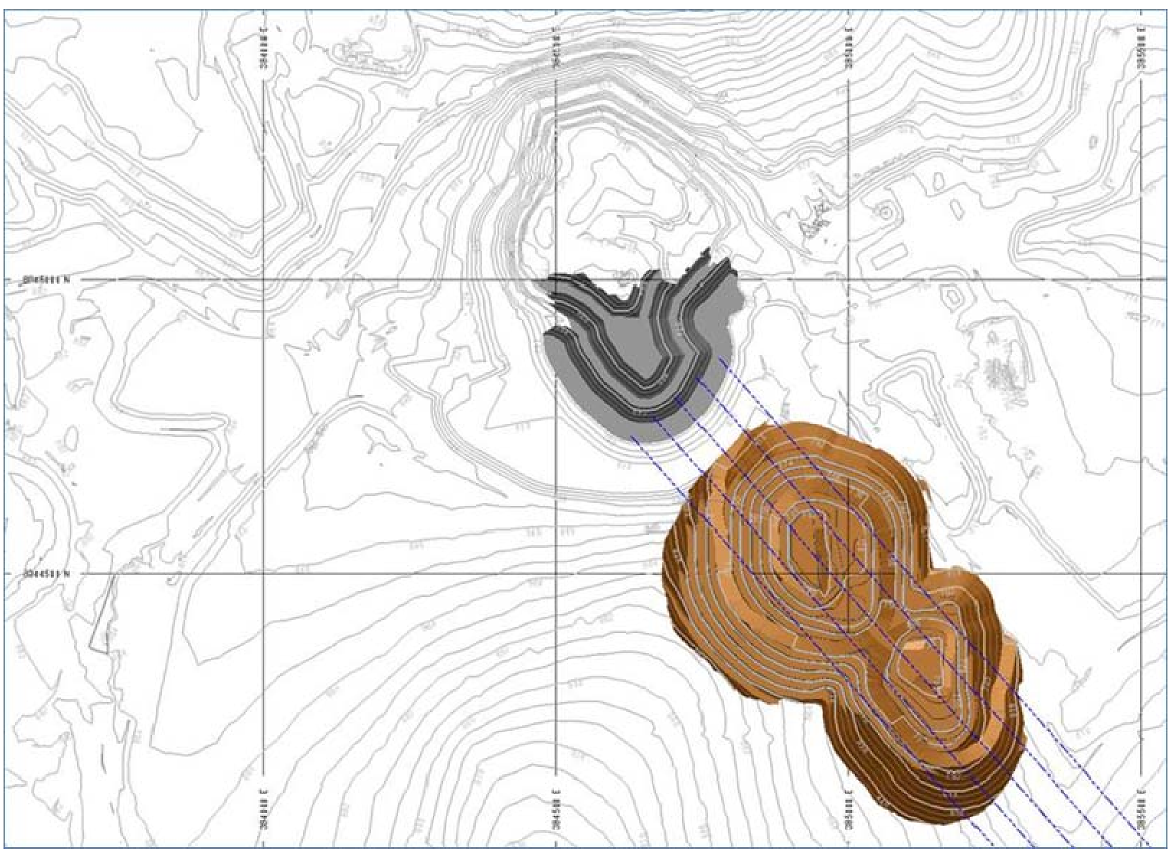


Figure 18.21: Site Topography and Proposed Pit and Buttress Layouts

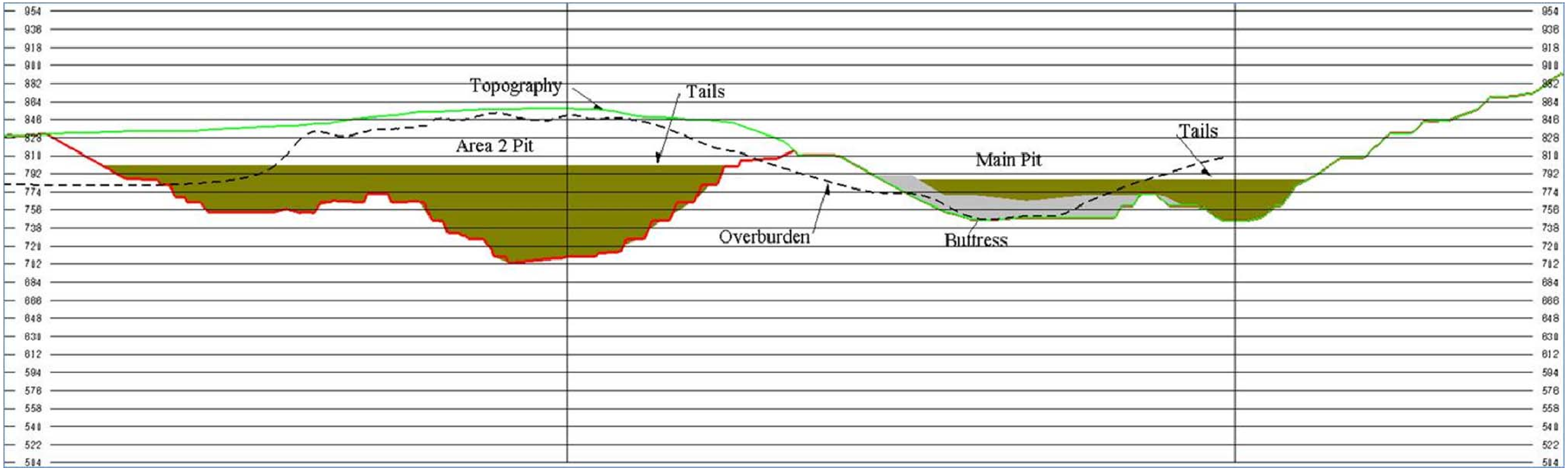


Figure 18.22: Section 2 looking SW

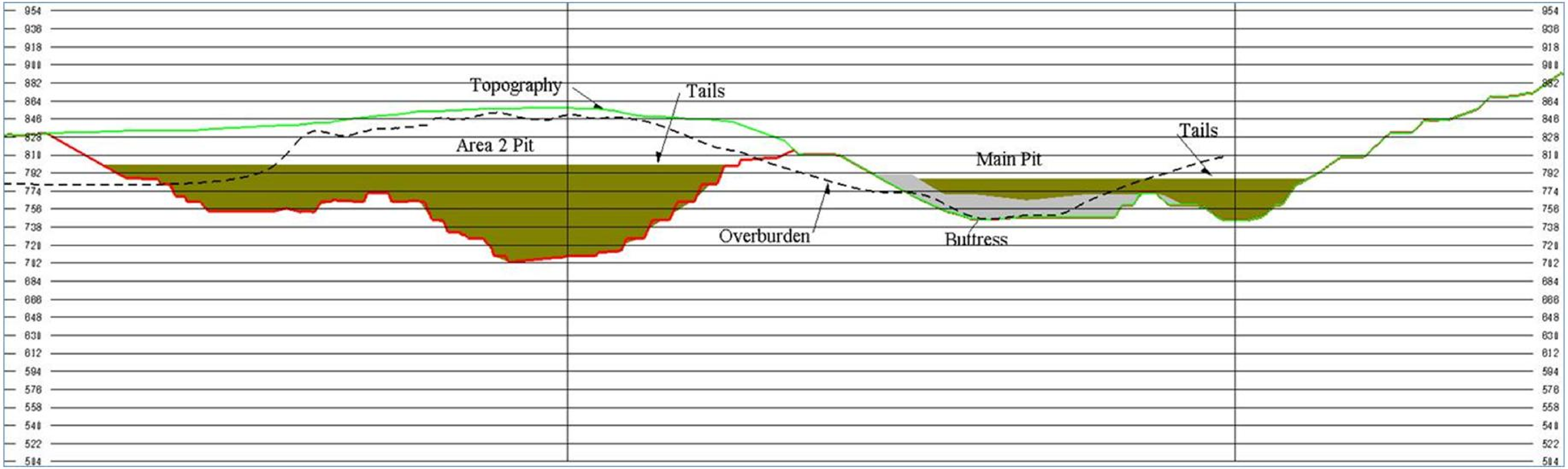


Figure 18.23: Section 3 looking SW

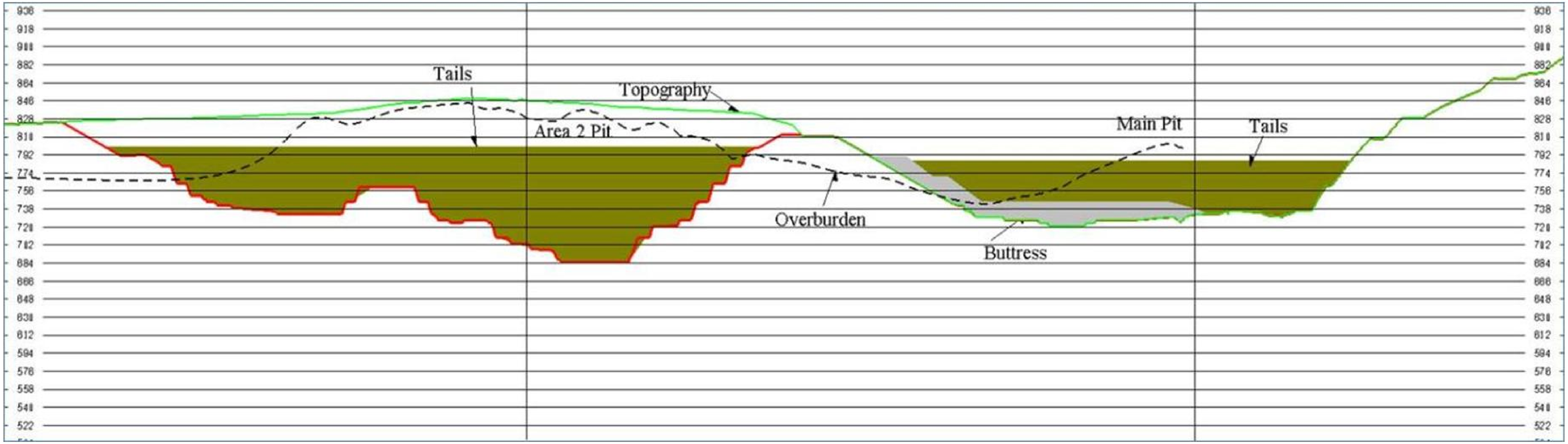


Figure 18.24: Section 4 looking SW

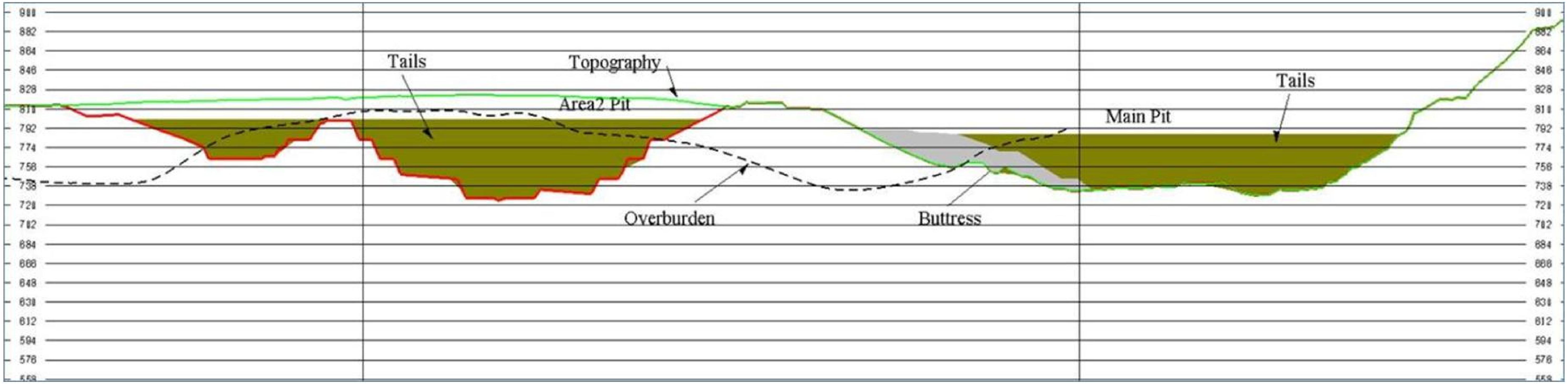


Figure 18.25: Section 5 looking SW

The LOM processing schedule along with the various stage curves (rates of rise) for the in-pit tailings storage options were used to determine the sequence and location of tailings levels within the various mined out pits. Figures 18.26 and 18.27 illustrate the various stage curves for the Main Pit tailings storage options.

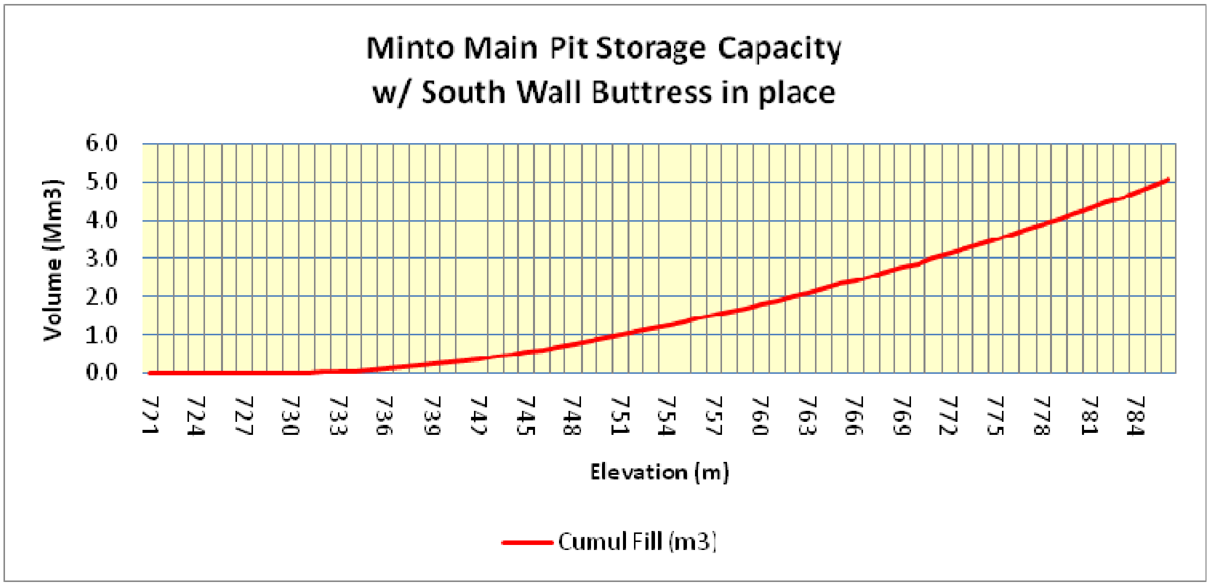


Figure 18.26: Minto Main Pit Stage Curve

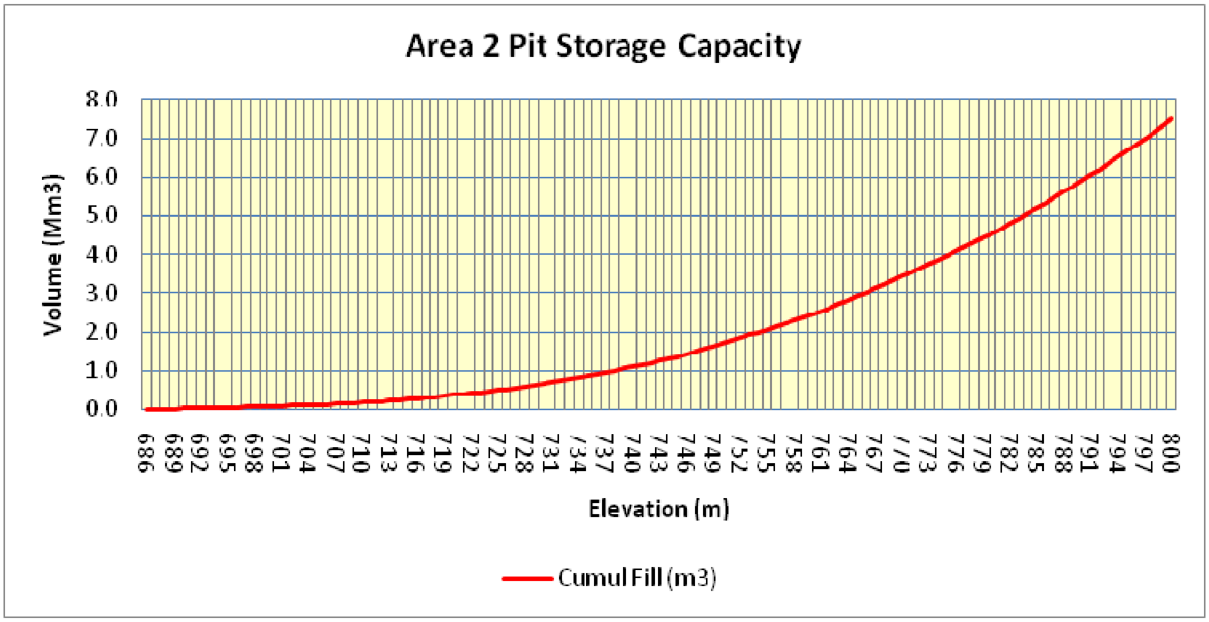


Figure 18.27: Area 2 Pit Stage Curve

Table 18.30 summarizes this information in terms of tailings produced and where it is planned to be deposited on a yearly basis. Tailings quantities along with tailings elevation levels are included.

Table 18.30: Tailings Production and Storage Schedule

Item	Unit	Total	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020
O/P Mining schedule	(pit)		MM/A2stg1	A2stg1/North	North	A2stg2/118	A2stg2	RGS/RGN	RGN/A2stg3/A2stg4	A2stg3/A2stg4		
Waste mined	(kt)	**11,236**	828	1,765	2,256	1,482	1,555	1,515	1,013	822		
Ore mined	(kt)	**58,520**	9,647	9,383	6,423	7,921	7,459	7,525	7,819	2,342		
Total O/P Material Mined/Day	(t/day)	**26,214**	28,426	28,312	21,711	23,752	23,664	24,768	24,196	8,669		
Ore process inpit only	(t)	**12,238,279**	628,165	1,372,500	1,368,750	1,368,750	1,368,750	1,372,500	1,368,750	1,368,750	1,368,750	652,614
Total inpit tails only	(m^3)	**9,414,061**	483,204	1,055,769	1,052,885	1,052,885	1,052,885	1,055,769	1,052,885	1,052,885	1,052,885	502,011
Main pit tails w buttress	(m^3)	**5,057,894**	483,204	1,055,769	1,052,885	1,052,885	1,052,885	360,267				
Main pit tails cumulative	(m^3)		483,204	1,538,973	2,591,858	3,644,742	4,697,627	5,057,894				
Freshet	(m^3)		700,000	700,000	700,000	700,000	700,000					
Total Main pit storage req'd cumul. (max 5.1M)	(m^3)		1,183,204	2,238,973	3,291,858	4,344,742	5,397,627					
Main pit tails elevation (max. 786m)	(m^3)		753	764	773	781	786					
Buttress elev req'd w 2m freeboard	(m^3)		755	766	775	783	790					
Buttress dumped volume req'd cumul.	(m^3)		503,660	696,055	860,253	999,875	1,147,919					
Area2 Stg1&2 inpit tails	(m^3)	**1,311,449**						695,502	615,947			
Area2 Stg1&2 inpit tails cumulative	(m^3)							695,502	1,311,449			
Freshet	(m^3)							700,000	700,000			
Total Area2 Stg1&2 storage req'd cumul. (max. 2.0M)	(m^3)							1,395,502	2,011,449			
Area2 Stg1&2 tails elevation (max. 758m)	(m^3)							747	758			
RGN inpit tails	(m^3)	**752,803**							436,938	315,865		
RGN inpit tails cumulative (max 1.1M)	(m^3)								436,938	752,803		
RGN tail elevation (max. 860m)	(m^3)								838	849		
Area2 Final inpit tails	(m^3)									737,019		
Area2 Final inpit tails	(m^3)	**3,603,364**								2,048,468	1,052,885	502,011
Area2 Final inpit tails cumulative	(m^3)									2,048,468	3,101,353	3,603,364
Freshet	(m^3)									700,000	700,000	700,000
Total Area2 Final storage req'd cumul. (max. 7.5M)	(m^3)									2,748,468	3,801,353	4,303,364
Area2 Final tails elevation (max. 800m)	(m^3)									763	773	777
Total inpit storage	(m^3)	**9,414,061**	483,204	1,055,769	1,052,885	1,052,885	1,052,885	1,055,769	1,052,885	1,052,885	1,052,885	502,011

In-pit tailings deposition commences in mid-2011 upon completion of mining in Main pit. The tailings deposition into Main pit continues until 2015 where the maximum estimated capacity of Main pit storage is reached. Area 2 Stage 1&2 mining is completed at beginning of 2016 and allows tailings to then be stored at the bottom of this mined out area until late in 2017. Upon the completion of mining in Ridgetop North (mid 2017), tailings deposition will then move to this pit until the end of 2018. All open pit mining is completed by mid-2018. The final two year of processed tails are deposited into the ultimate mined out Area 2 pit.

19 Recoverability

19.1 Mineral Processing

Prediction of the flotation performance was determined following analysis of the locked cycle tests. The design recoveries of the target metals as selected by Ausenco are generally in line with, or slightly lower than those achieved in the locked cycle tests suggesting a degree of conservatism.

Table 19.1: Grade/Recovery from Locked Cycle Test work

Ore Type	Primary Grind (P_{80})	Secondary Grind (P_{80})	Feed Grade		Concentrate Recovery		Concentrate Grade	
			Cu (%)	Au (g/t)	Cu (%)	Au (%)	Cu (%)	Au (g/t)
Minto North	234	99	2.7	1.3	97.3	78.3	41	16
Area 118	198	70	2	0.6	95	77	39.6	9.4
Ridgetop	168	66	1.1	0.3	88	55.7	39	5.9
Area 2	216	72	2.1	0.8	93.2	71.7	38.7	11.8
Main (South) Primary	197	97	2.6	1.2	88.8	73	39.5	15.8
Main (South) Partially Oxidized	223	76	1.7	0.6	84	64.7	36.9	10.5
Minto East	271	73	2.4	1.3	94	85	40.4	20.5

The average silver recovery to final concentrate for the locked cycle test work was 78.2%.

The locked cycle tests in Table 19.1were completed at a wide range of primary and secondary grind sizes. The average primary grind size of 80% passing 210 micron is lower than the design of 250 micron. Also, a number of locked cycle tests did not include a regrind stage. As a result there was insufficient locked cycle test work data at the design conditions to determine a statistical relationship between feed grade, recovery and final concentrate grades.

The actual recovery for the Minto plant treating Minto main ore for the period of January 09 – May 09 was reviewed to further verify test work results on the new ore bodies against the current plant performance on Minto main ore. Findings from the review are:

- Copper recovery was 93.1% with a final concentrate grade of 41.9%;
- Gold recovery of was 69.5%; and
- Silver recovery of 81.9%.

The overall project economics for the Study were based on:

- Copper recovery of 92%;
- Gold recovery of 69.5%; and
- Silver recovery of 78%.

In light of the test work to date, Ausenco believes these recoveries are reasonable for a copper concentrate from the new Minto orebodies based on the flowsheet selected for the upgraded plant. The overall design grade and recovery numbers predicted by Ausenco are shown in Table 19.2. The values selected are generally lower than the actual test work values shown in Table 19.1. This is due to the finer primary grind size used in the majority of locked cycle tests and typical scale-up issues resulting in target metals misreporting during the separation stages.

Table 19.2: Final Grade/Recovery Used For Minto Ores

	Concentrate Grade			Concentrate Recovery			Comments
	Cu %	g/t Au	g/t Ag	Cu %	Au %	Ag %	
Whittle Mine Design	42	Variable	Variable	92	70	80	
Ausenco Prediction	38	Variable	Variable	92	70	78	Prediction based on test work

20 Markets

The Minto concentrate is deemed highly desirable by smelters due to its high copper grade (plus 38% Cu), its low contaminant levels and relatively low sulphur content. These attributes enable the Minto concentrate to be marketed at a favourable smelter terms.

20.1 Concentrate Sales

MintoEx has an established concentrate purchase contract with a metal trading company. MRI Trading AG (“MRI”). The terms of the contract are confidential; however, SRK confirms that the appropriate terms were used in the economic model. Under the terms of the contract, MRI has the obligation to buy all of MintoEx’s concentrate production and MintoEx has the obligation to sell all of its concentrate production to MRI. The contract is in effect from July 201007 to the end 2013June 2010. The contract may be extended by mutual agreement one or more years. This study assumes that long-term treatment charges will be US$40.00/dmt of concentrate and refining charges will be US$ 0.04/lb of payable copper after 2013. These assumptions are based on the continuation of a general supply shortage of copper concentrate and in particular, high-quality concentrates from Minto.

20.2 Copper Price Contract

MintoEx has a copper price guarantee contract with Macquarie Bank for copper production that is valid until the third quarter of 2011. The contract tonnages and prices are shown in Table 20.1.

Table 20.1: Copper Price Hedging Contract Summary

Year	Total Hedged Copper	Average Contract Price
2011	8,312 tonnes Cu	US$ 2.26/lb Cu

20.3 Precious Metal Price Contract

MintoEx sold most of its gold and all of its silver production to Silverstone Resources in November 2008. Silverstone was subsequently bought by Silver Wheaton Corp. (“SLW”) who now owns the Minto mine precious metal stream. SLW pays Minto US$300/oz Au and US$3.90/oz Ag through the mine life.

21 Contracts

MintoEx has several contracts for the supply of goods and services to the mine, concentrate sales and metal price guarantees. SRK reviewed the material MintoEx contracts and found them to be reasonable and within industry norms. A summary of some of the main contracts is shown in Table 21.1.

Table 21.1: Significant Minto Contracts

Company	Contract Coverage	Comments
Metal Trading Company	Concentrate purchase	Agreement to purchase all Minto concentrates up to the end of 2013. See Marketing Section
Macquarie Bank Ltd	Metal price guarantee (hedge)	Agreement to pay Minto pre-set metal prices for a portion of its metal production. This is only valid for a portion of the 2011 copper production and expires in 2011. See Marketing Section
Silver Wheaton Corp. (“SLW”)	Precious Metal Stream	Silver Wheaton has an agreement with MintoEx to purchase the LOM gold and silver production at Minto for US$300/oz Au and US$3.90/oz Ag. All of Minto’s silver production is committed to SLW. Up to 30,000 Au oz/annum is committed to SLW and thereafter, 50% of the gold production is committed to SLW.
Yukon Energy Corporation	Grid power	Minto agrees to purchase power from YEC, pay for a portion of the new transmission line in exchange for YEC building the main new transmission line.
Dyno Nobel Canada Inc.	Explosive and accessory supply	Supply, storage, transportation and placement of explosives and accessories
Pelly Construction	Mining and mobile equipment supply	Pelly Construction currently performs all open pit mining functions at Minto and uses its ancillary equipment for various jobs on site. The mining costs shown in 2012 and beyond assume owner/operator mining, not contractor.
Canadian Lynden Transport Co.	Concentrate transport	Provides terms and conditions for the road transportation of concentrates to Skagway, AK. Valid until the 2nd Qtr, of 2014.
Great Northern Oil	Diesel supply	Transport and supply of diesel fuel.
Sodexo	Camp Services	Lodging and catering

Contract Mining

Minto and Pelly Construction have entered an agreement to the end of 2011 for open pit mining and support activities. The unit rate per bank cubic metre (“bcm”) for loading, hauling and dumping (“LHD”) is based on two standard haul criteria; haul distance and road gradient. Variations to the haul criteria greater than 10% lead to change in contract costs. LHD rates are exclusive of fuel, explosives and force account charges. Drilling and blasting costs for waste vary based on powder factor (“PF”) requirements for various types of material but is generally categorized as ore, waste and overburden (PF = kg of explosive per m^3 of material blasted), as well as the drilling equipment used.

The blasting is conducted by another contractor, Dyno Nobel Canada, who also ships, stores, blends delivers explosives on site and performs blast hole loading services.

The work performed by the contractors appeared to be of good quality and they have been an integral part of the mining operation since the mine opened. Pelly and Dyno combined, maintain a workforce at Minto of between 30 and 40 people depending on the amount of work being done.

22 Environmental Considerations

22.1 Environmental Assessment and Licensing

In the Yukon, mining projects require an environmental assessment prior to the issuance of significant operating permits for mining, including a Type A Water Use Licence and a Quartz Mining Licence.

As the Minto Project was originally submitted to DIAND for environmental assessment in December 1994, the project was assessed and a positive determination made under the Environmental Assessment Review Process Guidelines Order (EARPGO). In January 1995, the Canadian Environmental Assessment Act (CEAA) was enacted and project assessments related to the Type B Water Use Licence for the Big Creek bridge construction and Land Use Permit for the access road construction were conducted under this assessment regime by DIAND.

In April 2003, the Yukon Territory Government (YTG) assumed responsibility for management of minerals, water, lands and forestry resources in the Yukon, including the environmental assessment of development projects as part of the devolution transfer agreement with the Federal Government. Mirror environmental assessment legislation was created and subsequent assessments were then carried out by the YTG under the Yukon Environmental Assessment Act (YEAA). In November 2005, the Yukon Environmental and Socio-economic Assessment Act (YESAA) legislation created under the Umbrella Final Land Claims Agreement was formally enacted and this legislation now guides developmental assessments in Yukon. Any activities that trigger environmental assessment in the Yukon are now conducted in accordance with this legislation (see http://www.yesab.ca/ for more information.)

Once an environmental assessment process is completed, the project moves through the regulatory permitting phase to obtain a Type A Water Use Licence, Quartz Mining Licence and other minor approvals. Water Use Licences (i.e. Type A Water Use Licence) are issued by the Yukon Water Board under the Yukon Waters Act (YWA) and *Waters Regulations*, with the approval of the YTG Minister of Executive Council Office. The Quartz Mining Licence is issued by YTG Minister of Energy Mines and Resources under the Yukon Quartz Mining Act (YQMA).

Elements of the Minto Project have undergone environmental assessment under EARPGO, CEAA and YEAA. A previous milling and mining rate increase (2008) and water management amendments have also been assessed under YESAA. These previous environmental assessment activities undertaken for the Minto Project are summarized in the following Table 22.1. The project is currently (January 2011) in the assessment process under YESAA again for Phase IV Expansion amendments to the major authorizations.

Table 22.1: Previous Environmental Assessments of the Minto Project

Activity	Period	Sources
Minto Explorations Ltd. Minto Mine Development, Operation and Closure	1996 to 1998	Government and company reports, 1996. DIAND EARP screening and Decision Report, Water Licence QZ96-006.
Minto Explorations Ltd. Minto Mine Development, Operation and Closure Cumulative Effects Assessment	1999	Company report on Cumulative Effects, 1999. Quartz Mining Licence QLM-9902.
Minto Explorations Ltd. Minto Mine Development, Operation and Closure Licence Amendments – Expiry Extensions and Temporary Closure Modifications	2004 to 2005	Government and company reports, 2004. YG AO Development Assessment Branch YEAA Screening Water Licence Amendment and Quartz Mining Licence QML-0001
Minto Explorations Ltd. Mining and Milling Rate Increase, Minto Project	2008	Company Reports, 2008. YESAA DO Assessment Quartz Mining Licence QML-0001 Amendment
Minto Explorations Ltd. Water Management and Milling Rate Amendments, Minto Project	2009 to 2010	Company Reports, 2009 YESAA DO Assessment
Minto Explorations Ltd. Minto Mine Expansion- Phase IV	2010 to 2011	Company Reports, 2010, YESAB Project Proposal, August 2010

22.2 Environmental Authorizations

Several government agencies, both federal and territorial, are involved in reviewing, assessing, authorizing and monitoring Minto Mine in the form of regulatory and guideline based environmental instruments. The major instruments or authorizations and their attendant assessment and amendment histories are summarized below.

Type A Water Use License

In February 1997, MintoEx submitted a Type A Water Use Licence application (QZ96-006). The Yukon Water Board (YWB) convened a public hearing into the application in May 1997, and after deliberations by the YWB, the Type A Water Use Licence was subsequently issued in April 1998 pursuant to the Yukon Waters Act (YWA) and Regulations for the mine and milling operations. The Type A Licence was supported by the Selkirk First Nation (SFN) and contained typical licence terms and conditions to ensure that mitigation measures identified during the environmental assessment were implemented. The expiry date for the Type A Water Use Licence was June 30, 2006. This licence was subsequently extended up to 2016, as discussed below.

Type B Water Use License

In August 1995, the company submitted a Type B Water Use Licence application, which was filed with the YWB for construction of the Yukon River barge landing sites, the Big Creek Bridge, and Minto Creek road culvert installations. In October 1995, a land use and quarry permit application for the access road construction was filed with DIAND Land Resources.

An integrated Canadian Environmental Assessment Act (CEAA) screening of the Type B and land use applications was completed and a positive determination was made in August 1996. Type B Water Use Licence MS95-013 and Land and Quarry Permit YA5F045 were issued in August 1996 and the initial 16 km of the Minto project access road, barge landings and Big Creek Bridge were installed in September and October 1996.

Yukon Quartz Mining License

In 1999, the Yukon Quartz Mining Act (YQMA) was amended and Section 139 of the Act required that all development and production activities related to quartz mining in the Yukon be carried out in accordance with a licence issued by the Minister. In June 1999, the company filed an application with DIAND Minerals for a Yukon Quartz Mining Licence, which included a cumulative effects assessment for the project to ensure that the provisions of CEAA were met. DIAND issued Yukon Quartz Mining Licence QLM-9902 in October 1999 with a licence expiry date of June 30, 2006. This licence was subsequently extended until June 30, 2016 (see chronology of amendments in the following section).

Amendments and Current Licensing

Water Use Licence QZ96-006 was amended (Amendment #1) to revise the decommissioning requirements for the project, and to request the submission of an interim plan as the project was not yet constructed. The project is still subject to Water Use Licence QZ96-006.

In addition, the Federal *Metal Mining Effluent Regulations* (MMER) under the Fisheries Act currently apply to the Minto mine. These Regulations are a law of general application and the requirements of this legislation are the responsibility of MintoEx. Generally, the Type A Water Use Licence is considered more restrictive that the MMER; however, separate reporting for effluent discharge and receiving water monitoring is required by the Federal Department of Environment Canada.

As the Type A Water Use Licence (QZ96-006), Type B Water Use Licence (MS95-013), and Yukon Quartz Mining Licence (QML-9902) were set to expire in June 2006, and in recognition of the project development delays, licence amendment applications to extend the licences to June 30, 2016 were filed with the YWB and Yukon Government (YG), Department of Energy, Mines & Resources (EMR) in October 2004. In response to the amendment applications, YTG Development Assessment Branch completed a Yukon Environmental Assessment Act (YEAA) screening of the Type A Water Use Licence using the previous EARPGO screening and issued their screening report in March 2005.

The YWB completed a YEAA screening of the Type B application and subsequently issued the amended Type B Water Use Licence (MS04-227) in February 2005. YTG Development Assessment Branch completed a YEAA screening of the Type A Water Use Licence and Yukon Quartz Mining Licence using the previous EARPGO screening and issued their screening report in March 2005.

The YWB issued the amended Type A Water Use Licence (QZ04-064) in September 2005 (Amendment #2) and YTG EMR issued amendments to the Yukon Quartz Mining Licence QLM-0001, Amendment No. 05-001 in December 2005 and Amendment No. 05-002 to change the mill rate to 2,500 today in October 2006. The Type A Water Use Licence (WUL) was further amended on April 6, 2006 (Amendment #3) following an application by MintoEx to address an apparent inconsistency in the original licence regarding the milling of sulphide ore.

In July 2008, the MintoEx submitted a Project Proposal to Yukon Environmental and Socio-Economic Assessment Board (YESAB) that outlined a proposed increase in the project mining and milling rate. The Mayo Designated Office (DO) issued a recommendation that the project proceed, and YTG EMR as decision body released a decision document that concurred with the assessment recommendations. Subsequently, Quartz Mining Licence QML-0001 was amended to increase the milling rate (and associated mining rate) to 3,200 tpd on July 24, 2008.

In response to exceptional precipitation received in the site area in late August 2008 and an imminent release of water from the Water Storage Pond (WSP) that did not meet water licence discharge standards, MintoEx applied on August 25, 2008 to the YWB for an emergency amendment to the Water Use License QZ96-006 under section 21 (4), c.19 of the Yukon Waters Act. The application to release 350,000 m^3 of water from the WSP using the Metal Mining Effluent Regulations (MMER) effluent discharge criteria was approved and Amendment #4 to the WUL was issued on August 26, 2008.

The melting of significant snowpack accumulations in the winter of 2008/09 required the retention of freshet runoff in the open pit and prompted concern about stability of the south pit wall should additional summer precipitation events need to be directed there as well. As a result, MintoEx applied again for an amendment to the Water Use Licence QZ96-006 under the same provision in June of 2009, to allow the release of water that would provide additional capacity for such an event. On June 26, 2009, the Yukon Water Board approved Amendment #5 which authorized the release of 300,000 m^3 of water from the site, subject to the same MMER criteria and additional monitoring requirements.

On August 3, 2009, MintoEx received an Inspector’s Direction from YTG EMR to empty the pit of accumulated runoff water prior to October 15, 2009. Subsequently, MintoEx, in order to remain in compliance both with the Inspector’s Direction and with its water use licence, applied for another amendment to WUL QZ96-006, again under the emergency provision of the Yukon Waters Act. The Yukon Water Board approved this amendment (Amendment #6) and MintoEx was permitted to release an additional 705,000 m^3 of water from the Minto Mine site provided it met amended discharge standards.

MintoEx applied for an additional amendment (Water Use License Application #QZ09-094) to WUL QZ96-006 in June of 2010 to authorize changes to the water management plan and effluent discharge standards. A minor milling rate increase and other housekeeping items are also included. The Yukon Water Board is currently reviewing the application and information presented in a formal Public Hearing conducted December 6-8, 2010.

All of the above noted licences have an expiry date of June 30, 2016.

Assessment and Licensing for Phase V

The expansion of the Minto Mine in the Phase V development will require environmental assessment and major licence amendments. Environmental and socio-economic assessments under YESAA are conducted at different levels of review, depending on the project scope and thresholds of project elements. Most projects are assessed at the Designated Office (DO) level, while more complex projects are assessed at the Executive Committee (ExComm) level. Information requirements for project proposals at the ExComm level are more comprehensive than those required for DO assessments, and ExComm review and assessment timelines are longer.

Although the Phase V development plans may not trigger an Executive Committee review (to be determined based on project details), a Designated Office reviewer may forward a project to the Executive Committee review level if it is determined that the Project Proposal is too complex to be fairly assessed at the DO level or if significant public concern is demonstrated by the public or local First Nation during the review period.

22.3 Selkirk First Nation

On May 29, 1993, the Government of Canada, the YTG, and Yukon First Nations as represented by the Council of Yukon Indians (now the Council of Yukon First Nations) signed the Umbrella Final Agreement (UFA) after approximately 20 years of negotiation. The UFA provided a comprehensive land claim agreement for all Yukon First Nations and an outline for community based social well-being, political autonomy, and economic independence.

On July 21, 1997, Selkirk First Nation (SFN), became the fifth First Nation to sign a comprehensive land claim agreement. The Selkirk First Nation Final Agreement and the Selkirk First Nation Self Government Agreement (LCA) was negotiated by SFN, YTG and the Government of Canada. Through the LCA, the SFN was allocated 1,830 sq. miles of land over which the SFN has ownership and control. Of this land total, 930 sq. miles are Category A Settlement Lands, of which the SFN has the ownership of the surface and subsurface, including minerals and oil and gas, and exclusive fish and wildlife harvesting rights. The balance of the land allocation is 900 sq. miles of Category B, on which SFN has ownership of surface only, and a small amount of land, (2.62 sq. miles) in the form of site-specific parcels.

Three years before the start of land claims negotiations, the Minto and DEF mineral claims were staked by two competing exploration syndicates. These claims were extensively explored between 1971 and 1974 and feasibility studies were completed in 1975-76, but thereafter, activities ceased. Ownership was somewhat restructured in 1984 and 1989, which resulted in limited exploration in 1989, after which the property became dormant again. In 1993, MintoEx purchased the claims for the purposes of initiating mining in the area, and was active until 1999. During this time, SFN signed the LCA, which placed the MintoEx claims within Category A Settlement Lands. Recognizing that, pursuant to land claims agreement, the SFN were afforded the rights to exercise certain powers over land use and environmental protection.

MintoEx claims continue to lie within SFN Category A Settlement Lands (Parcel R-6A), where both surface and mineral rights are reserved for SFN. In addition, the mine access road lies within parcels Parcel R-6A and Parcel R-44A, and the east barge landing access point lies on Parcel R-43B. However, under the LCA, certain rights are reserved, including:

- All rights to mines (opened and unopened) and minerals (including precious and base metals) within settlement land are ceded to the Crown except on Category A lands, where mines and minerals are owned fee simple by SFN excepting pre-existing rights such as those that form the Minto property (SFN Final Agreement, Chapter 5.4.2);
- Where pre-existing rights lie within Category A land, such as the Minto mineral claims, the government will continue to administer those rights as though they were still Crown Land (SFN Final Agreement, Chapter 5.6.2) except that any royalties collected from those mineral rights will be paid to SFN (SFN Final Agreement, Chapter 5.6.3);
- A 30 m right of way within land parcels R-6A, R-40B and R-44A covering the existing access road from Minto Landing to the project, with the right to construct, maintain, upgrade and use the right of way and road for as long as the company holds its mineral rights (SFN Final Agreement); and
- The right of YTG to grant a surface lease over the mineral rights, subject to the consent of SFN, not to be unreasonably withheld (SFN Final Agreement).

If any of the claims are allowed to lapse, they could not be re-staked, and the surface and mineral rights would revert to the SFN. In September 16, 1997, MintoEx and the SFN entered a Cooperation Agreement concerning the Minto Project with respect to the development of the Minto Mine. This agreement was amended November 4, 2009. In addition to establishing cooperation with respect to permitting and environmental monitoring, this confidential document deals with other economic and social measures and communication between Selkirk First Nation and the company. This agreement will continue to guide SFN involvement in the project as mine expansion planning and development proceeds.

22.4 Environmental Conditions

Table 22.2 below summarizes existing environmental conditions in the Minto Mine area. The information was compiled from various published and unpublished reports. This table is not intended to provide a thorough reflection of the environmental setting, but rather a succinct overview of the key environmental parameters.

A more detailed description of the environmental conditions in the Minto Mine area was presented as part of the environmental assessment and licensing process associated with the 1996 WUL application. Updates to these conditions (presented in the Project Proposal for the 2009 Water Management and Mining and Milling Rate Amendments and the Project Proposal for the 2010 Minto Mine Expansion- Phase IV) have been based on further information collected at the Minto Mine site during the Interim Closure monitoring and more recently from monitoring associated with license conditions and operational management during mine construction and operations.

Table 22.2: Minto Mine Setting Summary

Project Area Attribute	Description
Region:	Yukon
Topographic Map Sheet:	NTS 115 I/10, 115 I/11
Geographic Location Name Code:	Minto Project
Latitude:	62° 36' N
Longitude:	137° 15' W
Drainage Region:	Yukon River
Watersheds:	Yukon River, Big Creek, Wolverine Creek, Dark Creek, Unnamed Creek B and Minto Creek.
Nearest Community:	Pelly Crossing, Yukon, approx. 33 km north on Klondike Highway.
Access:	Klondike Highway, Barge crossing on Yukon River at Minto Landing, Minto mine access road. Airstrip on site.
Traditional Territory:	Northern Tutchone, Selkirk First Nation peoples. Traditional use for hunting, trapping and fishing.
Surrounding Land Status:	Selkirk First Nation Settlement Lands and Federal Crown Land.
Special Designations:	Lhutsaw Wetland Habitat Protection Area located approx. 17 km NE of Minto Landing (outside the project area).
Ecoregion:	Yukon Plateau (Central) - Pelly River Ecoregion.
Study Area Elevation:	Rolling hills above mine site at 1131 m to 600 m at the Yukon River Valley bottom.
Site Climate:	Recorded site air temperature ranges from –43.2°C (Nov. 2006) to 25.9°C (Jun. 2006). Mean annual temp. of -3.0°C. Mean annual rainfall is 131 mm.
Vegetation Communities:	Riparian, black spruce, white spruce, paper birch, lodgepole pine, buck brush/willow and ericaceous shrubs, feather moss, sedge, sagewort grassland, mixed, aspen, balsam, and sub-alpine. Discontinuous permafrost is present on site. Site has been subject to recent forest fires.
Wildlife Species:	Moose, caribou, Dall sheep, mule deer, grizzly and black bear, varying hare, beaver, lynx, marten, ermine, deer mouse, fox, mink, wolverine, least weasel, wolf, squirrel, porcupine coyote, muskrat, otter and wood frog. Bird species include: spruce, blue, ruffed, and sharp-tail grouse, waterfowl, raptors, and a variety of smaller birds.
Fish Species:	In the Yukon River, chinook, coho, and chum salmon, rainbow trout, lake trout, least cisco, bering cisco, round whitefish, lake whitefish, inconnu, arctic grayling, northern pike, burbot, longnose sucker and slimy sculpin; In Big Creek, Chinook and chum salmon, arctic grayling and whitefish species; In Wolverine Creek, chinook salmon, arctic grayling, and slimy sculpins; In Minto Creek and project area watershed (lower reaches only), chinook salmon, slimy sculpin, round whitefish, arctic grayling.
Known Heritage Resources:	East side of Yukon River in the vicinity of Minto Landing four historic sites designated KdVc-2 (Minto landing), KdVc-3 (Minto Resort), KdVc-4 (Old Tom's Cabin), and KdVD-1 (Minto Creek).

(*Table adapted primarily from Hallam Knight Piesold Ltd. 1994. Minto Project, Initial Environmental Evaluation, Supporting Volume II, Environmental Setting.*)

Environmental conditions pre-mine development have been compiled, assessed and referenced in previous environmental assessments, but the environmental assessment and permitting process for the Phase V expansion will require that these conditions be further updated based on recent site monitoring program results. Specifically, baseline environmental conditions of the drainage to the north of the Minto Creek drainage will be of interest to assessors, as the Minto North deposit is located approximately 100 m into the drainage.

Although physically there will likely be minimal disturbance in this drainage from the mining activities, there is potential for there to be effects to the aquatic receiving environment downstream. An updated Environmental Conditions report will be prepared to support the Phase V development and will update all environmental data for the project area and will be used for the assessment and permitting processes.

This watercourse and its drainage area north of Minto Creek (McGinty Creek) have been the subject of intermittent biophysical study since the 1970s, often as a reference (undisturbed) area for aquatic and geochemical investigations of Minto Creek. Additional water quality, hydrology, benthic invertebrate and fisheries investigations were all conducted in the McGinty Creek drainage in 2009 and 2010 by Access Consulting Group, and a summary report on the baseline conditions will be prepared for inclusion with the Environmental Conditions report for the environmental assessment project proposal. The McGinty Creek and Minto Creek Drainages are presented in Figure 22.1 below, with the approximate location of the Minto North Deposit.

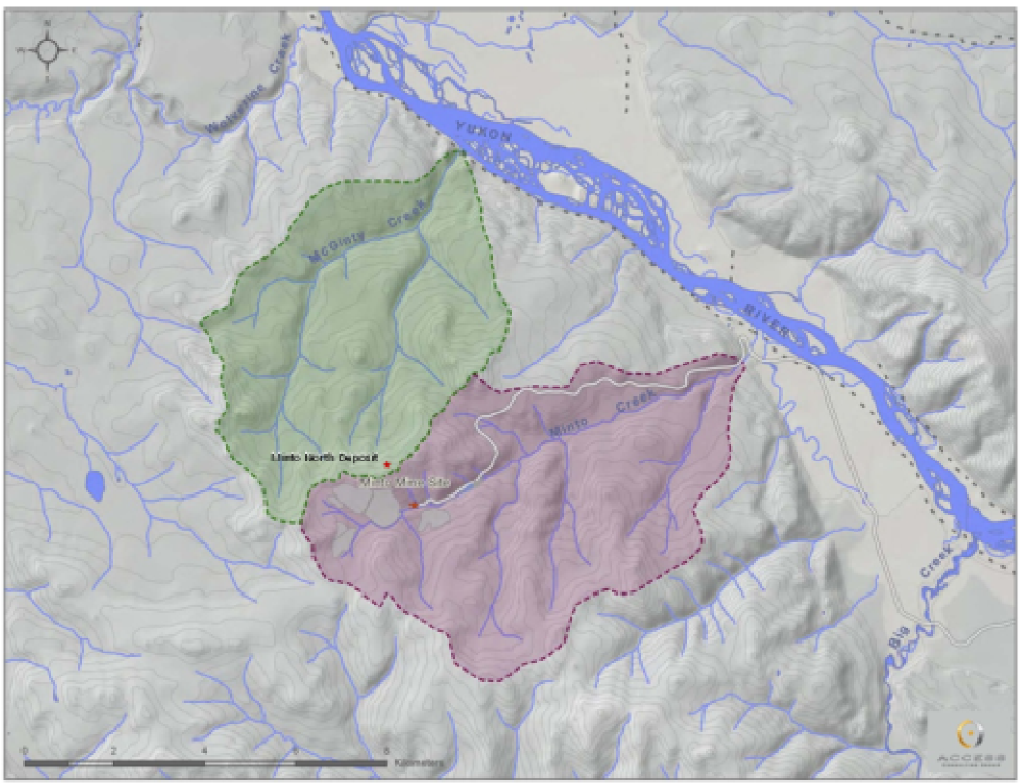


Figure 22.1: Minto Creek and McGinty Creek Drainages relative to Minto Mine Site, Minto North Deposit and Yukon River

Groundwater monitoring information is limited to two multi-level wells at the Minto site. Installation of additional wells to augment the groundwater monitoring network at the site is currently being planned, with the intention of characterizing the groundwater conditions in all areas of current and proposed mining activities.

22.5 Water Management and Effluent Discharge

MintoEx, in its original WUL application submitted in 1996, outlined a water management plan based on the limited baseline information and project projections available for the Minto Mine at the time. This information included hydrology and water balance information, operational water requirements, water storage, treatability studies and a diversion strategy for discharge to lower Minto Creek.

This 1996 WMP and supporting information formed the basis for the existing WUL QZ96-006 conditions that govern water use, treatment and effluent discharge at the Minto Mine, which include stringent effluent discharge standards relative to other major mining projects in the Yukon licensed around the same time (late 1990s). These WUL discharge standards are presented in Table 22.3 below.

Table 22.3: Water Use Licence QZ96-006 Effluent Quality Standards for Minto Mine Project

Parameter	Units	WUL QZ96-006 Effluent Quality Standards	
		Frequency	Daily Limit
pH	pH units	weekly	6.5 - 9.0
Suspended Solids	mg/L	weekly	15
Aluminium	mg/L	weekly	0.5
Iron	mg/L	weekly	1
Copper	mg/L	weekly	0.01
Lead	mg/L	weekly	0.002
Manganese	mg/L	weekly	0.2
Nickel	mg/L	weekly	0.065
Zinc	mg/L	weekly	0.03
Total Ammonia	mg/L	weekly	1
Oil and Grease	visibility	weekly	no visible oil or grease
Rainbow Trout Acute Lethality Test	<50% mortality in 100% effluent	monthly	Pass

In the intervening period since the application, screening and issuance of the Type A water use licence, significant additional baseline and operational data have been collected. These data show that the conditions upon which the initial water management and treatment assumptions were predicated were not representative of actual conditions observed.

Since commencement of commercial production at the Minto Mine in 2007, MintoEx has responded to this discrepancy between modelled water quality and observed conditions with a number of progressively intensive and expensive measures aimed at maintaining compliance with the WUL QZ96-006 discharge criteria. These measures have only been partially and temporarily successful, and are not sustainable in the long term.

As a result, in the summers of 2008 and 2009, MintoEx sought and received authorizations to release significant volumes of stored runoff subject to adjusted discharge standards, as discussed previously.

In 2010, MintoEx was able to consistently meet the WUL QZ96-006 effluent discharge standards at low discharge rates through the adjusted use of the new site water treatment plant. Although this allowed the release of excess site water, it was first stored in the pit and required the temporary cessation of mining activities again. MintoEx has therefore revised the site Water Management Plan and expects the authorization for its implementation (Water Licence amendment) shortly. This includes the construction and use of storm water diversions, a water treatment plant and revised project effluent discharge standards.

This water management plan should provide the project with much improved flexibility in how it manages site runoff water and effluent discharge, while still protecting downstream aquatic resources. Although the major elements of these water management revisions were designed to be functional beyond the mining of the Main Pit and into mine expansion plan currently being assessed for the Phase IV developments, the fundamentals of the approved amended plan will still apply for the Phase V development activities. However, the plan will still require further reassessment and adjustment (new drainage area) during the Phase V development planning process.

The critical considerations with respect to operational water management for Phase V planning will be contingency runoff storage of water requiring treatment of settling prior to discharge, and ensuring that effects to the unnamed drainage for the Minto North deposit are minimized and fully mitigated. Waste management plans for waste rock, tailings and wastewater will be scrutinized and these management plans must be fully integrated for the Phase V development. The currently proposed water management plan identifies the Main Pit as a contingency storage location. As the mine planning progresses for Phase V, this contingency storage requirement should be reassessed, as it may have implications on pit sequencing and/or waste deposition in the Main Pit.

22.6 Closure Planning

History

A Decommissioning and Reclamation Plan for the Minto Project was filed with the Yukon Water Board in April 2001 in accordance with WUL QZ96-006. This plan included cost estimates for closure activities. A review of the 2001 plan by YTG Water Resources guided the preparation, as required in Part G – Decommissioning and Reclamation of WUL QZ96-006, of an Interim Care and Maintenance & Interim Closure Plan which was filed with the Yukon Water Board in November 2003. The Interim Plan addressed two scenarios:

- Continued care and maintenance of project infrastructure; and
- Closure issues related to the decommissioning of existing site developments at the Minto mine and reclamation of the site, including reclamation and security costs associated with the then dormant property.

The 2003 plan presented closure scenarios based on existing conditions in the construction phase at the time.

The submission of a detailed closure plan was also required under QML-0001, Section 14.1. Both the 2001 and 2003 plans were drawn upon in the preparation of the Detailed Decommissioning and Reclamation Plan (DDRP), which was submitted in November 2006 and approved in June 2007. A program was presented for site management and monitoring both during implementation of closure and after decommissioning and reclamation measures are completed.

Decommissioning and reclamation cost estimates were provided and financial security requirements were reviewed, leading to the provision of security to YTG based on approved closure cost estimates.

The first update to this DDRP was submitted in September 2009. The updated DDRP addresses the long-term physical and chemical stability of the site, including reclamation of surface disturbances, and the unanticipated water quality and water quantity issues at the Minto Mine. Revised closure cost estimates for interim and final closure scenarios were also submitted and reviewed, and the security held with YTG was adjusted accordingly.

Section 14.3 of QML-0001 requires that this DDRP be updated again in 2011, but discussions are underway with YTG to formulate a way of integrating this update with the closure planning details regarding the Phase IV Expansion Plan. The Phase IV Expansion proposal currently in the environmental assessment stage included a Conceptual Closure Plan, and the concepts presented will need to be advanced into detailed planning as a part of the amendment of QML-0001 to authorize the Phase IV development activities.

Closure Philosophy

A principle tenet of the philosophy followed during the development of the DDRP was to work towards an eventual passive closure, with eventual walk-away closure after long term chemical and physical stability has been demonstrated. It is anticipated that final determination of the effectiveness of closure measures for passive and eventual walk-away status will be the subject of review and concurrence with regulatory agencies, First Nations and the public. Under the Quartz Mining Act (QMA), the company would then apply for a certificate of closure from the YTG.

MintoEx has indicated in the 2009 DDRP update its continued intent to implement progressive reclamation measures where possible during mine construction and operations. This approach should provide valuable reclamation success feedback for use in advanced/final closure and would reduce final reclamation liability and costs and shorten the overall reclamation implementation schedule. Progressive efforts will also help reduce slope erosion through physical slope stabilization of revegetation efforts, enhancing ultimate reclamation success.

Although the closure of the Dry Stack Tailings Storage Facility is scheduled to proceed in concert with the Phase IV development activities, no other substantial progressive reclamation was conducted on the site in the first three years of operations according to the proposed schedule in the 2007 DDRP. The YTG may therefore be less willing to offset security requirements through future commitments to progressive reclamation.

Current Closure Plan

Under the current plan, decommissioning of the site infrastructure will see some key diversions left in place and drainage of upper Minto Creek and minor tributaries re-established in channels where required. Proposed reclamation measures are primarily traditional in nature, i.e. re-contour, cover, and re-vegetate. This will apply to waste rock dumps, stockpile pads, lay-down areas and the mill complex and camp areas. Water treatment facilities will remain on site as long as required to maintain project water quality control, as will the main water dam. Re-vegetation prescriptions are being tested at various trial plot locations around the site to optimize revegetation success of progressive and final reclamation seeding.

Phase IV Closure Plan

Closure philosophies and conceptual closure measures for the Phase IV mine plan mirrored those presented in the previously submitted and approved DDRPs, but were expanded based on results of a Water Quality Prediction for Closure Conditions. This prediction indicated that additional source load mitigation measures would be required to reduce loading from the key sources – waste dumps and dry stack tailings in particular. Detailed plans to support the concepts of infiltration-reducing dump covers and passive treatment systems are currently being developed, and the cost estimate and security will also be adjusted accordingly.

Closure Planning for Phase V Expansion

Closure philosophies and measures for the Phase V mine plan will mirror those presented in the previously submitted and approved DDRPs. Although closure and reclamation concepts will be required for the Phase V environmental assessment and attendant authorization amendments, it is expected that actual details (including closure cost estimates) will be presented in a subsequent revision to the DDRP to support the QML amendment, as is currently being conducted for the Phase IV closure planning. Closure measures for the site following the completion of the Phase V mine plan are expected to generally follow those currently authorized and/or authorized for the Phase IV development phase.

22.7 Metal Leaching/ Acid Rock Drainage Characterization

Introduction

The Phase V mine plan will introduce the following components to the presently-permitted facilities at the Minto Mine:

- Waste rock from the Area 2/118, Ridgetop, and Minto North open pits;
- Development rock from the access drifts to the Area 2/118 and Minto East underground; and
- Tailings from processing ore from Area 2/118, Ridgetop, Minto North and Minto East.

Geochemical characterization of metal leaching/ acid rock drainage (ML/ARD) potential has been carried out to inform the development of waste management plans for the planned Phase V operations. The results are presented in the following sections.

Phase V Waste Rock Characterization

Sample Selection

- Area 2: Two rounds of Area 2 waste rock testing were carried out.
  - For the first round of testing, 36 samples were selected by SRK to include: host rock surrounding the ore horizons, unmineralized rock between the ore horizons, weakly mineralized rock, and ore grade material. Details of the sample selection process for the first round of Area 2 rock characterization, including the origins of the samples selected, can be found in SRK (2007).
  - Drilling in the vicinity of the southwest portion of the Area 2 Pit had not been completed at the time of the first round of Area 2 waste rock characterization. The second round of Area 2 testing was carried out in 2008 utilizing newly-available drill core from this southwest pit region. Samples were selected from drill core intervals by SRK based on metal and sulphur contents from exploration assays; intervals were selected to target bulk waste (11 samples), mineralized waste (7 samples), and ore (2 samples) (based on metal and sulphur contents from MintoEx' exploration assays) and to ensure vertical and lateral coverage within the southwest region of the Area 2 Pit. A total of 20 samples were selected for the second round of Area 2 testing.
- Area 118: No Area 118 waste rock has been tested to determine ML/ARD characteristics. Waste rock from the Area 118 open pit is expected to have similar characteristics to Area 2 waste rock. There will be little to no waste rock brought to surface during underground mining of the Area 118 deposit, as there will be minimal quantities produced and that which is produced will be used as backfill in mined-out areas.
- Underground Development Rock: ABA testing was carried out in 2010 on drill core samples from rock along or adjacent to the alignment of the proposed decline. Using the exploration drill hole database provided by MintoEx and the preliminary decline alignment, SRK selected appropriate diamond drill hole intervals for testing. Intervals were selected from 2006, 2007, 2008 and 2009 drill holes, where the interval was within 20 m lateral distance of the decline centreline and within 10m elevation of the decline floor. Sample frequency was based on an approximate target of one sample for each 100 m length of decline. Twenty-seven samples were tested.
- Ridgetop: The current understanding of the Ridgetop deposit geology is summarized in Chapter 8, and is similar to the geology of the Area 2 deposit. In general, there are several shallow-dipping mineralized horizons separated by barren granodiorite. Contacts between ore and bulk waste are sharp, and mineralized waste consists of portions of the mineralized zones with sub-ore concentrations of the metals of economic interest. A distinction between the geology of the Ridgetop and Area 2 deposits is the present of a conglomerate unit within the Ridgetop pit shell.

This conglomerate unit is made up primarily of detrital clasts of local granodiorite, and is likely to be geochemically similar from a ML/ARD perspective.

– Two rounds of Ridgetop waste rock testing were carried out

- For the first round of testing, 20 drill core intervals from the 2007 Ridgetop drilling were selected from available core for ML/ARD testing by Dylan MacGregor of SRK. Sixteen intervals of bulk granodiorite waste were selected, along with two intervals of mineralized waste and two ore-grade intervals.
- For the second round of testing, 12 drill core intervals from the 2007, 2008 and 2009 Ridgetop drilling were selected from intersections of the conglomerate unit by Dylan MacGregor and Andrew Hoskin of SRK.
- Minto North: The current understanding of the Minto North deposit geology is summarized in Chapter 8. The ore consists of shallow-dipping mineralized horizons separated by barren granodiorite, similar to the other Minto-area deposits. Contacts between ore and bulk waste are sharp. A late basaltic to andesitic dyke crosscuts the mineralized horizons; this material is barren and post-dates the mineralization. The late dyke will make up a small proportion of the Minto North waste, and it has not been characterized for ML/ARD potential.
  Twenty-three drill core intervals were selected for ML/ARD testing by Dylan MacGregor of SRK. Sample intervals (18 in total) were chosen from 5 vertical diamond drill holes to provide lateral and vertical coverage of the porphyritic granodiorite that makes up the Minto North hanging wall rock (most of the Minto North waste rock will originate from excavation of the hanging wall). In addition, five drill core intervals were selected to characterize waste rock in the deposit footwall.

Testing Methods

Two rounds of ML/ARD testing were carried out on Area 2 waste rock.

- The ML/ARD testing on Area 2 samples in 2007 was carried out at ALS Chemex in North Vancouver BC. ABA analyses were carried out using the Sobek et al. (1978) procedure with sulphur speciation and additional determination of inorganic carbon content. Elemental analyses were performed according ALS Chemex method ME-MS41 (aqua regia digestion followed by elemental determination by a combination of ICP-MS and ICP-AES).
- The ML/ARD testing on samples from the southwest region of the Area 2 Pit in 2008 were carried out at SGS CEMI in Burnaby BC. ABA analyses were carried out CEMI according to the Sobek et al. (1978) procedure with sulphur speciation and additional determination of inorganic carbon content. Elemental analyses consisted of aqua regia digestion followed by elemental determination by ICP-MS.

Ridgetop and Minto North waste rock samples were tested for ML/ARD characteristics at SGS CEMI according to the procedures noted above for the Area 2 samples tested in 2008. Underground development rock samples were tested for ABA characteristics only by those same methods.

ABA Characteristics

Potential for development of acid weathering conditions is evaluated by categorizing waste materials based on the ratio of neutralization potential (NP) and acid potential (AP). A common categorization approach is: materials with NP:AP<1 are designated as potentially acid generating (PAG); materials with 1<NP:AP<2 are designated as having uncertain acid generating potential; and materials with NP:AP>2 are designated as not potentially acid generating (NPAG).

The following sections summarize the ML/ARD characterization results for each Phase V pit. A plot of NP and AP values for all Phase V samples tested is shown in Figure 22.2. A line showing NP/AP = 3 is included for reference purposes only, due to this value being referenced in the existing water licence for waste rock from the Minto Pit.

Area 2

- 34 samples of Area 2 bulk waste were tested. NP/AP values ranged from 7.6 to 180, and all bulk waste samples were therefore classified as NPAG.
- 17 samples of Area 2 mineralized waste were tested. NP/AP values ranged from 0.6 to 61, with one sample classified as PAG (NP/AP of 0.6), one sample classified as uncertain (NP/AP of 1.96) and the remaining 15 samples classified as NPAG.
- Five samples of Area 2 ore were tested. NP/AP values ranged from 1.5 to 42, with two samples classified as uncertain (NP/AP of 1.5 and 1.8) and the remaining 3 samples classified as NPAG.

Ridgetop

- 16 samples of Ridgetop bulk waste were tested. NP/AP values ranged from 24 to 185, and all bulk waste samples were therefore classified as NPAG.
- Two samples of Ridgetop mineralized waste were tested. NP/AP values were 4.2 and 9.9, and both samples were classified as NPAG.
- Two samples of Ridgetop ore were tested. NP/AP values were 2.1 and 10.9, and both samples were classified as NPAG.
- Twelve samples of Ridgetop conglomerate were tested. Sulphur content of all samples was at or below the detection level of 0.02% total sulphur. NP/AP values were calculated by adopting the detection level total sulphur value as a conservative indicator of AP, and those NP/AP values ranged from 41 to 125. All Ridgetop conglomerate samples were therefore classified as NPAG.

Underground Development Rock

- 27 samples of underground development rock were tested.
- Sulphide sulphur content of most of the samples is low, with corresponding AP values less than 5 kg $CaCO_3$ equiv./ tonne for 23 of 27 samples tested. NP/AP values ranged from 5.7 to 130. All samples with AP<5 kg $CaCO_3$ equiv./ tonne were be classified as NPAG.

- Four of 27 samples tested had NP/AP values ranging from 0.5 to 1.4 would be classified as either Uncertain or PAG- three of these four are some of the deepest samples tested (from drill holes 07SWC211, 06SWC080, and 06SWC116), and represent a location where the decline passes through mineralization.
- All PAG and Uncertain samples have copper content near 1%. This rock would either be sent directly to the mill or stockpiled for later processing.
- Overall, underground development rock results were similar to those from Area 2 rock testing, which showed that bulk rock will not generate ARD and that mineralized rock has some potential to generate ARD.

Minto North

- 18 samples of Minto North hanging wall waste were tested. NP/AP values ranged from 20 to 55, and all hanging wall waste samples were therefore classified as NPAG.
- Five samples of Minto North footwall waste were tested. NP/AP values ranged from 39 to 62, and all footwall waste samples were classified as NPAG.

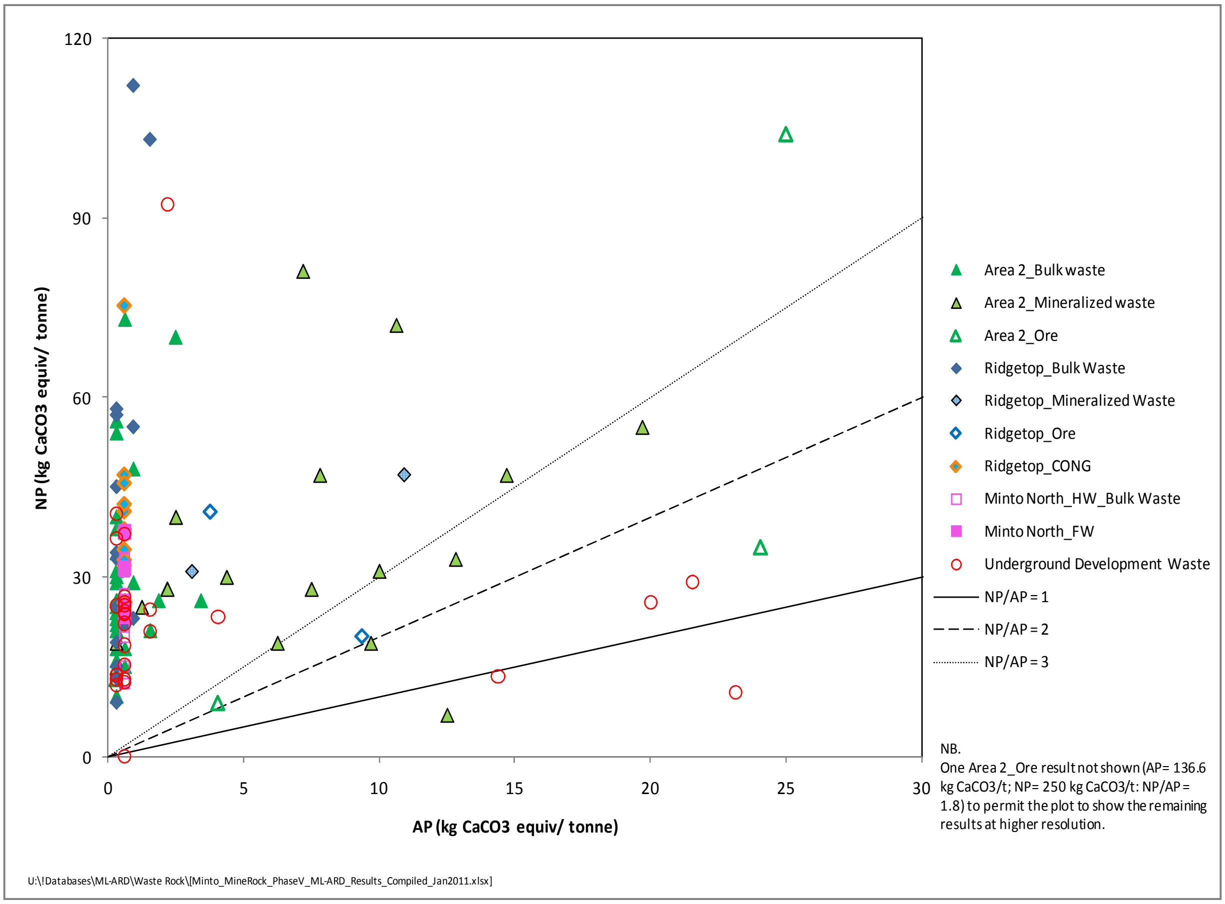


Figure 22.2: NP/AP Results for Phase V Mine Rock Samples

Elemental Content

Elemental content of mine rock tested in Phase V was compared with crustal average concentrations of granitic rocks (Price, 1997). A value of three times (3x) the crustal average concentration was used as a screen to determine whether Phase V mine rock contained anomalous elemental concentrations (based on median test results) that might indicate the potential for leaching at environmentally-significant rates. For bulk waste, median antimony concentrations in Ridgetop (median 1.1 ppm) and Area 2 waste rock (median 1.0 ppm) were reported to exceed 3x the crustal average concentration of 0.2 ppm. No other elements (copper included) had median concentrations exceeding 3x crustal average concentrations in bulk waste.

For Ridgetop conglomerate, the 12 samples tested had a median copper concentration of 153 ppm which was greater than 3x the crustal average concentration of 30 ppm. The remaining elements that were determined had median concentrations less than the 3x crustal average screening criteria.

For mineralized waste and ore samples tested, median concentrations of copper, molybdenum, and antimony exceeded 3x crustal average concentrations. Antimony concentrations in ore and mineralized waste were similar to bulk waste concentrations described above. Copper and molybdenum concentrations were elevated in mineralized waste and ore relative to bulk waste, with median molybdenum concentrations ranging from 10 to 15x the crustal average range of 0.6 to 1.3 ppm and median copper concentrations ranging from 50 to 140x the crustal average range of 5 to 30 ppm.

The elevated copper content of mineralized waste suggests that there is a risk copper may leach from these materials at environmentally-significant concentrations- implications for waste management are discussed above.

Phase V Tailings Characterization

Sample Selection

Tailings samples were selected from residues from metallurgical testing of ores from the Area 2, Area 118, Ridgetop, Minto North and Minto East deposits. The follow points summarize the samples tested.

- Area 2: Residues from locked cycle testing on ores from each of the seven discrete ore horizons (G&T, 2007) were tested for ML/ARD potential, along with a composite sample composed of 37% 272 horizon tails and 63% 280 horizon tails to evaluate the characteristics of a mixed tailings product (SRK, 2007). Ore samples were selected from drill core by MintoEx personnel, and metallurgical testing was carried out by G&T Metallurgical Services of Kamloops, BC.

- Area 118: Residues from locked cycle testing on two master composite ore samples from each of the upper and lower Area 118 ore zones (G&T, 2009a) were tested for ML/ARD potential. Area 118 ore samples were selected from drill core by Gordon Doerksen, P.Eng., of SRK, and metallurgical testing was carried out by G&T Metallurgical Services of Kamloops, BC.

- Ridgetop: Residues from locked cycle testing on master composite ore samples from the upper and lower portions of the Ridgetop East deposit were tested for ML/ARD potential. Three samples in total were tested, one from the Ridgetop East lower zone, and two from the Ridgetop East upper zone (one at a primary grind sizing of 100 µm K_{80} and the second at a primary grind sizing of 200 µm K_{80}) (G&T, 2009a). Ridgetop East ore samples were selected from drill core by Gordon Doerksen, P.Eng., of SRK, and metallurgical testing was carried out by G&T Metallurgical Services of Kamloops, BC.

- Minto North: Residues from locked cycle testing on a single master composite ore sample from the Minto North ore zone (G&T, 2009b) was tested for ML/ARD potential. Minto North ore samples were selected from drill core by Gordon Doerksen, P.Eng., of SRK and metallurgical testing was carried out by G&T Metallurgical Services of Kamloops, BC.

- Minto East: Residues from locked cycle testing on a single master composite ore sample from the Minto East ore zone (G&T, 2010) was tested for ML/ARD potential. Metallurgical testing was carried out by G&T Metallurgical Services of Kamloops, BC Testing Methods

- Area 2: Aliquots of rougher and cleaner tails from each ore horizon were combined, according to the 'as-produced' mass ratio, and submitted to ALS Chemex for ABA and elemental analysis. ABA analyses were carried out using an in-house version of the Sobek *et al.* (1978) procedure with sulphur speciation and additional determination of inorganic carbon content. Elemental analyses were performed according ALS Chemex method ME-MS41 (aqua regia digestion followed by determination of 51 elements by a combination of ICP-MS and ICP-AES).

- Area 118, Ridgetop, Minto North and Minto East: aliquots of rougher and cleaner tails from each sample were combined, according to the 'as-produced' mass ratio, and submitted to SGS CEMI for ABA and elemental analysis. ABA analyses were carried out according to the Sobek et al. (1978) procedure with sulphur speciation and additional determination of inorganic carbon content; for the Minto East sample, NP was also determined by the Modified NP (MEND 1991). Elemental analyses consisted of aqua regia digestion followed by determination of 36 elements by ICP-MS.

Results

Phase V tailings samples were assigned ARD classifications based on the categories described for waste rock in the section above (PAG, uncertain, or NPAG).

All Phase V tailings tested were classified as NPAG, with NP/AP values ranging from 3.8 to 62. A plot of NP and AP values for all Phase V samples tested is shown in Figure 22.3. NP/AP = 3 and NP/AP = 4 lines are shown for reference purposes only, due to these values being referenced in the existing water licence for tailings from Minto Pit ore.

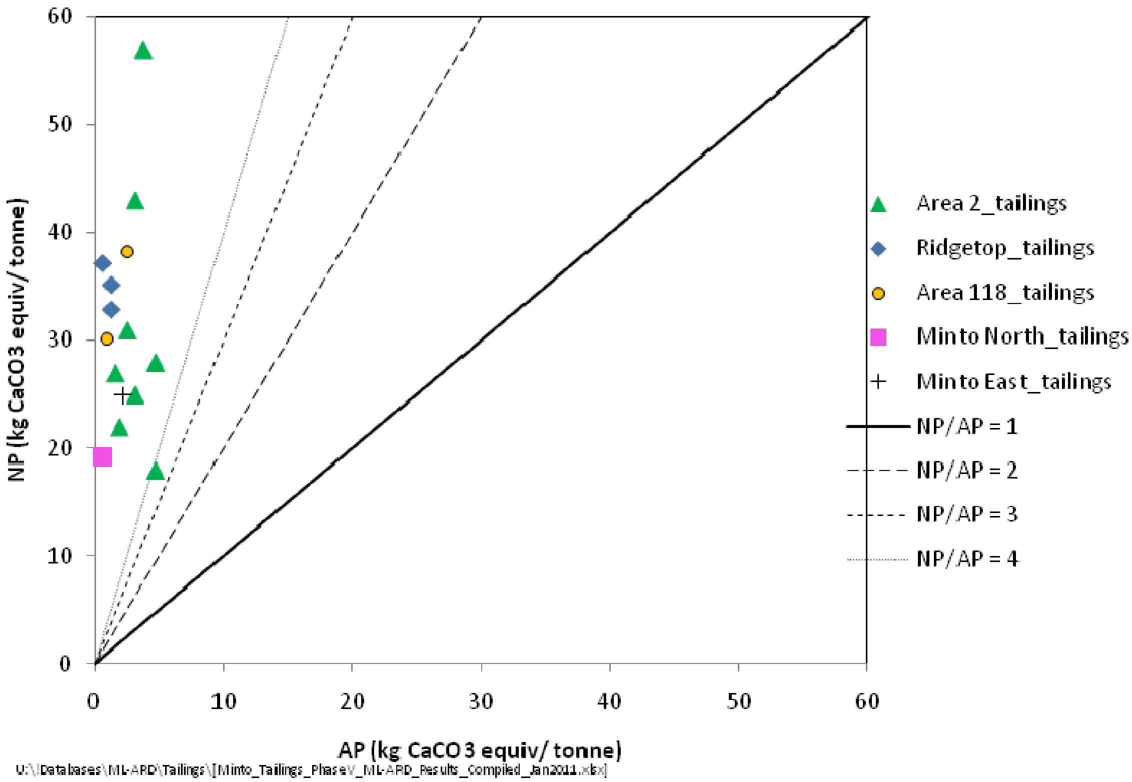


Figure 22.3: NP/AP Results for Phase V Tailings Samples

Implications of Phase V ML/ARD Characterization for Waste Management

Waste rock produced during mining of the Phase V pits and from underground is expected to have a substantial excess of NP overall. As such, prevention of ARD does not need to be considered in developing management plans for Phase V waste rock.

Detailed characterization of mineralized waste and ore from within the Area 2 Pit shell has shown that a small proportion of mineralized waste may either be PAG or have an uncertain potential to generate ARD, and it is likely that small quantities of similar material will be encountered during mining of the other Phase V pits as well. The abundant neutralizing potential present in the bulk waste is expected to consume any acidity produced locally within the waste rock dumps by small volumes of mineralized waste. The waste management plan submitted as part of the 2010 Minto Mine Expansion- Phase IV project proposal included plans for segregation and segregated disposal of mineralized waste to reduce the magnitude of long term neutral pH metal leaching. This same approach of segregation of mineralized waste and selective disposal has been factored in to the Phase V waste management plan. The final disposal location for mineralized waste will either be below the long term water table elevation or in a discrete location suitable for capping with a very low permeability cover.

Due to the nature of underground excavations, it is difficult to appropriately characterize development waste in advance. Although most of the development rock samples tested were found to be NPAG, there will be a need to carry out verification testing on development rock during temporary storage at the portal laydown stockpile if the rock is to be disposed with open pit waste in surface dumps. The use of development waste as backfill for mined-out voids will be maximized, and any remaining development waste will either be co-disposed with thickened tailings in a mined-out pit or appropriately characterized and, where geochemically suitable, disposed in the upland dumps with open pit waste rock. The overall quantity of development waste will be minor in comparison with the quantity of both tailings and open pit waste rock.

Tailings produced during processing of Phase V ore are expected to have a substantial excess of NP, and are therefore classified as NPAG. As such, prevention of ARD does not need to be considered in developing management plans for Phase V tailings.

Neutral pH leaching of copper and other trace elements from mine rock and tailings is expected to continue to require management to meet authorized effluent quality limits for the mine during operations. The mine commissioned a contractor to design, build and operate a water treatment plant to ensure mine effluent met licensed discharge criteria; this plant operated at or above design thresholds during 2010 and will remain in service for the duration of mining and beyond as necessary to meet effluent standards. Although leaching of blasting residues has not been a major problem, loadings have increased over time and as a proactive measure, an explosives handling plan has been developed and adopted by the mine to minimize leaching of blasting residues.

SRK prepared a post-closure site-wide water quality prediction as part of the Phase IV YESAB application which highlighted the need to manage neutral pH trace element leaching. The site-wide water quality prediction has not been updated for either the Phase V operational period or for the post-closure conditions. It is expected that an updated site wide water quality prediction will be a component of the Phase V environmental assessment.

23 Taxes

Federal and Provincial tax calculations start with the before tax cash flow amounts from the cash flow portion of the model and essentially deducts the cost of building the mine and mill (Class 41 UCC, CEE and CDE) as would be expected over the life of the mine as allowed by the Canadian tax rules. Generally Class 41 UCC and CEE can be deducted 100% against profit from the mine while CDE can only be deducted on a declining balance basis at 30% per year. The losses that are generated in the first few years of mine operation are deducted against income in later years.

The Yukon Quartz Mining Royalty ("Yukon mining tax") is a much different tax calculation than would normally be expected. It also starts with before tax cash flow from the cash flow portion of the model and deducts depreciation at 15% per year on a straight- line basis for the mine capital assets and mill capital assets. It deducts deferred pre-operating costs that are not capital assets on a units of production method. The Yukon mining tax does not have a loss carryover or carry back provision. Taxes are paid at rates that increase as income increases to a maximum of 12%.

The opening balances for the tax pools for both taxes are included in the cash flow model.

Since the model is based on operating cash flow the actual tax results may differ between periods from the model as concentrate shipment dates vary from the model.

23.1 Royalties

MintoEx pays royalties to the Selkirk First Nation ("SFN"). The SFN agreement is based on a percentage of net smelter return ("NSR") and is incremented based on copper price. The specifics of the royalty agreement are not shown due to confidentiality but are applied correctly in the economic model.

24 Cost Estimation

24.1 Operating Cost Estimate

A summary of the Phase V LOM operating cost estimates is shown in Table 24.1.

Table 24.1: Summary of LOM Unit Operating Cost Estimates

Area	Unit	Cost Estimate
Open Pit Mining	$/t mined	**2.57**
	$/t milled	**13.37**
Underground Mining	$/t milled	**35.17**
Total Mining	$/t milled	**20.04**
Processing	$/t milled	**12.94**
General, administration, camp, royalties	$/t milled	**12.13**
Total	**$/t milled**	**45.11**

24.1.1 UG Mining Operating Cost Estimate

The underground operating costs were developed in on an annual throughput basis. The estimate was prepared at a pre-feasibility study level of accuracy of ± 25%.

The underground mining operating costs were calculated for each cost category such as development, production, haulage, maintenance, mine services, and labour required for the 2,000 t/d operating mine. The cost was estimated using a combination of first principles calculations, experience and factored costs.

Table 24.2 shows the input data for cost estimation that were assumed in this study.

Table 24.2: Operating Cost Input Data

Operating Factors	Unit	Quantity
Underground Production		
Mine Days	d/a	365
Nominal Mining Rate	t/d	2,000
Average Mining Rate	t/a	730,000
Rock Characteristics		
In Situ Density Ore	t/m^3	2.76
In Situ Density Waste	t/m^3	2.65
Swell Factor	%	50
Loose Density Ore	t/m^3	1.84
Loose Density Waste	t/m^3	1.77
Shift Data		
Working Days per Week	ea.	7
Shifts per Day	ea.	2
Shift Length	h	12
Shift Change	h	0.5
Lunch Break	h	1.0
Equip Inspection	h	0.25
Subtotal Non-productive	h	1.75
Usable Time per Shift	h	10.25
Shift Efficiency	%	85
Usable Minutes per Hour	min	50
Hour Efficiency (50 min/h)	%	83
Effective Work Time per Shift	h	8.5

Productivities, equipment operating hours, labour, and supply requirements were estimated for each type of underground operation. Supply costs were based on estimated supply consumption and recent Canadian supplier's prices for drill and steel supplies, explosives, ground support, and services supplies, and were included in development and stoping costs. Maintenance consumables, such as parts, tyres, etc., as well as fuel ($0.85 /L) and power ($0.11 /kWh) were included in equipment operating costs and are part of mine development, stoping, haulage, or services costs.

The stope production cost was estimated based on estimated cycle times for each operation, supplies and equipment requirements for an average stope size. As RAP and PPCF are development type stoping, the cycle times and costs were estimated per each type of development as shown in Table 24.3

Table 24.3: Average Unit Development Cost

Cost Distribution	Unit	Cross-cut 5 m x 5 m	RAP Drifting 8 m x 5 m	RAP Break Through 8 m x 5 m	RAP Benching 8 m x 5 m
Equipment Cost	$/m	707.29	456.23	435.98	274.15
Drill Steel and Bits	$/m	172.52	224.98	196.97	131.37
Explosives	$/m	329.83	364.20	264.08	292.76
Ground Support	$/m	289.31	393.03	423.40	109.05
Piping	$/m	69.46	24.37	0.0	0.0
Electrical	$/m	64.91	64.91	0.0	0.0
Ventilation	$/m	13.79	13.79	0.0	0.0
Miscellaneous	$/m	6.89	6.89	6.89	6.89
Total without Labour	$/m	**1,654**	**1,548**	**1,327**	**814**

Then costs per metre of development were converted to cost per tonne of ore production. The underground stoping operating cost was estimated based on an average stope size of 8.0 m(W) x 5.0 m (H) is shown in Table 24.4.

Table 24.4: Average Underground Stoping Operating Cost Summary

Cost Distribution	RAP Drifting ($/t)	RAP Benching ($/t)	PPCF ($/t)
Stope Production	12.96	7.09	12.96
Brine	0.63	0.63	0.63
Stope Backfill	0.00	0.00	1.43
Sub-total Stoping Cost	**13.59**	**7.72**	**15.02**

The cost of production development per ton of ore was estimated based on the amount of secondary development required for stope production.

Table 24.5: Stope Development Cost

Area	Stope	Crosscut (m)	Lift / Bench (m)	Unit Cost ($/m)	2012 (K$)	2013 (K$)	2014 (K$)	2015 (K$)	Total Cost (K$)
118	101	212	238	1,654	744				744
	103	55	83	1,654				227	227
	104	38	19	1,654				94	94
	105	48	15	1,654		50	55		105
	106	55	40	1,654		74	66		140
	107	82	30	1,654		103	50		153
2	201	417		1,654	690				690
	203	67		1,654	111				111
	204	57		1,654	94				94
	205	35		1,654				58	58
	206	40		1,654				66	66
	207	65		1,654			108		108
	208	140		1,654			232		232
	209	60		1,654			99		99
	210	40		1,654				66	66
	211	105	14	1,654				196	196
East	301	275	159	1,654	198	256			454
Total Cost					**1,837**	**483**	**609**	**708**	**3,636**
Average Cost ($/t)					**2.92**	**0.66**	**0.83**	**2.10**	**1.49**

The haulage cost was estimated for each production stope based on an average haulage distance, stope tonnage and stope backfill requirements.

Table 24.6: Underground Truck Haulage Operating Cost

Area	Stope	Ramp Distance (m)	Stope Backfill (%)	Haulage Cost ($/t)	2012 (K$)	2013 (K$)	2014 (K$)	2015 (K$)	Total Cost (K$)
118	101	1,020	16	2.50	1,572	18			1,590
	103	1,020	29	2.59				128	128
	104	1,580	0	2.81				42	42
	105	1,900	0	3.19			25		25
	106	1,880	0	3.19			95		95
	107	1,800	0	3.07			37		37
2	201	1,410	33	3.85		1,447			1,447
	203	1,680	5	2.99		63	718	20	800
	204	1,400	24	3.52		196	281		477
	205	1,980	0	3.32				131	131
	206	1,760	0	3.07				34	34
	207	1,620	0	2.94				65	65
	208	1,760	0	3.07				63	63
	209	1,790	0	3.07				71	71
	210	1,890	0	3.19				62	62
	211	1,88,	12	3.33				174	174
East	301	2,380	33	4.12		1,112	1,483	325	2,921
Total Cost (K$)					**1,572**	**2,836**	**2,640**	**1,114**	**8,162**
Average Cost ($/t)					**2.50**	**3.89**	**3.62**	**3.30**	**3.34**

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

The mine maintenance cost was estimated based on required maintenance equipment, tools and supplies for maintenance shop. Maintenance consumables, such as parts, tyres, etc., were included in equipment operating costs and are part of mine development, stoping, haulage, or services costs. The maintenance labour costs were included in the overall underground mine labour costs.

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

The underground labour requirements and rates used for determining the overall mining cost were based on experience for similar operations of this size, and were divided into salaried and hourly personnel. The labour costs include overtime and shift premiums, leave pay, bonuses, pension and superannuation benefits, insurance coverage and educational assistance.

Table 24.7: Average Salaried Personnel Cost

Staff Description	Quantity	Base Salary ($/year)	Loaded Salary ($/year)	Total Cost ($/year)
Mine Superintendent	1	133,000	159,600	159,600
Senior Mining Engineer	1	105,000	126,000	126,000
Mine Ventilation/Project Engineer	1	80,000	96,000	96,000
Geotechnical Engineer	1	100,000	120,000	120,000
Mine Technician	1	65,000	78,000	78,000
Surveyor	2	65,000	78,000	156,000
Geologist	1	80,000	96,000	96,000
Geological Technician	1	65,000	78,000	78,000
Mine Rescue/Safety/Training Officer	2	75,000	90,000	180,000
Mine Captain	1	115,000	138,000	138,000
Mine Supervisor/Shift Boss	4	95,000	114,000	456,000
Sub-total Mine Operating Staff	16			1,683,600
Maintenance Superintendent	1	120,000	144,000	144,000
Mechanical/Electrical G. Foreman	1	105,000	126,000	126,000
Maintenance Planner	2	80,000	96,000	192,000
Maintenance Supervisor/Shift Boss	2	95,000	114,000	228,000
Sub-total Mine Maintenance Staff	6			690,000
Total Mining Staff	**22**			**2,373,600**

Total hourly labour requirements were estimated to achieve the daily mining production rate based on 2 shifts at 12 hrs/day with 4 crews on rotation.

Table 24.8: Average Hourly Labour Cost

Staff Description	Quantity	Base Salary ($/hr)	Loaded Salary ($/year)	Total Cost ($/year)
Jumbo Operator	8	36.00	110,376	883,008
Ground Support/Services	16	33.00	101,178	1,618,848
Blaster	4	31.00	95,046	380,184
LHD Operator	8	28.00	85,848	686,784
Truck Driver	12	26.00	79,716	956,592
Utility Vehicle Operator/Nipper	4	26.00	79,716	318,864
General Labourer	4	24.00	73,584	294,336
Grader Operator/Road Maintenance	2	26.00	79,716	159,432
General Helper	4	22.00	67,452	269,808
Sub-Total Mine Operating Labour	54			5,567,856
Lead, Mechanic/Electric	2	40.00	113,880	227,760
HD Mechanic, mobile	4	35.00	99,645	398,580
Mechanic, stationary	2	35.00	99,645	199,290
Electrician	4	35.00	99,645	398,580
Tireman/Instrument Man	2	30.00	85,410	170,820
Welder	2	30.00	85,410	170,820
HD Mechanic Apprentice	2	26.00	74,022	148,044
Dry/Lampman/Bitman	2	28.00	79,716	159,432
Sub-Total Mine Maintenance Labour	20			1,873,326
Total Mine Labour	**22**			**7,441,182**

Summary of total underground mining operating cost by year is shown in Table 24.9.

Table 24.9: Underground Mining Operating Cost Summary

Item	Total Cost (M$)	2011 (M$)	2012 (M$)	2013 (M$)	2014 (M$)	2015 (M$)	Average per year ($/t)
Development	3.6		1.84	0.48	0.61	0.71	1.49
Stoping	32.9		8.22	10.35	9.84	4.45	13.46
Truck Haulage	8.2		1.57	2.84	2.64	1.11	3.34
Mine Services	6.8		1.77	2.05	2.05	0.95	2.80
Operating Labour	19.5		5.57	5.57	5.57	2.78	7.98
Maintenance Labour	6.6		1.87	1.87	1.87	0.94	2.69
Supervisory Labour	8.3		2.37	2.37	2.37	1.19	3.40
Total Operating Cost (M$)	**85.8**		**23.22**	**25.53**	**24.96**	**12.13**	
Total Operating Cost ($/t)			**36.85**	**34.98**	**34.19**	**35.91**	**35.17**

24.1.2 Open Pit Mining Operating Cost Estimate

The open pit mining activities for the Minto mine were assumed to transition from the current contract mining scenario to an owner-operated mine as the basis for this pre-feasibility study. The transition period is assumed to occur in late 2012 when both the contractor and Minto fleets are planned to be operating. The operating costs for the owner-operated scenario are presented in Q4-2010 C$ and do not include allowances for escalation or exchange rate fluctuations.

The mining unit rate was calculated based on equipment required for the mining configuration of the operation as described in the report, as well as a comparison to similar sized open pit operations. The open pit mining costs encompass pit and dump operations, road maintenance, and mine supervision. Technical services cost have been included in the G&A costs noted elsewhere in the report.

The open pit operating costs for a 1.4 Mtpa operation are presented in Table 24.10 by mining category.

Table 24.10: Average LOM Open Pit Operating Cost Estimate (Owner-Operated Fleet)

Cost Category	Estimated OPEX ($/t mined)
Drilling	0.22
Blasting	0.33
Loading	0.27
Hauling	0.55
Roads/Dumps/Support Equipment	0.56
General Mine/Maintenance	0.23
Supervision & Technical	0.14
Total	**2.31**

Open pit mining costs are a summation of operating and maintenance labour, supervisory labour, parts and consumables, fuel, and miscellaneous operating supplies. The open pit labour requirements and rates used for determining the overall mining cost is based on experience for similar operations of this size, and are divided into salaried and hourly personnel.

Parts, non-energy consumables, fuel, and miscellaneous operating costs were based on the mining fleet requirements described in the report. A diesel fuel cost of $1.00/litre delivered to site was used as a basis in the operating cost estimate.

Mining costs in 2011 and 2012 were estimated based on the continued use of Pelly as the mining contractor. Pelly costs were estimated to be $3.08/t mined in 2011, as per Minto's budget, and $2.90/t mined in 2012 assuming an extension and reduction in cost of the Pelly contract. The Minto Fleet operating costs in 2012 were adjusted up by 15% to account for fleet start-up costs.

24.1.3 Process Plant Operating Cost Estimation

The total process operating costs were developed in Canadian dollars (C$) on an annual throughput basis. An operating cost estimate was generated for the current plant and formed a baseline for projecting the operating cost for the plant upgrade scenario. This baseline was verified against the Minto plant operating costs budget for 2011.

A summary of the average operating costs per tonne of ore treated for the Project is outlined in Table 24.11. The costs were divided into the key cost centres and all figures are as of the last quarter 2010 (calendar year).

Table 24.11: Estimated Average Operating Costs ($/t)

Summary	Year 2011 (3442 tpd)	Year 2012 - 2020 (3750 tpd)
Labour	4.79	4.40
Power	3.08	2.86
Reagents and Consumables	3.97	3.39
Contract Secondary Crushing	2.50	0.00
Other Maintenance Materials	0.48	0.44
Assay and Met Lab	1.13	1.04
Re-handle of fine crushed material	0.50	0.00
Re-handle on coarse ore stockpile	0.06	0.06
Tails filtration and dry stacking	0.73	0.00
Consultants	0.07	0.06
TOTAL $/t	**17.32**	**12.24**

The operating costs are considered to have an overall accuracy in the order of ±25%. The assumptions listed in this section require validation during a subsequent detailed engineering phase of the project.

The calculated operating cost for the Minto process plant in 2011 is based on an annualised throughput of 1,256,330 tonnes was $17.32 /t. The 2011 operating cost accommodates the tailings filter and dry stacking operation placed on stand-by at the end of June.

The calculated operating cost for the plant upgrade based on an annualised throughput of 1,372,500 tonnes was $12.24 /t. The reduction in operating cost ($/t) is primarily due to an increase in tonnage and cease operation of the tailings filter and dry stacking circuits.

Basis of Plant Operating Cost Estimate

The operating cost estimate was developed from a number of sources. Cost determinations were based on fixed and variable components relating to ore throughput and plant flowsheet. The source of data used for the operating cost estimation is summarised in Table 24.12.

Table 24.12: Derivation of Plant Operating Costs

Cost Category	Source Of Cost Data
Labour	Manning schedules and rates provided by MintoEx (2009 budget).
Power	Consumption from load estimate and power unit rate from MintoEx.
Reagents	Consumptions and unit prices from MintoEx 2011 budget and test work results on new deposits.
Consumables	Consumptions based on actuals as reported by MintoEx and Ausenco experience; unit prices from actuals as reported in the 2011 budget by MintoEx.
Maintenance Materials	Based on actuals as reported by MintoEx for first half 2009.
Contract Secondary Crushing	Based on actuals as reported in the 2011 budget by MintoEx.
Tailings Filtration	Based on actuals as reported in the 2011 budget by MintoEx.
Assay and Metallurgical Laboratory	Based on actuals as reported in the 2011 budget by MintoEx.

Operating costs not considered in this section are listed as follows:

- General and administration costs;
- Community and environment costs;
- TSF construction;
- General site environmental management costs;
- Concentrate handling (including sea freight & insurance);
- Concentrate smelting & refining; and
- Government fees and charges.

Other miscellaneous items not considered in this section include:

- Commissioning support (included in capital estimate) and plant start-up labour costs;
- Sustaining capital;
- Ongoing exploration;
- Insurances;
- Inflation;
- Import duty and applicable taxes;
- Royalties;
- Interest and finance charges; and

- Contingency.

Plant Operating Cost Estimate Inclusions

Included in the operating cost estimate are:

- Labour for supervision, management and reporting of onsite organizational and technical activities directly associated with the processing plant;
- Labour for operating and maintaining plant mobile equipment and light vehicles, process plant and supporting infrastructure;
- Costs associated with direct operation of the processing plant, including all fuels, reagents, consumables and maintenance materials;
- Fuels, lubricants, tyres and maintenance materials used in operating and maintaining the plant mobile equipment and light vehicles;
- Operation of the TSF, including tailings discharge and management and return water, excluding construction and wall lifts;
- Cost of power as provided by MintoEx, supplied from the local hydro-power grid;
- Operation of raw water supply facility from site rivers;
- Labour and operational costs for the metallurgical and assay laboratories; and
- Labour and reagents for the future contract water treatment plant.

Labour

Labour costs for the plant were provided by MintoEx as part of the 2011 budget. The labour costs include all cost of travel, overtime and shift premiums, leave pay, bonuses, pension and superannuation benefits, insurance coverage, educational assistance and supply of uniforms and personal protective equipment.

A labour allowance of $0.10 million per year for the future water treatment plant has been included in the operating cost for the plant upgrade as summarised in Table 24.13.

Table 24.13: Site Labour Cost Summary

Labour ($M/year)	Year 2011 (3,442 tpd)	Year 2012 - 2020 (3,750 tpd)
Mill Operations	3.45	3.45
Mill Maintenance	2.45	2.45
Mill Administration	0.03	0.03
Water Treatment Plant	0.10	0.10
TOTAL $M/y	6.02	6.02
TOTAL $/t	**4.79**	**4.40**

For the purposes of estimating overall operating labour costs for the plant was not adjusted for the upgrade, despite the removal of the tailings filtration plant.

Power

Power will continue to be supplied to the mine site from the local hydro power grid. The cost of power was based on the MintoEx 2011 budget unit power rate value of $0.11 /kWh.

The power requirements for the current plant were developed based on a specific power load of 30kWh/t for the first 6 months of 2011, while the tailings filtration circuit is in operation. From July 2011 to 2020 the power load is reduced to 26kWh/t. The reduction in power load considers the standby operation of the filtration circuit which is offset by the plant modifications to accommodate the increase in throughput.

The plant power consumption is expected to vary slightly over the life of the mine primarily due to the variable comminution characteristics of the ore and resulting change in comminution energy requirement. A summary of power costs by area for the plant site is given in Table 24.14.

Table 24.14: Process Plant Power Cost Summary

Power Costs ($/t)	Year 2011 (3,442 tpd)	Year 2012 - 2020 (3,750 tpd)
TOTAL $/t	$3.08	$2.86
TOTAL $M/y	$3.87	$3.91

Maintenance Consumables

Maintenance consumables were split into materials (consumables) and tools/miscellaneous maintenance costs for the purposes of estimating the operating cost. The maintenance labour costs were included in the overall plant labour costs as previously reported.

The cost of maintenance tools was based on the actual costs incurred by the plant for the first half of 2009. The maintenance tools/miscellaneous costs include grinding disks, welding rods, paint, tape, and etcetera.

The Minto cost centres assigned to maintenance tools/miscellaneous costs include those shown in Table 24.15.

Table 24.15: Process Plant Power Cost Summary

Maintenance Tools Cost Centres Used
316002-663125 Tools
316002-663910 Operating Supplies
316002-663915 Maintenance Supplies

The total fixed cost estimated for maintenance tools/miscellaneous from 2011 to 2020 was $0.61 million per year. The cost of the PLC servicing contract at $0.48 million per year was included in this estimate.

Maintenance material costs were estimated based on benchmarking the current Minto maintenance material costs against other plants of similar size.

The maintenance material costs include:

- Mechanical equipment replacement parts;
- Pipes and fittings;
- Electrical equipment and replacement parts; and
- Instrumentation equipment and replacement parts.

The maintenance material costs for Minto were higher than expected based on similar plants mainly due to:

- Excessive failure of the installed flotation mechanisms. These will be replaced in 2011 with a new supplier and replacement frequency and costs are expected to reduce;
- Original installed pumps had a high failure rate. Subsequently, various pumps have been upgraded and a standby tailings pump expensed under operating cost budgets; and
- Various pumps have been upgraded and standby tailings pumps installed under operating cost budgets.

Exclusions from these costs are:

- Crusher wear components, mill liners and lifters, and other components included in reagents and consumables;
- Maintenance labour costs (included in the labour cost); and
- Sustaining capital costs.

Reagents and Consumables

Reagent consumptions rates used for the estimate were based on the Minto 2011 budget. Exceptions to this were:

- Consumption of the flotation reagents AM28 and NaSH were averaged at 420g/t and 250g/t of oxide material from 2011 to 2020; and
- Consumption of flotation reagents Pax and MIBC were calculated based on flotation test work results.

Reagent consumptions will vary according to metallurgical and production parameters. Generally the consumption rates used are in line with the average consumptions from the current plant performance.

Reagent unit costs were based on the actual unit rates used in the 2011 budget. The rates are freight in store (FIS) at the Minto mine. The site is reviewing other reagents than AM28 and NaSH that are required to enhance recovery associated with oxide and partially oxide ores. Therefore, there is good potential to decrease regent expenditures. For the purposes of this study, a conservative approach was taken.

The average LOM consumptions and the unit costs are presented in Table 24.16.

Table 24.16: Reagent Consumptions and Unit Costs

Reagent	Unit $/kg	Consumption kg/t milled	2011 M$/y	2012 – 2020 M$/y
Flotation Collector (PAX)	2.77	0.02	54,680	59,262
Flotation Frother (MIBC)	3.87	0.05	240,141	256,813
Nitric Acid (kg)	1.19	0.05	67,276	73,297
AM28	12.0	0.42	285,466	285,466
NaSH	1.32	0.25	18,691	18,691
Diesel consumption (litres)	0.92		182,349	182,349
Tailings Flocculant (AE4270)	5.80	0.03	218,601	238,163
Filtration Flocculant (AE4270)	5.80	0.03	108,497	0
Concentrate Flocculant (AE4330)	5.61	0.001	5,709	6,220
TOTAL $M/y (x 1000)			**1,181**	**1,120**
TOTAL $/t			**0.94**	**0.82**

Plant Consumables

Plant consumables include major items, such as crusher and mill liners and grinding media. Consumption rates were estimated from the 2011 budget consumption rates as well as benchmarking. Consumption rates and unit costs are summarised in Table 24.17.

Table 24.17: Crusher and Mill Liner Consumption

Item	2011 Consumption	2012 - 2020 Consumption
SAG Mill Liners	SAG liners - 3 shell, 2 grates, half feed end and discharge liners	SAG liners - 3 shell, 2 grates, half feed end and discharge liners
Ball Mill Liners	2.0 sets per year	2.0 sets per year
Jaw Crusher Liners	2.0 sets per year	2.7 sets per year
Secondary Crusher Liners	-	4.0 sets per year
Regrind Mill Liners	-	1.0 set per year

The cost of secondary crusher liners for the current contract secondary crushing plant are included in the total cost ($/t) estimated for the contract crushing.

The SAG and ball mill liner consumption rate is a function of the power drawn by the respective mills and the ore hardness properties. The mills on site are currently operated at or near maximum power draws and therefore it is not expected that the liner wear rates will increase in the future when treating moderately abrasive ores. The annual liner costs are presented in Table 24.18.

Table 24.18: Crusher and Mill Liner Costs

Item	Current Plant Cost ($M)	Plant Upgrade ($M)
SAG Mill Liners	0.69	0.69
Ball Mill Liners	0.15	0.15
Jaw Crusher Liners	0.05	0.06
Secondary Crusher Liners	0.05	0.05
Regrind Mill Liners	0.01	0.08
TOTAL $M/y	**0.96**	**1.03**
TOTAL $/t	**0.76**	**0.75**

Details of the grinding media and consumption rates for the SAG, ball and regrind mills are detailed in Table 24.19. The SAG and ball mill media consumption rates were estimated using the consumption rates for the 2011 budget. The regrind mill media consumption rate was estimated by benchmarking.

Table 24.19: Grinding Media Details Usage and Pricing

Mill	Diameter	Type	Cost $/kg	Current Plant Consumption Rate (kg/t)	Plant Upgrade Consumption Rate (kg/t)
SAG Mill	125 mm	Forged	1.03	0.48	0.48
Ball Mill	75 mm	Forged	1.10	0.35	0.35
Ball Mill	50 mm	Forged	1.02	0.30	0.30
Regrind Mill	12 mm	Forged	1.25	0.08	0.08

Table 24.20 shows the annual grinding media costs and the cost per ton of ore processed. The calculation of the media consumption was based on the SAG and ball mill media consumption rate (kg/t) not changing from the current rate, considering the new orebodies will have similar abrasive properties to the Minto Main material.

Table 24.20: Grinding Media Costs

Item	Current Plant Cost ($M)	Plant Upgrade ($M)
SAG Mill Balls	0.62	0.68
Ball Mill Balls (75 mm)	0.48	0.53
Ball Mill Balls (50 mm)	0.39	0.42
Regrind Mill Media	0.06	0.13
TOTAL $M/y	**1.55**	**1.75**
TOTAL $/t	**1.24**	**1.28**

Tails Filtration and Dry Stacking

The tails filtration and stacking plant is due to operate until July 2011 at which point it will be placed on standby. From July 2011 onwards tailings will be thickened and pumped direct to the open cut pit.

The following costs were excluded from the tails filtration and dry stacking operation but included elsewhere:

- Reagents for the tails filtration area and diesel for dry stacking of the tails, The cost for these items are included in the reagent costs Section;
- Power for the tailings filtration plant. This cost is included under power costs; and
- MintoEx supplied labour. This cost is included under labour costs.

The following costs were included in this estimation:

- Contractor labour and equipment;
- Filter cloths, scrapers and operating consumables;
- Mechanical replacement parts; and
- Lubes and glycol.

Tails filtration and dry stacking were removed from the plant flowsheet for the upgrade scenario based on backfilling the Minto main pit with tailings directly from the tailings thickener underflow pumps.

Assay and Metallurgical Laboratory

The operating cost for the Assay and Metallurgical Laboratory was estimated based on the 2011 budget cost as supplied by MintoEx. A fixed cost of $1.42 million per year was used for the operating cost calculations for both plant throughput scenarios.

24.2 Capital Cost Estimate

The Phase V LOM capital cost estimate is shown in Table 24.21.

Table 24.21: Summary of Capital Costs for All Cases (excluding closure costs)

Area	Unit	Estimated Cost
OP mining equipment fleet	M$	**32.0**
UG equipment (fixed and mobile)	M$	**18.3**
UG development	M$	**15.8**
Process plant	M$	**5.0**
Contingency	M$	**3.1**
Sustaining Capital	M$	**1.8**
TOTAL CAPITAL COST	M$	**76.0**

Mine closure costs were estimated to be $16M.

Contingency capital is relatively low due to good quality recent mobile equipment and UG development expenditure estimates. Sustaining capital is relative low due to the short mine life that avoids major re-builds on mobile equipment and also replacement of various components in the process plant during the mill expansion project.

24.2.1 Underground Mine Capital Cost Estimate

All costs are expressed in Canadian dollars, with no allowance for escalation or interest during construction. The estimate was prepared at a pre-feasibility study level of accuracy of ±25%. No contingency was applied to cost estimate but instead, was applied in the economic model.

The underground mining capital cost estimate was based on the following:

- Underground mining equipment list;
- Budget quotes for the major equipment obtained by SRK from equipment manufacturers;
- Cost for used drilling and hauling equipment provided by Capstone;
- Budget quotes for raise development obtained by SRK from a contractor;
- In-house database;
- Western Mining estimation references; and
- Preliminary project development plan.

Mining capital was divided into equipment capital cost and mine development cost categories.

Equipment Capital Cost

The purchase of a permanent mining equipment fleet would be required as the underground mining operation will be performed by the owner.

It was assumed that for a mine life of 5 year, no replacement of the major equipment fleet would be needed as most of underground equipment has the same life; however, annual purchasing of spare parts in amount of 5% of equipment cost was assumed to provide equipment maintenance. An additional 4% of equipment cost was applied to cover expenses for delivering equipment to the site.

Table 24.22 shows estimated prices for UG capital.

Table 24.22: Underground Mining Equipment Unit Costs

Equipment	Quantity	Unit Cost (K$)
Drilling Equipment		
Jumbo (2 boom)	2	800
Rockbolter	2	737.9
Loading & Hauling Equipment		
LHD, 5.4 m^3 (10 t)	2	800
Haulage Truck, 40 t	4	600
Service Vehicles		
Grader	1	330
Explosive Truck	1	350
ANFO Loader	2	218.5
Cassette Carrier	2	272.7
Personnel Cassette	2	83.3
Boom Cassette	1	68.4
Fuel / Lube Cassette	1	116.7
Mechanics Truck	1	303.5
Scissor Lift	1	303.5
Supervisor/Engineering Vehicle	3	87.5
Electrician Vehicle - Scissor Lift	1	115.5
Shotcrete Sprayer	1	656.8
Transmixer	1	323.6
Forklift	1	100
Primary Ventilation Fan, Intake	1	200
Primary Ventilation Fan, Exhaust	1	50
Primary Ventilation Fan Accessories	2	50
Fan Foundation and Installation	2	25
Auxilary Ventilation Fan 75 kW	2	50
Auxilary Ventilation Fan 50 kW	2	30
Auxilary Ventilation Fan 40 kW	2	20
Ventilation Doors and Regulators		50
Compressed Air		
Portable Compressor 500 cfm @ 100 psi	2	70.5
Compressed air system 1500 cfm elec.	1	200
Mine Water Management		
Main Dewatering Pump	2	35
Submersible Pump	3	8.2
Brine System	1	184
Mine Electrical		
Power Line	1	250
750kVA Portable Substation	4	115
5kV Distribution GND Check Panel	5	30

Equipment	Quantity	Unit Cost (K$)
5kV Junction Box	8	8
5kV Mine Power Feeder (PPC) 2,000 m	1	50
2kV Portable Power Cable 900 m	1	19.8
Jumbo GND Check Panel	4	5
Surface 5kV S&C	5	15
Miscellaneous (15% of total)	15%	150.6
Communication System		
Leaky Feeder Communication System	1	150
U/G Telephone	3	1
Mine Safety		
Portable Refuge Station (16 person)	2	79.4
Gas Monitoring System	1	10
Mine Rescue Equipment	1	50
First Aid Equipment	1	20
Cap Lamps (inc. radio)	114	1.8
Cap Lamp Charger (5 units each)	24	1.1
Charger Rack (15 units each)	8	0.64
Personal Protective Equipment	114	0.4
Fire Extinguishers (10 lb)	40	0.14
Sanitary Unit	4	5
Sanitary Pumping Tank System	1	5
Stench Gas System	2	22
Foam Generator	1	25
Mine Engineering Equipment		
Survey Equipment		75
PC, Printers, Network, Softawre		50
Mine Design Software		50
Geology Department Software		50
Miscellaneous		
Mining Tools	1	30
Jackleg	8	5.3
Stoper	8	4.5
Surface Repair Shop	1	200
Explosives Storage		
Underground Powder Magazines	1	25
Underground Cap Magazines	1	20
ANFO Kettle	1	10

Underground mining equipment capital cost by year is summarized in Table 24.23.

Table 24.23: Underground Mining Equipment Capital Cost Summary

Item	2011 (M$)	2012 (M$)	2013 (M$)	2014 (M$)	2015 (M$)	Total (M$)
Drilling Equipment	1.5	1.5				3.0
Loading and Hauling Equipment	1.4	2.0	0.6			4.0
Service Vehicles	3.2	1.2				4.4
Ventilation	0.4	0.1	0.1			0.6
Compressed Air	0.3	0.1				0.3
Mine Water Management	0.2	0.1				0.3
Underground Electrical	1.1	0.1				0.7
Communication	0.2	0.0				0.2
Safety	0.4	0.3				0.6
Underground Engineering Equipment	0.2	0.0	0.0	0.0	0.0	0.3
Underground Miscellaneous	0.3	0.0	0.0	0.0	0.0	0.4
Underground Explosives Storage		0.1				0.1
Spare Parts (5%)	0.4	0.3	0.5	0.4	0.4	2.0
Freight (4%)	0.4	0.2	0.1	0.0	0.0	0.7
Total Underground Mining Equipment Capital Cost	**10.09**	**6.0**	**1.3**	**0.5**	**0.5**	**17.6**

Development Capital Cost

All development in year 2011, including ore development, is included in capital costs as that period is considered as per-production. During mine production, the decline development, raise development, ventilation drifts and underground infrastructure were considered as capital costs, but stope crosscuts development was included in the mining operating cost.

It was assumed that all jumbo development would be done by owner and raise development by contractor. The estimated owner costs per metre of development excluding labour were applied for all jumbo development headings and the contractor rates were used for raise development. In year 2011, hourly and salaried labour, mine sevices and ore haulage costs were included in the capital cost.

Underground mine development capital costs is shown in Table 24.24.

Table 24.24: Underground Development Capital Cost

Development	Unit	Unit Cost (K$)	2011 Qty (m)	2011 Cost (M$)	2012 Qty (m)	2012 Cost (M$)	2013 Qty (m)	2013 Cost (M$)
Jumbo Crew Mobilization	ea	80.0	1	0.08				
Setup Services	ea	50.0	1	0.05				
Raise Crew Mobilization	ea	40.0	1	0.04				
Setup Services	ea	20.0	1	0.02				
Road to Portal	m	0.05	700	0.04				
Laydown Area	Km2	0.01	6,500	0.05				
OB Stripping	Kbc m	0.01	10,000	0.07				
Decline Portal	m	10.0	25	0.25				
Main Access Decline	m	1.8	950	1.71	1,510	2.71		
Remuck Bays	m	1.4	90	0.13	150	0.22		
Ventilation Drift	m	1.8	15	0.03	265	0.48		
Crosscut	m	1.8	85	0.15	75	0.13		
Ore Development	m	1.0	180	0.18				
Alimak Nest	m	2.2	15	0.03	15	0.03		
Vent Raise Collar	ea	50.0	1	0.05	1	0.05		
Alimak Setup	ea	91.0	1	0.09	1	0.09		
Vent Raise	m	4.2	172	0.71	273	1.13	188	0.78
Strip Rails	ea	20.0	1	0.02			1	0.02
Contractor Indirects	ea	50.0	1	0.05	1	0.05	1	0.05
Main Sump	ea	200.0			1	0.20		
Powder Magazine	ea	150.0	1	0.15				
Cap Magazine	ea	25.0	1	0.03				
Underground Miscellaneous		100.0		0.10		0.10		
Mechanical Shop	ea	200.0	1	0.20				
Mine Dry	ea	150.0	1	0.15				
Surface Miscellaneous		200.0		0.20				
Hourly Labour				3,04				
Salaried Personnel				1.10				
Ore Haulage				0.03				
Mine Services				0.96				
Site Tear-down	ea	20.0					1	0.02
Demobilization	ea	35.0					1	0.04
Total Development Capital Cost				**9.71**		**5.19**		**0.91**

24.2.2 Open Pit Mine Capital Cost Estimate

The capital cost estimate for the open pit operation is based on the ability of producing 1.4 Mtpa of ore. A transition from the current contract mining to an owner-operated fleet forms the basis of the estimate.

The major open pit equipment capital costs (based on new equipment) required to achieve the target processing rate of 1.4 Mtpa is summarized in Table 24.25.

Table 24.25: Open Pit Equipment Capital Cost Summary

Item	Unit	Total #	Total
Primary			
Crawler-Mounted, Rotary Tri-Cone, 9.875-in Dia.	M$	1	1.3
Crawler-Mounted, Rotary Tri-Cone, 6.5-in Dia.	M$	1	1.0
Crawler-Mounted, Rotary Tri-Cone, 4.5-in Dia.	M$	1	0.7
Diesel, 13-cu-yd Front Shovel	M$	1	2.9
Diesel 14-cu-yd Wheel Loader	M$	1	1.7
100-ton class Haul Truck	M$	6	9.9
D9-class 15.8' blade	M$	0	0.0
D9-class 15.8' blade	M$	3	2.3
824H-class 13.8' blade	M$	1	0.7
16H-class 16' blade	M$	2	1.5
14H-class 14' blade	M$	0	0.0
HD325-7R(40ton) 35m3 9000 gallon	M$	1	0.6
Subtotal Primary	M$		22.7
Ancillary			
ANFO/Slurry Truck, 12-ton	M$	1	0.2
Stemming truck, 15-ton	M$	1	0.1
Powder Truck, 1-ton	M$	1	0.0
AN Storage Bin, 60-ton	M$	1	0.1
Powder magazine, 24-ton	M$	1	0.1
Cap magazine, 3.6-ton	M$	1	0.0
Excavator (backhoe), 4 cu-yd	M$	1	0.5
Haul Truck (road constr), 35-ton	M$	3	1.7
Backhoe/Loader, 1.4 cu-yd	M$	1	0.2
Portable Aggregate Plant,30 tph	M$	1	0.3
All-terrain Crane, 60-ton	M$	1	0.5
Transporter w/Tractor, 100-ton	M$	1	0.4
Fuel truck, 5000-gal	M$	1	0.3
Lube/Service Truck	M$	1	0.3
Mechanic Field Service Truck	M$	3	0.1
Off-Road tire handling Truck	M$	1	0.4
Wheel Loader 8.5-cu-yd	M$	1	0.9
16 cu-yd Scraper	M$	1	0.6
Light Plant, 6-kW	M$	5	0.1
Pickup Truck, 0.75-ton, 4-WD	M$	10	0.3
Crew Van, 1-ton, 4-WD	M$	4	0.2
Mobile Radio, installed	M$	50	0.0
Subtotal Ancillary	M$		7.4
Miscellaneous			
Shop Equipment	M$	1	0.8
Eng & Office Equip plus Software	M$	1	0.5
Truck Dispatch System	M$	0	0.0
Radio Communications System + GPS	M$	1	0.5
Electrical Distribution System w/substation	M$	0	0.0
Subtotal Miscellaneous	M$		1.9
Total Equipment & Misc.			32.0
Spares Inventory @ 5%	M$		1.6
Contingency @ 5%	M$		1.6
Salvage @10%	M$		-3.2
TOTAL MINE CAPITAL, Pre-Tax	M$		32.0

24.2.3 Process Plant Capital Cost Estimation

General

The plant capital cost estimate is presented in Canadian dollars (C$) and has an overall accuracy of ±25% as of the fourth quarter 2010. Following is a breakdown of capital works planned for the Minto Mine.

The scope of work planned for 2011 includes:

- Installation of a new crushed ore scalping screen with belt feeder;
- Upgrade scavenger cell mechanisms and launders, and replacement of the existing 2nd cleaner cells with new tankcell-10 cells;
- Rerouting of the tailings thickener underflow pipework to bypass the tailings filter circuit; and
- A new 2,600 metre long high density poly-ethylene tailings pipeline from the tailings thickener to the Minto Main pit.

The estimate scope of work planned for 2012 includes:

- Installation of a new regrind circuit including regrind tower mill, new cleaner flotation cells and associated secondary equipment (pumps etc.)

The estimate for upgrading the primary crushing area, flotation circuit, rerouting the tailings thickener pipework, and installation of the regrind mill, 1st cleaners and modification to the existing coverall building were supplied by B. Ross Design Inc. ("BRDI"). The estimate for this scope has not been verified by Ausenco and has been accepted as received.

The estimate for the 2,600 meter tailings line was supplied by Ausenco and sourced from the Phase IV PFS estimate.

Table 24.26 shows a summary of the costs which exclude any escalation or foreign currency fluctuations and are current day costs only.

Table 24.26: Cost Breakdown Summary – Process Plant

Facility	2011	2012
Primary Crushing Area	0.531	
Flotation Circuit Area	1.136	2.13
Tailings and Dewatering Area	0.337	
Spares and first fills		0.219
EPCM	0.301	0.352
Total	**2.305**	**2.701**

Scope of Estimate

The capital cost estimate includes the following:

- Mechanical equipment costs for new process plant equipment that has not been purchased by MintoEx;
- Tailings delivery pipeline and deposition spigot's;
- Freight allowance;
- EPCM costs;
- Excluded from the capital cost estimate:
- Mining, mining vehicles/equipment, mine infrastructure;
- New equipment required for the Phase IV plant upgrade already purchased by MintoEx;
- Owner's costs for the project;
- Tailings storage facility construction;
- Tailings dam (Minto Main pit) water decant and return system;
- Owner's contingency, owner's costs, escalation and foreign currency fluctuation;
- Licenses and permits; and
- On-going, future or deferred capital costs.

Detailed Cost Estimate Build-up

The estimated capital cost for the Minto plant upgrade and non-mining infrastructure was based on the following:

- The regrind tower mill was purchased by MintoEX so the mechanical equipment cost has not been included and will not form part of the project budget;
- Major mechanical equipment budget costs were sourced directly from suppliers. This includes the crushed ore scalping screen and feeder, and flotation cell mechanisms and new cleaner cells; and
- The equipment was selected on an area by area basis to suit the respective process requirements of each area.
- Concrete was estimated from preliminary material takeoffs generated from the general arrangement drawings.
- Structural steel was estimated from preliminary material takeoffs generated from the general arrangement drawings.
- Piping, electrical & instrumentation, plate work and freight were included as factors based on the overall supply and installation cost of the mechanical equipment.

- Cost for the new tailings pipeline to the Minto Main pit was estimated based on an overall length of 2,600 metres of high density poly-ethylene pipe. The deposition costs were estimated based on the use of spigots.
- EPCM costs were included in the estimate. The EPCM cost is 15% of the capital cost under management.
- The factors used for each area and discipline were established from similar and detailed past estimates for plants similar to Minto.
- Rates used for each area and discipline were established from similar and detailed past estimates for plants similar to Minto.
- A temporary construction camp was not factored into the estimate. This assumes that the existing Minto camp accommodation, messing, water supply and sewage treatment facilities are able to meet the increased demand during the construction period.
- An allowance for capital spares was included and calculated based on a percentage of process plant cost established from previous projects.
- An allowance was made for first fills and initial consumables which have been calculated based on a percentage of process plant cost established from previous projects.
- No allowance was made for temporary construction facilities. Existing facilities at Minto will be used during the construction period.
- No allowance for commissioning was made in the EPCM. Minto operating personnel would start-up the additional equipment.

Assumptions

Geotechnical

A detailed geotechnical and drainage assessment of the proposed site is not yet available. For the purpose of the study no special ground preparation has been considered.

Base Date and Exchange Rates

The base date of the cost estimate is 11th of November 2010.

The estimate is expressed in Canadian Dollars.

Electricity Supply

It is assumed that hydro-power is available to satisfy the increased demand for any new or upgraded plant equipment.

Water Supply

A water supply capable of supplying the required demand of the processing plant is assumed to be available. For this reason, costs associated with any increase in water supply have not been included within this estimate.

Contingency

Contingency is not included in the PFS capital cost estimate. Contingency is controlled by the client and as such the amount of contingency to be included in the project is ultimately the client's decision.

Owner's Costs

Owner's costs have been excluded from this estimate.

Project Fee

No project fee was included.

Escalation

Escalation provision past the fourth quarter 2010 was not included in the estimate.

25 Economic Analyses

25.1 Assumptions

An engineering economic model was compiled by SRK based on the Phase V LOM plan. Three cases were used to show the variability of metal prices and C$:US$ exchange rate:

- Case A assumes a constant copper price of US$2.75/lb and a constant exchange rate of C$1.09:US$1.00 US$2.75/lb Cu approximates the average copper price over the LOM to date for Minto. Copper prices are currently greater than US$4.00/lb and are projected by most financial institutions to decline to US$2.25/lb over time and, as a result, Case B may underestimates revenues in the earlier years and may overestimate revenues in later years, if predictions hold true.
- Case B assumes a constant copper price of US$2.25/lb, equivalent to the long-term price forecast by many financial institutions for 2016 and beyond. As with Case B, copper price is currently well above the assumed price and the mine is currently in production, therefore is able to benefit from current high prices. These factors make Case B very conservative. Case B uses comparable assumptions to the Phase IV PFS and prior studies and is included mainly for benchmarking purposes. Case B assumes a constant exchange rate of C$1.16:US$1.00.
- Case C estimates economic results at varied metal prices and exchange rates that approximates forward projections from a collection of financial institutions. Case C uses a declining value of unhedged copper ranging from US$3.60/lb in 2011 to US$2,25/lb in 2016 and beyond. The prices selected for Case C were chosen in late 2010 and, since then, copper prices have continued to rise giving Case C a degree of conservatism.

The detailed annual metal price and exchange rate assumptions are in Table 25.1.

It should be noted that payable gold, up to 30,000 oz/year, and all payable silver in the LOM plan are under contract to Silver Wheaton (“SLW”) at a price of US$300/oz and US$3.90/oz respectively.

All three economic model cases use identical mineral reserve estimates that were based on mine optimization and cut-off grades using US$2.25/lb Cu, which are similar to these used in the Phase IV PFS, thereby providing a useful comparison). On-site costs, off-site costs (except royalties) and production parameters were also held constant for each case. Royalty costs vary with metal price as per MintoEx’s agreement with the SFN.

Other main economic factors common to all three economic cases are:

- A discount rate of 7.5%;
- Closure allowance of $16M;
- Nominal 2011 dollars;
- Payment of capital from operating revenue;

- No inflation; and
- Costs, revenues and taxes calculated for each period in which they occurred rather than at the actual date of payment.

It must be noted that spot copper prices on the effective date of this report were over US$4.00/lb, significantly higher than the assumptions used in this report.

Table 25.1: Metal Pricing and Exchange Rates Assumptions by Case

Metal	Units	Case A	Case B (Reserve Case)	Case C									
		Constant Prices	Constant Prices	Variable Metal Prices and Exchange Rates									
				2011	2012	2013	2014	2015	2016	2017	2018	2019	2020
Hedged Copper*	US$/lb	2.26	2.26	2.26									
Unhedged Copper	US$/lb	2.75	2.25***	3.60	3.30	3.00	2.70	2.40	2.25	2.25	2.25	2.25	2.25
SLW Contract Gold**	US$/oz	300	300	300	300	300	300	300	300	300	300	300	300
Uncommitted Gold	US$/oz	1,000	850	1,250	1,200	1,100	1,050	1,000	950	950	950	950	950
SLW Contract Silver	US$/oz	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90
Exchange rate	C$/US$	1.16	1.09	1.03	1.05	1.06	1.08	1.09	1.09	1.09	1.09	1.09	1.09

**Hedged copper only applies to 8,312 tonnes of Cu in 2011.*

***SLW contract gold only applies for the first 30,000 oz/annum. Production beyond 30,000 oz/annum is split equally between SLW and uncommitted gold*

****US$2.65/lb in 2011 only*

Economic Results

The LOM economic model results common to all three cases are shown in Table 25.4. Economic model results specific to each case are shown in Table 25.5.

It must be noted that the net present value ("NPV") calculations in the economic models were done using 2011 as the starting year and do not take into account approximately $150M in capital spent for initial plant and mine construction (sunk costs) nor the revenue derived from operations from 2007-2010. As a result, the economic analyses show very high returns on the planned, future capital investments.

Results show the NPV at a 7.5% discount rate ("$NPV_{7.5\%}$") to be $284M before tax. Table 25.2 shows the NPV results at various discount rates for each case.

Case B clearly produces a very robust case for the support of mineral reserve estimates.

Table 25.2: Discount Factors and Related Net Present Values for All Cases

Case	Discount Rate	Pre-tax NPV (M$)	Post-tax NPV (M$)
Case A	0%	369	262
	5%	309	223
	7.5%	284	206
	10%	262	191
Case B (Mineral Reserve Estimation Case)	0%	234	180
	5%	196	153
	7.5%	180	142
	10%	166	131
Case C	0%	323	231
	5%	283	205
	7.5%	266	194
	10%	250	183

Comparison of Phase IV PFS and Phase V PFS Economic Results

An incremental analysis was done to compare the economic results of the Phase IV PFS (SRK 2009) and the Phase V PFS. The Phase IV NPV results were subtracted from the Phase V NPV results and are shown in Table 25. The comparison shows that the Phase V Case A NPVs are higher than the Phase IV Case 1 NPVs by $58M for the undiscounted pre-tax case and $34M for the discounted pre-tax case.

The two studies used many different assumptions and, as a result, the incremental analysis only provides a cursory comparison. Some of the main assumption differences from the Phase IV PFS and the Phase V PFS are:

- $24M in capital that was to be spent in 2010 in the Phase IV PFS was transferred to 2011 in the Phase IV Net Cash Flow in Table 25.3

- Delayed mining of the high grade Minto North deposit in the Phase V PFS due to a revised permitting schedule;
- Variances in metal prices and exchange rates;
- Variances in OPEX and CAPEX assumptions;
- Delayed plant production capacity ramp-up in the Phase V study; and
- Inclusion of debt repayment in the Phase V study ($27M).

Table 25.3: Pre-Tax Incremental Economic Results

Net Present Value	Unit	Total	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020
Undiscounted	**M$**	**58**	**(3.6)**	**(23.0)**	**(75.2)**	**116.5**	**4.8**	**34.6**	**(35.9)**	**49.2**	**23.1**	**(32.5)**
Discounted @7.5%	M$	**34**	(4)	(21)	(65)	94	4	24	(23)	30	13	(17)

Table 25.4: Summary of Economic Model Results for All Cases

			2011-2020	Y E A R									
SECTION	**ITEM**	**UNIT**	**Total**	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**
Open Pit Mining	OVB and Waste	Mt	**58.18**	8.94	10.37	8.05	5.71	7.42	7.53	7.82	2.34	-	-
	Ore	Mt	**8.80**	0.81	1.34	0.28	1.80	1.22	1.51	1.01	0.82	-	-
UG Mining	Waste	Mt	**0.34**	0.09	0.18	0.02	0.00	0.04					
	Ore	Mt	**2.44**	0.01	0.63	0.73	0.73	0.34					
Total Mining	Ore	Mt	**11.2**	0.83	1.77	2.26	1.48	1.55	1.51	1.01	0.82	-	-
Mill Feed	Mill Feed	dmt/day	**3,718**	3,442	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,751
	Mill Feed	Mt	**12.9**	1.256	1.373	1.369	1.369	1.369	1.373	1.369	1.369	1.369	0.653
	Head grade	% Cu	**1.53**	1.60	1.86	1.70	2.86	1.62	1.68	1.11	0.96	0.78	0.78
	Head grade	g/t Au	**0.6**	0.6	0.7	0.7	1.5	0.6	0.7	0.3	0.3	0.2	0.2
	Head grade	g/t Ag	**5.2**	6.0	7.2	5.4	10.0	5.8	5.7	2.9	3.2	2.2	2.2
Recovery to Conc.	Cu recovery	% Cu	**92.0**	92	92	92	92	92	92	92	92	92	92
	Au recovery	% Au	**70.0**	70	70	70	70	70	70	70	70	70	70
	Ag recovery	% Ag	**78.0**	78	78	78	78	78	78	78	78	78	78
Conc. Grade	Cu grade of concentrate	% Cu	**39**	41.5	38	39	38	38	38	38.6	38.6	38.7	38.7
Conc. Production	Cu Conc. Produced - Dry	dmt	**470,478**	44,633	61,728	54,846	94,937	53,519	55,956	36,066	31,201	25,449	12,134
Conc. Metal	Cu in Concentrate	Mlb Cu	**400.4**	40.8	51.7	47.2	79.5	44.8	46.9	30.7	26.6	21.7	10.4
	Au in concentrate	oz Au	**173,146**	16,807	22,531	22,259	45,118	19,488	20,538	9,375	8,246	5,948	2,836
	Ag in concentrate	oz Ag	**1,673,940**	188,612	246,680	184,654	341,791	197,647	194,509	100,627	110,290	73,897	35,234
Payable Metal	Payable Copper	Mlb	**387**	39.5	50.0	45.7	76.9	43.4	45.4	29.7	25.7	21.0	10.0
	Gold Payable	oz	**167,435**	16,303	21,855	21,592	43,764	18,903	19,921	8,906	7,834	5,651	2,694
	Silver Payable	oz	**1,220,153**	145,563	187,142	131,753	250,223	146,027	140,538	65,841	80,196	49,350	23,530
Off-site Costs	Unit conc. transport cost	US$/wmt	**154.84**	155.46	152.29	153.37	149.45	153.56	153.15	158.17	160.38	164.08	184.19
	Treatment Charge Cu Conc.	US$/dmt	**30.01**	5.00	10.00	20.00	40.00	40.00	40.00	40.00	40.00	40.00	40.00
	Refining charge Cu	US$/payable lb	**0.030**	0.005	0.010	0.020	0.040	0.040	0.040	0.040	0.040	0.040	0.040
	Refining charge Au	US$/payable oz	**5.00**	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
	Refining charge Ag	US$/payable oz	**0.40**	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40
	Unit Offsite costs (ex royalty)	**US$/lb Cu**	**0.27**	**0.20**	**0.23**	**0.25**	**0.29**	**0.30**	**0.30**	**0.30**	**0.30**	**0.31**	**0.33**
OPEX	Open Pit Mining	$/OP t mined	**2.57**	3.08	2.77	2.30	2.63	2.56	2.21	2.25	2.71		
		$/t milled	**13.37**	23.89	23.61	14.03	14.45	16.14	14.59	14.55	6.26	-	-
	UG Mining	$/UG ore t mined	**35.17**	-	36.85	34.98	34.19	35.91					
	Combined OP and UG mining	$/t milled	**20.04**	23.89	40.53	32.69	32.69	25.00	14.59	14.55	6.26	0	0
	Milling cost	$/t milled	**12.94**	19.49	12.23	12.23	12.23	12.23	12.23	12.23	12.23	12.23	12.23
	G&A	$/t milled	**12.13**	14.24	11.90	11.90	11.90	11.90	11.90	11.90	11.90	11.90	11.90
	Total Unit OPEX	**$/t milled**	**45.11**	**57.62**	**64.66**	**56.82**	**56.82**	**49.13**	**38.72**	**38.68**	**30.39**	**24.13**	**24.13**
	Cu unit cost (without Au & Ag credits)	$/lb Cu	**1.50**	1.83	1.77	1.70	1.01	1.55	1.17	1.78	1.62	1.57	1.57
	Case A Cu unit cost (with Au & Ag credits)	$/lb Cu	**1.33**	1.68	1.62	1.54	0.74	1.39	1.02	1.68	1.50	1.47	1.47
LEASE, INTEREST	**Total**	M$	**26.4**	**1.8**	**1.7**	**4.4**	**4.4**	**4.4**	**4.4**	**4.1**	**0.3**	**0.3**	**0.3**
CAPEX	**Total (inc. Closure)**	**M$**	**92.0**	**23.8**	**46.7**	**2.6**	**2.6**	**2.9**	**0.3**	**0.3**	**- 3.2**	**-**	**16.0**
Debt Repayment	**Debenture and bank debt principal**	**M$**	**31.4**	**0**	**5**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**27**

Table 25.5: Economic Results by Case

				YEAR									
SECTION	**ITEM**	**UNIT**	**TOTAL**	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**
CASE A													
Metal Price	Wt. Ave. Cu Price	US$/lb	**2.73**	2.52	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75
	Wt. Ave. Au Price	US$/oz	**331**	300	300	300	417	300	300	300	300	300	300
	Wt. Ave. Ag Price	US$/oz	**3.90**	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90
Net Revenue from Smelter	Cu Revenue	M$	**1,034.3**	**99.7**	**137.3**	**124.5**	**205.9**	**115.8**	**121.1**	**79.2**	**68.5**	**55.9**	**26.4**
	Au Revenue	M$	**59.3**	**5.2**	**7.0**	**6.9**	**19.6**	**6.1**	**6.4**	**2.9**	**2.5**	**1.8**	**0.9**
	Ag Revenue	M$	**4.6**	**0.6**	**0.7**	**0.5**	**1.0**	**0.6**	**0.5**	**0.3**	**0.3**	**0.2**	**0.1**
	Total Revenue	M$	**1,098.2**	**105.4**	**145.0**	**131.9**	**226.4**	**122.4**	**128.0**	**82.3**	**71.3**	**57.9**	**27.3**
Exchange Rate	Exchange rate	C$:US$	**1.09**	1.09	1.09	1.09	1.09	1.09	1.09	1.09	1.09	1.09	1.09
Royalties	Royalties	M$	**13.7**	1.3	1.8	1.6	2.8	1.5	1.6	1.0	0.9	0.7	0.3
NPV	**Pre-tax, undiscounted**	**M$**	**369**	**6**	**2**	**49**	**142**	**50**	**72**	**27**	**31**	**23**	**(33)**
	After tax, undiscounted	**M$**	**284**	**7**	**2**	**42**	**114**	**37**	**50**	**17**	**19**	**13**	**(17)**
CASE B													
Metal Price	Wt. Ave. Cu Price	US$/lb	**2.25**	2.25	2.25	2.25	2.25	2.25	2.25	2.25	2.25	2.25	2.25
	Wt. Ave. Au Price	US$/oz	**324**	300	300	300	392	300	300	300	300	300	300
	Wt. Ave. Ag Price	US$/oz	**3.90**	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90
Net Revenue from Smelter	Cu Revenue	M$	**894.5**	**94.6**	**118.1**	**106.9**	**176.0**	**99.0**	**103.5**	**67.7**	**58.5**	**47.8**	**22.5**
	Au Revenue	M$	**62.1**	**5.6**	**7.5**	**7.4**	**19.7**	**6.5**	**6.8**	**3.1**	**2.7**	**1.9**	**0.9**
	Ag Revenue	M$	**5.0**	**0.6**	**0.8**	**0.5**	**1.0**	**0.6**	**0.6**	**0.3**	**0.3**	**0.2**	**0.1**
	Total Revenue	M$	**961.6**	**100.7**	**126.4**	**114.9**	**196.8**	**106.0**	**110.9**	**71.0**	**61.5**	**49.9**	**23.5**
Exchange rate	Exchange rate	C$:US$	**1.16**	1.16	1.16	1.16	1.16	1.16	1.16	1.16	1.16	1.16	1.16
Royalties	Royalties	M$	**9.6**	1.0	1.3	1.1	2.0	1.1	1.1	0.7	0.6	0.5	0.2
NPV	**Pre-tax, undiscounted**	**M$**	**234**	**2**	**(18)**	**32**	**113**	**34**	**55**	**16**	**22**	**15**	**(36)**
	After tax, undiscounted	**M$**	**180**	**2**	**(17)**	**28**	**91**	**25**	**38**	**10**	**13**	**9**	**(19)**
CASE C													
Metal Price	Wt. Ave. Cu Price	US$/lb	**2.65**	2.98	3.30	3.00	2.70	2.40	2.25	2.25	2.25	2.25	2.25
	Wt. Ave. Au Price	US$/oz	**333**	300	300	300	426	300	300	300	300	300	300
	Wt. Ave. Ag Price	US$/oz	**3.90**	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90	3.90
Net Revenue from Smelter	Cu Revenue	M$	**986.8**	**112.9**	**161.8**	**133.9**	**199.5**	**99.3**	**96.5**	**63.1**	**54.6**	**44.5**	**20.9**
	Au Revenue	M$	**58.8**	**5.0**	**6.8**	**6.8**	**19.8**	**6.1**	**6.4**	**2.9**	**2.5**	**1.8**	**0.9**
	Ag Revenue	M$	**4.6**	**0.5**	**0.7**	**0.5**	**0.9**	**0.6**	**0.5**	**0.3**	**0.3**	**0.2**	**0.1**
	Total Revenue	M$	**1,050.2**	**118.4**	**169.2**	**141.2**	**220.2**	**105.9**	**103.4**	**66.2**	**57.4**	**46.5**	**21.9**
Exchange rate	Exchange rate	C$:US$	**1.07**	1.03	1.05	1.06	1.08	1.09	1.09	1.09	1.09	1.09	1.09
Royalties	Royalties	M$	**12.8**	1.8	2.5	1.8	2.8	1.1	1.0	0.7	0.6	0.5	0.2
NPV	**Pre-tax, undiscounted**	**M$**	**323**	**19**	**26**	**58**	**136**	**34**	**48**	**11**	**17**	**12**	**(38)**
	After tax, undiscounted	**M$**	**266**	**19**	**25**	**50**	**109**	**25**	**33**	**7**	**10**	**7**	**(20)**

Sensitivities

The project was evaluated for sensitivity to the operating costs, capital costs, grade and metal price. All sensitivities were assessed for a range of -20% to +20% with the resulting pre-tax ("PT") $NPV_{7.5\%}$ value shown. Figures 25.1 to 25.3 graphically depict results of the sensitivity analyses.

All sensitivities were done as mutually exclusive variations. A combination of variable changes was not conducted nor was an analysis of the probability of any variations.

The economic models show the project is most sensitive to variances in Cu grade. This sensitivity is somewhat mitigated in the mine plan by the significant use of stockpiles to allow the early processing of higher grade ore and the ability to blend different grades to provide a consistent mill feed. These two features of the LOM plan are important in maximizing the economics of the project. In Case A, a 20% drop in Cu grade yields a $175 M (-62%) decrease in PT-$NVP_{7.5\%}$. Diligent grade control practices will be important in achieving mill feed with minimal dilution, especially in Area 2 where the mineralized zones are smaller and more numerous than are found elsewhere.

Metal prices demonstrate an almost identical sensitivity to grade. In Minto's case, the metal prices are buffered fractionally some hedged copper production in 2011. Gold and silver prices are fixed as per the Silver Wheaton contract. A 20% decrease or increase in Cu price changes the Case A PT-$NPV_{7.5\%}$ by approximately 59% or $168M.

Exchange rate is another factor in which the project is sensitive. In Case A, changing the C$:US$ exchange rate from 1.09 to parity decreases the PT- $NPV_{7.5\%}$ by approximately 23% or $67M.

A 20% reduction in OPEX in Case A yields a $92 M (32%) increase in PT-$NPV_{7.5\%}$. Conversely, a 20% increase in OPEX in Case A yields a $92 M (32%) decrease in PT-$NPV_{7.5\%}$. Some of Minto's operating expenses including TCs, RCs, concentrate transport and short-term open pit mining are covered by contracts and, therefore, offer some protection from variances in the next several years. The mining OPEX used in this report is based on an owner-operated fleet starting in late 2012 and presents a significant change from the current contract mining scenario, both operationally and in terms of predicted costs. If the owner-operator fleet cannot achieve the costs that are estimated there will be a substantial change in NPV.

As a relatively large percentage of the capital expenses have already been incurred in the project, the CAPEX sensitivity is reduced. A 20% increase in CAPEX in Case A represents a $16M (6%) reduction in PT-$NPV_{7.5\%}$.

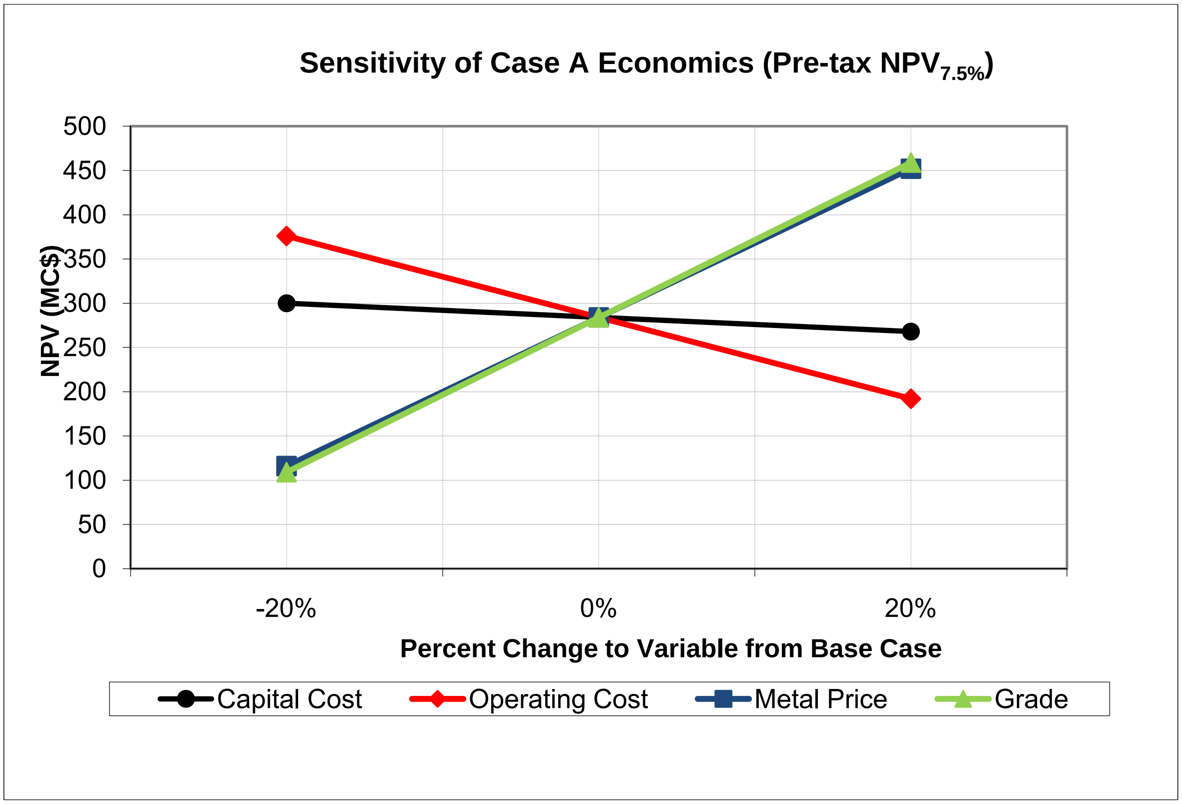


Figure 25.1: Case A Sensitivities

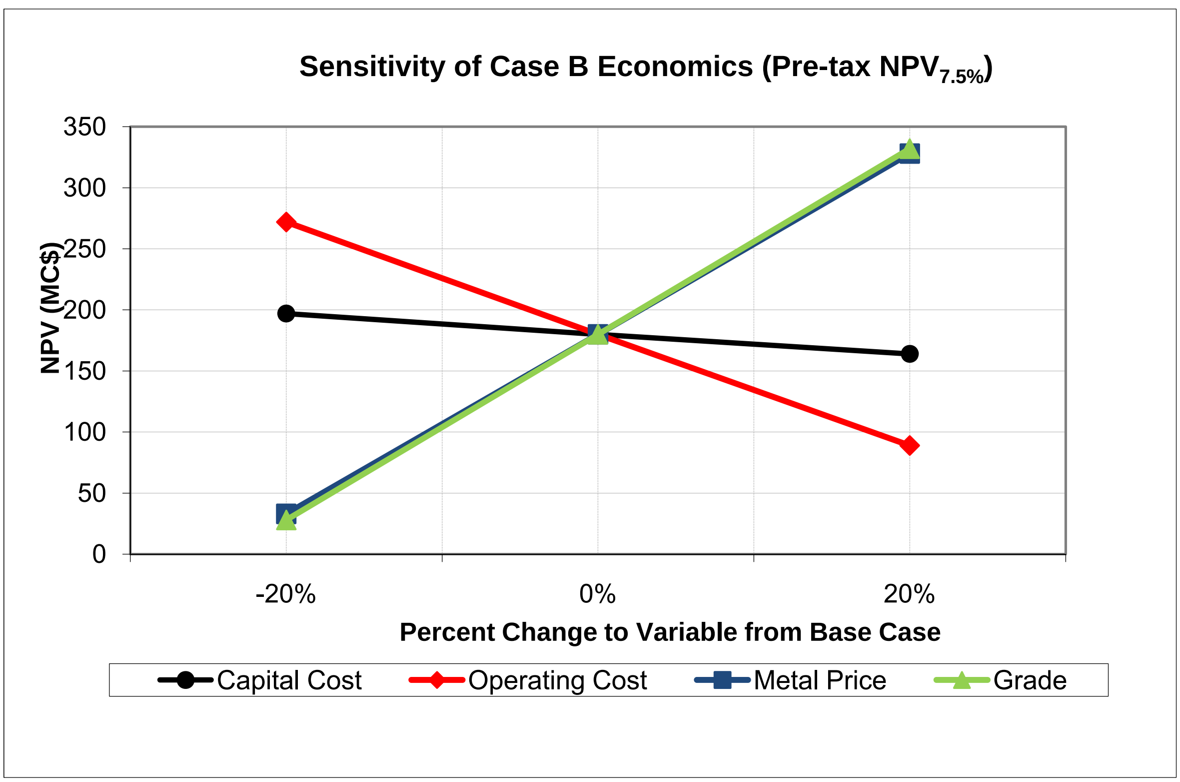


Figure 25.2: Case B Sensitivities

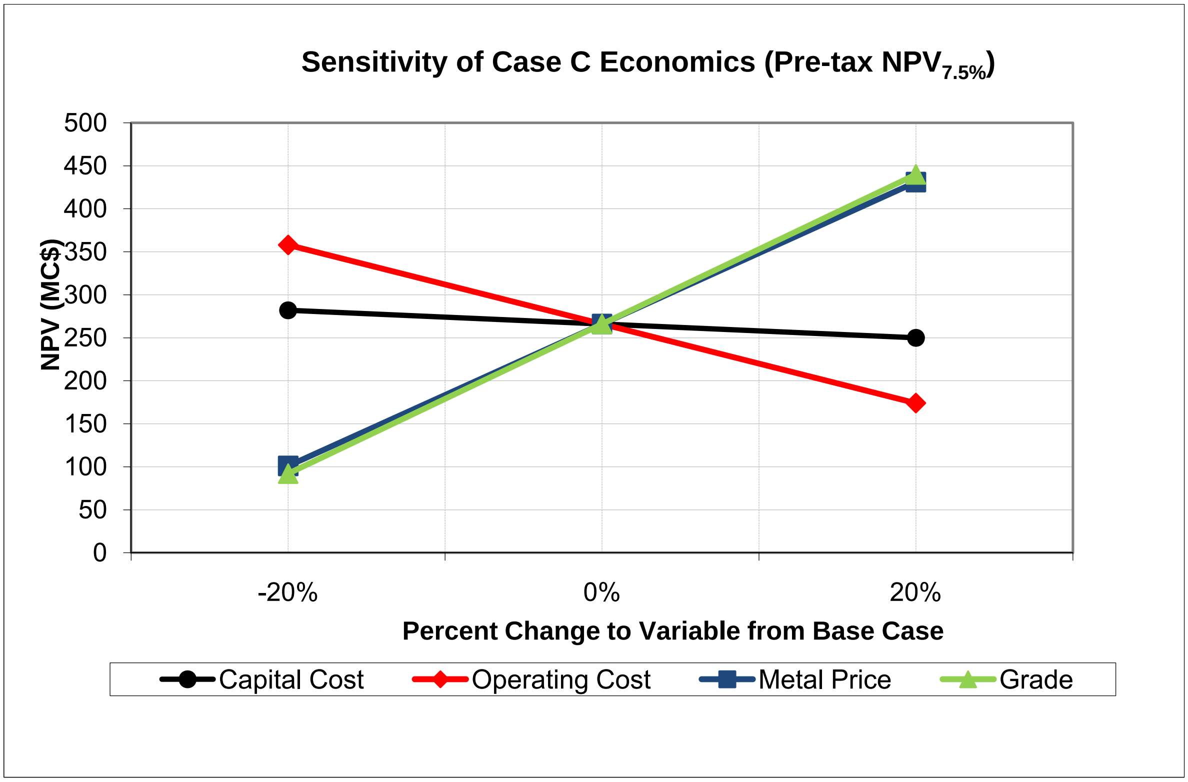


Figure 25.3: Case C Sensitivities

25.2 Payback

In Case A, the revenue the mine generates each year more than covers the capital expenditure for that year.

25.3 Mine Life

The Phase V LOM plan has mine operations finishing in the first half of 2018. The mill would continue to run on stockpiled ore until the middle of 2020 at which time all operations would cease.

It is SRK's opinion that there is potential for the mine to extend its life if additional resources can be turned into reserves. While there is no guarantee that this will happen, MintoEx has experienced a high level of success with its exploration drilling and there are still several exploration targets that show promise and are worth pursuing

In addition to finding more mineralized material, the improvement in long-term copper price or the reduction of operating costs could increase the mine life by lower in cut-off grades and allowing more resources to be converted to reserves. The reserves generated for this study were based on a copper price of $US2.25/lb, which is about 50% of the current price.

26 Interpretations and Conclusions

26.1 Processing Plant Risk and Opportunities

There are risks associated with the plant upgrade flowsheet, design criteria and equipment selection that may result in below design performance. Therefore opportunities exist to reduce the risk of below design performance.

Crushing Circuit Risks and Opportunities

The sizing of the existing jaw crusher is not seen as a risk for the plant upgrade. The published capacity of 37' x 49' jaw crusher with un-scalped feed and a closed side setting of 115 mm is around 290 tph and therefore the crusher is expected to achieve the design 228 tph.

The secondary crusher (S4800) has risk associated with the flowsheet design. There is no facility to screen the feed material prior to the crusher to remove fines. Therefore, the published de-rated (no fines scalping prior to crushing) crusher performance for a Sandvik S4800 with a closed side setting (CSS) of 25 mm is expected to be below the design throughput requirement of 228 tph. The risk with the secondary crusher flowsheet is that the CSS will be opened to achieve the required throughput which will increase the product size with a resultant decrease in SAG mill throughput. An opportunity exists to incorporate screening prior to the secondary crusher to reduce the load on the secondary crusher and provide the required final crushed product size for the milling circuit.

Crushed Ore Stockpile and Reclaim Risks and Opportunities

Previously, approximately 50% of the feed to the existing SAG mill was secondary crushed. The plant upgrade design is based on 100% of the SAG mill feed being secondary crushed and, hence, substantially finer. The current stockpile consists of a single apron feeder. The risk with this design is the finer crushed product on the stockpile will be different from the existing drawdown resulting in a variation in live stockpile capacity. An opportunity exists to review the crushed ore properties through further test work and/or experience in operating the recently installed secondary crusher. A second reclaim feeder will improve the amount of recoverable material on the stockpile. A second feeder will have the added benefit of providing improved blending to the SAG mill and operating redundancy.

Comminution Circuit Risks and Opportunities

The risks associated with modelling of the Minto comminution circuit are:

- The limited ore samples tested for ore competency are not "representative" of the range of ore competency characteristics;
- The limited ore samples tested for ore hardness are not "representative" of the range of ore competency characteristics;

- The actual plant observations by Starkey do not align well with the power based mill performance modelling by Ausenco and current plant performance.
- The ore competency data is limited for deposits outside of the Main deposit and further work is recommended to confirm the assumptions made in this report with regard to SAG mill throughput. Further test work is recommended to mitigate risk associated with the new ore bodies being either, on average more competent than main ore, or, containing areas of ore that may be localised but may limit plant production in the future.

The modelling of the comminution circuit indicates that the existing mills will need to operate at maximum power draw to achieve the design throughput. Whilst this is normal practice for a ball milling circuit, operational control of a SAG mill at sustained maximum power draw increases the risk of mill overloads and potential downtime.

Optimum Grind Size Risks and Opportunities

A point of discussion from the test work reports is the optimum primary grind size target. Table 26.1 below summarises the effect of primary grind size as studied in the test work.

Table 26.1: Summary of Effect of Grind Size on Recovery

Orebody	Impact of P_{80} on Cu and Au Recovery
Minto North	Primary grind size had no impact on rougher tailings grade but flotation kinetics slower with coarser grind.
Ridgetop East	The partially oxidized upper zone is sensitive to the primary grind size, and a grind P_{80} below 200 μm is required.
Area 118	Primary grind size had no significant impact on rougher tailings grade.
Area 2	Primary grind size had no significant impact on copper recovery but gold recoveries were 10% worse at P_{80} 270 compared with P_{80} 150 μm.
Main	A primary grind size of 200 μm appears optimum. Grind sizes coarser than 200 μm have poorer gold (5 - 10%) recoveries.
Main (South)	Copper and gold recoveries appeared to decrease at primary grind sizes higher than 150 μm.

The test work indicated some potential benefits of a finer primary grind size for certain deposits.

Regrind Mill Capital Cost Risks and Opportunities

Ausenco searched for a used VTM300 regrind mill however there were none located at the time of this study. The cost for a new VTM300 mill is around $1.2 million. A second hand VTM200 was sourced at the time of the pre-feasibility study at a cost of around $0.3 million. There is an opportunity to use this second hand regrind mill to reduce the overall capital expenditure however the risks are:

- The mill was not inspected by Ausenco during the pre-feasibility study and therefore refurbishment costs would need to be included; and
- The VTM200 would not provide the required regrind size of 80% passing 60 micron for the plant upgrade scenario. The P_{80} that would be achieved is around 72 micron for the nominal regrind circuit throughput of 21 tph (based on 171 tph fresh feed to the plant and 12.3% rougher/scavenger mass recovery).

Tailings Treatment Risks and Opportunities

The suitability of the existing thickener for the plant upgrade was determined. A summary of the current Minto thickener design against thickeners benchmarked by Ausenco in similar applications is included in Table 26.2.

Table 26.2: Tailings Thickener Sizing

	Thickener Solids Throughput (tph)	Thickener Unit Settling Rate (m^2/t/day)	Thickener Diameter (m)	Thickener Diameter Required (m)	Comments
Current Minto Tailings Thickener	113	0.024	9.14	-	
Similar Thickeners Benchmarked	149	0.052	-	13.5	
KM 2420 preliminary settling test work	149	0.045	-	12.5	Preliminary settling work on Minto North at 5 g/t floc and 55% solids U/F
KM 2351 preliminary settling test work	149	0.080	-	16.5	Preliminary settling work on Ridgetop East/Area 118 at 5 g/t floc and 55% solids U/F

The 13.5 m diameter thickener that Ausenco would recommend for the application is around 218% larger in surface area than the existing Minto tailings thickener and therefore would provide improved settling and thickener performance. A comprehensive settling test work campaign is recommended to confirm the suitability of the current tailings thickener during the next phase of the project.

Until mid-2011, the Minto Main Pit will not be available for direct discharge of thickened tailings. During this period the tailings will continue to be filtered and dry stacked. An evaluation into the capacity of the current tailings treatment circuit is required during the next phase. This evaluation should confirm if the existing tailings treatment circuit can handle the increased throughput.

Phase V Plant Expansion Opportunities

A high level trade-off study for a plant expansion to 7,500 tonnes per day was completed as part of the Phase IV Study. The preliminary throughput selection for the Phase V plant upgrade was based on:

- Current power supply and distribution constraints; and
- 7,500 tonnes per day is approaching the maximum volumetric throughput capability of existing facilities down stream of the grinding circuits. It is expected that pump box residence times, pipe

lines and other ancillary equipment will become limited above 7,500 tonnes per day. Above 7,500 tonnes per day a new process plant should be considered.

It is anticipated that the following major equipment would be required in addition to that installed as part of the Phase IV upgrade:

- A new single stage jaw crushing plant to replace the Phase IV crushing plant capable of treating 450 t/h and producing a product size of 80% passing 115 mm;
- A new single stage SAG mill capable of treating 240 t/h. The existing milling circuit would treat 102 t/h to provide an overall plant throughput of 342 t/h;
- A new reclaim feeder and SAG mill feed conveyor to supply ore to the new single stage SAG mill;
- An additional 3 x 40 m^3 rougher/scavenger flotation cells;
- A new flotation tailings thickener to replace the existing tailings thickener;
- Addition of a new flotation air blower; and
- A general upgrade of water, air and reagent services as well as slurry pumps as required.

A high level conceptual capital and operating cost was calculated for the Phase V plant expansion to 7,500 tonnes per day.

- Plant capital cost is expected to be in the order of $27 million. The exclusions from this estimate are per those listed in the capital cost section for the Phase IV upgrade in this report; and
- The process plant operating cost for the Phase V expansion is in the order of $9.20 /t. The basis for this estimate is similar to that described for the Phase IV estimate in this report.

26.2 Resource Estimation Interpretations and Conclusions

SRK reviewed and audited the exploration data available for Area 2/118 and Ridgetop deposits. This review suggests that the exploration data accumulated by MintoEx personnel is reliable for the purpose of resource estimation.

SRK, guided by MintoEx geologists, modelled mineralized domains based on up-to-date interpretation of mineralization on three deposits. A total of 30 (Area 2/118) and 70 (Ridgetop) separate wireframes were constructed in GEMS to represent ore zones alone. SRK considers that the geological model is a very good interpretation of the mineralized domains and is more than adequate for the resource estimation.

Following geostatistical analysis, SRK constructed new mineral resource block models for Area 2/118 and Ridgetop deposits constraining grade interpolation to within the modelled mineralization domains. After validation and classification, SRK considers that the mineral resources for the all three deposits are appropriately reported at a 0.5% Cu cut-off considering the open pit mining scenario discussed in the report.

Mineral resources for Area 2/118 and Ridgetop deposits have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" Guidelines. In the opinion of SRK, the block model resource estimate and resource classification reported herein are very representative of the copper, gold, and silver mineral resources found in the three deposits. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

Kirkham Geosystems reviewed and audited the exploration data available for the Minto North deposit. This review suggests that the exploration data accumulated by MintoEx personnel is reliable for the purpose of resource estimation.

Kirkham Geosystems, guided by MintoEx geologists, modelled mineralized domains based on up-to-date interpretation of mineralization the deposit. A total of three wireframes (i.e. 115, 120 and 130 zones along with a cross-cutting dyke) were constructed in MineSightTM. Kirkham Geosystems considers that the geological model is a very good interpretation of the mineralized domains and is more than adequate for the resource estimation.

Following geostatistical analysis, Kirkham Geosystems constructed new mineral resource block models for Minto North constraining grade interpolation to within the modelled mineralization domains. After validation and classification, SRK considers that the mineral resources for the Minto North deposit is appropriately reported at a 0.5% Cu cut-off considering open pit mining scenario as discussed in the report.

Mineral resources for the Minto North deposit has been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" Guidelines. In the opinion of Kirkham Geosystems, the block model resource estimate and resource classification reported herein are very representative of the copper, gold, and silver mineral resources found in Minto North. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

A number of factors may affect the quality and quantity of the current resource estimates, and thereby highlight opportunities for improvement:

- There are gaps in the understanding of the mineralization paragenesis. Improved understanding could benefit exploration models as well as the constraint on high-grade continuity and orientation. MintoEx are proactively making an effort in fundamental research to answer these questions.
- There are still some details that need to be constrained with respect to the structural geometries that are influencing the resource. Ductile and brittle fault structures and folding on various scales deform the ore horizons. The deformation history needs to be better constrained, and again research is currently on-going in order to answer these questions.
- There is poor control on the brittle structures that could impact the geotechnical assessment. It would be beneficial to undertake a mapping exercise of the current pits to determine the brittle fault and joint pattern. This information should be combined with drill hole logs, modelled structural information, mineralization offsets, exploration data and geophysical data (e.g. Titan 24 MT) to determine the structural patterns and position of major faults and folds.

26.3 Mining Conclusions and Risks

Conclusions

- The Minto deposit, encompassing Main Pit and Phase V OP and UG deposits, represents a significant ore reserve. The current mining in the Main Pit has helped confirm the expected grade and extent of the ore reserves and the detailed drilling has provided a good level of confidence in the reserve estimate.
- The Phase V deposits are estimated to be economic to exploit by both OP and UG mining methods and, according to the assumptions of this study, adds value to the Minto mine by increasing the NPV of the overall project.
- There are strong exploration targets in the immediate vicinity of the Main and Phase V pits.
- Based on test work conducted to date, the Phase V waste rock does not appear to have any ARD issues.

Risks

The flowing risks have been identified for the Minto Phase IV project:

- Mine Permit revisions: The mine is not currently permitted to carry out the mine plan as presented in the report. Changes to the permit involve a production increase, an increase in tailings deposition volume; and a change in tailings deposition modality.
- Exchange rates, metal prices and external influences: MintoEx has no control over exchange rates and their impact on the economics of the operation is significant. Metal prices are also not controllable, other than by forward sales contracts, and can have an appreciable effect on project return.
- Grade control: The Phase V pits (and in particular Area 2) are made up of several zones of ore that are not as continuous and thick as the Main zone currently being mined. As a result, a very thorough and proactive grade control program will be necessary in order to minimize dilution. Excessive dilution will have a negative impact on the project economics.
- The mining operating cost used in this study is based on an owner-operated fleet and is a significant departure from the current contract mining scenario, both operational and in terms of predicted unit costs.

Opportunities

The most significant opportunities that should be investigated are listed below:

Optimization of the Mine Plan

- Optimization of the mine plan: The mine plan has not been fully optimized and it is likely that further scheduling work will smooth out some of the grade and ore extraction variations seen in this study. The optimized mine plan may mean that higher grade ore is available to the mill sooner in the schedule, thus having a positive effect on the discounted cash flow.

Use of Higher Metal Prices in Mine Optimization

- Mineral reserves shapes were defined using a US$2.25/lb copper price; at higher copper prices, re-optimization of the UG and open pit shapes would yield more ore tonnes, all other things being equal and would improve the project NPV and lengthen the mine life.

  SRK conducted a preliminary mine optimization comparison using US$2.25/lb Cu vs. US$2.75/lb Cu, and US$:$C exchange rates of 0.86 and 0.92, respectively. The comparison revealed that using US$2.75/lb Cu increased mineable tonnes by approximately 3 Mt or 30% more than the US$2.25/ lb case. Using US$2.75/lb Cu dropped the average diluted grade from 1.4% Cu to 1.3% Cu and increased contained copper from 285 Mlb to 340 Mlb.

  The difference in NPV when comparing the US$2.25/lb pit results but using a copper price of US$2,75/lb and the US$2.75/lb pit was approximately $30M. The use of higher metal prices in the pit optimizations of near-term pits should be considered and further investigated.

Additions to Mineral Resources

- Minto has a consistent history of exploration success in both the exploration of new deposits and in the conversion of those deposits into mineral resources and reserves. There is no guarantee that this success will continue, however, the property still has a number of high-quality exploration targets that adjacent to existing reserves that are worth pursuing. It is SRK's opinion that some of those targets will be converted to resources and ultimately reserves with continued definition drilling.

Expansion Potential

In order to assess the opportunity of a potential large scale open pits and their potential impact on future permitting requirements, a preliminary study was conducted where an optimistic copper price, lower operating costs and increased mill throughputs were used to understand these potential pit limits.

Table 26.3 below compares the parameters that were modified for this expansion potential options analysis.

Table 26.3: Open Pit Optimization Parameters

Item	Unit	Case 1	Case 2	Case 3	Case 4
Plant Throughout					
Annual	Mtpa	2.74		6.75	
Daily Average	tpd	7,500		18,500	
Metal Prices					
Copper	US$/lb	2.25	2.75	2.25	2.75
Gold[1]	US$/oz	300			
Silver[1]	US$/oz	3.9			
Exchange Rate					
Exchange Rate[2]	C$/US$	1.16	1.09	1.16	1.09
Recovery to Cu Concentrate					
Copper	% max	92			
Gold	% max	70			
Silver	% max	80			
Cu Concentrate Grade					
Copper[3]	%	38			
Gold	g/t	variable with Cu grade			
Silver	g/t	variable with Cu grade			
Moisture content	%	8			
Smelter Payables					
Copper in Cu conc	%	96.75			
Gold in all cons	%	per MRI contract			
Gold deduction in all cons	g/t in conc	0			
Silver in all cons	%	100			
Silver deduction in all cons	g/t in conc	30			
Off-site Costs					
Cu concentrate treatment[4]	US$/dmt conc	40			
Cu refining charge[4]	US$/lb pay Cu	0.04			
Au refining charge	US$/oz pay Au	5			
Ag refining charge	US$/oz pay Ag	0.4			
Ocean freight to Japan[5]	US$/wmt conc	60			
Truck freight to Skagway[5]	US$/wmt conc	57.33	61.01	57.33	61.01
Skagway port charges[6]	US$/wmt conc	10.5		10	
Skagway quarterly user fee	US$/wmt conc	6.13		2.49	
Port maintenance[6]	US$/wmt conc	1.5		1.25	
Survey, assay, umpire	US$/wmt conc	3.15			
Customs fee	US$/wmt conc	1.85			
Port property taxes	US$/wmt conc	0.51		0.21	
Marine/transportation insurance[7]	%	0.049% x 110% of shipment value			
Transport, marketing, ins, etc.	US$/wmt conc	140.97	144.65	136.27	139.95
Transport, marketing, ins, etc.	US$/dmt conc	153.23	157.23	148.12	152.12
Other Parameters					
NSR Royalty	%	1	1.25	1	1.25
Pit Slope Angles	overall	As per SRK 2009 PFS			
Dilution	%	8			
Mining recovery	%	100			
Operating Costs					
Waste Mining Cost[8]	C$/t waste	2.18		1.5	
Ore Mining Cost	C$/t ore	2.18		1.5	
Processing Cost	C$/t milled	9.12		10.5	
G&A Cost	C$/t milled	6.51			

1 Based on previous metal stream transaction with Silver Wheaton
2 Based on the historical relationship between Cu price and C$:US$ exchange rate
3 Capstone estimate based on bulk tonnage likely being less efficient than current operation
4 Capstone estimated below benchmark pricing for good quality concentrate
5 Based on Capstone's estimate of efficiencies on shipping larger loads
6 Based on historical port charges and maintenance costs and efficiencies of bulk tonnage
7 MRI charge
8 Based on previous studies and first principles estimate for bulk mining scenario

A revised NSR model was created based on the parameters noted above. The revised operating costs and throughput rates were then used in the Whittle optimization to determine the potential open pit limits. The revised operating costs were based on a factoring of the costs used for the PFS as well as on experience for similar sized large scale open pit operations.

It should be noted that this large open pit scenario is preliminary in nature and only serves as a rough indication of potential pit size. Further detailed work would need to be carried out in order to increase the level of confidence of the results. Many technical issues were not considered in this review but would be important to investigate should any of these cases move ahead. The 7,500 tpd case will, for the most part, take advantage of existing infrastructure, whereas the 18,500 tpd case requires a broader assessment of these technical issues. They include, but are not limited to:

- Tailings disposal, waste rock disposal and water management;
- Geotechnical and hydrogeological characteristics of the deeper, larger pits, for both overburden and rock;
- New mill and camp location;
- Power supply; and
- Environment and permitting.

Also, this large open pit scenario encompasses mineral resources that are not mineral reserves and have not currently demonstrated economic viability. There is no certainty that the tonnages noted will be converted to the measured and indicated resource category through further drilling, or into mineral reserves, once economic considerations are applied.

Expansion Potential Results

The results of the preliminary study of the potential large scale open pit, based on the parameters noted above, are summarized in Table 26.4 below. These include estimates on the U/G contribution for each case, considering the UG tonnes remaining under the pits.

Table 26.4: Expansion Potential Results

Parameter	Unit	Case 1	Case 2	Case 3	Case 4
Ore Mined - OP	Mt	19.7	32.4	48.5	67.4
Cu Grade	%	1.1	0.93	0.77	0.68
Au Grade	g/t	0.37	0.3	0.24	0.21
Ag Grade	g/t	3.35	2.9	2.46	2.23
Waste Material Mined	Mt	77.3	148.2	189.3	284
Overburden	Mt	13.6	26.2	33.4	50.1
Total Material Moved	Mt	110.6	206.8	271.3	401.5
Stripping Ratio	t:t	4.6	5.4	4.6	5
Underground Ore Mined	Mt	2.2	1.4	1.4	1.4
Cu Grade	%	2.05	2.08	2.08	2.08
Au Grade	g/t	0.74	0.68	0.68	0.68
Ag Grade	g/t	6.96	5.83	5.83	5.83
Total Material Mined	Mt	112.8	208.3	272.7	403
Contained Metal					
Copper	Mlb	580	732	886	1,073
Gold	Koz	287	347	403	488
Silver	Koz	2,620	3,297	4,103	5,095
MILLING					
Ore Tonnes	Mt	21.9	33.9	49.9	68.8
Mill Head Grades					
Cu Grade	%	1.2	0.98	0.8	0.71
Au Grade	g/t	0.41	0.32	0.25	0.22
Ag Grade	g/t	3.72	3.03	2.55	2.3

As can been seen from the results summarized in the above table there are significant increases in material tonnages for the large scale pits versus the pits defined in the mineral reserve section of this report.

Preliminary schedules were created using a basic assumption of mining the higher grade deposits first. Only average grades for each deposit were used in the schedules and schedules were not validated for bench advance or minimum mining widths and were not optimized. Preliminary Opex and Capex cost estimates were made and a summary of the earnings before interest, taxation, depreciation and amortization (“EBITDA”) analysis are shown in Table 26.5.

Table 26.5: EBITDA Summary

Item	Unit	Case 1	Case 2	Case 3	Case 4
Waste mined	Mt	91	174	223	334
Ore mined	Mt	22	34	50	69
Total mined	Mt	113	208	273	403
Strip ratio	W:O	4.6	5.4	4.6	5
Mill Feed	Kt/day	7,500	7,500	18,500	18,500
Mine Life	Production Years	8	13	8	11
Copper millhead grade	% Cu	1.2	0.87	0.8	0.71
Gold millhead grade	g/t Au	0.41	0.29	0.25	0.22
Silver millhead grade	g/t Ag	3.72	2.79	2.55	2.3
Copper in cons	Mlb	533	674	815	987
Gold in cons	Koz	201	243	282	341
Silver in cons	Koz	2,096	2,638	3,282	4,075
Concentrate Grade	% Cu	38	38	38	38
Copper Price	US$/lb	2.25	2.75	2.25	2.75
Gold price	US$/oz	300	300	300	300
Silver price	US$/oz	3.9	3.9	3.9	3.9
Exchange rate	US$/C$	0.86	0.92	0.86	0.92
NSR (inc. royalties)	$/t milled	56.59	53.47	37.89	38.57
Unit Mining Costs	$/t milled	14.58	14.79	9.15	9.47
Unit Total OPEX	$/t milled	30.21	30.42	19.65	19.97
Unit On-site OPEX	$/lb Cu payable	1.28	1.58	1.24	1.44
Unit OPEX net by-product credits	$/lb Cu payable	1.16	1.46	1.13	1.33
Unit Off-site OPEX	$/lb Cu payable	0.33	0.31	0.32	0.31
Total Capital (initial & sustaining)	$M	118	134	398	429
Allowance for closure cost	$M	22	34	50	69
$NPV_{0\%}$ pre-tax	$M	460	647	513	851
$NPV_{5\%}$ pre-tax	$M	374	475	377	606
$NPV_{7.5\%}$ pre-tax	$M	339	413	322	513
$NPV_{10\%}$ pre-tax	$M	309	362	274	435

Although the large scale pits provide the potential for more tonnage through the mill, they do so at a reduced copper grades (due to lower operating costs and higher copper prices) and also would require significant increases in waste dump capacities as well as tailings storage requirements. A significant increase in capital expenditures would also be required, both from a mining and mill processing standpoint, and again further studies would be required to determine the economics of this bulk mining scenario.

The results of this study indicate that there is upside potential to expand the existing Minto Mine operation. All cases considered had positive $NPV_{7.5\%}$s in spite of having schedules and production throughput that were not optimized. Further study is warranted and should be done in greater detail to consider the impact of larger pits on the Minto property especially for tailings, waste and water management. The study shows that a key element of any further expansion work at Minto will require a trade-off between gaining economies of scale by production expansion must be and the efficient use of capital.

26.4 In-pit Tailings Disposal – Conclusions, Risks and Opportunities

Conclusions

- In-pit tailings disposal methods can be used to store the entire volume of tailings associated with the development of the Phase V pits into Main, Area 2 and Ridgetop North mined out pits.
- A stability buttress of waste rock that will be created within the limits of the Minto Main pit has been accounted for when determining in-pit storage requirements.
- The buttress will be constructed in stages, depending on deposition and water balance requirements, commencing with a starter embankment prior to the commencement of in-pit tailings disposal.
- In-pit management of tailings and water (including annual freshet inflows to the pit of approximately 700,000 cubic metres) will result in the tailings being inundated for the entire operational life, resulting in the requirement for subaqueous tailings deposition.
- Slurry deposition would be performed from variable locations around the pit perimeters and within the pit “basin” to facilitate uniform distribution of tailings and avoid the formation of a “peak and valley” tailings surface.
- During winter, the deposition plan may have to be modified to account for temperatures significantly below 0 degrees C.
- Excess water would be pumped from the pits using a floating barge that would have sufficient capacity to accommodate both mill operational requirements (continuous recycle at an assumed rate of 150 m^3/hr) and annual freshet disposal requirements (approximately 100 to 250 m^3/hr for 5 months per year). It is expected that the annual freshet disposal water will require treatment prior to disposal.
- Seepage through the divider embankment (and potentially the pit sidewalls) can be controlled through embankment design and construction, tailings management (pre-sliming) and vertical dewatering wells.
- Storage of water and tailings behind the buttress could enhance stability of the buttress since the hydraulic pressure is maximized at the same location as the toe of the south wall slide. This will be reviewed in a FLAC model that the site is to complete in 2011.

Risks

- As tailings and water are stored behind the buttress in the Main Pit, the North wall of the Area 2 pit will have to be monitored to ensure slope design assumptions for the overburden layer are confirmed. It would be prudent to undertake "high level" mitigation planning in conjunction with the final design of the buttress. Therefore, in the event the slope shows movement which exceeds that which conforms to the design assumption, the development of a mitigation plan can be conducted expeditiously.
- Reclaim water pumped from the barge may not be sufficiently clarified for immediate use in the plant, which leads, for example, to an incremental water treatment requirement at the plant.
- Operational difficulties with the reclaim barge lead to the need for adjustments to the operation plan and/or the implementation of a different reclaim system.
- Operational difficulties with tailings deposition lead to a highly uneven tailings surface that, in turn, leads to significant incremental closure costs associated with creating an appropriately covered and graded tailings surface at closure.
- Due to possible additional inflows linked to seepage management and Area 2 Pit dewatering requirements, the anticipated annual surplus water treatment/disposal requirements could increase.

Opportunities

- The water collected in the wells which are likely to be installed in the buttress turns out to be appropriate in volume and quality that it can be pumped directly to the plant for re-use in the mill circuit.
- Cyclones could be used to deposit sand tailings on the benches around the pit in order to increase the storage capacity of the various pits.
- Once the use of the various pits for active tailings deposition has stopped, the concept of storing additional waste rock or overburden on the tailings surface (beyond what would be needed within the current closure concepts), could be considered.

27 Recommendations

27.1 Further Metallurgical Test Work

Work carried out to date is sufficient to support the PFS design and costing. Further work will be required for a Feasibility Study in order to confirm certain aspects of the design criteria. For a detailed Feasibility Study flotation and comminution variability test work across the ore body is required to develop detailed models of plant throughput and grade/recovery that take into account variations in competency, mineralogy and head grade.

Further Comminution Test Work

The test work undertaken to date on the ore competency (impact breakage for SAG Mill sizing) and ore hardness (abrasion breakage for ball mill sizing) is limited. It is recommended that further test work be completed to confirm the similarities between the current plant feed (Minto Main ore) and the new orebodies. The test work should comprise of:

- SMC and ball mill work index tests on current plant feed;
- Associated throughput and SAG and ball mill specific energy measurement (average over 2 hours); and
- Ball mill cyclone overflow P_{80} measurement sampled over the same 2 hours.

SMC tests should be conducted on Area 2/Ridgetop/North drill core over a larger range of holes.

At least 6 mill feed samples are recommended (over a one week period of normal and typical operation) and around 10 drill core samples from across the future ore bodies.

The purpose of further comminution test work is to mitigate risk associated with the new orebodies being either on average harder than the current Minto Main ore, or containing localised zones of harder ore.

Further Flotation and General Plant Design Test Work

Recommended additional test work identified as part of a feasibility study includes:

- A program of locked cycle test work specifically at the plant up-grade conditions (primary grind size of 250 micron with rougher/scavenger concentrate regrind at 60 micron) to determine the validity of the assumptions used for the overall recoveries and final concentrate grades;
- Test work to confirm tailings thickening rates for tailings thickener selection;
- Test work to confirm concentrate filtration rates to verify the suitability of the current concentrate filter for the finer re-ground flotation concentrate;
- Rheology test work to confirm tailings pumping, pipeline and distribution design at the TSF;

- Bulk materials handling test work to optimise design of the chutes, conveyors, crushed ore stockpile and reclaim facility; and
- Confirmation of geotechnical conditions for engineering design purposes in the plant, particularly in the locations of heavy structures such as the Vertimill.

The overall cost for the recommended comminution, flotation and general plant design test work is in the order of $300K.

27.2 Mining and Exploration

- Further exploration drilling is recommended to further define drilled targets that indicate anomalous metal values, in particular, deeper targets that could have further underground mining potential:
- Down-hole geophysical surveys be carried out in any future drill holes in order to better vector exploration in the Copper Keel and Airstrip Southwest areas.
- Further optimization studies should be conducted to attempt to smooth out the mill-feed grade profile and the mining schedule;
- A comprehensive study of the brittle structural geology would be beneficial to better quantify mineralization zone displacements, and to increasing confidence in geotechnical rock mass characterization.
- Continue geotechnical investigations with respect to the Main Pit buttress. It may be feasible to decrease the size of the buttress that would increase tailings deposition.

27.3 Geotechnical Work

Additional geotechnical characterization and analyses should be conducted at the feasibility and design levels for each of the areas. Analyses and recommendations presented herein are based on ultimate pit designs as described in this report, and, as such, any significant changes to mine plans or pit architecture should be reviewed by SRK to verify that recommendations will remain valid for the new mine plans.

Geologic structure should be further evaluated to more accurately characterize the rock mass which, according to the current mine plans, will comprise the toe of the Area 2 western slope walls and which will better ascertain the likelihood of the existence and orientation of major structures that may adversely impact stability of that western wall.

To do so, two additional geotechnical drill holes are recommended at Area 2 to investigate the potential for such major structures and to further characterize the variability in orientation of joint sets.

Additional geotechnical characterization and analysis will also be necessary at Minto North, to better define rock mass conditions and structural impacts on bench stability as the project advances. To accomplish this, one additional geotechnical corehole is recommended at Minto North drilled into the northwest wall for evaluation of rock mass conditions and structure.

The underground areas will also require additional geotechnical drilling for rock mass characterization at the feasibility and design levels. Underground mapping from the access drift and horizontal (low angle diamond drill holes) would be desirable to better capture and characterize steep dipping structures.

The Area 118 and Ridgetop open pits most likely will not require additional geotechnical drilling unless major changes are made to the current plans.

Tailings Solids

- Grain size distribution with hydrometer (-#200 fraction) and Atterberg limits (to evaluate cycloning potential, settlement characteristics, in situ permeability and potential for use of underflow as a drainage layer);
- Modified Proctor testing (cyclone underflow - to evaluate constructability and define parameters for direct shear and permeability testing);
- Specific gravity (to facilitate evaluation of slurry rheology);
- Shear strength (cyclone underflow and overflow fractions for embankment stability evaluation);
- Flexible wall permeability (total tailings permeability, or cyclone underflow and overflow drainage characteristics);
- Settled density and consolidation testing to evaluate the initial settled dry density and the dry density of the tailings under self-weight.

Overburden and Waste Rock

- The following recommendations apply to overburden, waste rock or other borrow material that may be used for embankment construction:
- Grain size distribution and Atterberg limits (characterization of material for suitability as filter material and/or embankment core material and constructability)
- Modified Proctor testing (to evaluate constructability and define parameters for direct shear and permeability testing)
- Shear strength (direct shear for embankment stability evaluation)
- Flexible wall permeability (to determine drainage characteristics and necessity for low-permeability embankment liner or core)

Foundation Properties of Main Pit/Area 2 Pit Dividing Ridge

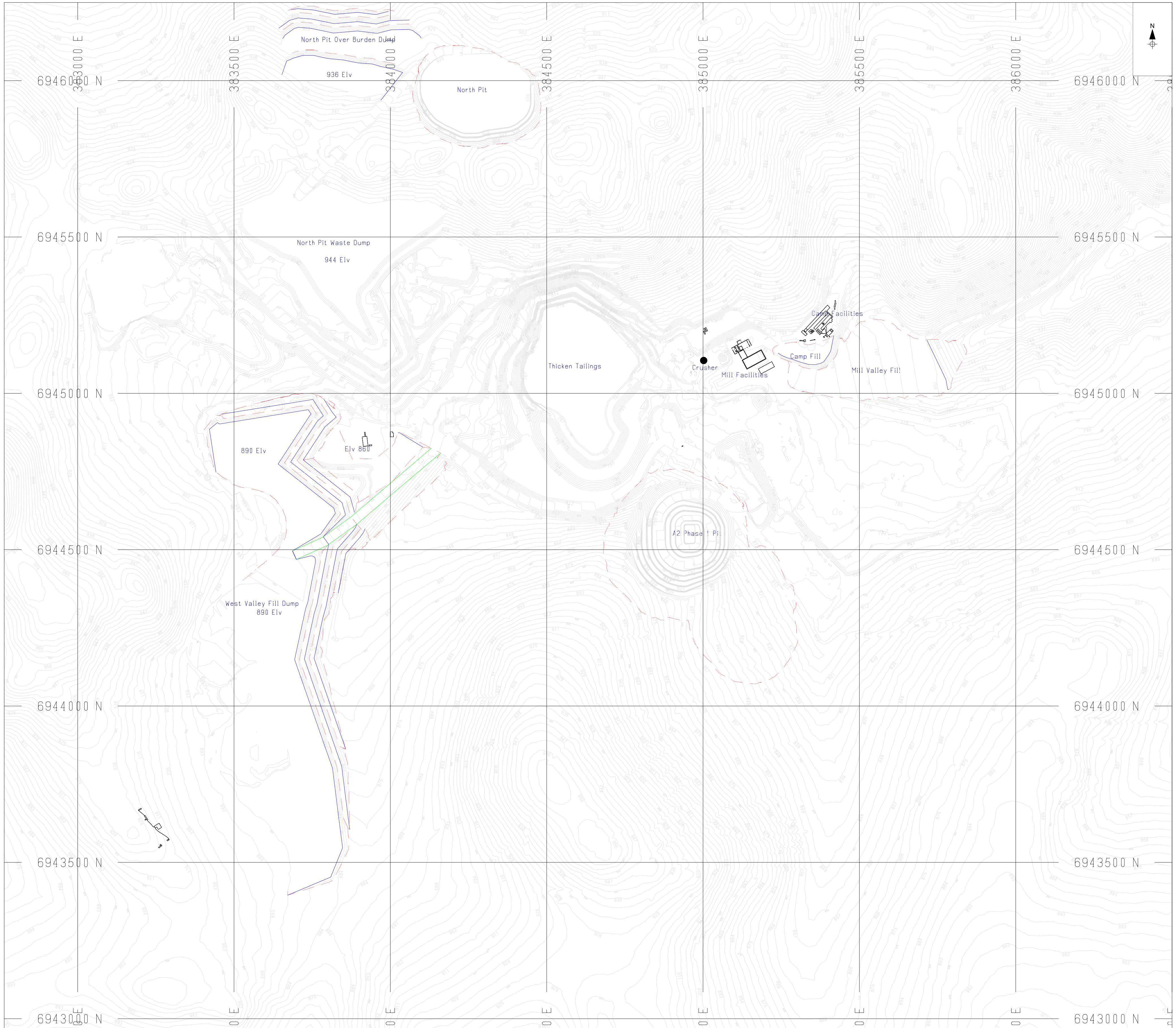
- Foundation evaluation of native soil and/or rock that will form the residual dividing ridge between the final configurations of the Main Pit and Area 2 Pit, to include the same material characterization as described above for overburden and waste rock, together with an evaluation of the potential for settlement or foundation failure due to the planned embankment construction (stability analysis, rock fracture evaluation, etc. as required depending on nature of in situ material)

Pit Area Surface and Subsurface Hydrogeology

- Definition of surface drainage characteristics and the potential to divert run-on flows
- Depth of active layer
- Depth of base of permafrost
- Depth to groundwater and the shape of the potentiometric surface
- Evaluation of potential pit groundwater inflows (i.e. inflows into Main Pit and potential dewatering requirement for Area 2 Pit)

28 Illustrations

End of Period Maps



SRK Consulting Engineers and Scientists

SRK PROJECT NO: 2CM022.016

FILE NAME:

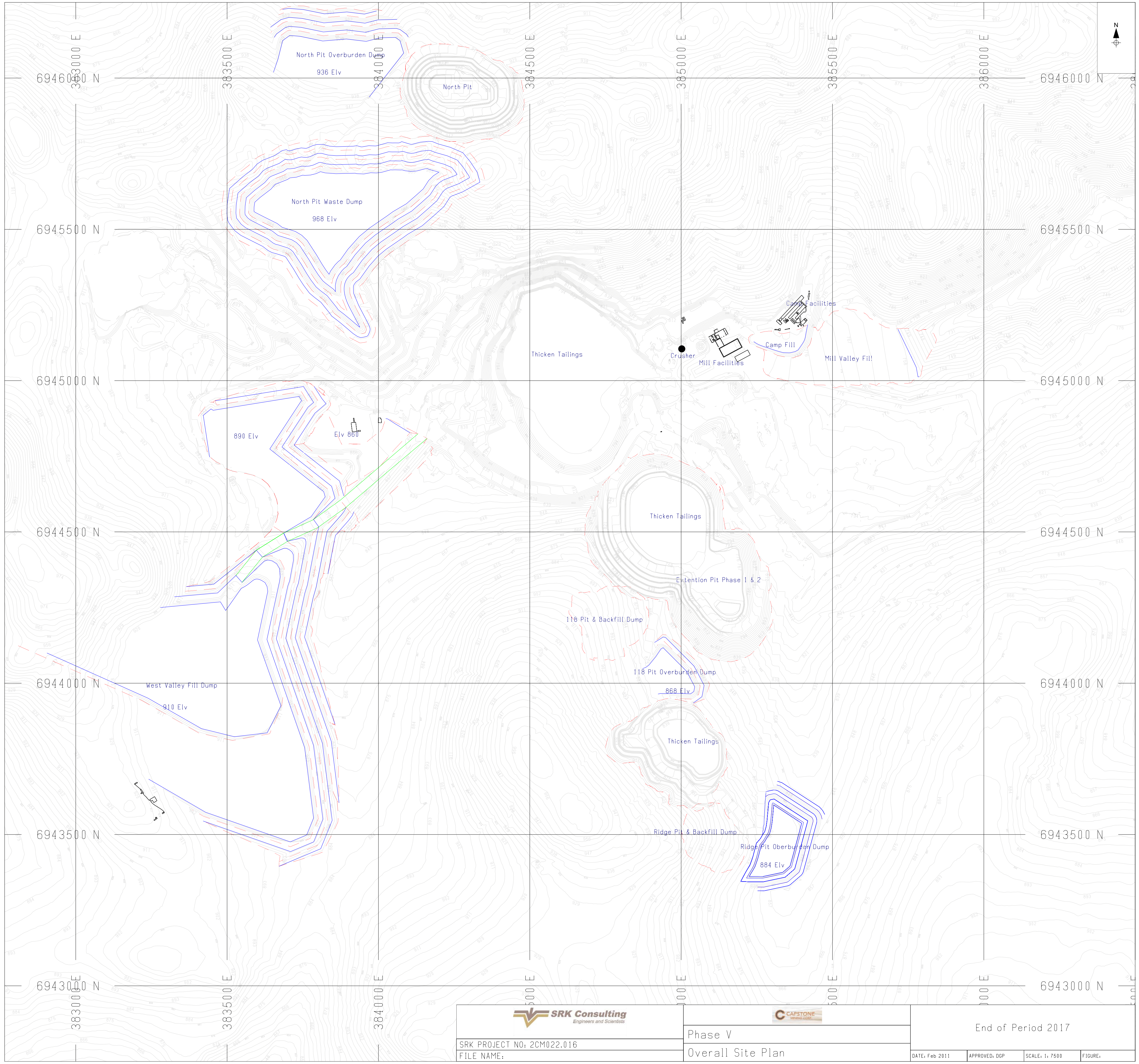
CAPSTONE MINING CORP.

Phase V

Overall Site Plan

End of Period 2012

DATE: Feb 2011	APPROVED: DGP	SCALE: 1: 7500	FIGURE:


North Pit Overburden Dump
936 Elv
North Pit
North Pit Waste Dump
968 Elv
Camp Facilities
Thicken Tailings
Crusher
Mill Facilities
Camp Fill
Mill Valley Fill
890 Elv
Elv 860
Thicken Tailings
Extention Pit Phase 1 & 2
118 Pit & Backfill Dump
118 Pit Overburden Dump
868 Elv
West Valley Fill Dump
910 Elv
Thicken Tailings
Ridge Pit & Backfill Dump
Ridge Pit Oberburden Dump
884 Elv
6946000 N
6945500 N
6945000 N
6944500 N
6944000 N
6943500 N
6943000 N
383000 E
383500 E
384000 E
384500 E
385000 E
385500 E
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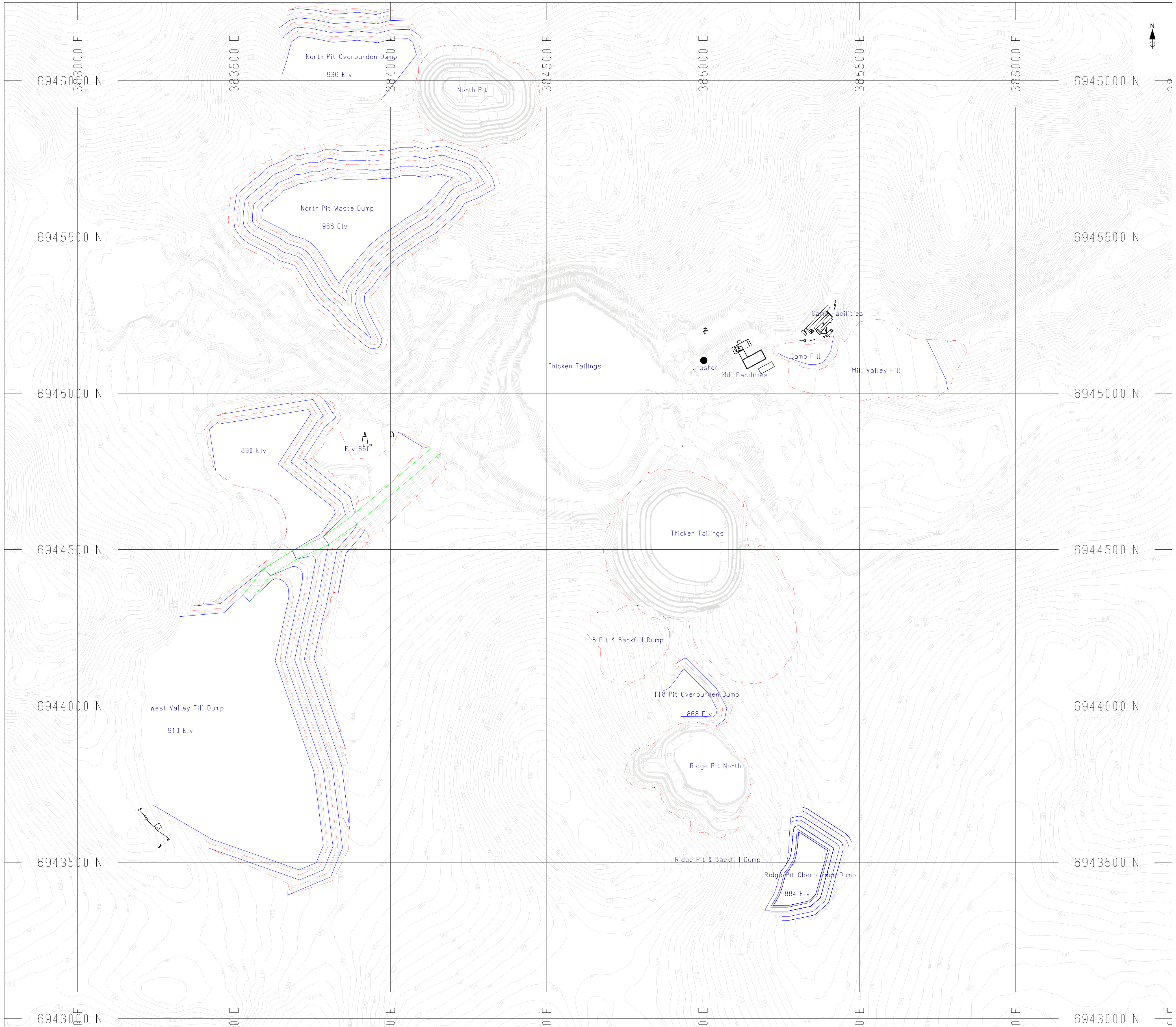
N
SRK Consulting
Engineers and Scientists
SRK PROJECT NO: 2CM022.016
FILE NAME:
CAPSTONE
Phase V
Overall Site Plan
End of Period 2017
DATE: Feb 2011
APPROVED: DGP
SCALE: 1: 7500
FIGURE:



SRK Consulting
Engineers and Scientists

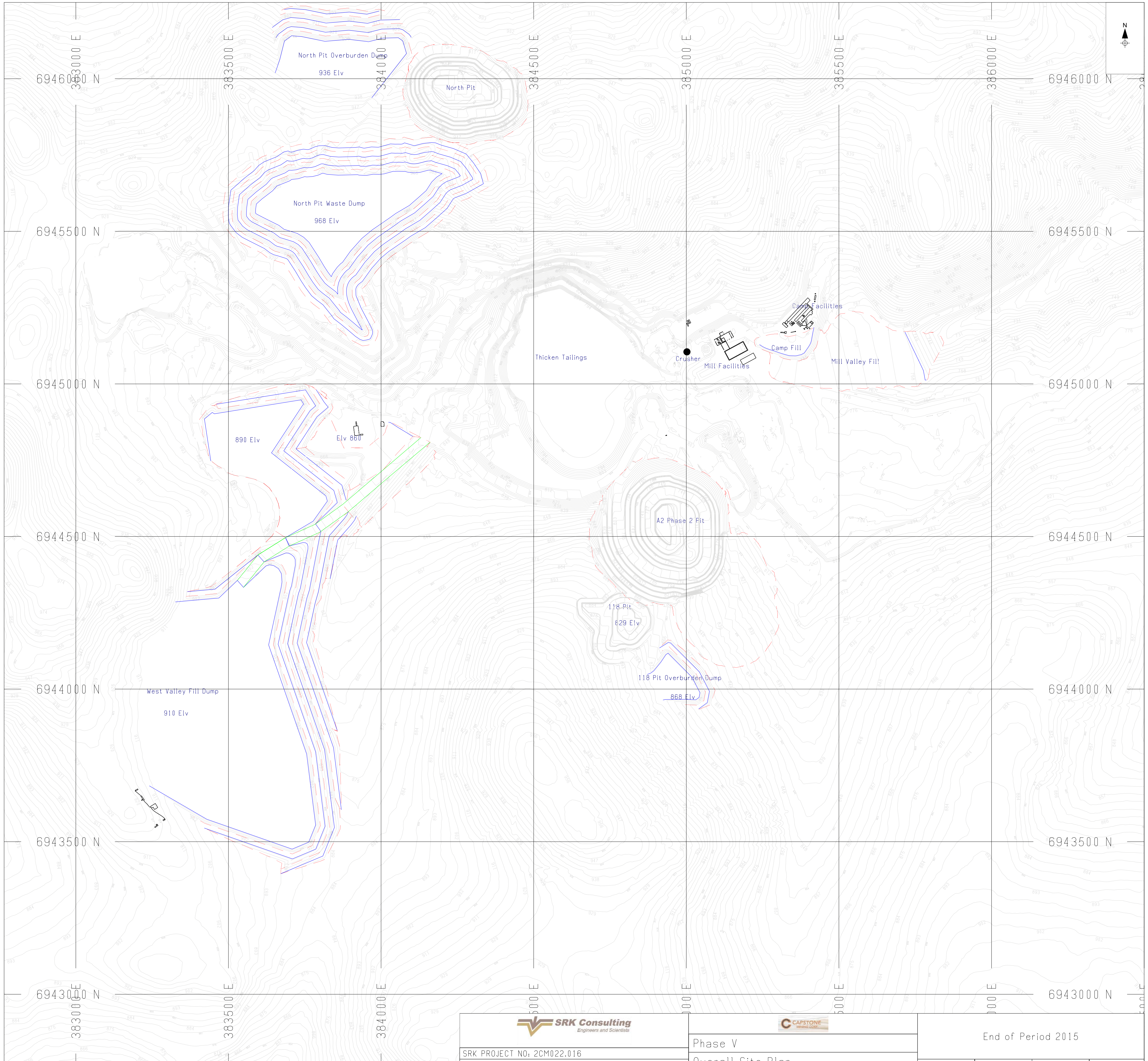
SRK PROJECT NO: 2CM022.016
FILE NAME:

Capstone Mining Corp.

Phase V
Overall Site Plan

End of Period 2016

DATE: Feb 2011	APPROVED: DGP	SCALE: 1: 7500	FIGURE:



North Pit Overburden Dump
936 Elv
North Pit
North Pit Waste Dump
968 Elv
Camp Facilities
Thicken Tailings
Crusher
Mill Facilities
Camp Fill
Mill Valley Fill
890 Elv
Elv 880
A2 Phase 2 Pit
118 Pit
629 Elv
118 Pit Overburden Dump
868 Elv
West Valley Fill Dump
910 Elv
6946000 N
6945500 N
6945000 N
6944500 N
6944000 N
6943500 N
6943000 N
383000 E
383500 E
384000 E
384500 E
385000 E
385500 E
386000 E
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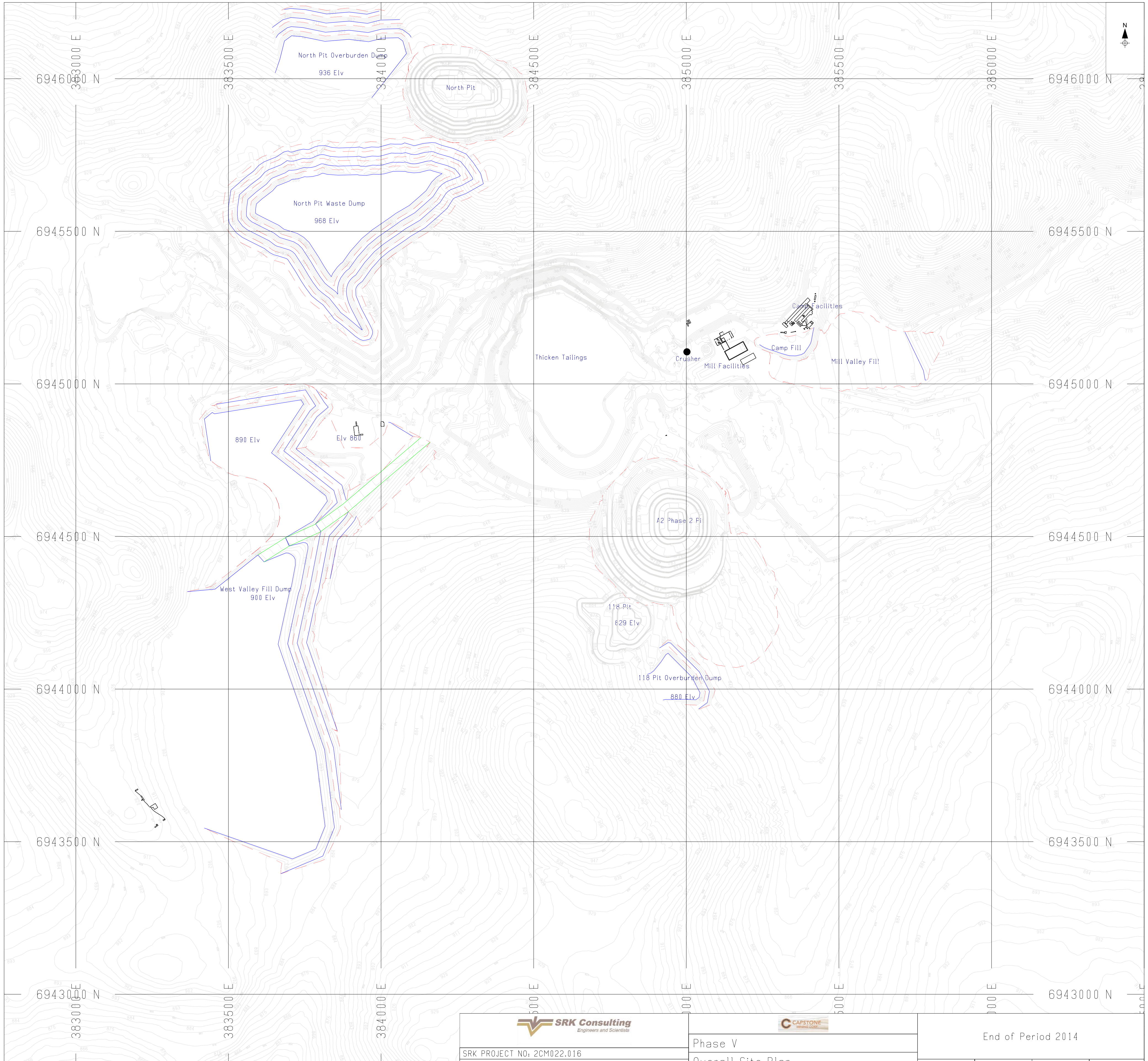
SRK Consulting
Engineers and Scientists
SRK PROJECT NO: 2CM022.016
FILE NAME:
CAPSTONE
Phase V
Overall Site Plan
End of Period 2015
DATE: Feb 2011
APPROVED: DGP
SCALE: 1: 7500
FIGURE:


North Pit Overburden Dump
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968 Elv
Thicken Tailings
Crusher
Mill Facilities
Camp Facilities
Camp Fill
Mill Valley Fill
890 Elv
Elv 860
West Valley Fill Dump
900 Elv
118 Pit
629 Elv
118 Pit Overburden Dump
880 Elv
6946000 N
6945500 N
6945000 N
6944500 N
6944000 N
6943500 N
6943000 N
383000 E
383500 E
384000 E
384500 E
385000 E
385500 E
386000 E
N
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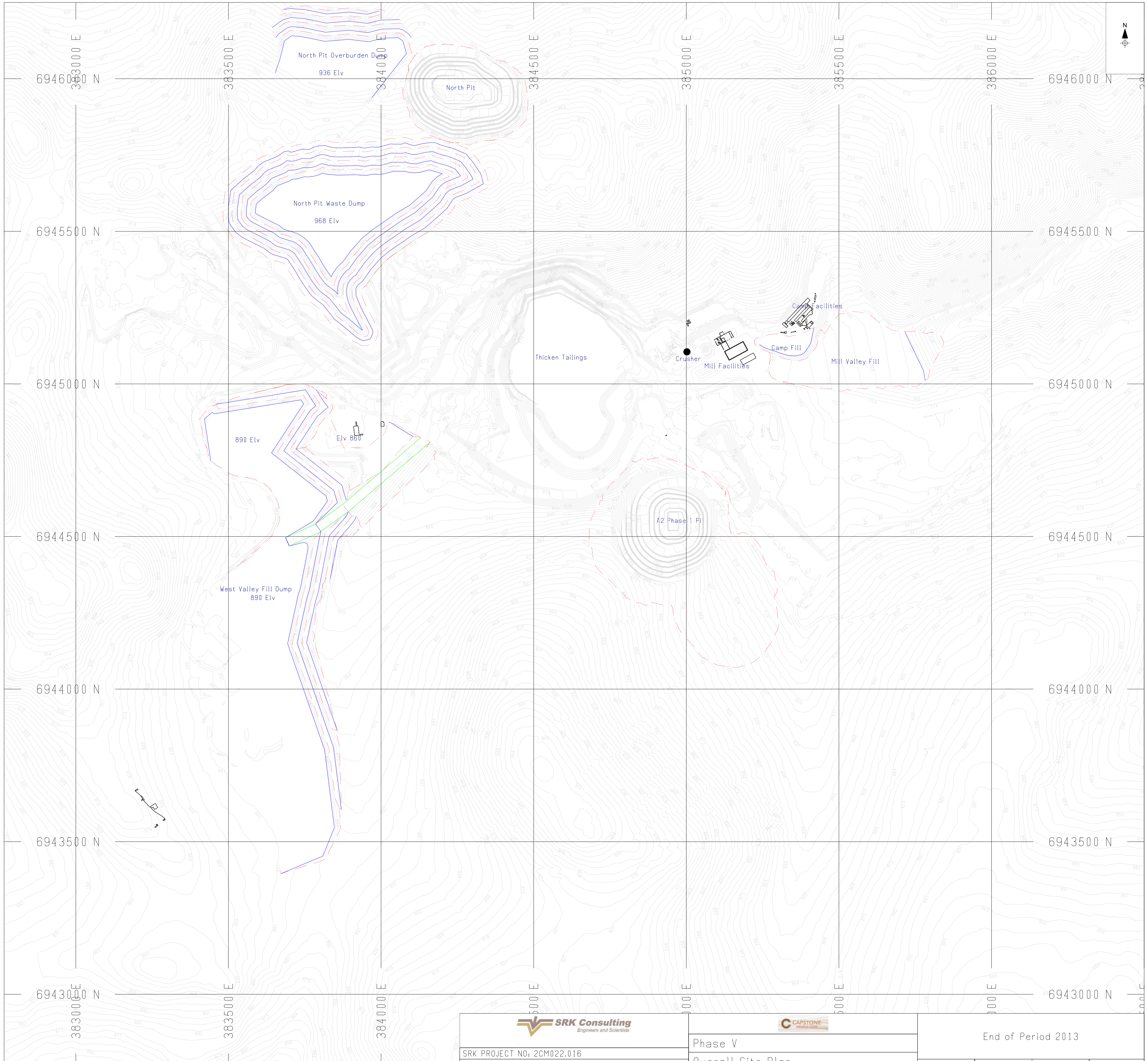
SRK PROJECT NO: 2CM022.016
FILE NAME:
Phase V
Overall Site Plan
End of Period 2014
DATE: Feb 2011
APPROVED: DDP
SCALE: 1: 7500
FIGURE:


North Pit Overburden Dump
936 Elv
North Pit
North Pit Waste Dump
968 Elv
Camp Facilities
Thicken Tailings
Crusher
Mill Facilities
Camp Fill
Mill Valley Fill
890 Elv
Elv 860
A2 Phase 1 Pit
West Valley Fill Dump
890 Elv
6946000 N
6945500 N
6945000 N
6944500 N
6944000 N
6943500 N
6943000 N
383000 E
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384500 E
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386000 E
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Engineers and Scientists
SRK PROJECT NO: 2CM022.016
FILE NAME:
Phase V
Overall Site Plan
End of Period 2013
DATE: Feb 2011
APPROVED: DDP
SCALE: 1:7500
FIGURE:

29 References

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30 Standard Acronyms and Abbreviations

Distance	
µm	micron (micrometre)
mm	millimetre
cm	centimetre
m	metre
km	kilometre
”	inch
in	inch
’	foot
ft	foot
Area	
m^2	square metre
km^2	square kilometre
ac	Acre
Ha	Hectare
Volume	
l	litre
m^3	cubic metre
ft^3	cubic foot
usg	US gallon
yd^3	cubic yard
bcm	bank cubic yard
Mbcm	Million bcm
Mass	
kg	kilogram
g	gram
t	metric tonne
Kt	Kilotonne
lb	pound
Mt	Megatonne
oz	troy ounce
wmt	wet metric tonne
dmt	dry metric tonne
Pressure	
psi	pounds per square inch
Pa	Pascal
kPa	kilopascal
MPa	megapascal
Elements and Compounds	
Au	gold
Ag	silver
Cu	copper
Hg	lead
Zn	zinc
$CaCO^3$	Calcium carbonate
ANFO	Ammonium Nitrate/Fuel Oil

Other	
^{o}C	degree Celsius
^{o}F	degree Fahrenheit
Btu	British thermal unit
cfm	cubic feet per minute
elev	elevation above sea level
amsl	above mean sea level
hp	horsepower
hr	hour
kW	kilowatt
kWh	kilowatt hour
Ma	Million years
mph	miles per hour
ppb	parts per billion
ppm	parts per million
s	second
s.g.	specific gravity
usgpm	US gallon per minute
V	volt
W	watt
Ω	ohm
A	ampere
tph	tonnes per hour
tpd	tonnes per day
Ø	diameter
Acronyms	
SRK	SRK Consulting (Canada) Inc.
CIM	Canadian Institute of Mining
NI 43-101	National Instrument 43-101
ABA	Acid- base accounting
AP	Acid potential
NP	Neutralization potential
NPTIC	Carbonate neutralization potential
ML/ARD	Metal leaching/ acid rock drainage
Conversion Factors	
1 tonne	2,204.62 lb
1 oz	31.1035 g

31 Date and Signature Page

This technical report was written by the Qualified Persons listed below. The effective date of this technical report is December 15, 2010.

Qualified Person	Signature	Responsible Section
Cam Scott, P.Eng	Original signed	Sections 18.4.3 and 26.4
David Brimage	Original signed	Sections 15, 19, 24.1.3, 24.2.3, 26.1 and 27.1
Dino Pilotto, P.Eng	Original signed	Sections 16.7, 16.8, 18.2, 18.3.1 to 18.3.3, 18.4.1, 18.4.2, 24.1.2, 24.2.2 and 26.3
Gordon Doerksen, P.Eng	Original signed	Sections Executive summary, 1 to 4, 14, 20 to 23, 24.1, 24.2, 25, 27.2 and 28 to 31
Garth Kirkham, P.Geoph.	Original signed	Sections 16.5 and 16.6
Iouri Iakovlev, P.Eng.	Original signed	Sections 16.9, 18.1, 18.3.4, 24.1.1 and 24.2.1
Marek Nowak	Original signed	Sections 13 and 16.1 to 16.4
Mike Levy, PE	Original signed	Sections 17.1 and 27.3
Scott Carlisle, P.Eng.	Original signed	Section 17.2
Wayne Barnett, Pr.Sci.Nat	Original signed	Sections 5 to 12 and 26.2

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CERTIFICATE OF QUALIFIED PERSON

Cameron C. Scott, P. Eng.

I, Cameron C. Scott, am a Professional Engineer, employed as a Principal Geotechnical Engineer with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Minto Phase V Preliminary Feasibility Study Technical Report" with an effective date of December 15, 2010 ("Technical Report").

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia and the Association of Professional Engineers of Yukon, amongst others. I graduated with a B.A.Sc. Degree in Geological Engineering granted by the University of British Columbia in 1974 and an M.Eng. Degree in Civil Engineering (Geotechnical Option) granted by the University of Alberta in 1984.

I have practiced my profession continuously since 1974 and, over this period, have been involved in the geotechnical, geoenvironmental and waste management aspects of mining projects throughout the mine life cycle, i.e. from scoping, pre-feasibility and feasibility studies, through detailed design and construction, to closure planning and closure implementation.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I visited the Minto Project site on 29 and 30 October 2008.

I am responsible for Sections 18.4.3 and 26.4 of the Technical Report.

I am independent of Minto Explorations Ltd. and Capstone Mining Corp. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Minto Project since approximately 2006. The initial involvement, in July and August 2006, consisted of a review of the tailings and water dam design on behalf of the Yukon Territorial Government. In the first half of 2008, on behalf of Minto, I was responsible for the detailed investigation of the foundation conditions at the proposed location of the southwest waste rock dump. In the first quarter of 2009, again on behalf of Minto, I was involved with others in a failure modes and effects analysis for the tailings storage facility. And in mid 2009, I was involved in the assessment and remediation design of an overburden slide in the south wall of the Main Pit on behalf of Minto.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

ORIGINAL SIGNED AND STAMPED
FEBRUARY 24, 2011

Ausenco Solutions Canada Inc.
855 Homer Street
Vancouver, BC V6B 2W2
Canada

T +1 778 374 6999
F +1 778 374 6998
W www.ausenco.com

To accompany the report entitled, "Minto Phase V Preliminary Feasibility Study Tech
effective date of December 15, 2010 ("Technical Report")..

I, David Brimage, MAusIMM, do hereby certify that:

1. I am Manager Process for Ausenco Solutions Canada Inc. 855 Homer Street, Vancouver, BC V6B 2W2, Canada.

2. I graduated with a degree in Metallurgical Engineering (Metallurgy) from the University of South Australia in 1993.

3. I am a member of AusIMM.

4. I have worked as a Metallurgist continuously since my graduation from University. For the past 15 years I have been employed with Ausenco Minerals and Metals. During this period I have fulfilled roles as senior process engineer, principal process engineer, engineering manager, and am currently employed as Manager Process.

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 education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of this NI 43-101.

6. I have participated in the preparation of Sections 15, 19, 24.1.3, 24.2.3, 26.1 and 27.1 of this technical report.

7. I have not visited the site.

8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Minto Explorations Ltd. and Capstone Mining Corp., or any associated or affiliated entities.

9. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Minto Explorations Ltd. and Capstone Mining Corp., or any associated or affiliated companies.

10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Minto Explorations Ltd. and Capstone Mining Corp., or any associated or affiliated companies.

11. I have read National Instrument 43-101 and Form 43-101F1, and confirm that the Technical Report has been prepared in compliance with that instrument and form.

12. To the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make this technical report not misleading.

Dated this 24th day of February

– Original Signed –

"D.J. Brimage"
David John Brimage, MAusIMM
Manager Process

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vancouver@srk.com
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CERTIFICATE OF QUALIFIED PERSON

Dino Pilotto, P.Eng.

I, Dino Pilotto, am a Professional Engineer, employed as a Principal Consultant - Mining with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Minto Phase V Preliminary Feasibility Study Technical Report" with an effective date of December 15, 2010 ("Technical Report").

I am a member of the Association of Professional Engineers and Geoscientists of Saskatchewan and Alberta. I graduated with a B.A.Sc. (Mining & Mineral Process Engineering) from the University of British Columbia in May 1987.

I have practiced my profession continuously since June 1987. I have been involved with mining operations, mine engineering and consulting covering a variety of commodities at locations in North America, South America, Eastern Europe, and Africa.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have visited the Minto Project site several times in the past four years, most recently on September 21-22, 2009.

I am responsible for open pit mine engineering aspects of Sections 16.7, 16.8, 18.2, 18.3.1 to 18.3.3, 18.4.1 to 18.4.2, 24.1.2, 24.2.2 and 26.3 of the "Minto Phase V Preliminary Feasibility Study Technical Report" with an effective date of December 15, 2010.

I am independent of Minto Explorations Ltd. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Minto Project since 2006 participating in various independent studies and reviews.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

ORIGINAL SIGNED

____
Dino Pilotto, P.Eng. Dated: February 24, 2011

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CERTIFICATE OF QUALIFIED PERSON

Gordon Doerksen, P.Eng.

I, Gordon Doerksen, am a Professional Engineer, employed as a Principal Consultant - Mining with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Minto Phase V Preliminary Feasibility Study Technical Report" with an effective date of December 15, 2010 ("Technical Report").

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I graduated with a BS (Mining) degree from Montana College of Mineral Science and Technology in May 1990.

I have been involved in mining since 1985 and have practised my profession continuously since 1990. I have been involved in mining operations, mine engineering and consulting covering a wide range of mineral commodities in Africa, South America, North America and Asia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have visited the Minto Mine site several times in the past three years, most recently on September 21-22, 2009.

I am responsible for the Executive Summary and Sections 1 to 4, 14, 17, 20 to 23, all parts of 24, 26 and 27 of the Technical Report.

I am independent of Minto Explorations Ltd. and Capstone Mining Corp. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Minto Mine since 2006 participating and managing various independent studies and reviews.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"ORIGINAL SIGNED AND STAMPED"

Gordon Doerksen, P.Eng.   Dated: February24, 2011

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Tucson 520.544.3688

CERTIFICATE OF QUALIFICATION

I, Garth David Kirkham, do hereby certify that:

1) I am a consulting geoscientist with an office at 6331 Palace Place, Burnaby, BC, V5E 1Z6.

2) This certificate applies to the "Minto Phase V Preliminary Feasibility Study Technical Report" with an effective date of December 15, 2010.

3) I am a graduate of the University of Alberta in 1983 a B. Sc. in Geophysics. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of Alberta, the Association of Professional Engineers and Geoscientists of BC, and the Northwest Territories and Nunavut Association of Engineers and Geoscientists. I have continuously practiced my profession performing computer modelling since 1988, both as an employee of a geostatistical modelling and mine planning software and consulting company and as an independent consultant.

4) I have not visited the property.

5) In the independent report titled "Minto Phase V Preliminary Feasibility Study Technical Report" with an effective date of December 15, 2010, I am responsible for the Sections 16.5 and 16.6.

6) I have been involved with the Minto Mine as an author of the previous technical Report titled "Minto Phase IV Preliminary Feasibility Study Technical Report"

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

8) 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 and that this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

9) I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

"ORIGINAL SIGNED AND STAMPED"

____

Garth Kirkham, P.Geo. Dated: February 24, 2011

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CERTIFICATE OF QUALIFIED PERSON

Iouri Iakovlev, P.Eng.

I, Iouri Iakovlev, am a Professional Engineer, employed as a Senior Mining Consultant with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Minto Phase V Preliminary Feasibility Study Technical Report" with an effective date of December 15, 2010 ("Technical Report").

I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #: 32213). I am a graduate of the Siberian State Industrial University, Novokuznetsk, Russia (Mining Engineer, 1983).

I have practiced my profession for more than 20 years. I have been involved in mining operations, mine engineering and consulting covering a wide range of mineral commodities in North and South America, South Africa, Europe and Asia.

As a result of my education, affiliation with a professional association and relevant work experience, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have not visited the Minto Mine.

I am responsible for the preparation of Sections: 16.9, 18.1, 18.3.4, 24.1.1 and 24.2.1 of the Technical Report.

I am independent of Minto Explorations Ltd. and Capstone Mining Corp. as independence is described by Section 1.4 of NI 43-101.

I have no prior involvement with the Property that is the subject of the Technical Report.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"ORIGINAL SIGNED AND STAMPED"

Iouri Iakovlev, P.Eng. Dated: February 24, 2011

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CERTIFICATE OF QUALIFIED PERSON

Marek Nowak, P.Eng.

I, Marek Nowak, am a Professional Engineer, employed as a Principal Consultant - Geostatistics with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Minto Phase V Preliminary Feasibility Study Technical Report" with an effective date of December 15, 2010 ("Technical Report").

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I have a Master of Science degree from the University of Mining and Metallurgy, Cracow, Poland, and a Master of Science degree from the University of British Columbia, Vancouver, Canada

I have over 25 years of experience in the mining industry, as a mining engineer (in Poland), geologist and geostatistician (in Canada). I specialize in natural resource evaluation and risk assessment using a variety of geostatistical techniques. I have co-authored several independent technical reports on base and precious metals exploration and mining projects in Canada, and United States.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have not visited the Minto Mine site and relied on the site visit completed by other authors of this report.

I am responsible for Sections 13 and 16.1 to 16.4 of the Technical Report.

I am independent of Minto Explorations Ltd. and Capstone Mining Corp. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Minto Mine since 2006 participating and managing various independent studies and reviews.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"ORIGINAL SIGNED AND STAMPED"

Marek Nowak, P.Eng.   Dated: February 24, 2011

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CERTIFICATE OF QUALIFIED PERSON

Michael Levy, P.E., P.G.

I, Michael E Levy, am a Professional Engineer, employed as a Senior Geotechnical Engineer with SRK Consulting Inc.

This certificate applies to the technical report titled "Minto Phase V Preliminary Feasibility Study Technical Report" with an effective date of December 15, 2010 ("Technical Report").

I am a registered Professional Engineer in the states of Colorado (#40268) and California (#70578) and a registered Professional Geologist in the state of Wyoming (#3550). I graduated with a B.Sc. in Geology from the University of Iowa in 1998 and a M.Sc. in Civil-Geotechnical Engineering from the University of Colorado in 2004. I have practiced my profession continuously since March 1999 and have been involved in a variety of surface an underground geotechnical projects specializing in advanced analyses and design of soil and rock slopes for mining projects.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have visited the Minto Mine site several times in the past four years, most recently on January 8-10, 2011.

I am responsible for the Executive Summary and Sections 17.1 and 27.3 of the Technical Report.

I am independent of Minto Explorations Ltd. and Capstone Mining Corp. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Minto Mine since 2006 participating and managing various independent studies and reviews.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"ORIGINAL SIGNED AND STAMPED"

Michael E. Levy, P.E, P.G.

Dated: February 24, 2011

Group Offices:	Canadian Offices:		U.S. Offices:	
Africa	Saskatoon	306.955.4778	Anchorage	907.677.3520
Asia	Sudbury	705.682.3270	Denver	303.985.1333
Australia	Toronto	416.601.1445	Elko	775.753.4151
Europe	Vancouver	604.681.4196	Fort Collins	970.407.8302
North America	Yellowknife	867.445.8670	Reno	775.828.6800
South America			Tucson	520.544.3688

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Canada

vancouver@srk.com
www.srk.com

Tel: 604.681.4196
Fax: 604.687.5532

CERTIFICATE OF QUALIFIED PERSON

Scott P. Carlisle, P. Eng.

I, Scott Carlisle, P.Eng., do hereby certify that:

1. I am a Professional Engineer, employed as a Principal Consultant – Mining and Rock Mechanics with SRK Consulting (Canada) Inc.
2. I graduated with a BS Mining degree from University of Utah in May 1979. In addition I have obtained a MS Mining/ Rock Mechanics degree from the University of Idaho in 1987.
3. I am a member of the Canadian Institute of Mining (CIM), Society for Mining, Metallurgy, and Exploration (SME), and registered as a Professional Engineer in Idaho, Wyoming and with the Professional Engineers of Ontario (PEO).
4. I have worked as a mining engineer and rock mechanics specialist since my graduation 1979 (32 years). I have been involved in rock mechanics, mining operations, tunnelling, mine engineering and consulting covering a wide range of mineral commodities and ground in North America.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
6. I am responsible for the Underground Rock Mechanics Section 17.2 of the Technical Report titled “Minto Phase V Preliminary Feasibility Study Technical Report” with an effective date of December 15, 2010. I visited the Minto Mine October 5-7, 2010.
7. I have not been involved with the Minto Mine prior to this visit and report.
8. 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.
9. I am independent of Minto Explorations Ltd. and Capstone Mining Corp. as independence is described by Section 1.4 of NI 43-101.
10. I have read the NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

“ORIGINAL SIGNED AND STAMPED”

Scott P. Carlisle, P. Eng.

Dated: February 24, 2011

Group Offices:
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Europe
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South America

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CERTIFICATE OF QUALIFIED PERSON

Wayne Barnett, Pr.Sci.Nat.

I, Wayne Barnett, am a Professional Natural Scientist, employed as a Principal Geologist with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Minto Phase V Preliminary Feasibility Study Technical Report" with an effective date of December 15, 2010 ("Technical Report").

I am a member of the South African Council for Natural Scientific Professions, South Africa. I graduated with a geology honours degree from the University of Cape Town in 1996, and a doctorate degree from the University of Kwa-Zulu Natal in 2006.

I have been involved in mining and have practised my profession continuously since 1997. I have been involved in mining geology, exploration geology, geological modelling and estimation covering a wide range of mineral commodities in Africa, Australia, South America, North America and Asia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have visited the Minto Mine site on the 4-6 March, 2009.

I am responsible for Sections 5 to 12 and 26.2 of the Technical Report.

I am independent of Minto Explorations Ltd. and Capstone Mining Corp. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Minto Mine since 2009 undertaking geological modelling and estimation.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"ORIGINAL SIGNED AND STAMPED"

Wayne Barnett, Pr.Sci.Nat. Dated: February 24, 2011

Group Offices:	Canadian Offices:		U.S. Offices:	
Africa	Saskatoon	306.955.4778	Anchorage	907.677.3520
Asia	Sudbury	705.682.3270	Denver	303.985.1333
Australia	Toronto	416.601.1445	Elko	775.753.4151
Europe	Vancouver	604.681.4196	Fort Collins	970.407.8302
North America	Yellowknife	867.445.8670	Reno	775.828.6800
South America			Tucson	520.544.3688

Appendix A

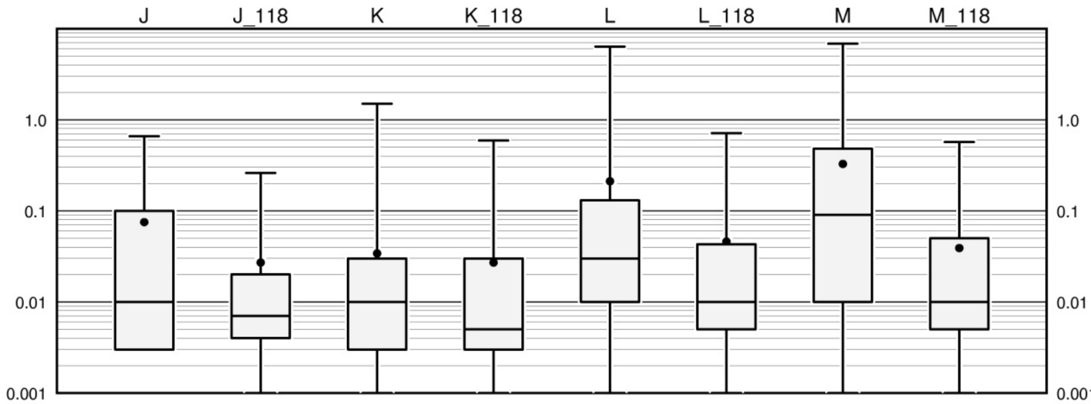
Statistics of Gold and Silver Assays

and

Variogram Models of Gold Grades

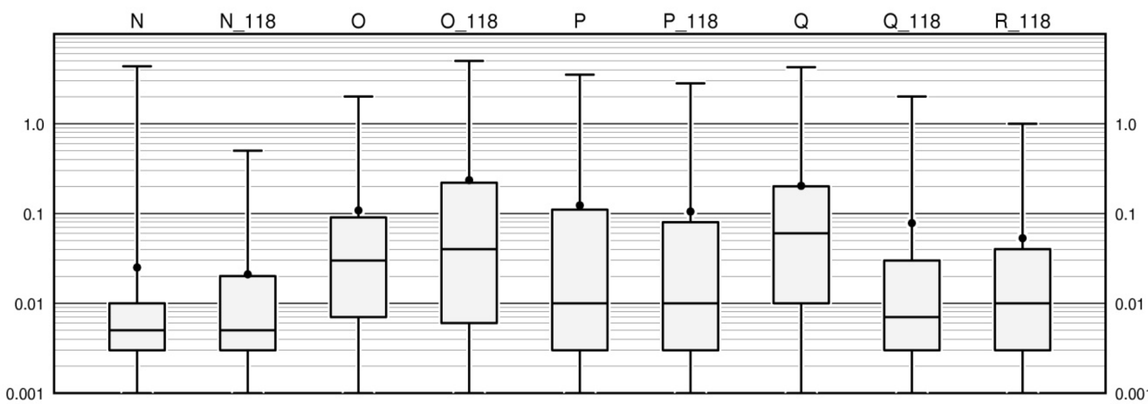
Area 2/118 Deposit

Au Declustered Composites



	J	J_118	K	K_118	L	L_118	M	M_118	
Number of data	224	140	585	342	1218	373	1891	463	Number of data
Mean	0.075	0.027	0.034	0.027	0.211	0.046	0.327	0.039	Mean
Std. Dev.	0.129	0.047	0.096	0.059	0.548	0.094	0.522	0.062	Std. Dev.
Coef. of Var.	1.732	1.77	2.837	2.151	2.598	2.023	1.596	1.611	Coef. of Var.
Maximum	0.66	0.26	1.5	0.59	6.35	0.71	6.84	0.57	Maximum
Upper quartile	0.1	0.02	0.03	0.03	0.13	0.043	0.48	0.05	Upper quartile
Median	0.01	0.007	0.01	0.005	0.03	0.01	0.09	0.01	Median
Lower quartile	0.003	0.004	0.003	0.003	0.01	0.005	0.01	0.005	Lower quartile
Minimum	0.003	0.001	0.001	0.001	0.001	0.001	0.001	0.001	Minimum

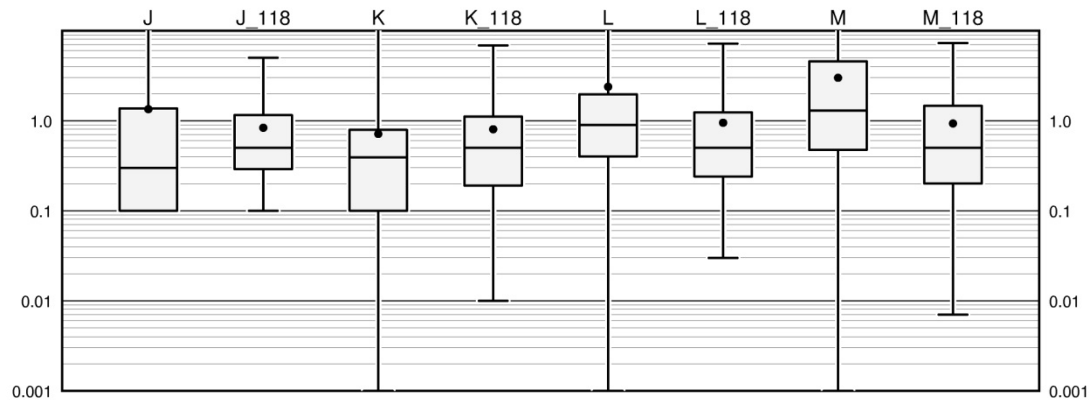
Au Declustered Composites



	N	N_118	O	O_118	P	P_118	Q	Q_118	R_118	
Number of data	4032	546	943	666	1328	374	908	207	319	Number of data
Mean	0.025	0.021	0.108	0.235	0.123	0.105	0.203	0.078	0.053	Mean
Std. Dev.	0.157	0.053	0.24	0.532	0.296	0.305	0.412	0.242	0.125	Std. Dev.
Coef. of Var.	6.308	2.485	2.223	2.258	2.41	2.923	2.024	3.104	2.352	Coef. of Var.
Maximum	4.35	0.5	2.0	5.0	3.5	2.81	4.24	2.0	1.0	Maximum
Upper quartile	0.01	0.02	0.09	0.22	0.11	0.08	0.2	0.03	0.04	Upper quartile
Median	0.005	0.005	0.03	0.04	0.01	0.01	0.06	0.007	0.01	Median
Lower quartile	0.003	0.003	0.007	0.006	0.003	0.003	0.01	0.003	0.003	Lower quartile
Minimum	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	Minimum

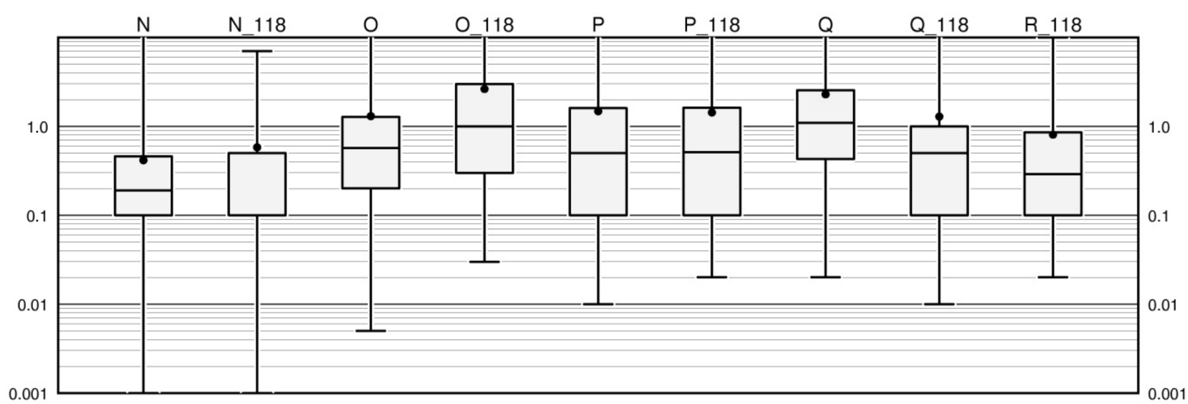
Area 2/118 Deposit

Ag Declustered Composites



	J	J_118	K	K_118	L	L_118	M	M_118	
Number of data	224	140	585	342	1218	373	1891	463	Number of data
Mean	1.342	0.834	0.712	0.803	2.381	0.949	2.993	0.931	Mean
Std. Dev.	2.807	0.823	1.356	0.883	5.17	1.138	3.747	0.937	Std. Dev.
Coef. of Var.	2.091	0.987	1.903	1.099	2.172	1.199	1.252	1.006	Coef. of Var.
Maximum	19.88	5.0	15.0	6.87	65.37	7.19	30.3	7.3	Maximum
Upper quartile	1.37	1.15	0.79	1.11	1.96	1.24	4.57	1.46	Upper quartile
Median	0.3	0.5	0.39	0.5	0.895	0.5	1.29	0.5	Median
Lower quartile	0.1	0.29	0.1	0.19	0.4	0.24	0.472	0.2	Lower quartile
Minimum	0.1	0.1	0.001	0.01	0.001	0.03	0.001	0.007	Minimum

Ag Declustered Composites



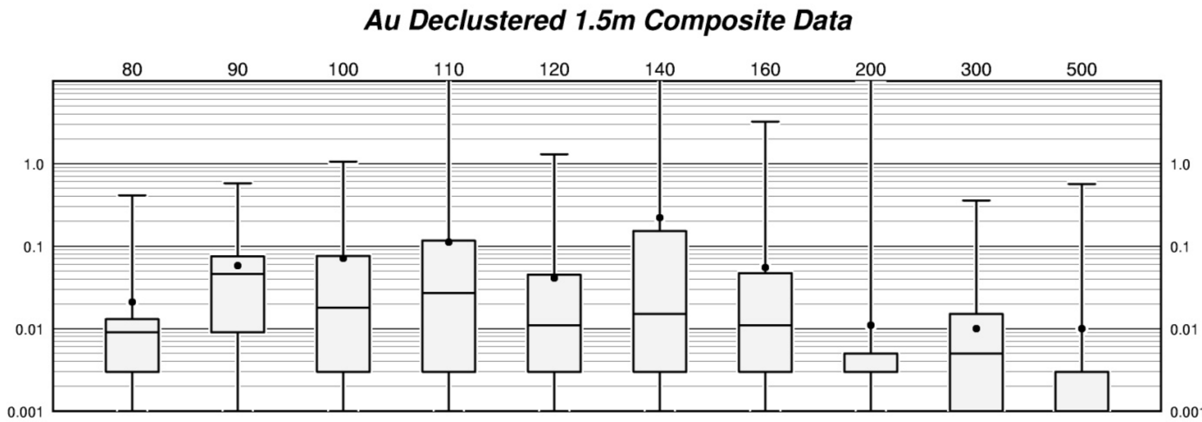
	N	N_118	O	O_118	P	P_118	Q	Q_118	R_118	
Number of data	4032	546	943	666	1328	374	908	207	319	Number of data
Mean	0.415	0.58	1.301	2.626	1.476	1.43	2.29	1.287	0.805	Mean
Std. Dev.	1.111	0.789	2.341	4.439	2.981	2.786	3.771	2.991	1.37	Std. Dev.
Coef. of Var.	2.679	1.361	1.799	1.69	2.02	1.948	1.646	2.325	1.702	Coef. of Var.
Maximum	29.53	7.0	20.0	35.81	35.0	25.0	40.0	25.0	10.0	Maximum
Upper quartile	0.46	0.5	1.278	3.0	1.6	1.62	2.545	1.0	0.855	Upper quartile
Median	0.19	0.5	0.57	1.0	0.5	0.51	1.09	0.5	0.29	Median
Lower quartile	0.1	0.1	0.2	0.3	0.1	0.1	0.43	0.1	0.1	Lower quartile
Minimum	0.001	0.001	0.005	0.03	0.01	0.02	0.02	0.01	0.02	Minimum

Area 2/118 Deposit – Variogram Models of Gold Grades

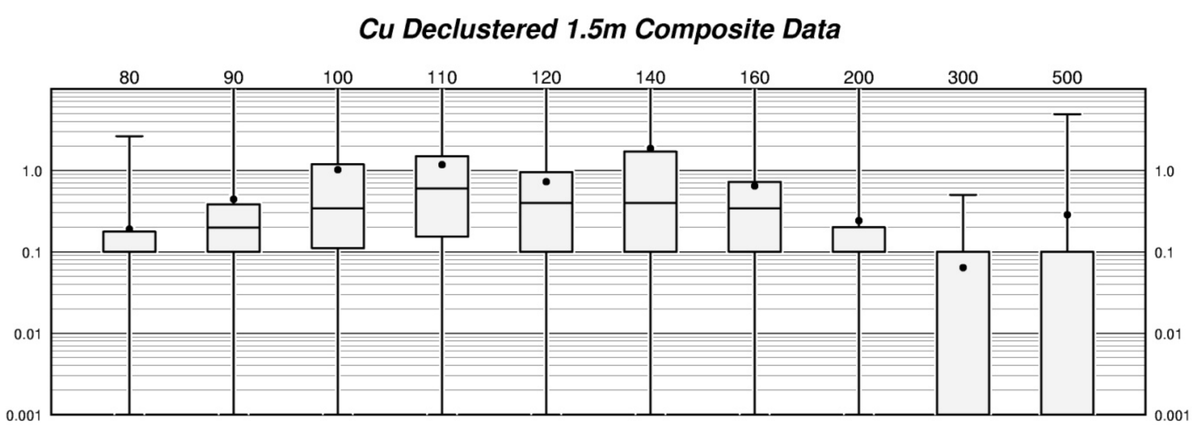
Zone	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
J	0.05	0.50	60	-15	0	100	30	12
		0.45				130	60	17
K	0.45	0.45	45	15	0	25	35	3
		0.10				100	150	8
L	0.10	0.45	45	15	0	45	100	25
		0.45				230	120	45
M	0.15	0.45	100	18	-37	70	70	25
		0.40				400	300	45
N	0.10	0.45	45	15	0	20	40	10
		0.45				50	100	25
O	0.17	0.45	45	15	0	50	100	15
		0.38				60	140	30
P	0.10	0.45	45	15	0	25	25	20
		0.45				350	350	60
Q	0.35	0.45	0	0	0	80	70	15
		0.20				180	120	30

Zone	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
J118	0.15	0.45	45	20	0	30	40	8
		0.40				60	120	10
K118	0.15	0.45	45	30	0	30	40	8
		0.40				60	120	10
L118	0.25	0.45	60	30	0	25	35	8
		0.30				70	100	30
M118	0.25	0.45	60	30	0	25	35	8
		0.30				70	100	30
N118	0.20	0.45	-30	75	-15	8	15	15
		0.35				30	70	70
O118	0.25	0.45	90	30	0	100	80	15
		0.30				430	200	30
P118	0.05	0.45	90	30	0	50	50	5
		0.50				100	100	15
Q118	0.05	0.45	0	-60	30	5	50	50
		0.50				15	100	100
R118	0.05	0.45	30	15	0	50	50	5
		0.50				100	100	15

Ridgetop Deposit



	80	90	100	110	120	140	160	200	300	500	
Number of data	169	262	837	1301	1048	1112	1219	7626	133	689	Number of data
Mean	0.021	0.058	0.071	0.112	0.041	0.221	0.055	0.011	0.01	0.01	Mean
Std. Dev.	0.047	0.07	0.137	0.29	0.082	1.122	0.176	0.102	0.019	0.033	Std. Dev.
Coef. of Var.	2.232	1.198	1.946	2.583	1.989	5.065	3.186	9.373	1.826	3.233	Coef. of Var.
Maximum	0.411	0.577	1.048	16.087	1.297	34.851	3.2	10.361	0.356	0.565	Maximum
Upper quartile	0.013	0.075	0.076	0.116	0.045	0.152	0.047	0.005	0.015	0.003	Upper quartile
Median	0.009	0.046	0.018	0.027	0.011	0.015	0.011	0.003	0.005	0.001	Median
Lower quartile	0.003	0.009	0.003	0.003	0.003	0.003	0.003	0.003	0.001	0.001	Lower quartile
Minimum	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	Minimum



	80	90	100	110	120	140	160	200	300	500	
Number of data	169	262	837	1301	1048	1112	1219	7626	133	689	Number of data
Mean	0.19	0.442	1.018	1.174	0.725	1.859	0.644	0.241	0.064	0.285	Mean
Std. Dev.	0.25	1.072	1.85	1.786	1.135	4.055	1.287	0.57	0.068	0.86	Std. Dev.
Coef. of Var.	1.315	2.425	1.818	1.521	1.566	2.182	1.999	2.363	1.063	3.015	Coef. of Var.
Maximum	2.618	30.275	77.44	23.712	12.908	38.9	24.272	15.471	0.5	4.9	Maximum
Upper quartile	0.177	0.384	1.187	1.498	0.953	1.7	0.721	0.2	0.1	0.1	Upper quartile
Median	0.1	0.198	0.34	0.6	0.399	0.4	0.34	0.1	0.1	0.001	Median
Lower quartile	0.1	0.1	0.111	0.153	0.1	0.1	0.1	0.1	0.001	0.001	Lower quartile
Minimum	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	Minimum

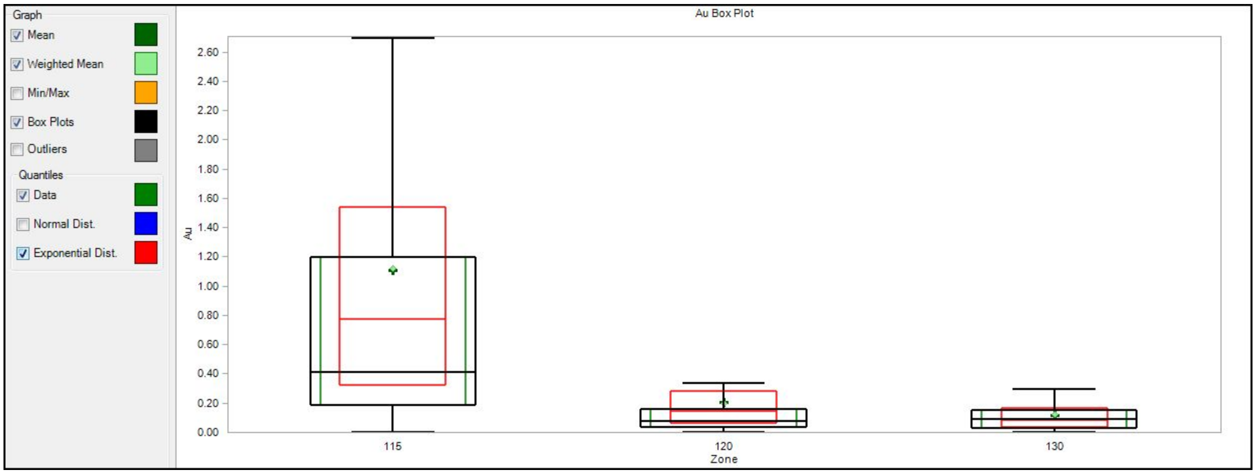
Ridgetop Deposit – Variogram Models of Gold Grades

Domain	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
80*	0.05	0.70	50	24	-48	35	25	18
		0.25				260	120	22
90*	0.05	0.70	50	24	-48	35	25	18
		0.25				260	120	22
100	0.05	0.70	50	24	-48	35	25	18
		0.25				260	120	22
110	0.05	0.40	50	24	-45	10	15	20
		0.55				50	140	25
120	0.10	0.40	50	24	-48	45	20	8
		0.50				100	70	12
140	0.05	0.50	50	24	-45	35	45	8
		0.45				150	250	25
160	0.05	0.35	50	24	-48	10	40	14
		0.60				70	100	15

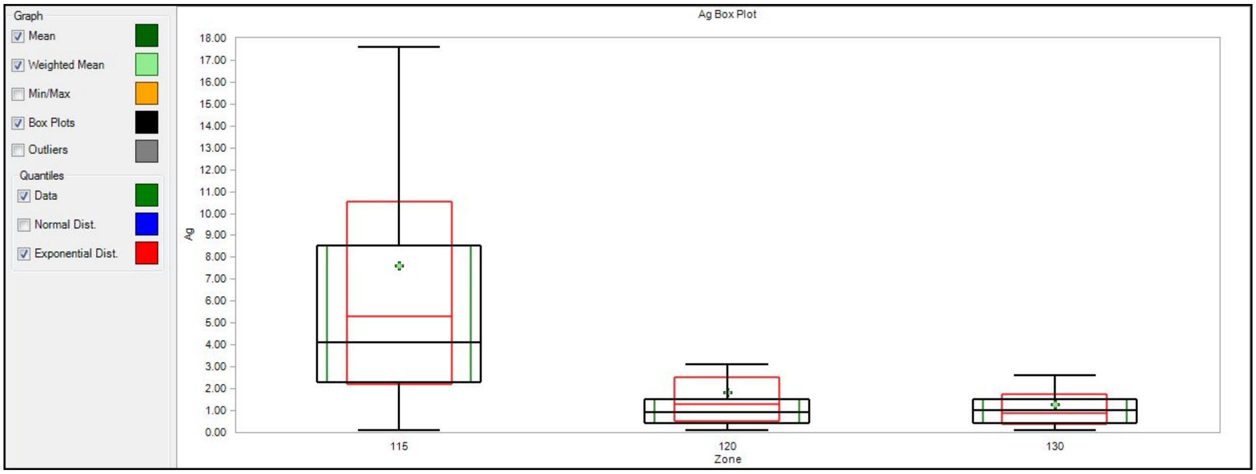
* Variogram models assigned from Domain 100

Note: Exponential variogram models have been used with practical ranges of continuity

Minto North Deposit



AU			
ZONE	**115**	**120**	**130**
#Samples	1081	430	83
Min	0.00	0.00	0.00
Max	72.18	16.60	0.64
Mean	1.11	0.20	0.12
First quartile	0.18	0.03	0.03
Median	0.41	0.08	0.09
Third quartile	1.20	0.16	0.15
SD	3.14	0.85	0.12
Variance	9.86	0.73	0.01
CV	2.83	4.22	1.00



AG			
ZONE	**115**	**120**	**130**
#Samples	1081	430	83
Min	0.10	0.10	0.10
Max	110.20	54.70	7.20
Mean	7.61	1.82	1.26
First quartile	2.30	0.40	0.40
Median	4.10	0.90	1.00
Third quartile	8.50	1.50	1.50
SD	10.11	4.72	1.36
Variance	102.19	22.29	1.84
CV	1.33	2.60	1.07

Minto North Deposit – Variogram Models of Gold and Silver Grades

	AU			**AG**		
Nugget (C0)	0.22			0.14		
C1	0.78			0.86		
	Range	Rotation	Angle	Range	Rotation	Angle
Major	60	R1	37	80	R1	115
Minor	30	R2	-11	60	R2	20
Vertical	37	R3	12	10	R3	-16