# Minto Phase IV
# Pre-Feasibility Technical Report

***Prepared for:***


**Minto Explorations Ltd.**
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***Vancouver BC, V6C 2W2***


***Prepared by:***


**SRK Consulting**
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Project No. 2CM022.06

**December 15, 2009**

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# Executive Summary

## Introduction

Minto Explorations Ltd. ("MintoEx"), a wholly owned subsidiary of Capstone Mining Corp. ("Capstone"), 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. This pre-feasibility and technical report was compiled for MintoEx by SRK Consulting (Canada) Inc. ("SRK") to describe a new mineral resource and reserve estimate and describe the new life-of-mine plan with cost and plant capacity improvements.

A preliminary feasibility study and technical report ("2007 PFS") 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 additional mineral resources are described in this report, and form the basis of the life-of-mine plan. Exploration on the Minto property is ongoing, diamond drilling is currently suspended for the season but is planned to start again in early 2010 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 early 2010 in order to further increase production and modify operating parameters to accommodate other proposed operational improvements, as well as incorporate the mining of the Area 2, Area 118, Ridgetop and Minto North deposits.

## Geology and Exploration

The Minto Project is found in 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 while the most recently discovered deposit to be reported is the Minto North deposit located about 700 m north of the 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 forms sub-horizontal horizons and can be traced laterally for more than 1000 m in the drill core. They are often stacked in parallel to sub-parallel sequences and it is postulated that the foliated granodiorite represent healed, shallowly dipping shear zones within the Granite Mountain Batholith, that are 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 batholiths to eventual exhumation. However, there is on-going debate as per the stratigraphic, intrusive or structural nature of the zones hosting the foliation and mineralization, and MintoEx have engaged the Mineral Deposits Research Unit 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 our 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 and biotite mineralization. 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 centimeters 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 modeled 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 northeastern boundary of the Ridgetop deposit, and defines the outcrop of Cretaceous conglomerates.

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 Main deposit, now the producing open pit Minto mine, 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 1000 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 2009 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 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 PEC in future there will likely be more reliance 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 Minto East 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 Minto East prospect in late 2009

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.

## Resources

A primary objective of SRK's work was to produce a revised independent resource evaluation for the Area2/118 and for the Ridgetop deposits. The Minto North Zone, another integral part of the Minto Deposit, has been evaluated by Kirkham Geosystems Ltd (Kirkham Geosystems).

The mineral resource evaluation reported herein supersedes earlier resource estimates prepared by LGGC in 2008 and reported in the SRK Technical Report, June 2008.

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 this term is defined in National Instrument 43-101. The effective date of this resource estimate is June 1, 2009. Marek Nowak, P.Eng., analyzed the data, reviewed and validated the mineral resource estimates. The Minto North deposit resource estimate was completed by Garth Kirkham, P.Geo., of Kirkham Geosystems, 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 global mineral resources at Area2/Area 118, Ridgetop, and Minto North 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.

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 Geosystems audited the Minto North 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. In order to meet this requirement, SRK considers that the Area 2/118, Ridgetop, and Minto North deposits are amenable for open pit extraction.

In order to constrain the overall mineral resource to demonstrate reasonable prospects for economic extraction, for the Area 2/118, and Ridgetop deposits the mineral resources are 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. At Minto North, a project at its relatively early stage of exploration, global resources have been reported. The mineral resource statements for the Area2/118, Ridgetop, and Minto North are presented in Tables 1-3. A combined resource from all three deposits is presented in Table 4.

**Table 1: Mineral Resource Statement at 0.5% Cu Cut-off for the Area 2/118 Deposit, SRK Consulting June 9, 2009**

| Classification | Tonnes (Kt)* | Copper (%) | Gold (g/t) | Silver (g/t) | Contained Copper (K lb.)* | Contained Gold (K oz)* | Contained Silver (K oz)* |
|---|---|---|---|---|---|---|---|
| Measured (M) | 6,936 | 1.25 | 0.47 | 4.29 | 190,638 | 104 | 956 |
| Indicated (I) | 11,301 | 0.92 | 0.29 | 3.36 | 230,198 | 106 | 1,220 |
| **Sub-total (M+I)**** | **18,237** | **1.05** | **0.36** | **3.71** | **420,836** | **210** | **2,176** |
| **Inferred** | **5,116** | **0.91** | **0.24** | **2.99** | **102,420** | **40** | **492** |

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

**Table 2: Mineral Resource Statement at 0.5% Cu Cut-off for the Ridgetop Deposit, SRK Consulting June 9, 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) | 1,568 | 0.98 | 0.26 | 2.12 | 33,719 | 13 | 107 |
| Indicated (I) | 2,355 | 0.98 | 0.33 | 3.30 | 50,926 | 25 | 250 |
| **Sub-total (M+I)**** | **3,923** | **0.98** | **0.30** | **2.83** | **84,645** | **38** | **357** |
| **Inferred** | **686** | **0.90** | **0.26** | **2.38** | **13,644** | **6** | **53** |

**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 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** |

**Table 4: Combined Mineral Resource Statement at 0.5% Cu Cut-off for Area 2/118, Ridgetop, and Minto North Deposits, 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) | 10,348 | 1.37 | 0.55 | 4.57 | 311,887 | 183 | 1,519 |
| Indicated (I) | 13,920 | 0.94 | 0.30 | 3.39 | 287,179 | 136 | 1,519 |
| **Sub-total (M+I)**** | **24,267** | **1.12** | **0.41** | **3.89** | **599,066** | **319** | **3,038** |
| **Additional Inferred** | **5,827** | **0.91** | **0.25** | **2.93** | **116,520** | **46** | **548** |

**This table excludes the Minto Main Deposit mineral resource*

## Mine Production and Mineral Reserve Estimate

The Area 2, 118, Ridgetop and Minto North ("Phase IV") deposits are proposed to be developed as open pits following completion of mining in the Minto Main deposit. The planning for this Pre-feasibility study assumes a start date of January 1, 2010. The proposed Main Pit mine plan (as provided by MintoEx) was incorporated into this pre-feasibility study.

Based on a start date of January 2010, the Main/Phase IV mine will produce a total of 10.9 million tonnes (Mt) of ore (includes Main Pit stockpile balance at end of 2009) and 70.4 Mt of waste over approximately an 8-year mine operating life ending in early 2018.

The 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 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 IV pits was initiated with the development of a Net Smelter Return ("NSR") model. The model included estimates of metal prices ($2.00/lb Cu price used), 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 IV. Gemcom Whittle™ software was then used to determine the optimal mining shells for each of the deposits. Detailed mine planning and scheduling was then conducted on the optimal pit shells to produce the current pit designs used in the mineral reserves estimate summarized in Table 5 below. The mineral reserve for Main Pit includes the ore stockpile balance predicted for the end of 2009 as well as proposed mining from 2010 going forward. The various estimated copper cut-off grades used within the planned pits are noted in Table 5.

**Table 5: Minto – Mineral Reserves by Class for Main/Phase IV**

| Deposit | Reserve Class | Tonnes ('000s) | Cut-off Grade (%Cu equiv.) | Diluted grade (%Cu) | Diluted grade (g/t Au) | Diluted grade (g/t Ag) | Contained Metal Cu (Mlb) | Contained Metal Au (oz) | Contained Metal Ag (oz) |
|---|---|---|---|---|---|---|---|---|---|
| Main Pit | Proven | 3,920 | 0.62 | 1.64 | 0.58 | 6.51 | 142 | 72 | 820 |
| | Probable | 206 | 0.62 | 1.20 | 0.45 | 5.25 | 5 | 3 | 35 |
| | Sub-total | 4,126 | 0.62 | 1.62 | 0.57 | 6.45 | 147 | 75 | 855 |
| North Pit | Proven | 1,346 | 0.55 | 2.50 | 1.37 | 9.04 | 74 | 59 | 391 |
| | Probable | 3 | 0.55 | 2.91 | 1.07 | 13.11 | 0 | 0 | 1 |
| | Sub-total | 1,349 | 0.55 | 2.50 | 1.37 | 9.05 | 74 | 60 | 393 |
| Ridgetop Pit | Proven | 802 | 0.58 | 1.17 | 0.31 | 2.33 | 21 | 8 | 60 |
| | Probable | 522 | 0.58 | 1.39 | 0.50 | 4.90 | 16 | 8 | 82 |
| | Sub-total | 1,324 | 0.58 | 1.26 | 0.38 | 3.34 | 37 | 16 | 142 |
| Area2/118 Pit | Proven | 3,707 | 0.56 | 1.56 | 0.59 | 5.36 | 127 | 71 | 639 |
| | Probable | 387 | 0.56 | 1.09 | 0.19 | 2.79 | 9 | 2 | 35 |
| | Sub-total | 4,094 | 0.56 | 1.51 | 0.56 | 5.12 | 137 | 73 | 674 |
| **Total** | **Proven** | **9,775** | **0.58** | **1.69** | **0.67** | **6.08** | **364** | **211** | **1,911** |
| | **Probable** | **1,118** | **0.58** | **1.25** | **0.38** | **4.26** | **31** | **14** | **153** |
| | **Total** | **10,893** | **0.58** | **1.64** | **0.64** | **5.89** | **395** | **224** | **2,064** |

The post-2009 mining sequence was divided into eight stages. The first stage sees the completion of mining in the Main Pit followed by Minto North, the two stages in Ridgetop, Area 118 and finally three stages in Area 2. 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 IV pits are most economical when mined in sequence with the stripping of the Phase IV pits beginning near the completion of mining in the current or Main Pit. Waste rock will be placed in the valley fill dumps to the west and tailings from Phase IV will be placed in the mined out Main Pit. The LOM mine production schedule is shown in Table 6 with the processing schedule summarized in Table 7.

## Table 6: LOM Mine Production Schedule

| Parameter | Units | Total | Year | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | 2010 | 2011 | 2012 | 2013 | 2014 | 2015 | 2016 | 2017 |
| | | | Main Pit | | Phase IV Pits | | | | | |
| **Mining** | | | | | | | | | | |
| Ore | Mt | **10.0** | 2.0 | 1.3 | 0.3 | 1.4 | 1.2 | 1.4 | 1.3 | 1.1 |
| Overburden | Mt | **16.9** | 4.9 | 3.4 | 2.3 | 1.2 | 1.6 | 1.0 | 1.9 | 0.7 |
| Waste Rock | Mt | **53.5** | 3.3 | 3.0 | 7.1 | 6.0 | 8.6 | 7.9 | 9.7 | 8.0 |
| Total Waste | Mt | **70.4** | 8.2 | 6.3 | 9.4 | 7.2 | 10.2 | 8.9 | 11.6 | 8.6 |
| Total Material | Mt | **80.4** | 10.2 | 7.6 | 9.7 | 8.6 | 11.4 | 10.3 | 12.9 | 9.8 |
| Strip ratio | Wt:Ot | **7.0** | 4.1 | 5.0 | 33.2 | 5.1 | 8.6 | 6.3 | 8.7 | 7.6 |
| Daily production | Kt/day | **27.5** | 27.8 | 20.9 | 26.4 | 23.5 | 31.1 | 28.3 | 35.3 | 26.8 |
| Mined Cu grade | % | **1.66** | 1.71 | 1.59 | 1.20 | 2.43 | 1.28 | 1.42 | 1.42 | 1.80 |
| Mined Au grade | g/t | **0.65** | 0.52 | 0.67 | 0.50 | 1.24 | 0.43 | 0.51 | 0.51 | 0.73 |
| Mined Ag grade | g/t | **5.93** | 7.04 | 6.23 | 2.27 | 8.71 | 3.76 | 5.23 | 4.48 | 6.00 |
| Mined Contained Cu | Mlbs | **367** | 74 | 45 | 7 | 75 | 33 | 44 | 42 | 45 |
| Mined Contained Au | Koz | **210** | 33 | 28 | 5 | 56 | 16 | 23 | 22 | 27 |
| Mined Contained Ag | Koz | **1,912** | 447 | 257 | 21 | 394 | 143 | 238 | 192 | 221 |

## Table 7: LOM Process Production Schedule

| Parameter | Units | Year | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Total | 2010 | 2011 | 2012 | 2013 | 2014 | 2015 | 2016 | 2017 | 2018 |
| | | | Main Pit | | Phase IV Pits | | | | | | |
| **Processing** | | | | | | | | | | | |
| Processed Ore | Mt | **10.9** | 1.2 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 0.1 |
| Process rate | dmt/day | **3,704** | 3,334 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 |
| Proc. Cu grade | % | **1.64** | 2.33 | 1.68 | 1.10 | 2.47 | 1.22 | 1.44 | 1.40 | 1.64 | 0.81 |
| Proc. Au grade | g/t | **0.64** | 0.80 | 0.67 | 0.35 | 1.27 | 0.40 | 0.52 | 0.50 | 0.65 | 0.25 |
| Proc. Ag grade | g/t | **5.89** | 9.84 | 6.48 | 3.64 | 8.88 | 3.66 | 5.32 | 4.44 | 5.52 | 2.67 |
| **Recovery** | | | | | | | | | | | |
| Copper | % | **92.8** | 94.0 | 94.0 | 93.6 | 92.0 | 92.3 | 92.0 | 92.0 | 92.4 | 92.0 |
| Gold | % | **73.8** | 80.0 | 80.0 | 77.9 | 70.0 | 71.3 | 70.0 | 70.2 | 71.8 | 70.0 |
| Silver | % | **81.3** | 86.7 | 86.7 | 84.9 | 78.0 | 79.1 | 78.0 | 78.2 | 79.6 | 78.0 |
| **Metal in Concentrates** | | | | | | | | | | | |
| Copper | Mlbs | **366** | 59 | 48 | 31 | 69 | 34 | 40 | 39 | 46 | 1 |
| Gold | oz | **164,814** | 24,961 | 23,470 | 12,163 | 39,168 | 12,529 | 16,028 | 15,594 | 20,407 | 494 |
| Silver | oz | **1,684,688** | 333,701 | 247,310 | 136,463 | 304,882 | 127,345 | 182,541 | 153,122 | 193,450 | 5,874 |

In order to assess the opportunity of potential large scale open pits and their potential impact on future permitting requirements, a preliminary study was conducted where an optimistic copper price and lower operating costs were used to understand these potential pit limits. 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. 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.

Exploration on the Minto project has historically been focused on finding near-surface deposits conducive to open pit mining. In the course of exploration, several deeper deposits have been discovered that may provide an opportunity to add mill feed material using underground mining methods. Both deep penetrating geophysical surveys and core drilling have provided some preliminary definition of deposits below 150 m in depth, and these deposits and targets may be amenable to underground exploitation.

### 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 end of 2011). Upon completion of mining in the Main Pit, thickened tailings generated from processing ores from other Phase IV pits will then be deposited into the Main Pit.

This plan is not yet permitted but offers a potentially viable solution to tailings disposal that provides backfill material for the Main Pit, reduces the amount disturbed land that would normally be required by mining of the Phase IV pits, and provides a significant cost savings over the current dry-stack method.

Waste rock from the current open pit 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. Phase IV waste rock is proposed to be placed in an adjacent Central Valley Fill waste dump.

# Mineral Processing

## Metallurgical Test Work

The mineralogy is relatively coarse grained and test work 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 by G&T Metallurgical Services Ltd. on Minto North, Ridgetop East and Area 118 in 2009 have demonstrated performance consistent with the current Main Pit ore flotation characteristics.

## 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 80% of the SAG feed material ($F_{80}$) being finer than 25 mm. An average of 3,750 tonnes per day will be processed at a design availability of 91.3%.

## Process Plant Capital Cost

The total process plant capital cost to facilitate the increase in plant throughput to a nominal 4,100 tonnes per day, or 3,750 tpd after allowances for availability, is C$9.1 million. This estimate has an overall accuracy of ±25% as of the fourth quarter 2009. This estimate excludes capital cost associated with the mine and associated infrastructure, water supply, access roads or tailings storage facility. This capital cost is exclusive of equipment purchased by MintoEx to date and therefore none of this capital cost is expected to be incurred before the end of 2009.

## Process Plant Operating Cost

The process plant operating cost for the plant upgrade based on an annualised throughput of 1,368,837 tonnes was calculated to be C$12.79/t. This operating cost was estimated at an accuracy of ±25% as of the fourth quarter 2009.

## Process Plant Design Risks and Opportunities

Risks associated with the project include:

- The secondary crusher (S4800) installed by MintoEx does not facilitate screening of the feed material prior to the cone 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 crushed ore product size ($P_{80}$) of 25 mm. 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:

- An opportunity exists to install a scalping screen prior to the secondary crusher. This will improve the overall operation and throughput of the crushing circuit.
- 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 communtion 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:
- The cost quoted for a new VTM300 concentrate re-grind mill was approximately C$1.2 million. A second hand VTM200 was identified at the time of the Pre-feasibility Study at a cost of around C$ 0.29 million.
- 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 during the Phase V pre-feasibility study.

## Conceptual Design In-pit Tailings Disposal

Using a spreadsheet-based tailings solids and surface water balance model, SRK has developed a conceptual design for the subaqueous disposal of 7.7 million tonnes of tailings in the Main Pit. Additional capacity is 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.

The design is based on the construction of a 2.1 million cubic metre divider embankment between the Main and Area 2 Pits so that tailings can continue to be contained within the Main Pit once the residual ridge crest between the two pits, at approximately elevation 766 m amsl, is exceeded. As a minimum a starter embankment will be required, followed by multiple stages of embankment raises in approximately 10-m increments.

Subaqueous deposition methods will be used with the expectation that slurry deposition would be performed from variable locations around the pit perimeter and within the pit “basin” to facilitate uniform distribution of tailings and avoid the formation of a “peak and valley” tailings surface.

It has been assumed that the excess water within the pit will be limited to a maximum depth of 10 m. This will be achieved by pumping from a floating barge located in the northeast quadrant of the pit. The pumping capacity will be sufficient 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). The excess water associated with the annual freshet will require treatment prior to discharge.

Seepage through the embankment (and potentially the pit sidewalls) can be controlled through embankment design and construction, tailings management (pre-sliming) and vertical dewatering wells.

## 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) has also been assessed under the current regime, the Yukon Environmental and Socioeconomic Assessment Board (YESAB). The project is currently (November 2009) entering the assessment process again for water management and mining and milling rate amendments to the major authorizations.

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 will require environmental assessment under YESAA and major licence amendments. Water management planning is expected to be of particular interest to the assessors given recent issues at the site.

### 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. 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.

In 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 recently 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.

Currently an updated Environmental Conditions report is in preparation to support the Phase IV development that updates all environmental data for the project area and will be used for the assessment and permitting processes.

### 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 developments, the plan will require further reassessment during the Phase IV development planning process.

The critical consideration with respect to water management for Phase IV 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 IV expansion, as it is in the currently proposed water management plan.

### Closure Planning

Closure philosophies and measures for the Phase IV 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 IV 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 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 IV 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 IV 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 IV 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 4 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, and metal leaching from bulk waste is not expected to be environmentally significant. Mineralized waste has elevated concentrations of copper, and care should be taken to ensure that mineralized waste in placed randomly with bulk waste to prevent the development of local ‘hot spots’ within the larger mass of bulk waste rock that lead to leaching of environmentally-significant quantities of copper.

## Economics

The estimated economic benefit of mining the Minto Phase IV deposits is sufficient to take the project to the next level. 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 8 presents a summary of the operating costs by major area, while Table 9 summarizes the capital costs. Table 9 shows the capital costs without closure costs. A closure cost allowance of $20M was used in the cash flow analysis, however, the end of mine life closure cost remains to be estimated once the requirements are defined. Table 10 shows the comparison of Phase IV PFS Base and Alternate Cases. The Phase IV deposits add economic benefit to the mine, yielding a Base Case pre-tax Net Present Value at a 7.5% discount rate (“$NPV_{7.5\%}$”) of $199 m. The Alternate Case models yield a substantial improvement in the project economics due to higher metal prices base on current forward projections.

### Table 8: Operating Costs by Major Area

| Area | C$/t |
|---|---|
| Mining (C$/t moved) | 2.31 |
| Mining (C$/t ore) | 17.02 |
| Processing | 13.90 |
| General, administration, camp, royalties | 11.94 |
| **Total** | **42.86** |

### Table 9: Capital Costs by Major Area

| Area | C$ millions |
|---|---|
| Plant Expansion | 9.1 |
| Open pit mining equipment | 33.7 |
| **Sub-total** | **42.8** |
| Sustaining Capital | 5.4 |
| **Life-of-mine capital** | **48.2** |

### Table 10: Comparison Phase IV Base and Alternate Cases

| Item | Unit | Phase IV PFS Base Case | Phase IV PFS Case 2 | Phase IV PFS Case 3 |
|---|---|---|---|---|
| Waste mined | Mtonnes | 70.4 | 70.4 | 70.4 |
| Ore mined | Mtonnes | 10.0 | 10.0 | 10.0 |
| Total mined | Mtonnes | 80.4 | 80.4 | 80.4 |
| Strip ratio | W:O | 7.0 | 7.0 | 7.0 |
| Mill Feed* | Ktonnes | 10.9 | 10.9 | 10.9 |
| Copper millhead grade | % Cu | 1.64% | 1.64% | 1.64% |
| Gold millhead grade | g/t Au | 0.64 | 0.64 | 0.64 |
| Silver millhead grade | g/t Ag | 5.9 | 5.9 | 5.9 |
| Copper in cons | Mlb | 366 | 366 | 366 |
| Gold in cons | Koz | 166 | 166 | 166 |
| Silver in cons | Koz | 1,685 | 1,685 | 1,685 |
| Concentrate Grade | % Cu | 40% | 40% | 40% |
| Copper Price (inc. hedging) | US$/lb | $2.25 | $2.55 | $2.90 |
| Gold price (inc. hedging) | US$/oz | $300.00 | $300.00 | $300.00 |
| Silver price (inc. hedging) | US$/oz | $3.90 | $3.90 | $3.90 |
| Exchange rate | US$/C$ | $0.91 | $0.91 | $0.91 |
| NSR | C$/t milled | $75 | $86 | $99 |
| Unit Mining Costs | $/t mined | $2.31 | $2.31 | $2.31 |
| Unit Mining Costs | $/t milled | $17.02 | $17.02 | $17.02 |
| Unit Total OPEX (inc royalties) | $/t milled | $42.86 | $42.92 | $42.98 |
| Unit On-site OPEX (inc. royalties) | US$/lb Cu payable | $1.20 | $1.20 | $1.20 |
| Unit Off-site OPEX | US$/lb Cu payable | $0.29 | $0.29 | $0.29 |
| Unit By-product Credit | US$/lb Cu payable | $0.15 | $0.15 | $0.15 |
| Unit OPEX net by-product credits | US$/lb Cu payable | $1.34 | $1.34 | $1.34 |
| Total Capital (initial & sustaining) | $M | $48 | $48 | $48 |
| Allowance for closure cost | $M | $20 | $20 | $20 |
| $NPV_{7.5\%}$ pre-tax | $M | $199 | $291 | $395 |
| $NPV_{7.5\%}$ after tax | $M | $160 | $218 | $281 |

*Note Mill Feed includes Ore Stockpile

Base case sensitivity analyses were run for Cu grade, Cu price, capital expense (“CAPEX”), and operating expense (“OPEX”). Each variable was changed from -20% to +20% of the base case value and the resultant $NSR_{7.5\%}$ values were graphed (Figure 1). Each variable was changed independently of the other variables so there is no compounding effect of multiple variable modifications.

The results show the project is most sensitive to Cu grade followed closely by Cu prices. Normally grade and metal price affects are equal but in Minto's case, the Cu price is hedged for some of the production so the effect of Cu price is tempered with some metal price certainty.

Most of Minto's costs are in Canadian dollars but metal prices and Minto's metal purchase agreement are in US dollars. This commercial situation makes the project sensitive to the US$:C$ exchange rate. For this study, an exchange rate of C$1.10: US$1.00 was selected based on a historical average relationship between C$ to US$ exchange ratio and copper price at US$2.25/lb of copper.

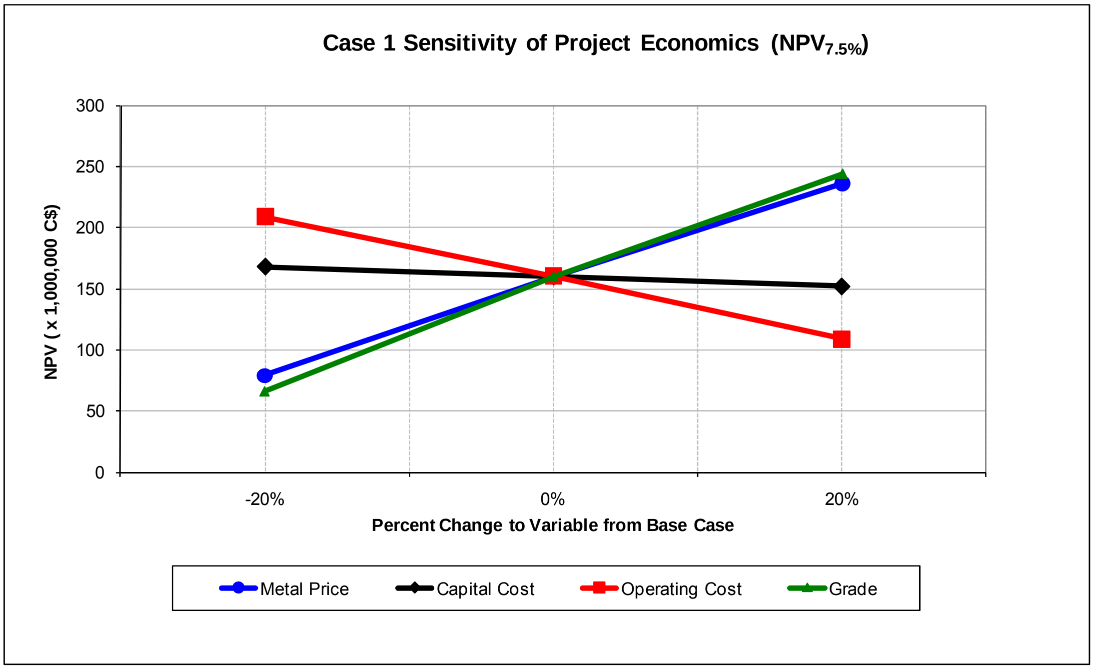


**Figure1: Base Case Pre-tax NPV$_{7.5\%}$ Sensitivities**

## Conclusions

The conclusions of note are:

- The Minto deposit, encompassing Main Pit and Phase IV pits (Area 2, North, Area 118 and Ridgetop), 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 further measure of confidence in the reserve estimate.
- The Phase IV deposits are estimated to be economic to exploit 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 IV pits and management has demonstrated its ability and commitment to explore for new deposits
- Based on test work conducted to date, the Phase IV waste rock does not appear to have any ARD issues.

The major risk areas identified in this study are:

- Timing and approval of mine permit revisions;
- Exchange rates, metal prices and external influences;

- Grade control.

The most important opportunities to improve the project are:

- Optimization of mine plan;
- Underground production potential, bringing ex-pit high grade feed to the mill relatively early in the mine life. A conceptual level review was completed for an alternative to the Phase V plant upgrade, that involves underground extraction of higher grade ore, eliminating the need for further plant expansions and allowing processing of higher grade ore sooner than in a open pit scenario.
- Conversion of inferred resources to higher classifications for reduction of strip ratios
- Discovering new mineral resources and mineral reserves

## 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;
- 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.

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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 and describe the new life-of-mine plan with cost and plant capacity improvements.

Personal visits to the Minto Mine were conducted by five of the seven Qualified Persons ("QPs") shown in Table 1.1.

**Table 1.1: QP Site Visits**

| Name of QP | Area Reviewed |
|---|---|
| Gordon Doerksen | Mine, dumps, tailings and general site |
| Dino Pilotto | Mine, dumps, tailings and general site |
| Cam Scott | Waste Dumps and Tailings |
| Mike Levy | Main Pit |
| Wayne Barnett | Geology |
| Clint Donkin | *No site visit* |
| Garth Kirkham | *No site visit* |

The following SRK employees are the QP under National Instrument 43-101 responsible for this project: Wayne Barnett, P.Eng. – Geology as well as resource estimates for Area 2/118 and Ridgetop; Cam Scott, P.Eng. – Waste dumps and Tailings Impoundments; Mike Levy, P.E. – Geotechnical; Dino Pilotto, P.Eng. – Mining and Reserves; Gordon Doerksen, P.E. – Project Overview.

Clint Donkin of Ausenco is the QP for the metallurgical plant design, capital and operating cost estimates. Clint has not been to the Minto site but relied upon the information gathered by Paul Staples, Tim Doddridge and Derek Elwin of Ausenco during their recent site visit.

Garth Kirkham of Kirkham Geosystems is the QP for the resource estimate for Minto North.

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 actual operating costs incurred for the first half of 2009. 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$") 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:

- Vivienne McLennan of MintoEx for providing 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;
- 2010 Minto Main Mine operating budget and forecast, supplied by Jaime Delgado of MintoEx, was used, as appropriate, for the remaining reserves of Main Pit for contribution to section 18 of this report;
- Gordon McKnight provided independent advice on concentrate sales and marketing for contribution to Section 20 of this report;
- Corporate tax information specific to Minto was obtained from Wentworth Taylor, CA of W.H. Taylor Inc., an independent taxation specialist for contribution to Sections 23 and 25 of the report;
- Metallurgical testing conducted by G&T Metallurgical Services Ltd;
- Projections of mill throughput including comminution test work managed by DJB Consultants Inc;
- Comminution test work completed by SGS Canada Inc;
- Assessment of the current Minto grinding operation and SAGDesign tests completed by Starkey & Associates Inc.

Names of the authors and their contributing sections are included in Section 31.

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 authors 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 |
| Y 61699 | Quartz | DEF | 7 | 8/23/1971 | 10/7/2028 | Active | OW00236 | 500057713 |
| Y 61700 | Quartz | DEF | 8 | 8/23/1971 | 10/7/2028 | Active | OW00237 | 500057714 |
| Y 61701 | Quartz | DEF | 9 | 8/23/1971 | 10/7/2028 | Active | OW00238 | 500057715 |
| Y 61702 | Quartz | DEF | 10 | 8/23/1971 | 3/1/2013 | Active | | 500057716 |
| Y 61703 | Quartz | DEF | 11 | 8/23/1971 | 10/7/2028 | Active | OW00239 | 500057717 |
| Y 61704 | Quartz | DEF | 12 | 8/23/1971 | 3/1/2013 | Active | | 500057718 |
| Y 61705 | Quartz | DEF | 13 | 8/23/1971 | 10/7/2028 | Active | OW00240 | 500057719 |
| 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 |
| Y 61711 | Quartz | DEF | 19 | 8/23/1971 | 3/1/2013 | Active | | 500057725 |
| Y 61712 | Quartz | DEF | 20 | 8/23/1971 | 3/1/2013 | Active | | 500057726 |
| Y 61713 | Quartz | DEF | 21 | 8/23/1971 | 3/1/2013 | Active | | 500057727 |
| Y 61714 | Quartz | DEF | 22 | 8/23/1971 | 3/1/2013 | Active | | 500057728 |
| Y 61715 | Quartz | DEF | 23 | 8/23/1971 | 3/1/2013 | Active | | 500057729 |
| Y 61716 | Quartz | DEF | 24 | 8/23/1971 | 3/1/2013 | Active | | 500057730 |
| Y 61717 | Quartz | DEF | 25 | 8/23/1971 | 3/1/2013 | Active | | 500057731 |
| Y 61718 | Quartz | DEF | 26 | 8/23/1971 | 3/1/2013 | Active | | 500057732 |
| Y 61719 | Quartz | DEF | 27 | 8/23/1971 | 3/1/2013 | Active | | 500057733 |
| Y 61720 | Quartz | DEF | 28 | 8/23/1971 | 3/1/2013 | Active | | 500057734 |
| Y 61721 | Quartz | DEF | 29 | 8/23/1971 | 3/1/2013 | Active | | 500057735 |
| Y 61722 | Quartz | DEF | 30 | 8/23/1971 | 3/1/2013 | Active | | 500057736 |
| Y 61723 | Quartz | DEF | 31 | 8/23/1971 | 10/7/2028 | Active | OW00246 | 500057737 |
| Y 61724 | Quartz | DEF | 32 | 8/23/1971 | 10/7/2028 | Active | OW00247 | 500057738 |
| Y 61904 | Quartz | MINTO | 17 | 8/31/1971 | 5/13/2018 | Active | OW00017 | 500057910 |
| Y 61905 | Quartz | MINTO | 18 | 8/31/1971 | 5/13/2018 | Active | OW00018 | 500057911 |
| Y 61906 | Quartz | MINTO | 19 | 8/31/1971 | 3/1/2013 | Active | | 500057912 |
| Y 61907 | Quartz | MINTO | 20 | 8/31/1971 | 3/1/2013 | Active | | 500057913 |
| Y 61908 | Quartz | MINTO | 35 | 8/31/1971 | 5/13/2018 | Active | OW00021 | 500057914 |
| Y 61909 | Quartz | MINTO | 36 | 8/31/1971 | 5/13/2018 | Active | OW00022 | 500057915 |
| Y 61910 | Quartz | MINTO | 37 | 8/31/1971 | 3/1/2013 | Active | | 500057916 |
| Y 61911 | Quartz | MINTO | 38 | 8/31/1971 | 3/1/2013 | Active | | 500057917 |
| Y 61914 | Quartz | MINTO | 23 | 8/31/1971 | 3/1/2013 | Active | | 500057920 |
| Y 61915 | Quartz | MINTO | 24 | 8/31/1971 | 3/1/2013 | Active | | 500057921 |
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| Y 61917 | Quartz | MINTO | 26 | 8/31/1971 | 3/1/2013 | Active | | 500057923 |
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| Y 61923 | Quartz | MINTO | 34 | 8/31/1971 | 5/13/2018 | Active | OW00020 | 500057929 |
| Y 61926 | Quartz | MINTO | 41 | 8/31/1971 | 3/1/2013 | Active | | 500057932 |
| Y 61927 | Quartz | MINTO | 42 | 8/31/1971 | 3/1/2013 | Active | | 500057933 |
| Y 61928 | Quartz | MINTO | 43 | 8/31/1971 | 3/1/2013 | Active | | 500057934 |
| Y 61929 | Quartz | MINTO | 44 | 8/31/1971 | 3/1/2013 | Active | | 500057935 |
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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 61935 | Quartz | MINTO | 48 | 8/31/1971 | 5/13/2018 | Active | OW00026 | 500057941 |
| Y 61936 | Quartz | MINTO | 49 | 8/31/1971 | 5/13/2018 | Active | OW00027 | 500057942 |
| Y 61937 | Quartz | MINTO | 50 | 8/31/1971 | 5/13/2018 | Active | OW00028 | 500057943 |
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| Y 61939 | Quartz | MINTO | 52 | 8/31/1971 | 5/13/2018 | Active | OW00030 | 500057945 |
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| Y 61998 | Quartz | DEF | 53 | 9/8/1971 | 3/1/2013 | Active | | 500057978 |
| Y 61999 | Quartz | DEF | 54 | 9/8/1971 | 3/1/2013 | Active | | 500057979 |
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| Y 62016 | Quartz | DEF | 71 | 9/8/1971 | 3/1/2013 | Active | | 500057996 |
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| Y 62023 | Quartz | DEF | 78 | 9/8/1971 | 3/1/2013 | Active | | 500058003 |
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| Y 62298 | Quartz | MINTO | 67 | 9/22/1971 | 5/13/2018 | Active | OW00033 | 500058006 |
| Y 62299 | Quartz | MINTO | 68 | 9/22/1971 | 5/13/2018 | Active | OW00034 | 500058007 |
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| Y 62301 | Quartz | MINTO | 70 | 9/22/1971 | 5/13/2018 | Active | OW00035 | 500058009 |
| Y 62302 | Quartz | MINTO | 71 | 9/22/1971 | 5/13/2018 | Active | OW00036 | 500058010 |
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| Y 62311 | Quartz | MINTO | 81 | 9/22/1971 | 3/1/2013 | Active | | 500058019 |
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| Y 62313 | Quartz | MINTO | 83 | 9/22/1971 | 3/1/2013 | Active | | 500058021 |

| Grant Number | Reg Type | Claim Name | Claim No. | Operation Recording Date | Claim Expiry Date | Status | Quartz Lease | Ops Number |
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| Y 62314 | Quartz | MINTO | 84 | 9/22/1971 | 3/1/2013 | Active | | 500058022 |
| 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 |
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| Y 76955 | Quartz | DEF | 86 | 8/31/1973 | 3/1/2013 | Active | | 500058313 |
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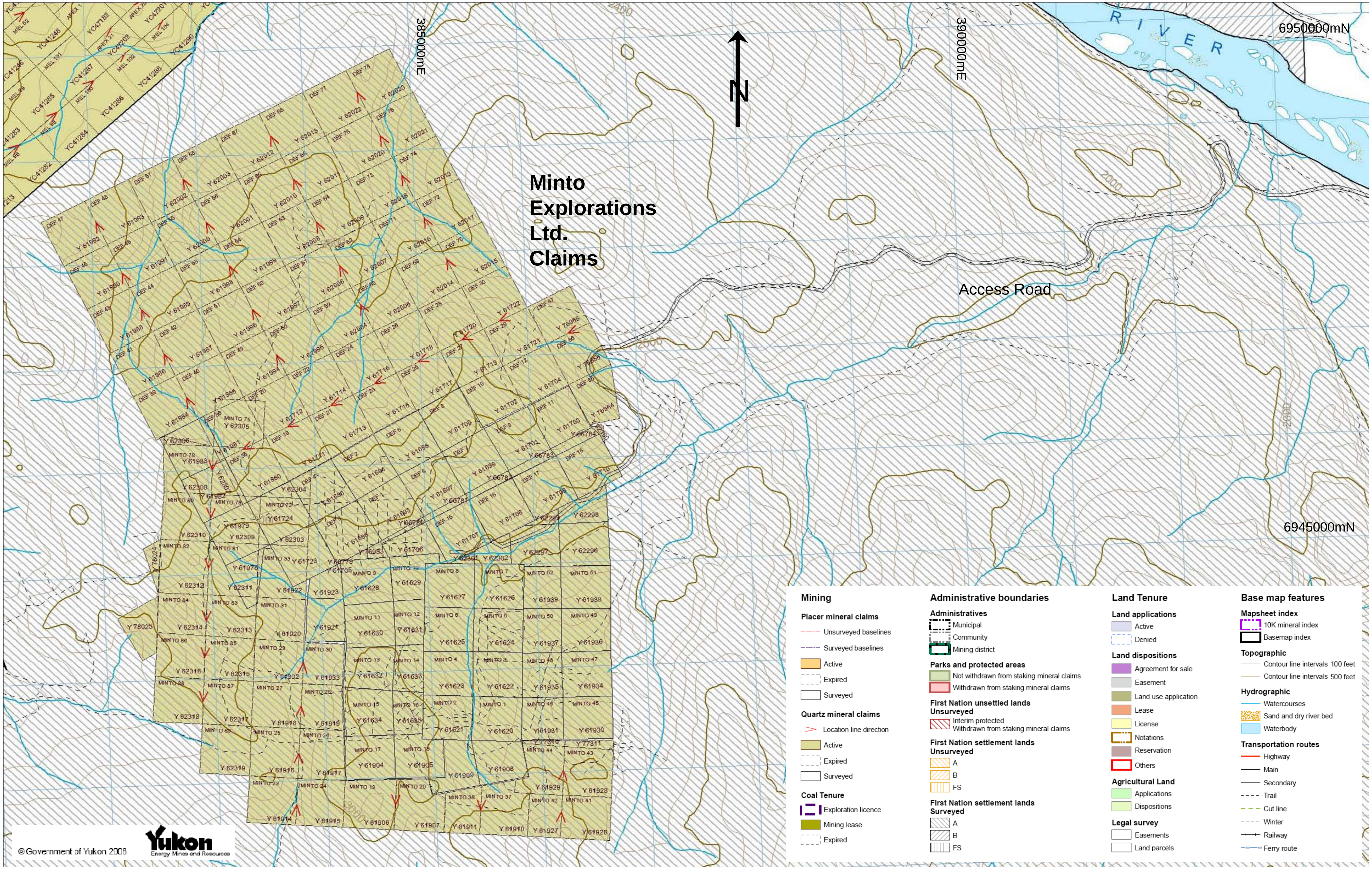
| Y 77311 | Quartz | MINTO | 95 | 10/1/1973 | 3/1/2013 | Active | | 500058316 |
| Y 78024 | Quartz | MINTO | 96 | 11/13/1973 | 3/1/2013 | Active | | 500058317 |
| Y 78025 | Quartz | MINTO | 97 | 11/13/1973 | 3/1/2013 | Active | | 500058318 |

*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.



Minto Explorations Ltd. Claims
Access Road
385000mE
390000mE
6950000mN
6945000mN
RIVER
N
Mining
Placer mineral claims
Unsurveyed baselines
Surveyed baselines
Active
Expired
Surveyed
Quartz mineral claims
Location line direction
Active
Expired
Surveyed
Coal Tenure
Exploration licence
Mining lease
Expired
Administrative boundaries
Administratives
Municipal
Community
Mining district
Parks and protected areas
Not withdrawn from staking mineral claims
Withdrawn from staking mineral claims
First Nation unsettled lands
Unsurveyed
Interim protected
Withdrawn from staking mineral claims
First Nation settlement lands
Unsurveyed
A
B
FS
First Nation settlement lands
Surveyed
A
B
FS
Land Tenure
Land applications
Active
Denied
Land dispositions
Agreement for sale
Easement
Land use application
Lease
License
Notations
Reservation
Others
Agricultural Land
Applications
Dispositions
Legal survey
Easements
Land parcels
Base map features
Mapsheet index
10K mineral index
Basemap index
Topographic
Contour line intervals 100 feet
Contour line intervals 500 feet
Hydrographic
Watercourses
Sand and dry river bed
Waterbody
Transportation routes
Highway
Main
Secondary
Trail
Cut line
Winter
Railway
Ferry route
©Government of Yukon 2008
Yukon
Energy, Mines and Resources

# 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 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 about 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, which 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 +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 1997, and has no old-growth trees remaining.

## 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, 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 2009 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. 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 YTD Oct. |
|---|---|---|---|---|
| Waste mining | Tonnes | 9,264,575 | 8,370,800 | 9,435,996 |
| Ore mining | Tonnes | 746,327 | 825,909 | 769,250 |
| Total material mined | Tonnes | 10,010,902 | 9,529,320 | 10,205,246 |
| Mined copper grade | % | 1.70 | 1.84 | 2.15 |
| Mined gold grade - estimated | g/t | 0.45 | 0.71 | 0.71 |
| Mined silver grade | g/t | 6.80 | 7.65 | 7.39 |
| Tonnes processed | Tonnes | 238,446 | 809,426 | 865,646 |
| Mill head copper grade | % | 2.16 | 2.91 | 2.47 |
| Mill head gold grade* | g/t | n/a | 1.28 | 0.91 |
| Mill head silver grade | g/t | 7.70 | 11.80 | 10.30 |
| Copper recovery | % | 85.1 | 91.9 | 92.8 |
| Gold recovery* | % | n/a | 77.7 | 72.9 |
| Silver recovery | % | 77.5 | 84.6 | 81.1 |
| Concentrate produced | Dmt | 12,630 | 53,148 | 47,918 |
| Concentrate grade – Cu | % | 34.7 | 40.7 | 41.4 |
| Concentrate grade – Au* | g/t | n/a | 15.9 | 13.4 |
| Concentrate grade – Ag | g/t | 113 | 152 | 155 |
| Copper in concentrate | K lb. | 9,661 | 47,686 | 43,448 |
| Gold in concentrate* | Oz | n/a | 27,202 | 20,887 |
| Silver in concentrate | Oz | 45,885 | 217,489 | 189,494 |

* 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 adapted and updated to describe recent events and information.*

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 2005-2009, 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 licence 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 licence 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 ;
- Dipole-dipole geophysical survey over northern regional targets;
- Titan 24 survey over the Minto Priority Exploration corridor;
- Drill discovery of the Minto East prospect.

## 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 five drill programs from 2005 to 2009 since Capstone's acquisition of MintoEx.

### 5.2.1 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.

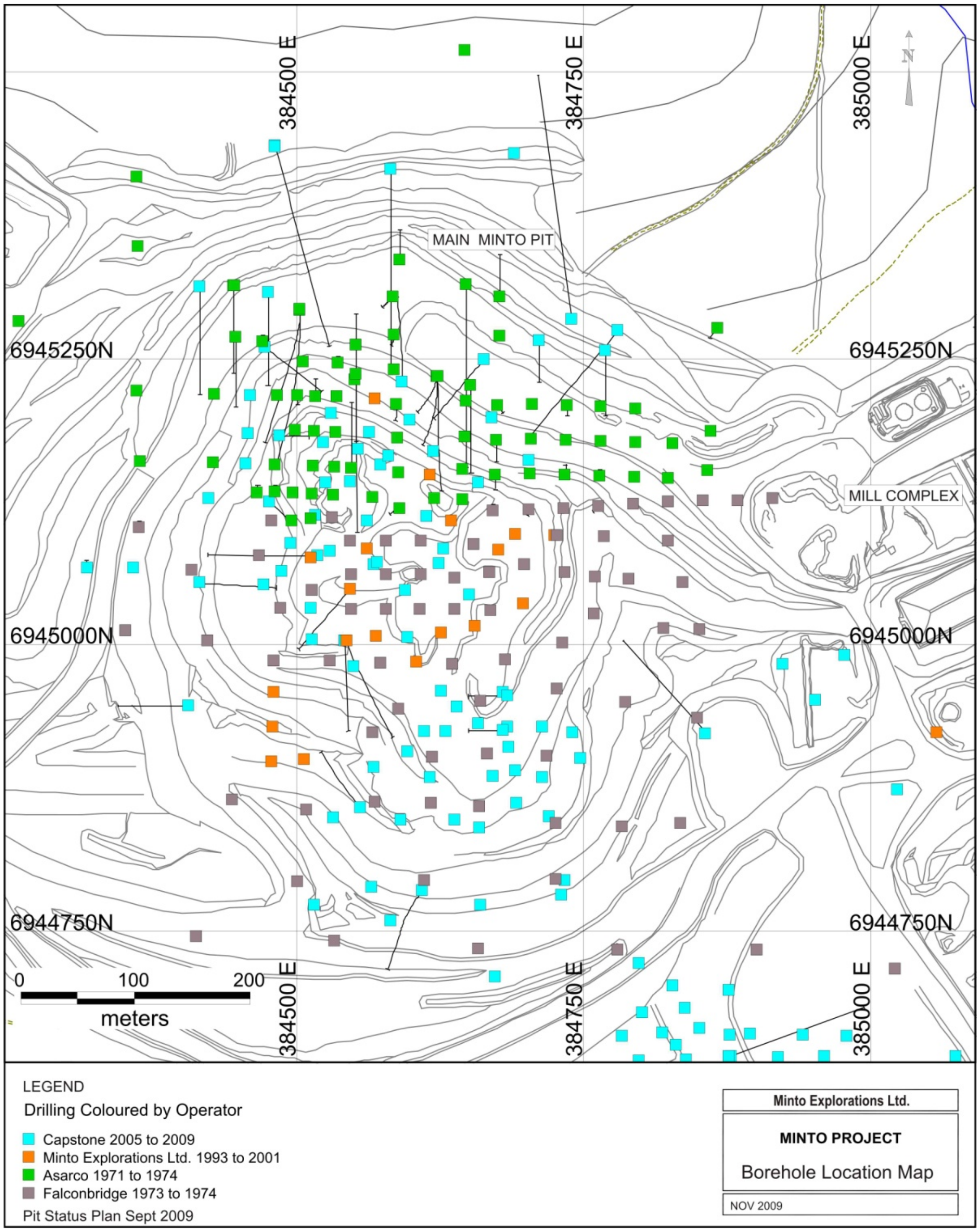



**Figure 5.1: Drill Hole Location Map – Minto Main Deposit**

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.

### 5.2.2 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 quartz-feldspathic 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**

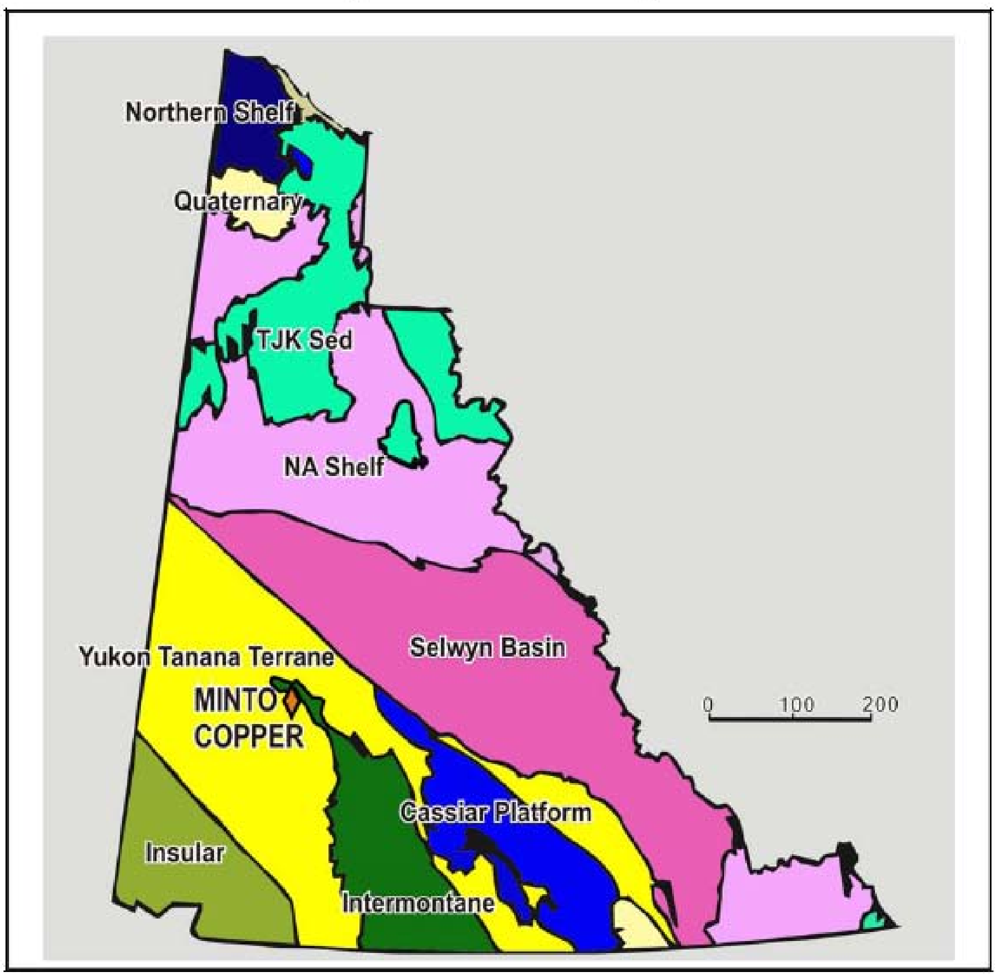
| Year | Source | Cut-off (%Cu) | Short Tons | Cu (%) | Au (oz/t) | Ag (oz/t) | Comments |
|---|---|---|---|---|---|---|---|
| 1976 | R.T Heard UKHM | unknown | 8,219,370 | 2.04 | | | |
| 1976 | L.A. Wigglesworth Falconbridge | unknown | 8,210,219 | 2.03 | | | |
| 1975 | R.J. Prevedi ASARCO | 0.60 | 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 5.3 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 terranes 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 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 18-20 km.

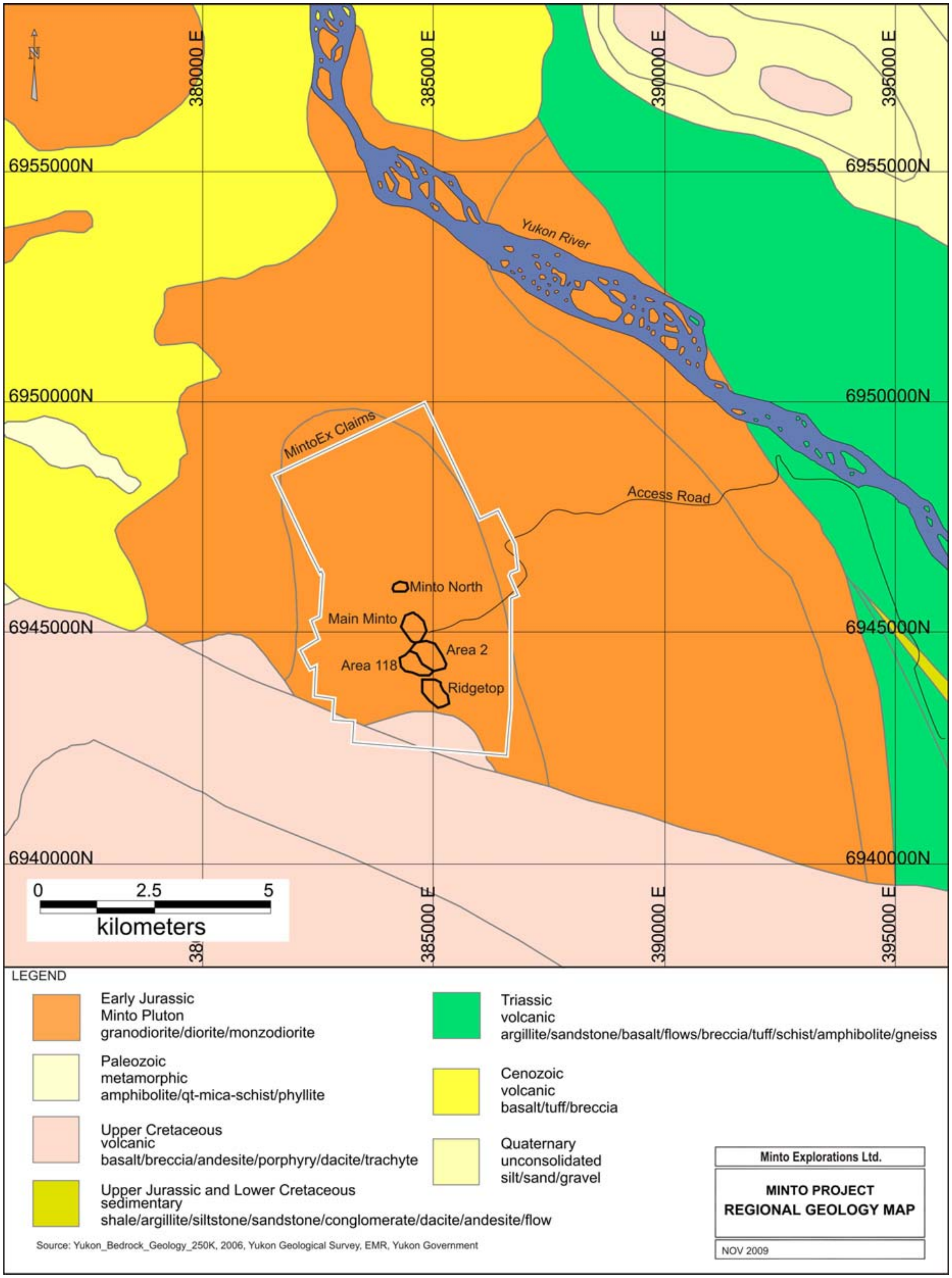



**Figure 6.2: Regional Geology**

## 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, due to deep weathering and oxidation of the exposed outcrop. The terrain was not glaciated during the last ice age event.

Four deposits of mineralization are reported in this document (Figure 6.2). Each of these deposit 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 Area 2/ 118 located immediately south of Main Minto. The Ridgetop deposit is located approximately 300 m south of the Area 2/118 deposit. The most recently discovered deposit to be reported is the Minto North deposit located approximately 700 m north of the Main Minto deposit. In addition to these mineral deposits which have NI43-101 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 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 an 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 folded 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 variously 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 gnessic textured granodiorite as a distinctly discernable unit. It is generally believed by MintoEx geologists that this 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 kilometers 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 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 pegmatititic/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 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 modelling 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 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).

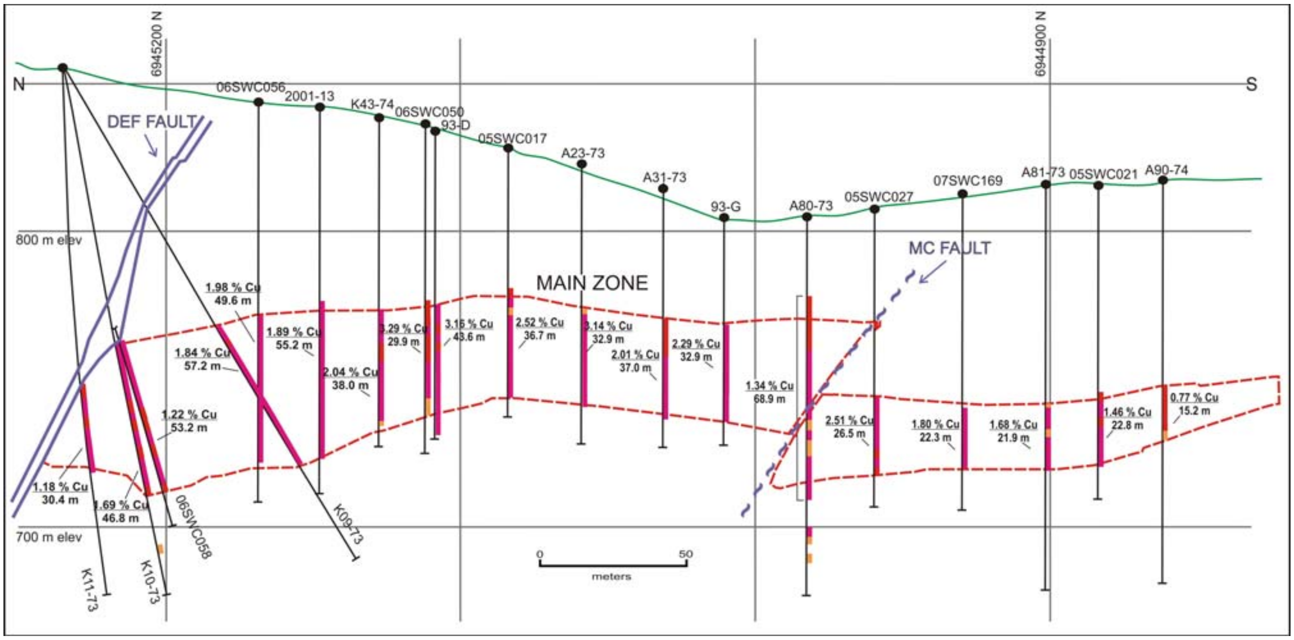



**Figure 6.3: North- South Cross Section through Minto Main Deposit showing DEF Fault and MC Fault**

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 degrees 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 “M” 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 conglomerates. 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 grandioritic pluton. MintoEx have engaged the Mineral Deposits Research Unit 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.

Some examples of IOCG mineralization the MintoEx geologists have been advised (in personal communications) exhibit some similar characteristics and setting to Minto include 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.

# 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 and biotite mineralization. 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 exhibit 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 Deposit also exhibits 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. Precious metal grades are elevated in the bornite zone, and visible gold has been observed on several occasions. Mineralization at Area 2 / 118 is distinct in that mineralization is predominantly disseminated (+ 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 be 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 veins. 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% 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 in the Minto deposits. At both Minto Main 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. Silicification 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 hematitization 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 Ridgetop, 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. Targets identified as Ridgetop Southwest, Copper Keel (North and South), Airstrip, Connector, DEF, and the newly discovered Minto East 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 Minto East prospect in late 2009.

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.

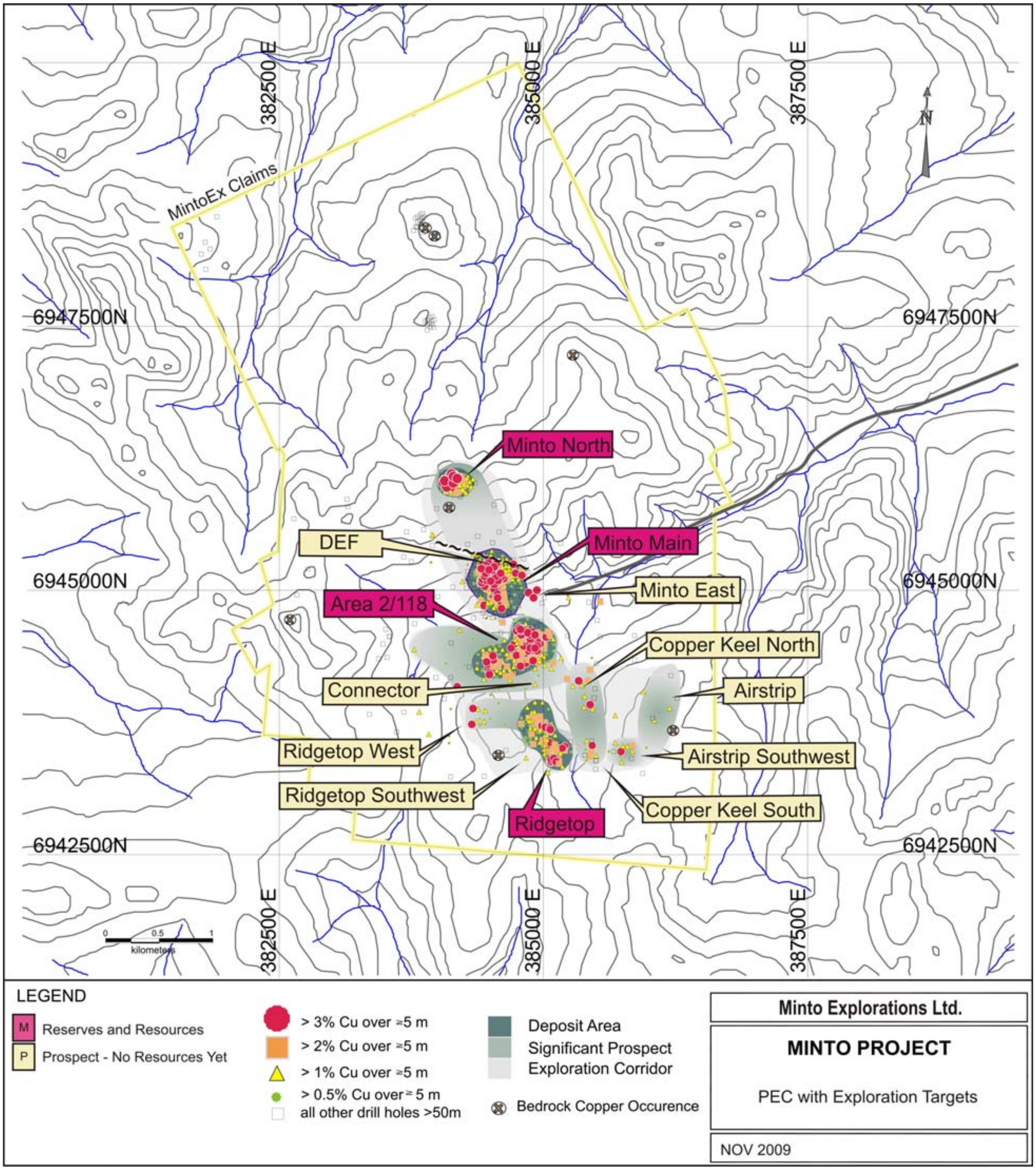



**Figure 8.1: Exploration Targets (Circa 2009) Minto East**

### 8.3.1 Minto East

The Minto East target was initially identified during the 2007 drilling in the gap between the Minto Deposit and Area 2, and is currently the highest priority target on the property. In 2007 a drill program was designed to test the "Gap" between the Main deposit and the Area 2 deposit mineral resource models. Drill hole 07SWC176 collared approximately 120 m east of the southeast corner of the Main deposit intersected 11.7 m of high grade copper-gold mineralization that looked remarkably similar to the 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 is 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, a "proof of concept" it was suspected the source of the anomaly was due to mineralization located some distance off the survey line. Drill holes 09SWC584 and 09SWC586 were collared further east and returned excellent copper grades (see table below) and thickness' confirming Minto East as a bona fide exploration target Figure 8.1. With four holes to date Minto East remains partially open to the west and south, and fully open to the north and east. Follow-up drilling is planned for 2010 along with down hole geophysical survey in 09SWC584 to vector further exploration on this high priority target. Select assay highlights from Minto East are presented in Table 8.1.

**Table 8.1: Select Assay Interval Highlights from Minto East Drilling**

| Hole Identification | From (m) | To (m) | Interval (m)* | Cu (%) | Au (g/t) |
|---|---|---|---|---|---|
| 07SWC176 | 291.9 | 303.6 | 11.7 | 2.95 | 1.07 |
| 08SWC286 | 288.1 | 290.8 | 2.7 | 0.82 | - |
| 09SWC584 | 302.0 | 315.6 | 13.6 | 3.45 | 1.18 |
| 09SWC586 | 279.8 | 306.8 | 27.0 | 2.75 | 0.97 |

**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 expected to be at or near true widths*

### 8.3.2 Copper Keel

Another priority exploration target, Copper Keel, is located southeast of the Minto Deposit, and is subdivided into Copper Keel North and Copper Keel South. Copper Keel North is located approximately 300 m south from 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). 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, although there is a gap in current drilling to support this conclusion definitively.

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 3D 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 and 2009 drill campaigns comprising an additional 9 drill holes (2,425 m) have been completed at the Copper Keel North target. 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 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. Many of the holes encountered significant faults but due to a lack of reliable marker horizons modelling the geology has been problematic. It is recommended that down hole geophysical surveys be carried out on any future drill holes in order to vector exploration in the area.

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 and the Airstrip Southwest target to the east. 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 Age appears to have removed sections of mineralization at the South Copper Keel and adjacent Airstrip prospects. Follow-up drilling in 6 drill holes as part of the 2008 (229 m) and 2009 (646 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.

It is recommended that down hole geophysical surveys be carried out on any future drill holes in order to better vector exploration in the area. Highlights of the drilling at Copper Keel South during 2007 to 2009 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 |

**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.*

### 8.3.3 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 (3323 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 Cretaceous Age erosion has now 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 directions, and further drilling is required to determine the extent of mineralization.

It is also recommended 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 |

## 8.4 Connector

The previous operators considered the northern part of the Connector area to be a continuation of the Area 2 Deposit. It is now considered to be a separate target, until such time as continuity with Area 2 can be demonstrated with core drilling. It is being treated as a separate target at a much deeper level than the near surface mineralization at Area 2, since it is 200 m below surface versus 100 to 120 m below surface in Area 2. The Connector target is identified in four historical holes that trace the unit over 550 m in an east-west direction. A fifth hole (A16-72) failed to intersect the target, as it was not drilled deep enough. Connector may be a down-faulted block of mineralization originally related to the Area 2 upper horizons. Despite the greater depth, the reported gold and copper grades make this an attractive drill target. Close proximity to Area 2 also provides development options that may mitigate its depth if sufficient tonnage could be outlined in both areas. Select highlights of historical and current assays results are presented in Table 8.4.

**Table 8.4: Select Assay Interval Highlights from Connector Historical Drilling**

| DDH ID | From (m) | To (m) | INT (m)* | Cu (%) | Au (g/t) |
|---|---|---|---|---|---|
| A16-7 | Not Applicable - Hole Too Shallow | | | | |
| A108-74 | 199.95 | 215.19 | 15.24 | 1.71 | 0.71 |
| A136-74 | 255.12 | 264.26 | 9.14 | 0.76 | 0.33 |
| A137-74 | 227.99 | 235.61 | 7.62 | 5.29 | 2.61 |
| A139-74 | 175.87 | 186.54 | 10.67 | 1.66 | 0.65 |

**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.*

### 8.4.1 DEF

This target, which lies along the DEF fault, is currently poorly understood due to the lack of angled drill holes in the area. MintoEx favours this area as a drill target as it appears to be an extension of the Main zone between two splays of the DEF fault zone. The northernmost fault appears to be a splay of the southern or main fault zone with the gap between the two widening up toward surface. While the extents of the target appear to be limited in the immediate area because of the fault geometry, sufficient room exists to warrant follow-up.

The target is open along strike, so any information gleaned from this area could help resolve both the magnitude and orientation of displacement along the DEF fault and vector toward any fault displaced portion of the main deposit further to the north.

Significant assay results for the Connector are presented in Table 8.5.

**Table 8.5: Select Assay Interval Highlights from DEF Area Drilling**

| DDH ID | From (m) | To (m) | INT (m)* | Cu (%) | Au (g/t) |
|---|---|---|---|---|---|
| 05SWC-049 | 93.75 | 99.12 | 5.37 | 3.59 | 2.56 |
| K01-73 | 118.57 | 129.54 | 10.97 | 1.74 | 0.18 |

** There is insufficient information to model this mineralization therefore the intervals in Table 8.5 are intersected widths, actual true widths are unknown.*

# 9 Exploration

Mineral exploration on the Minto property has been conducted intermittently since 1971. Subsequent to the discovery of the Main deposit, now the producing open pit Minto mine, 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 1000 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 2009 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 in future there will likely be more reliance 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. 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 four separate areas outside of the original or 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, and Minto North discovered in 2009. 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 (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.

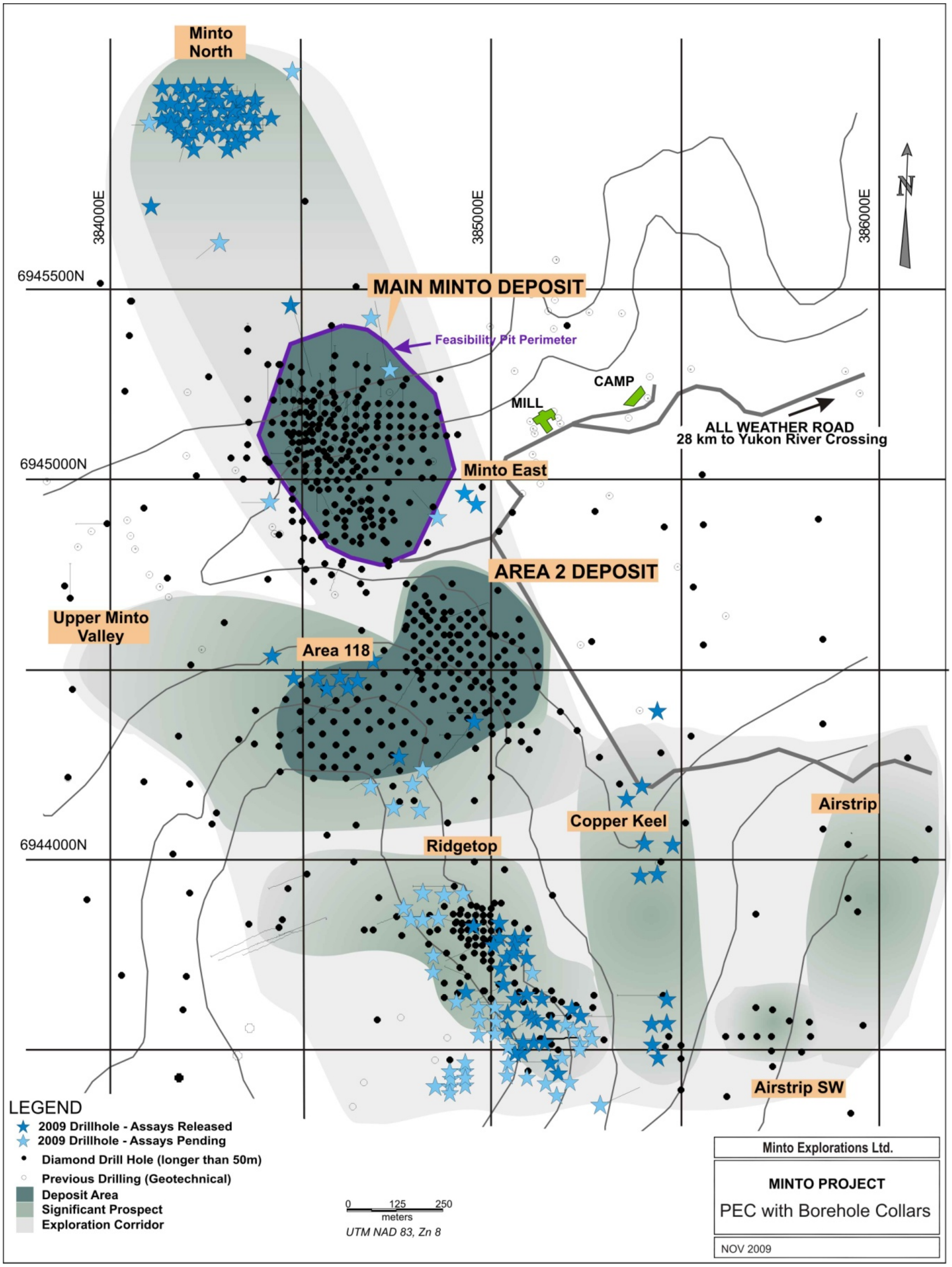



**Figure 9.1: Priority Exploration Corridor (PEC) with Drill Collars Current to November 17th**

In 2008 and 2009, 61 additional infill and margin step-out drill holes into the Area 2/118 deposit lead to a more robust NI43-101 resource calculation that was released June 9, 2009.

MintoEx geologists reassessed the Ridgetop area in 2007 (ASARCO's original Area 1 or Main discovery area) and drilled twenty-five new diamond drill holes, following up on sixteen historical holes between the 1970's and early 1990's. The s

ubsequent interpretation and drill density allowed for the completion of a 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 NI43-101 compliant resource calculation that was released June 9, 2009.

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 GIAP 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 (PEC).

In 2009, MintoEx geologists drilled 86 holes as a follow-up on two historic drill holes K88-74 and K91-74 in the immediate vicinity north of the target 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 author of the report suggests that the geophysical anomaly must have been misallocated in error. It now appears that he was correct. 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 in 2009.

The first drill hole at Minto North, hole 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 both hit significant mineralization. MintoEx geologists now know 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 NI43-101 compliant resource estimate completed June 9, 2009 contained herein.

The drilling at Minto North in 2009 returned some 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.

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 2010 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 3D 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 118/Area 2 deposit and Ridgetop deposits. At Minto North, 2009 drilling evolved from exploration, to delineation, to infill. 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. The Titan-24 surveys are discussed in more detail in section 9.3.

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

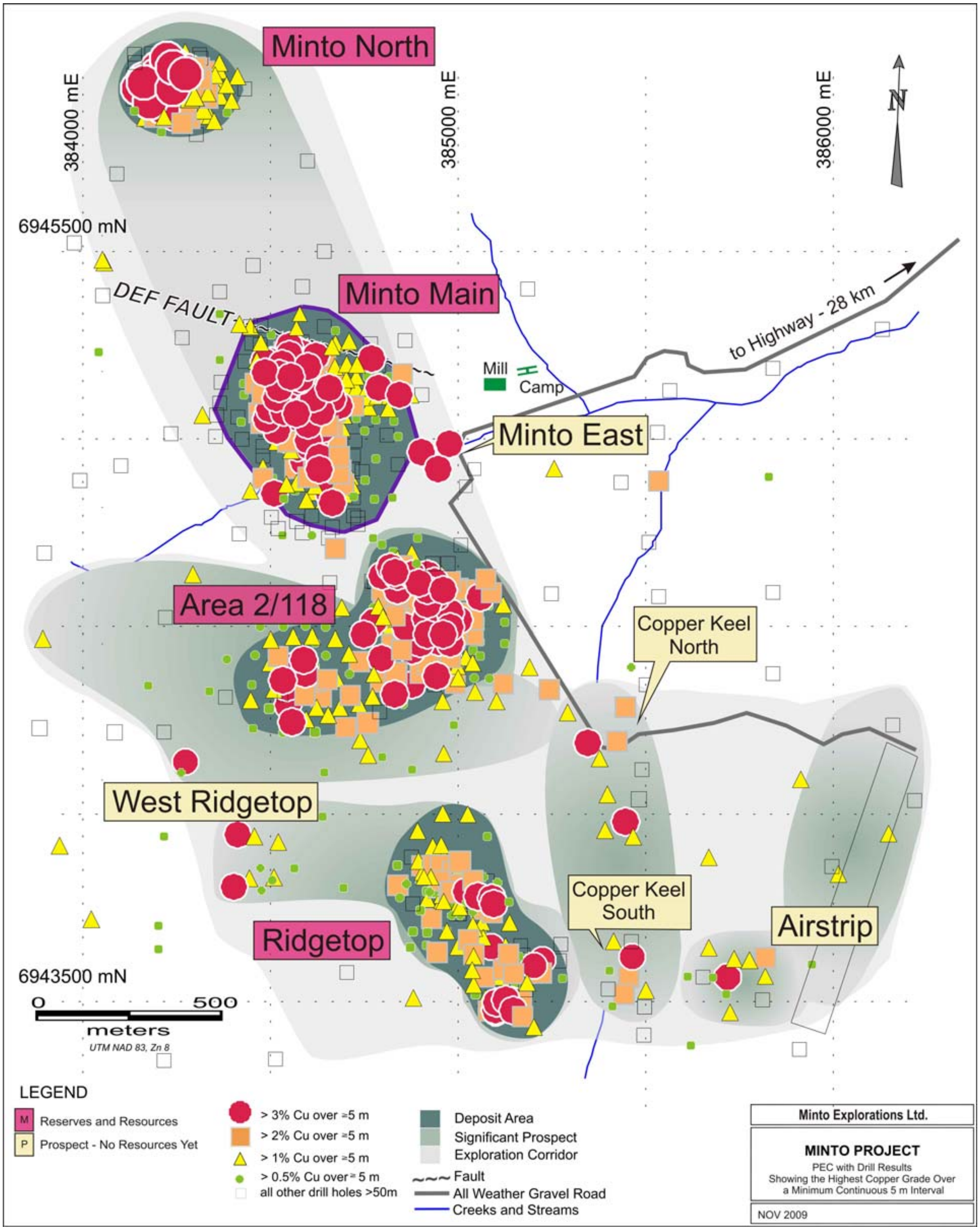



**Figure 9.2: Priority Exploration Corridor (PEC) with Drill Results Showing the Highest Copper Grade Over a Minimum Continuous 5 m Interval**

## 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. 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 Main Pit (now known as the Minto North Deposit) and the very large horseshoe shaped anomaly to the northeast of the Main Pit. Based on the success in 2009 drilling the coincident anomalies at Minto North, the horseshoe shaped anomaly northeast of Minto Pit 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

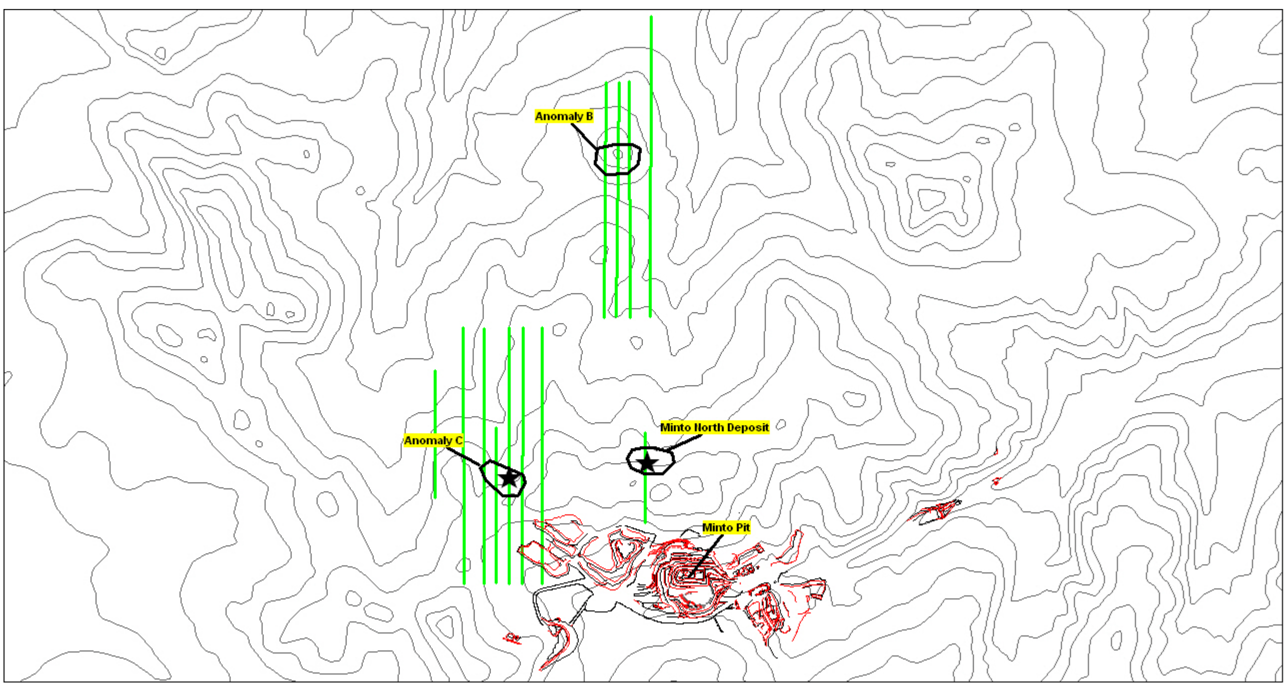



**Figure 9.3: Modified Pole-Dipole 2009 Survey Grid Location Map (Green Lines) and Location of modified Mise-a-la-Masse Drill hole Surveys (Black Stars)**

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.

Two of 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 downhole survey results (still pending as of November 5, 2009).

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

## 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).*

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 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 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 meters
- Number of Arrays/Line: 2
- Dipole Spacing: 100 meters
- Sampling Interval: Ex = 100 meters
  Ey = 200 meters
- 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
  Rx = Full wavefrom time-series acquisition
- The 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 meters
- Number of Arrays/Line: 2
- Dipole Spacing: 100 meters
- Sampling Interval: Ex = 100 meters
- Ey = 200 meters
- 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

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 Titan-24 survey grid is presented in Figure 9.4.

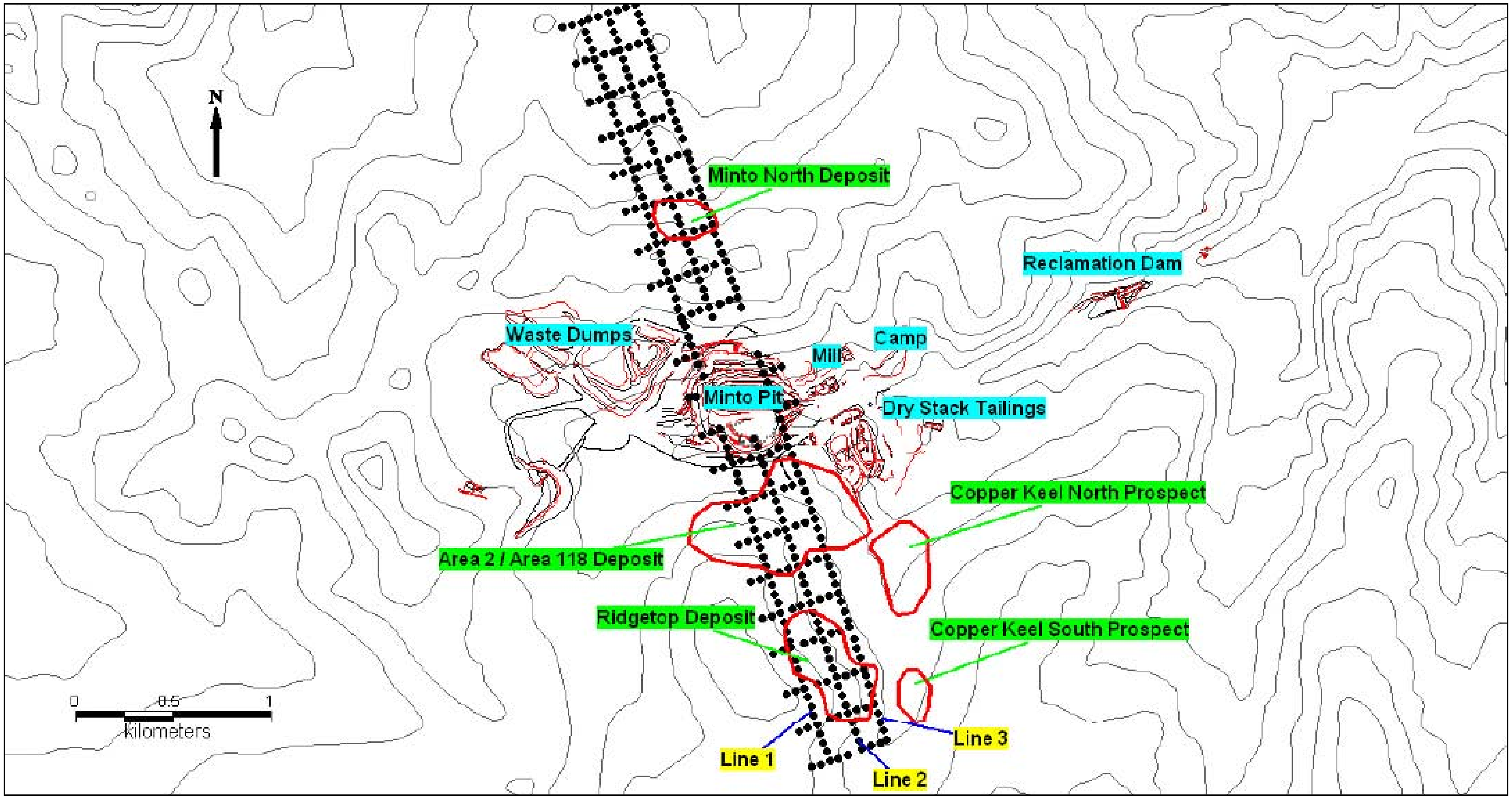



**Figure 9.4: Titan-24 2009 Survey Grid and Lines Location Map (idealized lines)**

The 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 degree 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 1st hole missed and a second hole drilled more than 130 m east of the actual survey line confirmed the discovery it is believe that a lack of success at some of these other anomalies is at least in part due to the limited coverage of the survey. The method appears 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 are considered to be poorly constrained.

Minor pyrite and trace chalcopyrite was sporadically encountered in these 9 unsuccessful holes, but it is believed that the low concentration of this mineralization does not satisfactorily explain the chargeability results. MintoEx geologists suspect that the poor intersections into the various targets may reflect a positioning problem with these specific anomalies; as mentioned above these anomalies flank either the eastern or western survey lines and the exact locations are thus poorly constrained. Casing was left in 7 holes in anticipation of follow-up downhole DC/IP surveys, and it is recommended that additional parallel survey lines are positioned to the east and the west in 2010 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.

## 9.4 Underground Exploration Targets

There are opportunities for increasing the resource accessible by underground mine development. The estimated resources around and below the Main and Area 2/118 deposits are shown in Table 9.1. Note that these estimates are MintoEx predictions and have not been verified by SRK.

### Table 9.1: Potential Underground Resources

| Label | Cut-off Grade (%) | Mined Tonnes | NSR ($2 UG) | Cu % | Au g/t | Ag g/t | Average Thickness (by Area) | Volume Drifting | Volume Benching |
|---|---|---|---|---|---|---|---|---|---|
| 1 | 0.6 | 382,373 | 92.71 | 2.26 | 0.91 | 8.98 | 7.4 | 73,330 | 63,405 |
| | 0.75 | 281,932 | 104.14 | 2.54 | 1.03 | 10.16 | 7.5 | 54,927 | 45,630 |
| | 0.9 | 226,397 | 112.42 | 2.74 | 1.13 | 10.87 | 7.3 | 46,177 | 34,403 |
| 2 | 0.6 | 339,378 | 79.87 | 1.95 | 0.81 | 9.4 | 7.5 | 71,791 | 50,789 |
| | 0.75 | 240,246 | 89.33 | 2.16 | 0.91 | 11.14 | 6.5 | 60,170 | 26,471 |
| | 0.9 | 137,537 | 102.52 | 2.46 | 1.05 | 14.62 | 6.4 | 34,344 | 15,131 |
| 3 | 0.6 | 83,029 | 72.4 | 1.72 | 0.65 | 6.13 | 7.1 | 17,177 | 13,352 |
| | 0.75 | 45,178 | 84.14 | 1.99 | 0.78 | 7.26 | 5.8 | 12,036 | 4,571 |
| | 0.9 | 12,392 | 106.87 | 2.52 | 1.16 | 11.35 | 5.4 | 4,155 | 410 |
| 4 | 0.6 | 89,117 | 62.98 | 1.48 | 0.57 | 5.6 | 6.0 | 23,497 | 9,375 |
| 5 | 0.6 | 75,868 | 60.55 | 1.52 | 0.41 | 5.69 | 5.5 | 23,452 | 4,503 |
| **Total 1-5** | **0.6** | **969,765** | **73.70** | **1.98** | **0.78** | **8.31** | **6.7** | **209,247** | **141,424** |

The Titan-24 geophysical survey also indicates the presence of underground targets that can followed up in future drilling campaigns. These target chargeability anomalies are summarized in Figure 9.5.

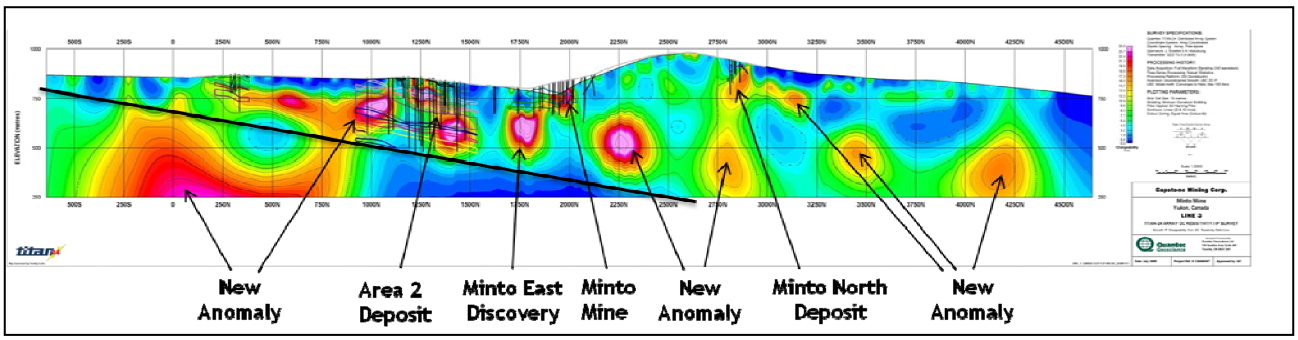



**Figure 9.5: North-South Cross-Section (Line 1 from Figure 9.4) showing Titan-24 Anomalies.**

# 10 Drilling

Up to October 17 2009, MintoEx had drilled a total of 55,319 m in 321 drill holes on the Minto Property as part of its 2008 and 2009 programs. The 2008 program was completed on August 29, 2008, and the 2009 program was stopped on October 17, 2009. More drilling in 2009 is tentatively planned for later in the year after the publication date of this technical report. The majority of the 2008 and 2009 drill holes were used in the resource estimations discussed in this report, however some drilling completed in the fall of 2009 is not incorporated and updating of the Ridgetop and Area 2 models and mineral resource estimates is planned for late 2009 or early 2010.

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.

In 2009, MintoEx drilled a total of 31,479 m in 201 diamond drill hole at the Minto North, Area 2, Area 118, and Ridgetop deposits, and at various other prospects. Drilling was conducted between 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.

A total of 209 (86 Minto North + 74 Ridgetop + 48 Area 2/118) holes or 31821 m of the 2008 and 2009 drilling were incorporated into the three resource models described in this report. 53 holes for 14,922 m were drilled specifically at exploration prospects outside of these resource models. The median length of 2008 MintoEx drill holes is 198 m (average 199 m), with the shallowest hole being 26 m in length and the deepest, 385 m. The median length of 2009 MintoEx drill holes is 123 m (average 157 m), with the shallowest hole being 54 m in length and the deepest, 752 m. MintoEx diamond drill holes by year and deposit, from 2005 through 2009, are summarized in Table 10.1 below.

**Table 10.1: Summary of MintoEx Drill holes by Deposit (2005 to 2009)**

| Company | Deposit | Year | No. DDH | Type | Core Size | Metres | Angled | Vertical |
|---|---|---|---|---|---|---|---|---|
| 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 | 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 | 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 |

The Area 2/ 118 resource estimation incorporates the majority of 2008 and 2009 drilling within Area 2 and Area 118 and 22 drill holes completed by ASARCO in the 1970s that were not included in the previous estimation (SRK, 2007).

At Area 2, MintoEx drilled at total of 4,162 m in 19 vertical diamond drill holes from May 11, 2008 to September 10, 2009. The size of the drill core is NQ. The Area 2 drill holes drilled in 2008 and 2009 range from 78 m to 339 m in length, with a median length of 255 m and an average length of 219 m. A total of 18 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 centers on a northeast striking grid. Mineralized zones, shown in Figure 10.1, undulate and dip shallowly to the northwest.

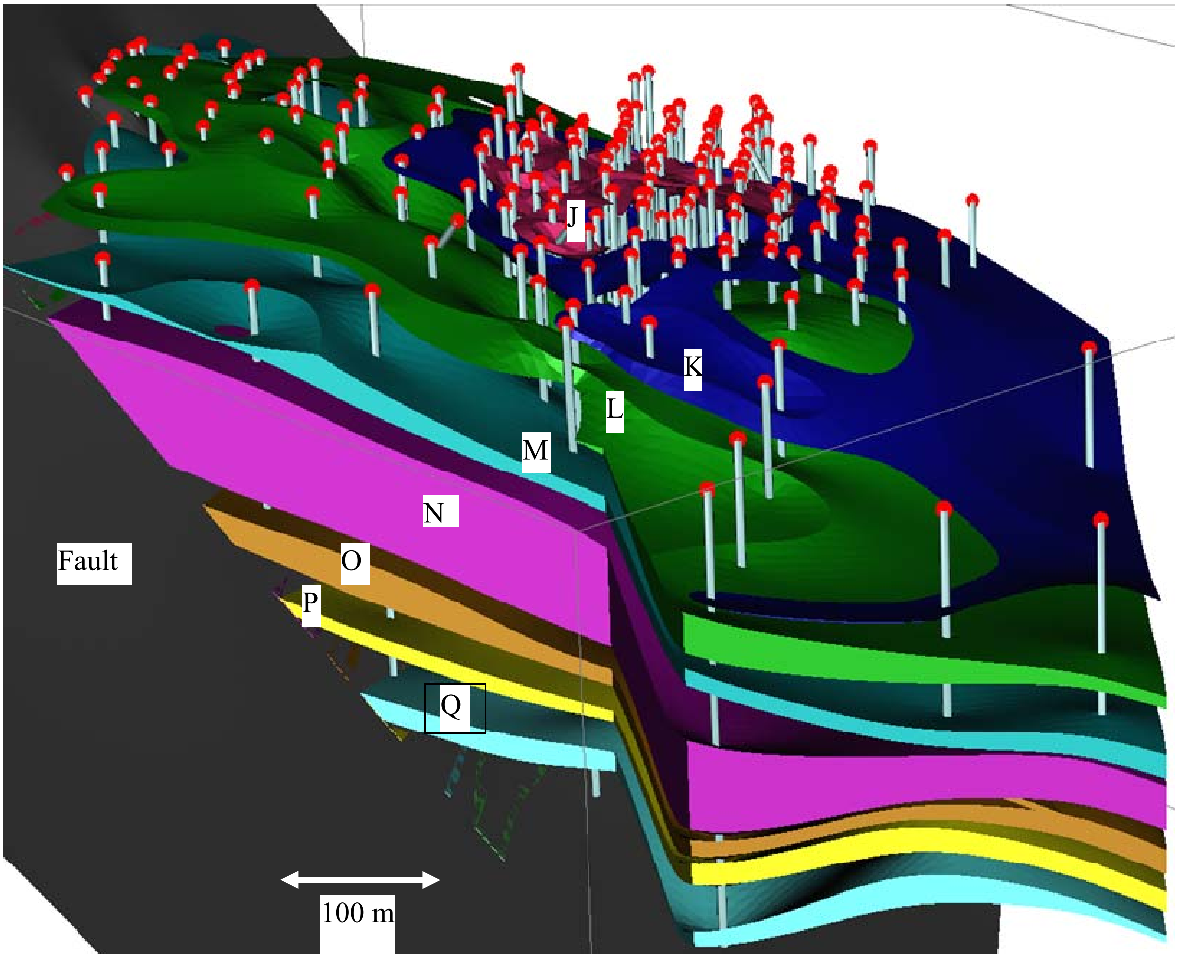



**Figure 10.1: Wireframes of Mineralized Domains with Drill Holes, Area 2. Fault Separates Area 2 from Area 118. View Northwest.**

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. 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 42 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.

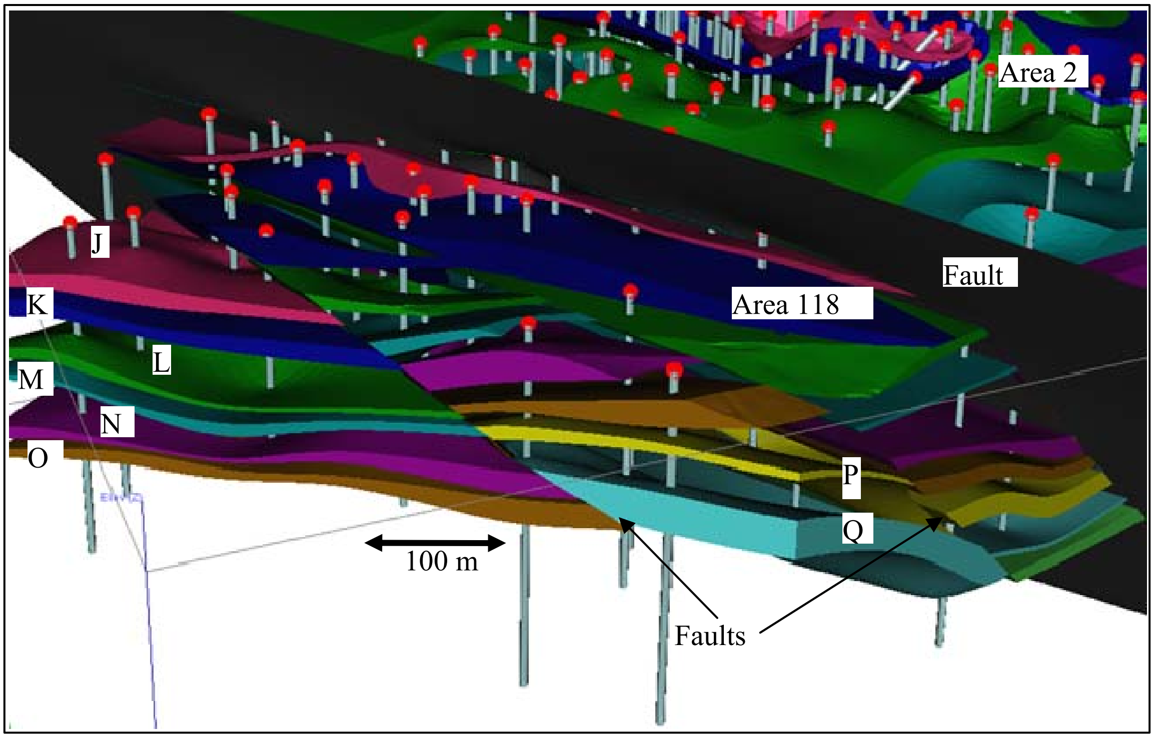



**Figure 10.2: Wireframes of Mineralized Domains with Drill Holes, Area 118. Faults Separate Area 2 from Area 118, and Subdivide Area 118 into Three Sub-domains.**

At Ridgetop, MintoEx drilled a total of 13,641 m in 113 vertical drill hole 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) 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).

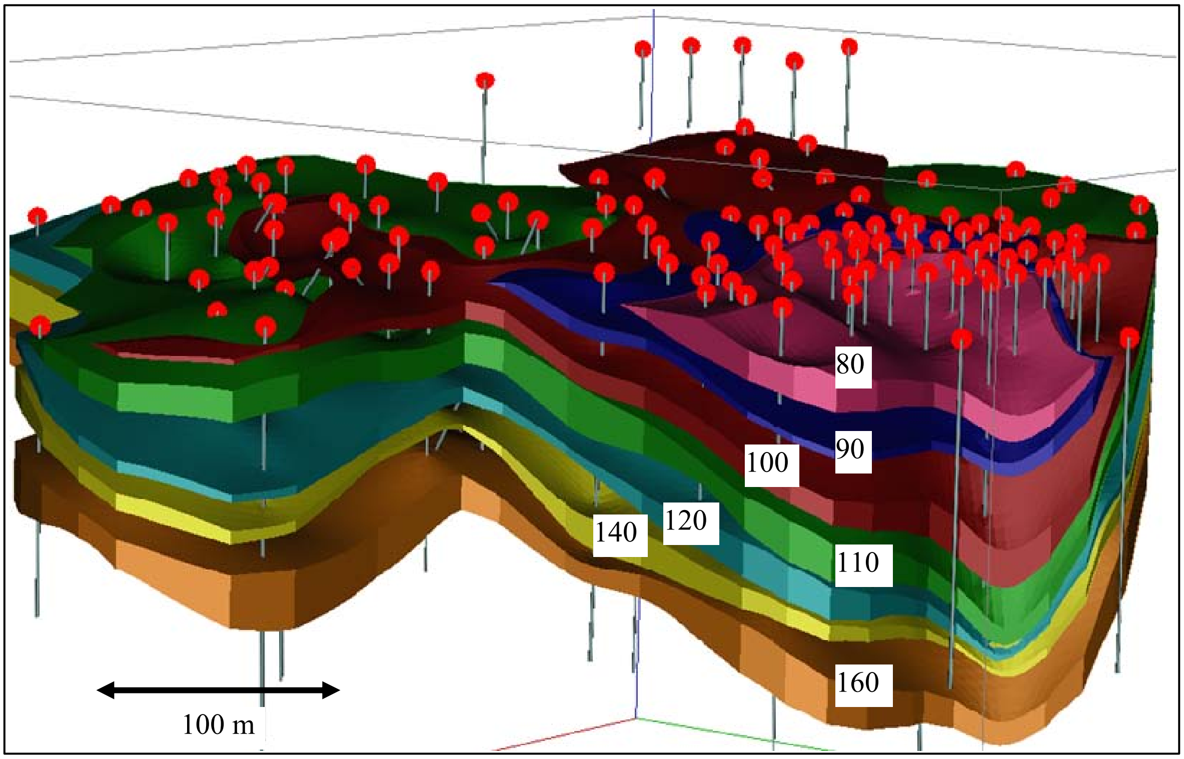



**Figure 10.3: Wireframes of Labelled Mineralized Domains with Drill Holes, Ridgetop**

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 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).

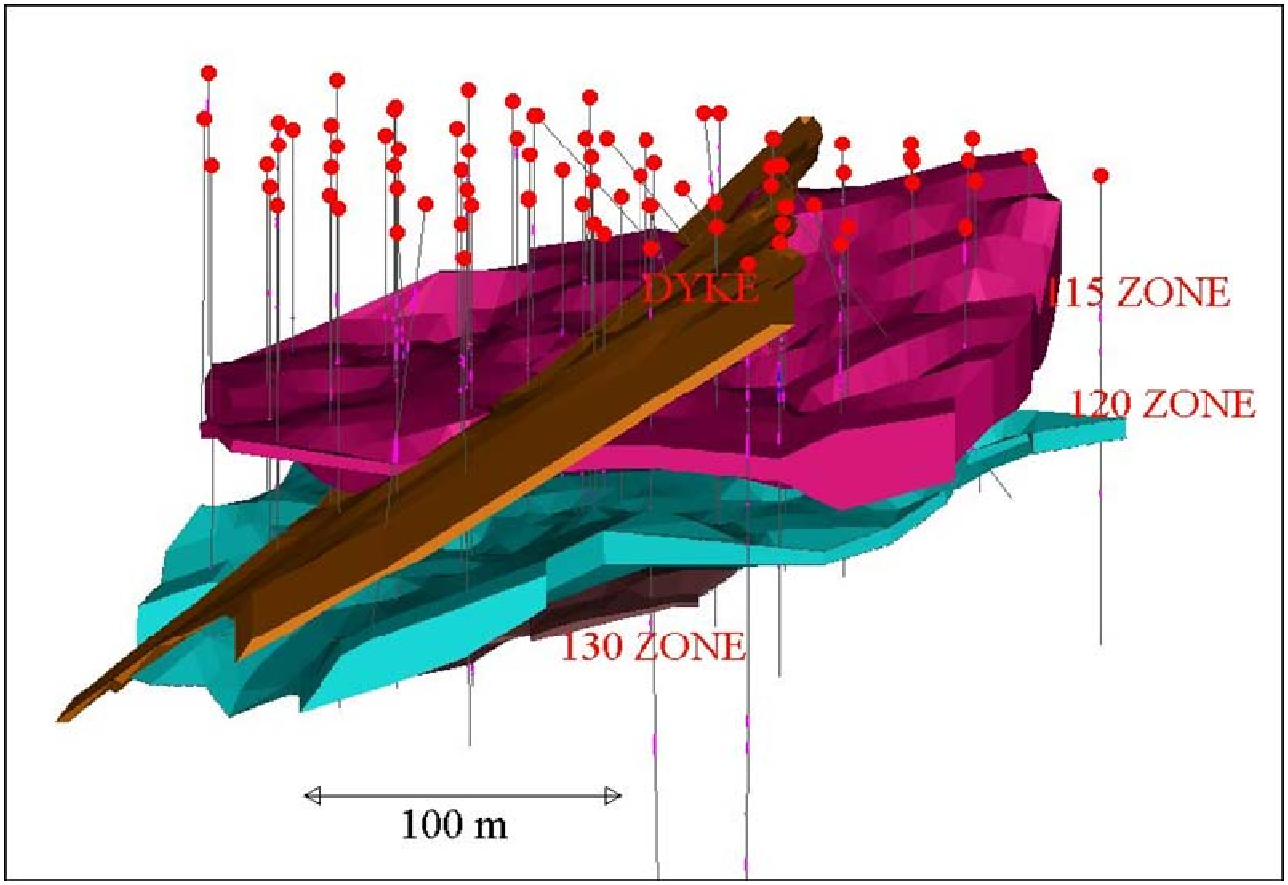



**Figure 10.4: Wireframes of Mineralized Domains with Drill holes, Minto North**

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. Minimal deviations were typical in all holes which were predominantly drilled at a vertical inclination. Since the spring of 2008, down hole surveys were performed using a FLEXIT downhole 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. 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 and 2009, 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 and 2009. 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 downhole 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 2009 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 2005-2007). 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 downhole 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.

# 12 Sample Preparation, Analyses and Security

## 12.1 Historic Samples

### 12.1.1 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.

### 12.1.2 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

### 12.2.1 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.

### 12.2.2 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.

### 12.2.3 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.

### 12.2.4 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 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.

## 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 in 2008 and 2009 (to the end of October) are summarized in Table 12.1. Quality control data are presented in graphical format in Appendix A.

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

| | | 2006 | | 2007 | | 2008 | | 2009 | |
|---|---|---|---|---|---|---|---|---|---|
| **Total Samples Collected** | | 13,121 | | 13,552 | | 15,119 | | 13,056 | |
| **SRM Used** | | CGS-5 | 36 | SRM-95 | 4 | SRM-95 | 13 | SRM-95 | 3 |
| | | Cu-115 | 47 | | | | | | |
| | | Cu-116 | 48 | | | | | | |
| | | CGS-10 | 116 | CGS-10 | 120 | CGS-10 | 24 | SRM-2 | 27 |
| | | CGS-7 | 103 | CGS-7 | 137 | CM-3 | 27 | SRM-1 | 27 |
| | | CGS-12 | 54 | CGS-12 | 139 | CGS-12 | 31 | SRM-3 | 24 |
| | | GSP-5 | 19 | CGS-8 | 17 | CGS-17 | 56 | CGS-17 | 12 |
| | | GS-2A | 17 | | | | | | |
| | | CM-1 | 6 | CM-2 | 8 | CM-2 | 99 | CM-2 | 117 |
| | | CGS-9 | 109 | CGS-9 | 51 | CGS-18 | 120 | CGS-18 | 190 |
| | | CGS-11 | 52 | CGS-11 | 175 | CGS-11 | 156 | CGS-11 | 123 |
| | | Cu-132 | 50 | CGS-13 | 17 | CGS-15 | 177 | CGS-15 | 191 |
| | | Cu-128 | 40 | Cu-128 | 15 | | | | |
| **Total SRM** | | 697 | | 682 | | 703 | | 714 | |
| **Blanks** | | 595 | | 674 | | 685 | | 698 | |
| **Paired Data** | **Coarse Reject Duplicate** | 404 | | 556 | | 590 | | 590 | |
| | **Pulp Reject Replicate** | 597 | | 702 | | 568 | | 568 | |
| **Total QC samples** | | 2293 | | 2614 | | 2582 | | 2570 | |
| **Frequency (percent)** | | 17 | | 19 | | 17 | | 20 | |
| **Umpire checks (percent)** | | 2 | | 1 | | 0.5 | | 0.5 | |

### 12.3.1 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.

### 12.3.2 Performance of Blanks in 2008 and 2009

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, Area 118 and Ridgetop 2008 drilling campaign. A total of 698 blanks were submitted with the sawn core samples from the Minto North, Area 2, Area 118 and Ridgetop 2009 drilling campaigns. 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.

### 12.3.3 Performance of SRM in 2008 and 2009

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 are presented in the Appendix B.

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, 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.

### 12.3.4 Performance of Pulp Reject and Coarse Reject Duplicates in 2008 and 2009

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.

Graphs of duplicate quality control data are shown in Appendix B.

### 12.3.5 Performance of Umpire Analyses in 2008 and 2009

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 and to a lesser extent for gold and silver (see Figures in an Appendix B). This level of precision is excellent for copper but warrants some improvement for gold and silver. In short, the results were shown to be sufficiently reproducible for resource estimates.

# 13 Data Verification

## 13.1 Verification by MintoEx

### 13.1.1 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 because both the Falconbridge and ASARCO core sheds have either collapsed and/or burned during regional forest fires. 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.

### 13.1.2 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 sited 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 |

### 13.1.3 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

### 13.3.1 Site Visits

In 2007, In accordance with NI43-101 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 4$^{th}$ and 6$^{th}$ 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.

### 13.3.2 Verification from Electronic Lab Files

SRK compared electronic lab files from 2008 and 2009 drill campaigns with the assays in the current database. The electronic lab files were sent to MintoEx by Chemex and SGS Labs. Overall, 80% of the Cu assays and 94% of the 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.

### 13.3.3 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.

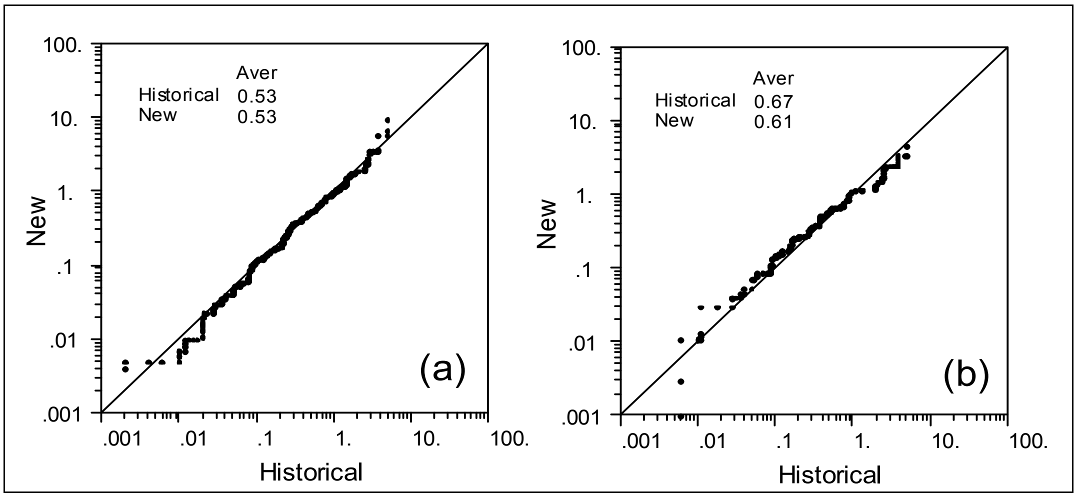



**Figure 13.1: Comparison of historical and new data in: (a) Area 2/ 118 and (b) Ridgetop**

# 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

Metallurgical test work on samples from Minto Main, Minto North, Minto Main (South), Ridgetop East, Area 2/118 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. 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 3,750 tpd while meeting a predicted metal recovery and concentrate grade.

Further test work was identified and risks and opportunities for the selected flowsheet were highlighted.

## 15.2 Test Work Review

For this class of study the reviewed flotation test work and comminution data are considered adequate for flowsheet development. The design of the processing circuits for the study is based on the design criteria derived from these data in conjunction with reasonable assumptions based on the current Minto plant performance.

The following technical reports and information were reviewed:

- Starkey and Associates Inc, "Final Report on Grinding Mill Design", 4th August 2009;
- DJB Consultants Inc, "Projections of Mill Throughput for the SAG and Ball Mill Circuits", October 2009; and
- Xstrata Process Report, "Minto Mine Site Visit Report", November 2008.

A number of test work reports were reviewed, as summarized in Table 15.1.

**Table 15.1: Metallurgical Test work Reports Reviewed**

| Orebody | Samples | Summary Comments |
|---|---|---|
| Minto North | 83 samples of quarter core combined to form a single master composite | KM 2420 test work report completed by G&T in June 2009. Rougher flotation kinetics completed with $P_{80}$s of 156 to 273 micron. A single locked cycle test was completed to assess the effect of cleaner recirculation loads. |
| Ridgetop East, Area 118 | A master composite generated representing upper and lower zones of each orebody. In addition 12 variability samples per orebody | KM 2351 test work report completed by G&T in May 2009. At a primary grind size of 200 micron two dimensional copper sulphide liberation was 55 to 65% for Area 118 and Ridgetop East. No sensitivity to primary grind size up to 250 micron for Area 118 and 200 micron for Ridgetop East. |
| Area 2 | A master composite was derived for zones L, M, N, O, P and Q from the individual samples | KM 1966 test work report completed by G&T in June 2007. Variability tests completed at various grind sizes. Gold recovery was lower for the coarse primary grind sizes. Regrind size of 100 micron for the rougher/scavenger concentrate was tested. Locked cycle test indicated a drop in final concentrate grade of 3% for the same overall recovery when the regrind stage was removed. |
| Minto Main Phases I, II and IV composites | Three composites for phase I, II and IV tested | KM 1867 test work report completed by G&T October 2006. Effect of primary grind size investigated on the Phase I composite. Copper recovery was not sensitive to coarser grind however gold recovery reduced by 5 – 10% with greater than 200 micron primary grind. |
| Minto Main | 2 composites based on 23 core samples | KM 1810 test work report completed by G&T in April 2006. Report recommends primary grind size greater than 150 micron and a regrind of 60 micron for the rougher/scavenger concentrate. |
| Minto Main | | KM 1742 test work report completed by G&T in November 2005. Single test completed with a primary grind size of 281 micron. Not sufficient data for determining the effect of coarser primary grind sizes on overall recovery. |
| Minto Main (South) | Two composite samples less oxidized than samples in KM 1937 test work campaign | KM 2024 test work report completed by G&T in August 2007. The flotation response was considerably more variable to increased grind sizes in comparison to the Minto Main Pit ore test work. |
| Minto Main (South) Partially Oxidized | Composite sample generated for Minto Main (S) pit ore | KM 1937 test work report completed by G&T in April 2007. Sample contained 20% non-sulphide copper as compared to 8% for non-partially oxidized Minto Main (S) ore for test work campaign KM 2024. Addition of a sulphidizing agent improved overall copper recovery by 2 – 4%. |
| Minto North | Minto North composite sample | KM 1937 test work report completed by G&T in March 2007. Varied primary and regrind sizes tested. Copper recovery was not sensitive to primary grind size however gold recovery reduced by 3% at 270 micron grind size. The optimum copper concentrate grade and recovery occurred with a rougher/scavenger concentrate regrind size of 79 micron. |

### 15.2.1 Test Work Program

The metallurgical test work program used as the basis for this report consisted of flotation test work on Minto North, Minto Main (South), Ridgetop East, Area 2/118 deposits. Comminution test work was completed on Area 2, Ridgetop East, Minto North and Minto Main orebodies. The test work programs were mainly completed on bulk composites designed to represent the complete orebody. Variability flotation test work was completed on the Area 2 orebody in the KM 1966 test work program completed by G&T. The test work on the orebodies generally consists of:

- Bench scale comminution testing, consisting of SAG design test work by Starkey and Associates, SAG Media Competency (SMC) test work, Bond ball and rod mill work indices testing as well as Bond abrasion indices; and
- Bench scale flotation testing consisting of rougher kinetic flotation, cleaner flotation and locked-cycle tests, supplemented with optical mineralogical examination.

### 15.2.2 Comminution Test Work Suite

The comminution data set used for the modelling of the existing Minto comminution circuit consisted of:

- 7 SAG design (test work specifically developed by Starkey & Associates) tests on Minto Main, Ridgetop East and Minto North samples. The two Minto Main samples were SAG mill feed conveyor sourced samples taken during comminution surveys by Starkey& Associates;
- 10 samples used for JK Drop Weight Testing and SMC ore competency tests on the Area 2 orebody drill core samples and two SMC tests completed on the Minto Main orebody;
- 37 Bond ball mill work indices based on samples from Area 2, Minto Main, Ridgetop East and Minto North; and
- 11 Bond rod mill work indices based on samples from Area 2 and Minto Main.

The JK Drop Weight Tests and SMC test results are summarized in Table 15.2.

**Table 15.2: JKDW and SMC Test Results**

| Sample Name | Location | S.G. ($g/cm^3$) | DWT Parameters | | |
|---|---|---|---|---|---|
| | | | A x b | DWI | ta |
| 07SWC197 - 84.47 to 90.50 | Area 2 | 2.85 | 61.9 | 4.6 | 0.57 |
| 07SWC197 - 90.50 to 95.46 | Area 2 | 2.80 | 66.1 | 4.2 | 0.62 |
| 07SWC197 - 128.00 to 133.93 | Area 2 | 2.69 | 72.4 | 3.7 | 0.70 |
| 07SWC197 - 133.93 to 138.90 | Area 2 | 3.05 | 86.4 | 3.5 | 0.74 |
| 07SWC197 - 138.90 to 145.37 | Area 2 | 2.87 | 93.8 | 3.0 | 0.86 |
| 07SWC197 - Waste dilution | Area 2 | 2.68 | 66.3 | 4.0 | 0.65 |
| 07SWC198 - 84.94 to 90.00 | Area 2 | 2.63 | 96.0 | 2.7 | 0.95 |
| 07SWC198 - 90.00 to 94.43 | Area 2 | 2.58 | 133 | 1.9 | 1.33 |
| 07SWC198 - 141.48 to 147.00 | Area 2 | 2.79 | 68.0 | 4.1 | 0.64 |
| 07SWC198 - 147.00 to 152.90 | Area 2 | 2.77 | 66.2 | 4.2 | 0.62 |
| 07SWC198 - 152.90 to 158.81 | Area 2 | 2.88 | 64.8 | 4.4 | 0.59 |
| 07SWC198 - Waste dilution | Area 2 | 2.68 | 75.0 | 3.6 | 0.73 |
| SAG Feed | Main | 2.68 | 69.4 | 3.9 | |
| SAG Charge | Main | 2.71 | 62.7 | 4.3 | |

The JK Drop Weight and SMC test work was organized by DJB Consultants as part of their mill throughput projections report (October 2007). Following a review of the data a DWI of 4.2 was selected to calculate the power draw required in the SAG milling stage.

The Bond ball mill work index (BWI) test work was completed at varying closing screen sizes. The results are summarized in Figure 15.1.

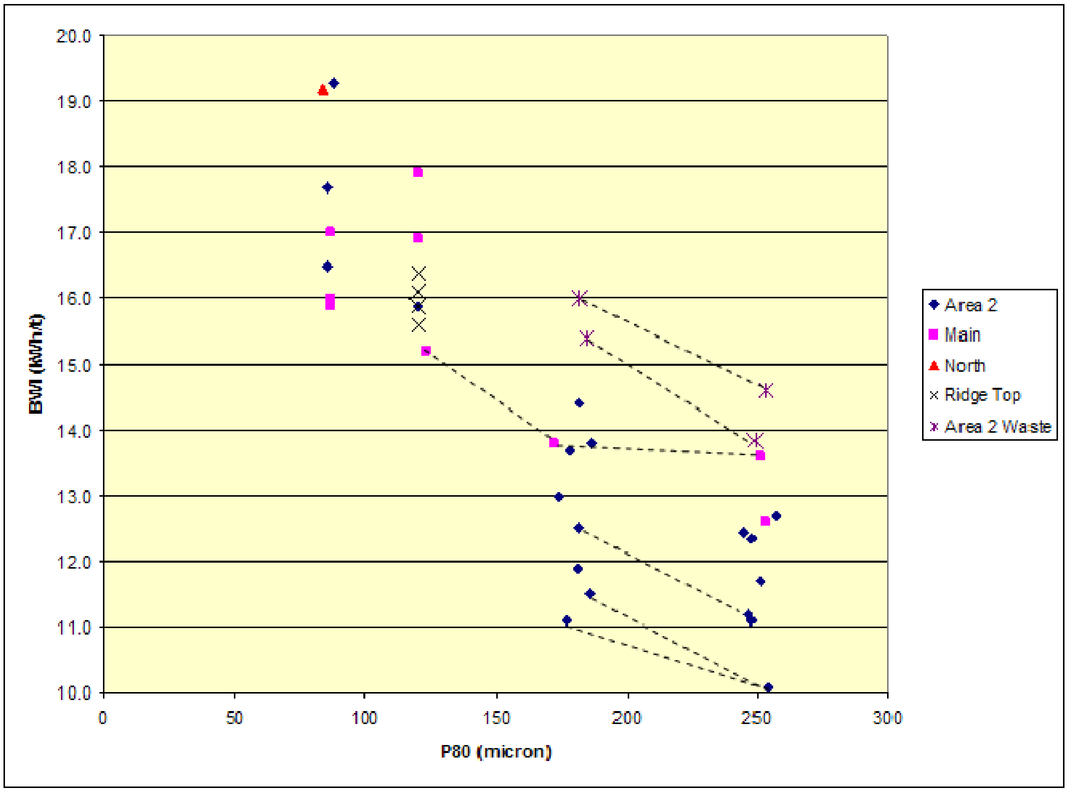



**Figure 15.1: Bond Ball Mill Work Indices**

The BWI is used to determine the power draw required in the ball milling stage. The figure above shows a strong correlation between the BWI and the closing screen, or final grind size.

Starkey & Associates completed SAGDesign™ test work as part of their report on a possible 5,000 tpd expansion project (August 2009). The results are summarized in Table 15.3.

### Table 15.3: SAGDesign Specific Energy Data

| Source | SAG Mill Specific Energy (kWh/t) |
|---|---|
| Belt sample - Minto Main | 6.1 |
| Belt sample - Minto Main | 5.9 |
| Ridgetop East Upper | 5.9 |
| Ridgetop East Lower | 6.2 |
| Area 118 Upper | 6.1 |
| Area 118 Lower | 6.1 |
| Minto North | 6.6 |

The SAG design test work procedure was specifically designed for the Starkey & Associates comminution model.

Starkey & Associates also completed two comminution circuit surveys on the existing plant whilst it was treating Minto Main Pit ore. The results from these surveys are shown in Table 15.4.

### Table 15.4: Plant Performance Data Based on Starkey & Associates Surveys

| Parameter | Units | Case 1 | Case 2 |
|---|---|---|---|
| Throughput | t/h | 140 | 92 |
| SAG mill power | kW | 504 | 574 |
| Ball mill power | kW | 1090 | 1050 |
| Total mill power | kW | 1594 | 1624 |
| SAG mill specific energy | kWh/t | 3.6 | 6.2 |
| Ball mill specific energy | kWh/t | 7.8 | 11.4 |
| Total specific energy | kWh/t | 11.4 | 17.7 |
| $F_{80}$ (nominal) | mm | 50.0 | 115 |
| $P_{80}$ (nominal) | um | 240 | 130 |
| Operating work index | kWh/t | 17.6 | 20.1 |

The feed size (F80) to the SAG mill was assumed for both mill surveys. The following are comments on the Starkey & Associates plant survey results:

- The data in Table 15.4 indicates that either the operating work index in the plant is higher than expected (and normal for ore of this competency) or that the estimated product sizes were incorrect; and
- The ball mill specific energy derived from the surveys is more than double the theoretical ball mill specific energy.

Only the general comments and observations by Starkey & Associates on their SAGDesign test work and comminution surveys were considered in future modelling.

### 15.2.3 Flotation Test Work Results

Flotation test work for the Minto Phase IV 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.

### 15.2.4 Phase IV 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;
- 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).

### 15.2.5 Minto North

Rougher kinetic tests were conducted for the North Minto ore in G&T test work program KM2420, with $P_{80}$ ranging from 156 micron to 273 micron. A locked cycle test was conducted at $P_{80}$ of 200 micron and 65 micron re-grind 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.

### 15.2.6 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).

### 15.2.7 Area 2

Ores from K, L, M, N, O, P& Q zones were tested. Variability tests were completed at approx $P_{80}$ of 130 to 150 micron. Copper was mainly present as bornite and chalcopyrite.

Locked cycle tests on composite samples were at primary grind sizes ($P_{80}$) 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 15.5.

**Table 15.5: KM 1966 Test work Summary by Zone**

| Composite | Rougher Performance as a Function of $P_{80}$ |
|---|---|
| L and M composites | $P_{80}$ 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.

### 15.2.8 Minto Main (South) Primary Ore

Report KM 2024 contains test work on two composite samples from the Minto Main (South) Pit that are less oxidized than the samples tested under the KM 1937 campaign (Figure 15.2). 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).

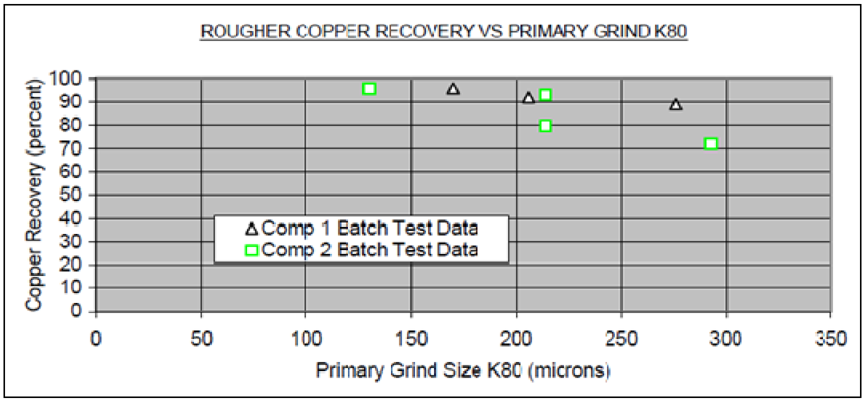



**Figure 15.2: KM 2024 batch rougher test work results**

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.

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.

### 15.2.9 Minto Main (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 Main (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.

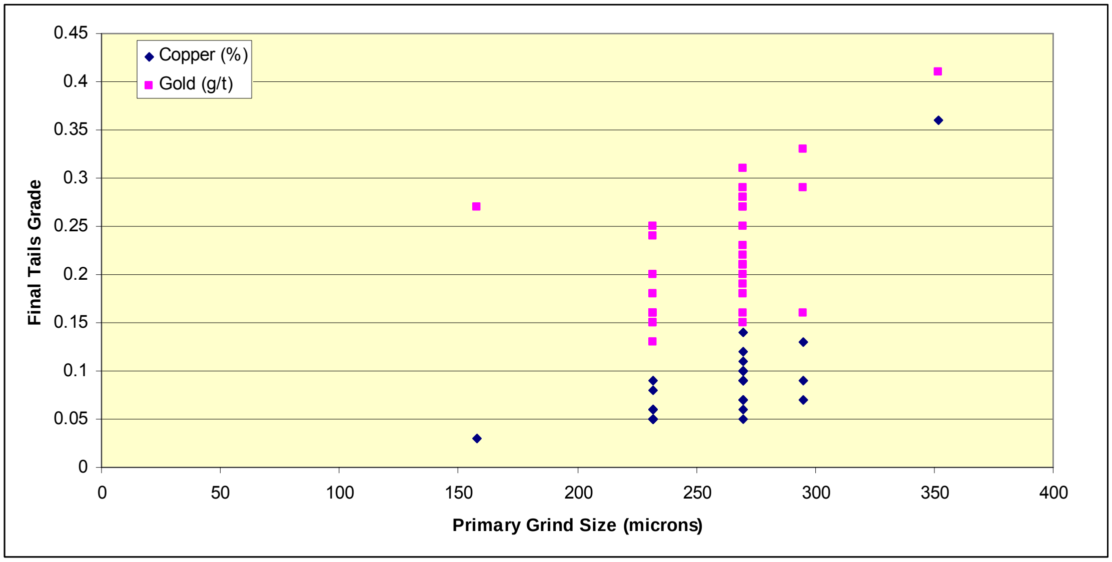



**Figure 15.3: KM 1937 Primary Grind Size vs. Tails Grade**

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.

### 15.2.10 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.

### 15.2.11 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 $P_{80}$ grind size of 212 to 250 micron. A regrind stage with a target cleaner feed $P_{80}$ grind size of 60 micron is required at this coarser primary grind.
- **Increased Cleaner Stages**: Three stages of cleaning provides improved circuit flexibility with regards to improving the final concentrate grade. The expansion will incorporate the increased cleaning stages and capacity.

## 15.3 Process Plant Design

### 15.3.1 General

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

### 15.3.2 Process Plant Design Basis

The key criteria selected for the plant design are:

- Treatment of an average 3,750 dry metric tonnes per day for 365 days per year, after allowance for availability;
- Design availability of 91.3%, being 7,998 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.

### 15.3.3 Throughput and Availability

An overall plant availability of 91.3% or 7,998 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;
- Original pipework around the milling area was not rubber lined. Pipework 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 the existing Minto plant milling circuit capacity. From the review of 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.

### 15.3.4 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 test work for the new orebodies as well as general plant observations by Starkey & Associates as well as DJB Consultants. Typically, ore hardness parameters were selected based on the 75th percentile, that is 75% of the ore to be processed is expected to be similar in hardness or softer than the ore hardness used for design.

### 15.3.5 Head Grade

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

### 15.3.6 Process Plant Design Criteria Summary

The overall approach was to review the current Minto plant throughput limitations and provide a robust process plant flowsheet that could handle the variability in the metallurgical performance of the new orebodies that has been evident from the test work.

The detailed process design criteria derived from the results of the metallurgical test work program are included in Appendix D.

## 15.4 Process Description

### 15.4.1 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 crushed in a new secondary crushing facility (as selected and installed by MintoEx) to a nominal 80% passing 25 mm and fed onto the stockpile stacking conveyor;
- 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 with the addition of a further two new 40 $m^3$ cells to provide a total of 33 minutes of retention time;
- Rougher/scavenger concentrate regrinding in a new 220 kW vertical stirred mill, grinding to a product size of nominally 80% passing ($P_{80}$) 60 micron;
- Cleaner 1 flotation consisting of the existing four 14 $m^3$ forced air tank flotation cells to provide a total of 36 minutes of retention time;
- Cleaner 2 flotation consisting of the existing four 10 $m^3$ forced air-tank trough shaped flotation cells to provide a total of 42 minutes of retention time;
- Cleaner 3 flotation consisting of the existing six 3 $m^3$ trough shaped flotation cells to provide a total of 25 minutes retention time;
- 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.
- The Phase IV plant flowsheets are included in Appendix D.

# 16 Mineral Resource and Mineral Reserve Estimates

## 16.1 Introduction

A primary objective of SRK's work was to produce a revised independent resource evaluation for the Area 2/ 118 and for the Ridgetop deposits. The Minto North Zone, another integral part of the Minto Deposit, has been evaluated by Kirkham Geosystems Ltd (Kirkham Geosystems).

The mineral resource evaluation reported herein supersedes earlier resource estimates prepared by Lions Gate Geological Consulting ("LGGC") in 2008 and reported in the SRK Technical Report, June 2008.

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 this resource estimate is June 1, 2009. Marek Nowak, P.Eng., analyzed the data, reviewed and validated the mineral resource estimates. The Minto North deposit resource estimate was 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, and Minto North 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 Area2/ 118, Ridgetop, and Minto North 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.

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 Geosystems audited the Minto North 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.

## 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 database was also prepared by MintoEx, and verified by Kirkham Geosystems. The mineralized domains of the deposits were modelled using Gemcom software.

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 | 2008 | 48 | 5,519 | 6,800 |
| | 2007 | 42 | 5,701 | 7,529 |
| | 2006 | 80 | 9,403 | 14,819 |
| | Historical | 22 | 604 | 1,672 |
| | **TOTAL** | **192** | **21,227** | **30,820** |
| Ridgetop | 2009 | 28 | 2,232 | 3,227 |
| | 2008 | 46 | 4,229 | 5,691 |
| | 2007 | 25 | 2,001 | 2,713 |
| | Historical | 20 | 788 | 1,915 |
| | **TOTAL** | **119** | **9,250** | **13,546** |
| 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.

In June 2009, a resource estimate was calculated for the Minto North (see Capstone Press Release dated June 9, 2009). The resource estimate calculated for the Minto North Deposit in this report is updated using the addition of the 56 in-fill and delineation drillholes and supersedes the previous resource estimate. The result is a slight change in tonnage due to adjustments made to the constraining solids and increasing confidence allowing for the conversion of resources to the measured and indicated categories.

## 16.3 Area 2/ 118 Deposit

### 16.3.1 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 one fault appears 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 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 Area 2/118 resource model was created using a commercial three-dimensional block modelling and mine planning software, GEMS version 6.1.3 (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.

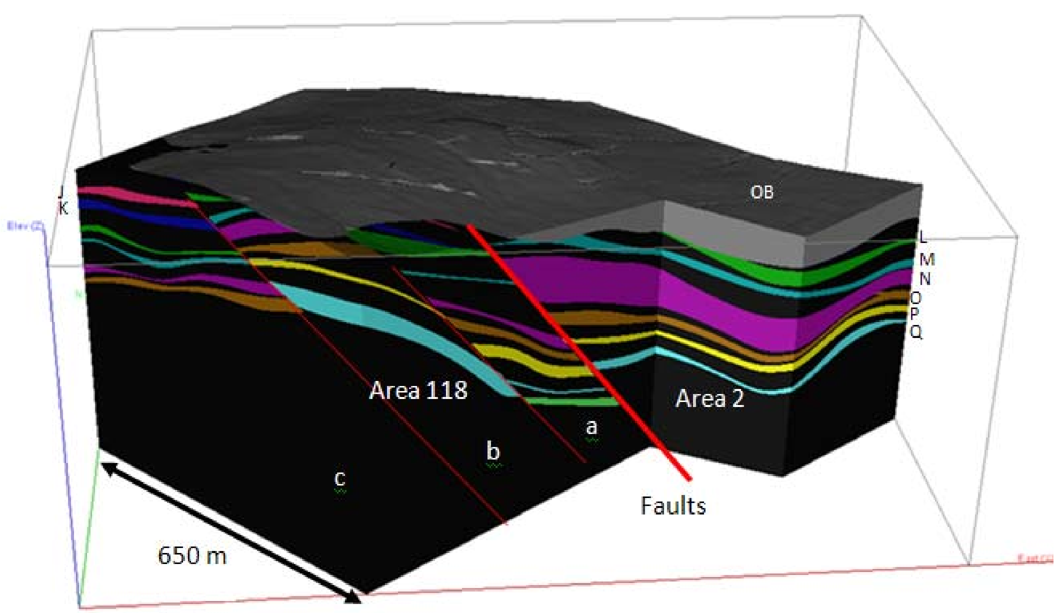
The contacts for a specific contact surface are made active by snapping polylines to the drill hole vertices, such that the polyline vertices are then used by GEMS as controls on the surface gridding. The grid triangulation vertices are then exported and re-imported as points. The final contact surface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

### 16.3.2 Data

At total of 12,109 grade measurements have been used in the design of mineralized domains from holes drilled roughly at 30 to 60 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. Note that previous resource estimates were based on 3.0 m composites.

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 14,188 composite assays were produced from 192 holes. The average thickness of highly mineralized horizons is 13 m (L, M, O, P) and 19 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.

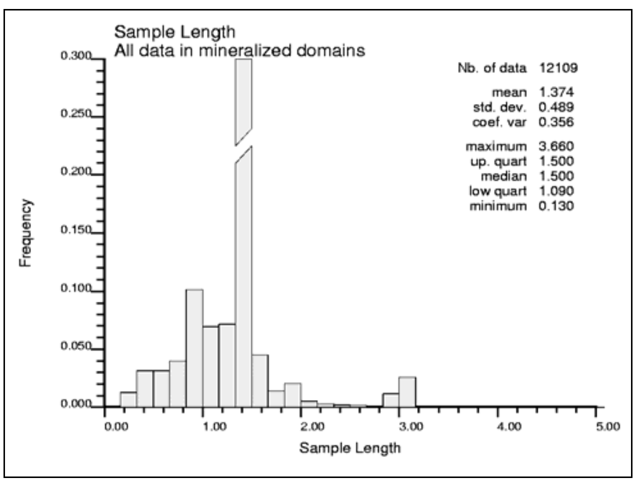
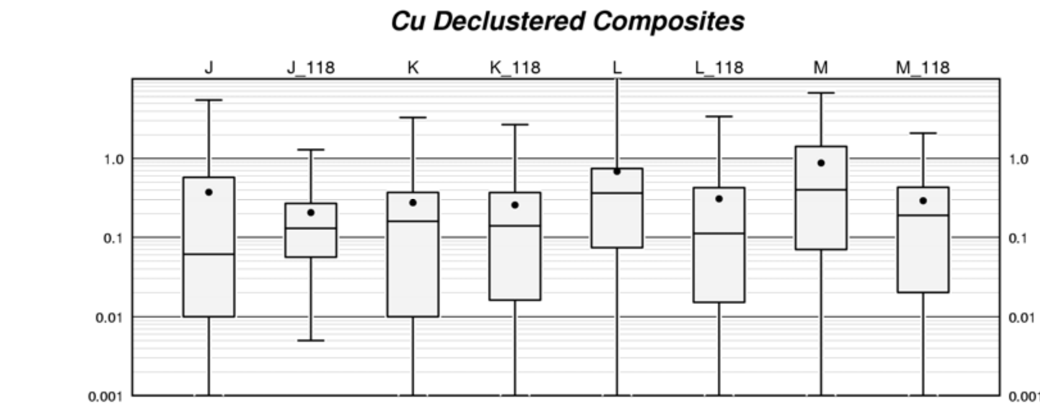
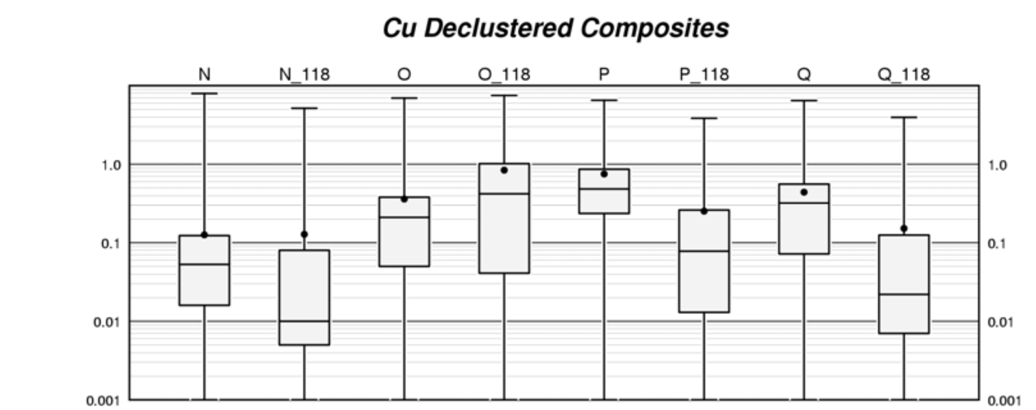



**Figure 16.2: Area 2/118 Histogram of Sample Lengths**




| | J | J_118 | K | K_118 | L | L_118 | M | M_118 | |
|---|---|---|---|---|---|---|---|---|---|
| Number of data | 248 | 113 | 675 | 332 | 1218 | 326 | 1915 | 373 | Number of data |
| Mean | 0.373 | 0.206 | 0.275 | 0.257 | 0.684 | 0.308 | 0.876 | 0.291 | Mean |
| Std. Dev. | 0.642 | 0.245 | 0.402 | 0.335 | 1.173 | 0.459 | 1.059 | 0.34 | Std. Dev. |
| Coef. of Var. | 1.72 | 1.19 | 1.464 | 1.307 | 1.715 | 1.49 | 1.209 | 1.169 | Coef. of Var. |
| Maximum | 5.5 | 1.29 | 3.308 | 2.682 | 15.646 | 3.4 | 6.674 | 2.091 | Maximum |
| Upper quartile | 0.577 | 0.269 | 0.37 | 0.369 | 0.744 | 0.426 | 1.42 | 0.43 | Upper quartile |
| Median | 0.061 | 0.13 | 0.16 | 0.14 | 0.364 | 0.112 | 0.4 | 0.19 | Median |
| Lower quartile | 0.01 | 0.056 | 0.01 | 0.016 | 0.074 | 0.015 | 0.07 | 0.02 | Lower quartile |
| Minimum | 0.001 | 0.005 | 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**




| | N | N_118 | O | O_118 | P | P_118 | Q | Q_118 | |
|---|---|---|---|---|---|---|---|---|---|
| Number of data | 3492 | 468 | 751 | 528 | 562 | 234 | 612 | 220 | Number of data |
| Mean | 0.126 | 0.128 | 0.361 | 0.841 | 0.752 | 0.252 | 0.441 | 0.152 | Mean |
| Std. Dev. | 0.322 | 0.343 | 0.625 | 1.219 | 0.916 | 0.528 | 0.603 | 0.413 | Std. Dev. |
| Coef. of Var. | 2.561 | 2.681 | 1.732 | 1.449 | 1.217 | 2.095 | 1.367 | 2.715 | Coef. of Var. |
| Maximum | 7.963 | 5.154 | 6.95 | 7.515 | 6.52 | 3.85 | 6.47 | 3.96 | Maximum |
| Upper quartile | 0.123 | 0.08 | 0.381 | 1.018 | 0.865 | 0.26 | 0.558 | 0.125 | Upper quartile |
| Median | 0.053 | 0.01 | 0.21 | 0.421 | 0.485 | 0.078 | 0.32 | 0.022 | Median |
| Lower quartile | 0.016 | 0.005 | 0.05 | 0.041 | 0.236 | 0.013 | 0.072 | 0.007 | Lower quartile |
| Minimum | 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**

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.

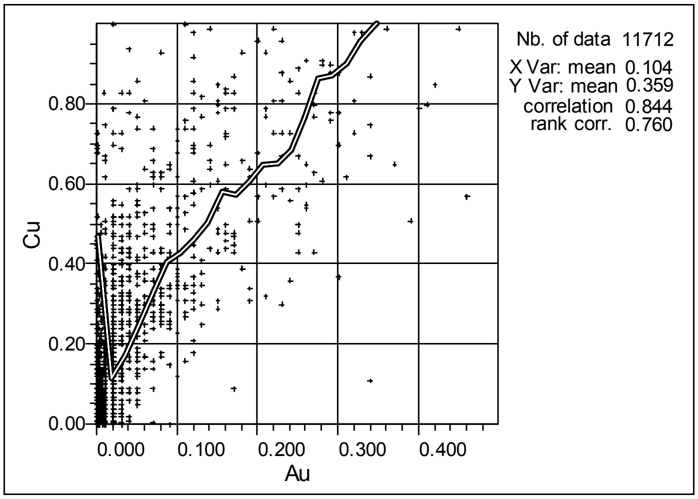



**Figure 16.5: Area 2/118 - Bivariate Statistics of Cu and Au Assays**

## 16.3.3 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.

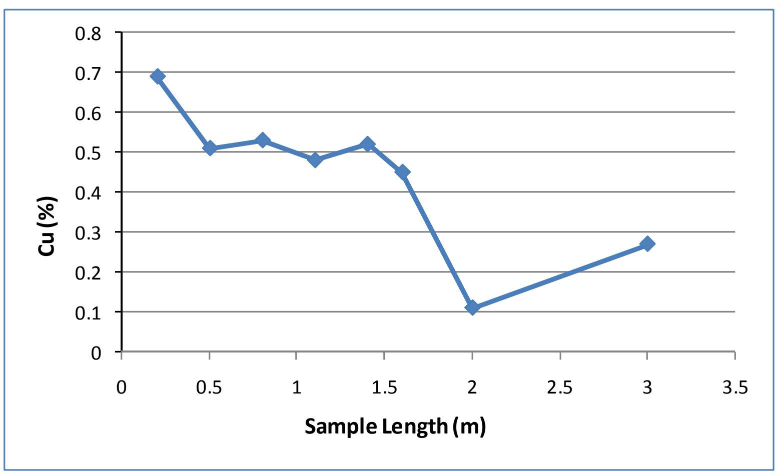



**Figure 16.6: Area 2/118 Grade Variation with the Sample Length**

### 16.3.4 Variogram Analysis

Experimental variograms and variogram models in the form of correlograms were generated for Cu and Au grades in the Area 2. For the Area 118, one generic variogram model was designed based on the results from the Area 2. The nugget effect values (i.e., metal variability at very close distance) were established from downhole variograms. The nugget values range from 5 to 20 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 Table 16.3. Note that no variogram models were designed for Ag grades. The Ag was estimated by the inverse distance squared method.

**Table 16.3: Area 2/118 - 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 |
| J | 0.05 | 0.55 | -60 | 0 | 0 | 110 | 30 | 12 |
| | | 0.40 | | | | 150 | 50 | 15 |
| K | 0.20 | 0.60 | 45 | 0 | 0 | 30 | 70 | 15 |
| | | 0.20 | | | | 200 | 100 | 20 |
| L | 0.20 | 0.60 | 45 | 0 | 0 | 55 | 75 | 17 |
| | | 0.20 | | | | 600 | 200 | 19 |
| M | 0.05 | 0.60 | 100 | 18 | -37 | 40 | 100 | 20 |
| | | 0.35 | | | | 350 | 160 | 60 |
| N | 0.15 | 0.60 | 45 | 15 | 0 | 30 | 45 | 15 |
| | | 0.25 | | | | 130 | 200 | 55 |
| O | 0.20 | 0.60 | 45 | 0 | 0 | 50 | 120 | 17 |
| | | 0.20 | | | | 100 | 170 | 22 |
| P | 0.10 | 0.45 | 45 | 15 | 0 | 25 | 25 | 20 |
| | | 0.45 | | | | 145 | 145 | 28 |
| Q | 0.10 | 0.50 | 75 | 15 | 0 | 25 | 25 | 15 |
| | | 0.40 | | | | 80 | 180 | 80 |
| All 118 | 0.05 | 0.85 | 60 | 15 | 0 | 90 | 70 | 18 |
| | | 0.10 | | | | 160 | 100 | 200 |

### 16.3.5 Resource Estimation Methodology

The geometrical parameters of the block model are summarised in Table 16.4.

### Table 16.4: Specifications for the Area 2/118 Block Model

| Description | Easting (X) | Northing (Y) | Elevation (Z) |
|---|---|---|---|
| Block Model Origin | 384,270 | 6,943,900 | 1000 |
| Parent Block Dimension | 10 | 10 | 3 |
| Number of Blocks | 132 | 86 | 225 |
| Minimum Sub- Block Dimension | No Sub-block | | |
| 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.

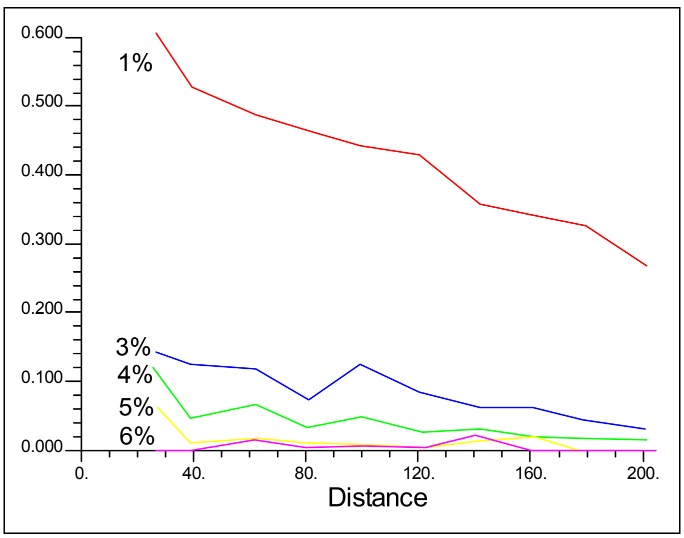



**Figure 16.7: Area 2/118 Continuity of High Grade Assays at Different Thresholds**

### 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 16.5). 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.

Note that in Area 2 long ranges of continuity were established separately for each mineralized domain. In Area 118 the long range of continuity was assumed at 135° azimuth.

**Table 16.5: Area 2/118 - Estimation parameters**

| Parameters | 80 | 90 | 100 | 110 | 120 | 140 | 160 |
|---|---|---|---|---|---|---|---|
| Rotated X (m) | 60 | 60 | 60 | 35 | 60 | 60 | 60 |
| Rotated Y (m) | 35 | 35 | 35 | 60 | 35 | 35 | 35 |
| Rotated Z (m) | 20 | 20 | 20 | 20 | 20 | 20 | 20 |
| Minimum data | 4 | 4 | 4 | 4 | 4 | 4 | 4 |
| Maximum data | 16 | 16 | 16 | 16 | 16 | 16 | 16 |
| Max number of samples per drill hole | 4 | 4 | 4 | 4 | 4 | 4 | 4 |
| Minimum number of octants | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Minimum number of holes | 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 and higher than 3.2 were excluded. 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.

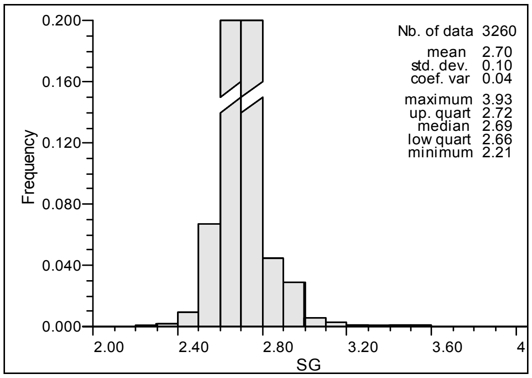



**Figure 16.8: Area 2/118 - Distribution of SG Values in the Mineralized Domains**

### 16.3.6 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. On average, the estimated blocks are very similar to the composite data, with high correlation between the estimates and the assays.

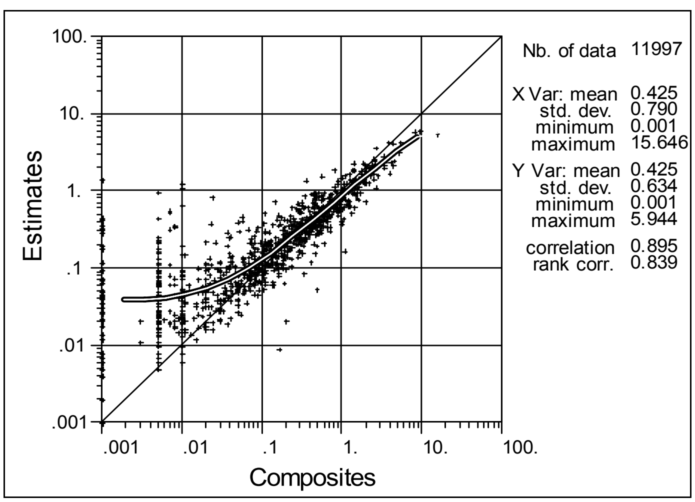



**Figure 16.9: Area 2/118 - Comparison of Cu Block Estimates with Composite Assay Data Contained Within the Blocks**

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 very good reflection of drill hole assay data.

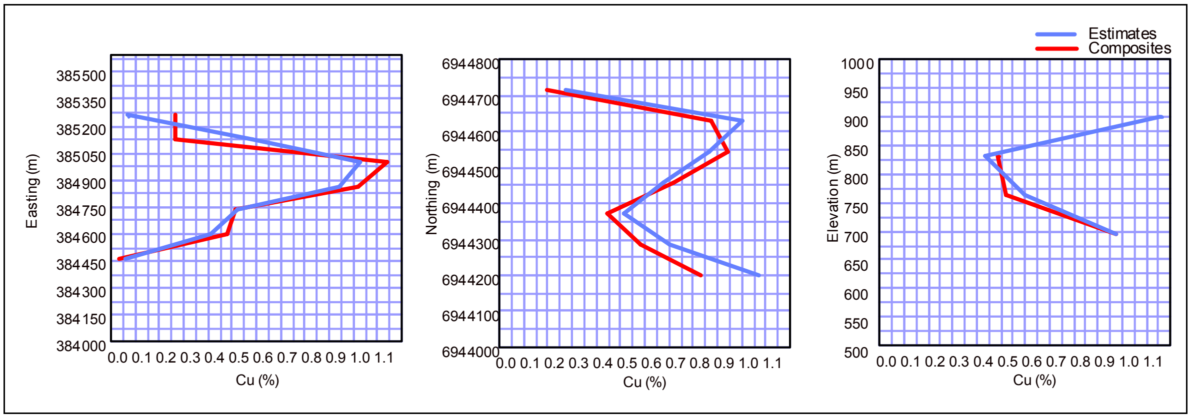



**Figure 16.10: Area 2/118 - Declustered Average Cu Composite Grades Compared to Cu Block Estimates in the M zone**

## 16.3.7 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 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;
- Average distance to the composites used to estimate a block

In order to classify mineralization as an 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 if informed from at least 12 composites from three or more separate drill holes within a search ellipse of the same orientation as used for estimating the blocks, but at a reduced size of 30x20x15 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 if informed from at least 8 composites from two or more separate drill holes within an ellipse of the same orientation as used for estimating the blocks, but a reduced size of 45 x 30 x 20 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.

### 16.3.8 Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 16.6 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 16.6: Area 2/118 - Sensitivity Analysis of Global Tonnage and Grades Deposit at Various Cu Cut-off Grades**

| Classification | Cut-Off (Cu%) | Tonnes (Kt)* | Copper (%) | Gold (g/t) | Silver (g/t) | Contained Cu (K lb.)* | Contained Gold (K oz)* | Contained Ag (K oz)* |
|---|---|---|---|---|---|---|---|---|
| Measured (M) | >2.0 | 1,014 | 2.61 | 1.15 | 9.36 | 58,250 | 38 | 305 |
| | >1.5 | 2,075 | 2.16 | 0.92 | 7.59 | 99,055 | 61 | 507 |
| | >1.0 | 3,461 | 1.79 | 0.73 | 6.26 | 136,790 | 81 | 696 |
| | >0.5 | 6,936 | 1.25 | 0.47 | 4.29 | 190,638 | 104 | 956 |
| | >0.4 | 8,301 | 1.12 | 0.41 | 3.81 | 204,095 | 109 | 1,017 |
| | >0.3 | 9,994 | 0.99 | 0.35 | 3.34 | 217,082 | 113 | 1,073 |
| | >0.2 | 12,604 | 0.83 | 0.29 | 2.79 | 231,223 | 117 | 1,132 |
| | >0.1 | 17,537 | 0.64 | 0.21 | 2.12 | 246,839 | 120 | 1,196 |
| Indicated (I) | >2.0 | 585 | 2.78 | 1.15 | 12.62 | 35,856 | 22 | 237 |
| | >1.5 | 1,189 | 2.24 | 0.89 | 9.65 | 58,688 | 34 | 369 |
| | >1.0 | 2,692 | 1.66 | 0.61 | 6.71 | 98,269 | 53 | 581 |
| | >0.5 | 11,301 | 0.92 | 0.29 | 3.36 | 230,198 | 106 | 1,220 |
| | >0.4 | 15,802 | 0.79 | 0.24 | 2.83 | 274,442 | 121 | 1,440 |
| | >0.3 | 21,914 | 0.67 | 0.19 | 2.37 | 321,347 | 136 | 1,673 |
| | >0.2 | 29,652 | 0.56 | 0.15 | 1.98 | 363,584 | 147 | 1,890 |
| | >0.1 | 41,085 | 0.44 | 0.12 | 1.58 | 400,145 | 157 | 2,093 |
| Total (M+I)** | >2.0 | 1,599 | 2.67 | 1.15 | 10.56 | 94,106 | 59 | 543 |
| | >1.5 | 3,264 | 2.19 | 0.91 | 8.34 | 157,743 | 95 | 875 |
| | >1.0 | 6,153 | 1.73 | 0.68 | 6.46 | 235,059 | 134 | 1,277 |
| | >0.5 | 18,237 | 1.05 | 0.36 | 3.71 | 420,836 | 210 | 2,176 |
| | >0.4 | 24,102 | 0.90 | 0.30 | 3.17 | 478,537 | 230 | 2,457 |
| | >0.3 | 31,908 | 0.77 | 0.24 | 2.68 | 538,429 | 249 | 2,746 |
| | >0.2 | 42,257 | 0.64 | 0.19 | 2.22 | 594,807 | 264 | 3,022 |
| | >0.1 | 58,622 | 0.50 | 0.15 | 1.75 | 646,985 | 277 | 3,289 |
| Inferred | >2.0 | 366 | 2.20 | 0.74 | 8.88 | 17,758 | 9 | 104 |
| | >1.5 | 591 | 2.02 | 0.69 | 8.08 | 26,282 | 13 | 154 |
| | >1.0 | 1,442 | 1.52 | 0.49 | 5.42 | 48,380 | 23 | 251 |
| | >0.5 | 5,116 | 0.91 | 0.24 | 2.99 | 102,420 | 40 | 492 |
| | >0.4 | 7,712 | 0.75 | 0.19 | 2.48 | 127,756 | 48 | 615 |
| | >0.3 | 11,334 | 0.62 | 0.15 | 2.08 | 155,655 | 55 | 756 |
| | >0.2 | 14,595 | 0.54 | 0.13 | 1.83 | 173,356 | 60 | 859 |
| | >0.1 | 21,026 | 0.42 | 0.10 | 1.48 | 193,801 | 65 | 999 |

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

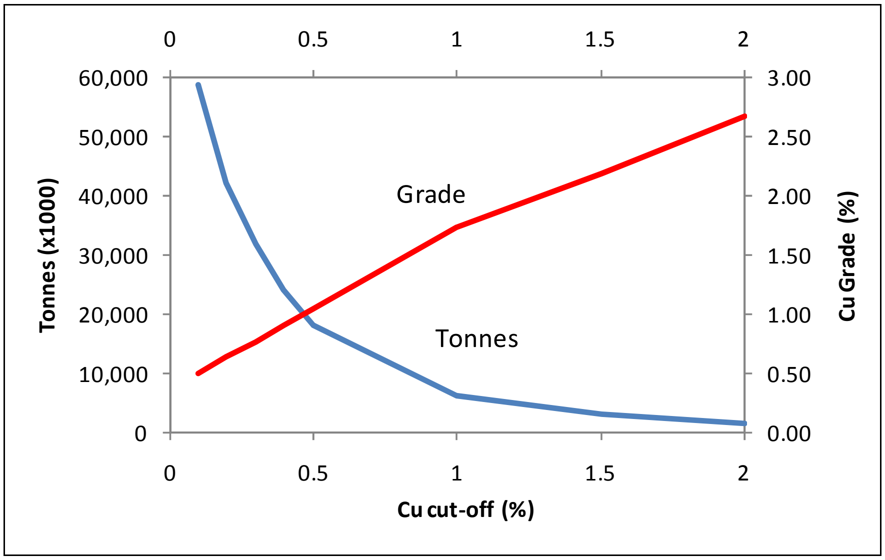



**Figure 16.11: Area 2/118 - Cu Grade Tonnage Curve for Measured and Indicated Resources**

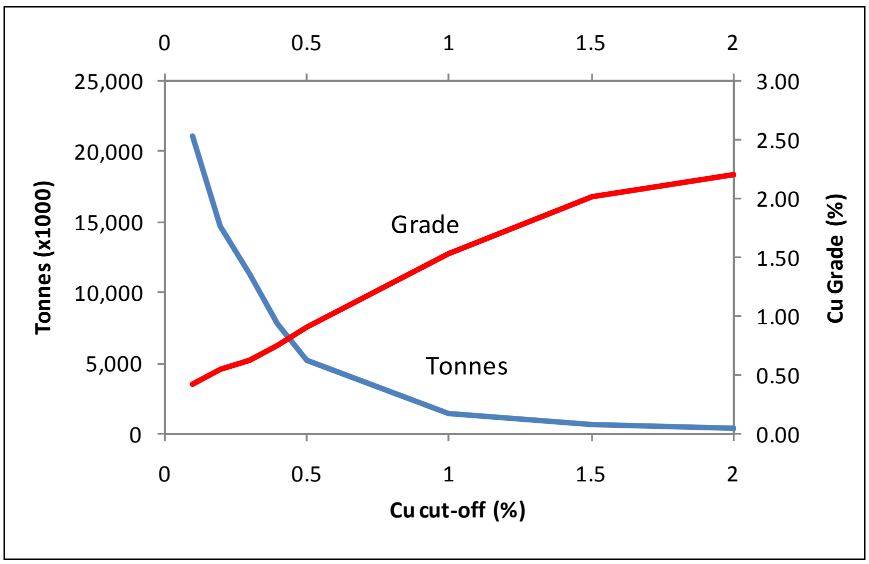



**Figure 16.12: Area 2/118 - Cu Grade Tonnage Curve for Inferred Resources**

### 16.3.9 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 16.7 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 16.7: Mineral Resource Statement at 0.5% Cu Cut-off for the Area 2/118 Deposit, SRK Consulting June 9, 2009**

| Classification | Tonnes (Kt)* | Copper (%) | Gold (g/t) | Silver (g/t) | Contained Cu (K lb.)* | Contained Gold (K oz)* | Contained Ag (K oz)* |
|---|---|---|---|---|---|---|---|
| Measured (M) | 6,936 | 1.25 | 0.47 | 4.29 | 190,638 | 104 | 956 |
| Indicated (I) | 11,301 | 0.92 | 0.29 | 3.36 | 230,198 | 106 | 1,220 |
| **Sub-total (M+I)**** | **18,237** | **1.05** | **0.36** | **3.71** | **420,836** | **210** | **2,176** |
| **Inferred** | **5,116** | **0.91** | **0.24** | **2.99** | **102,420** | **40** | **492** |

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

## 16.4 Ridgetop Deposit

### 16.4.1 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 6.2.1 (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 16.8 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 16.8: Ridgetop Modelled Domain Names and Block Model Codes

| Domain Name | Block Model Code | Comments |
|---|---|---|
| R80 | 80 | Chalcocite partial oxidation |
| R90 | 90 | Chalcocite partial oxidation |
| R100 | 100 | 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 | |

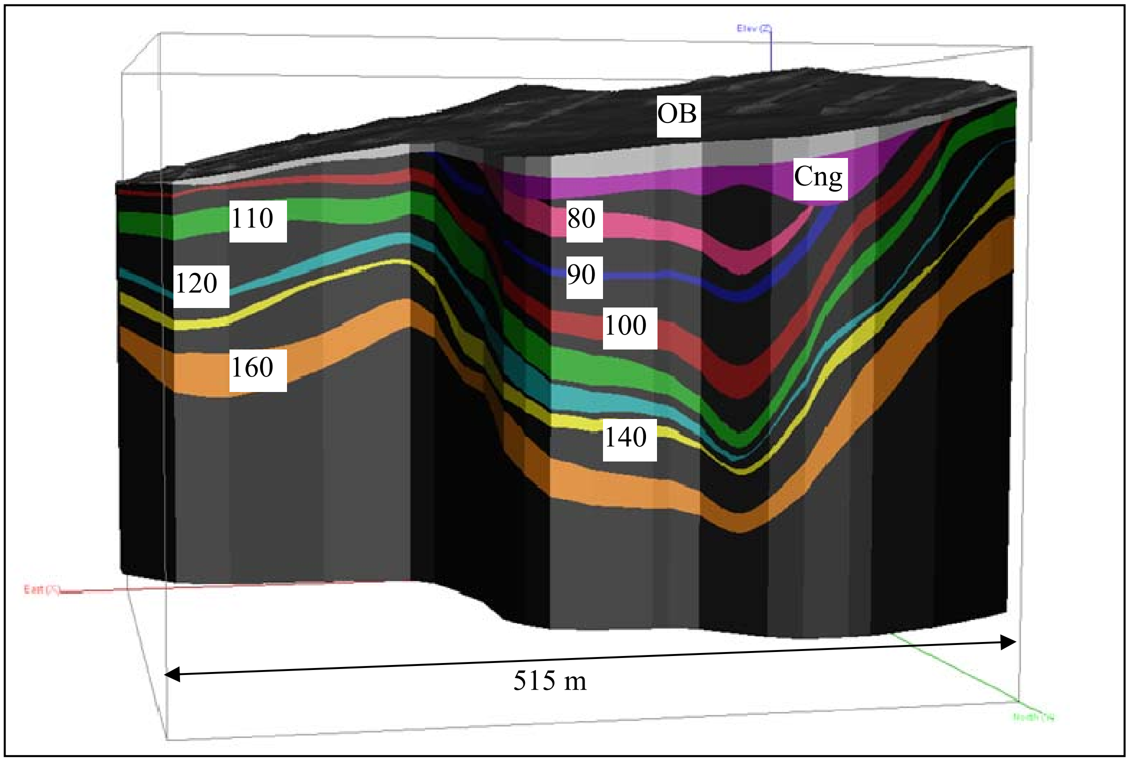



**Figure 16.13: View South of the Modelled Ridgetop Mineralized and Waste Domains**

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.

## 16.4.2 Data

A total of 4,831 grade measurements have been used in the design of mineralized domains (block model code 80-160) 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 22% 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 4,622 composite assays were produced from 119 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.

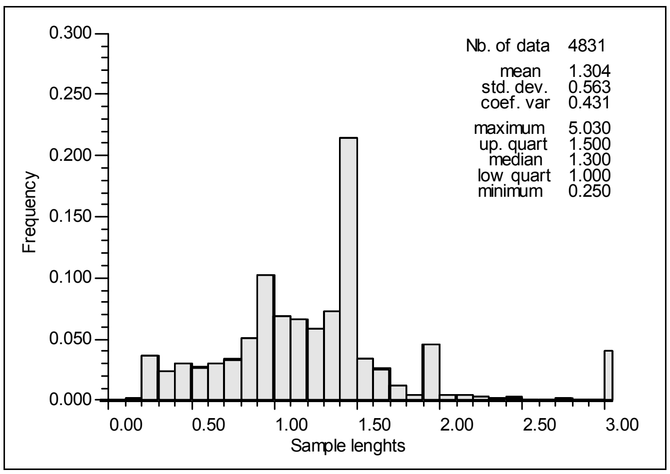
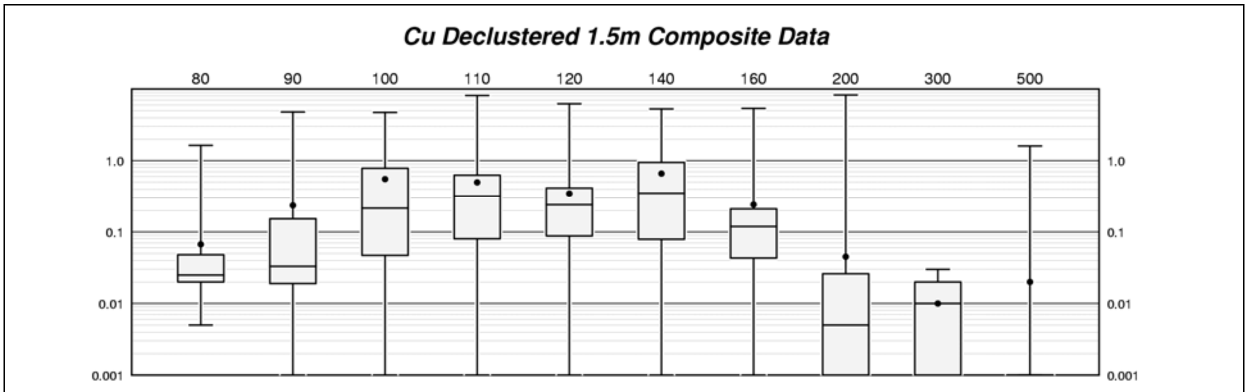



**Figure 16.14: Ridgetop - Histogram of Sample Lengths**




| | 80 | 90 | 100 | 110 | 120 | 140 | 160 | 200 | 300 | 500 | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Number of data | 169 | 259 | 778 | 1041 | 750 | 723 | 902 | 5928 | 100 | 484 | Number of data |
| Mean | 0.067 | 0.235 | 0.552 | 0.497 | 0.346 | 0.66 | 0.241 | 0.045 | 0.01 | 0.02 | Mean |
| Std. Dev. | 0.178 | 0.52 | 0.796 | 0.633 | 0.468 | 0.867 | 0.611 | 0.187 | 0.008 | 0.115 | Std. Dev. |
| Coef. of Var. | 2.665 | 2.21 | 1.443 | 1.275 | 1.352 | 1.315 | 2.541 | 4.106 | 0.81 | 5.641 | Coef. of Var. |
| Maximum | 1.635 | 4.793 | 4.72 | 8.142 | 6.23 | 5.281 | 5.37 | 8.27 | 0.03 | 1.603 | Maximum |
| Upper quartile | 0.048 | 0.153 | 0.784 | 0.626 | 0.413 | 0.947 | 0.21 | 0.026 | 0.02 | 0.001 | Upper quartile |
| Median | 0.025 | 0.033 | 0.215 | 0.318 | 0.24 | 0.35 | 0.119 | 0.005 | 0.01 | 0.001 | Median |
| Lower quartile | 0.02 | 0.019 | 0.047 | 0.08 | 0.088 | 0.079 | 0.043 | 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 three highest grade domains. Note quite good rank correlation, 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.

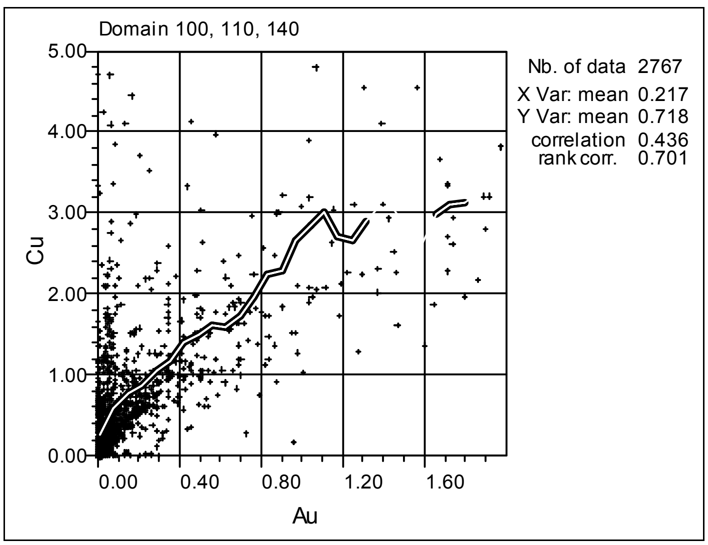



**Figure 16.16: Ridgetop - Bivariate Statistics of Cu and Au Assays**

### 16.4.3 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.

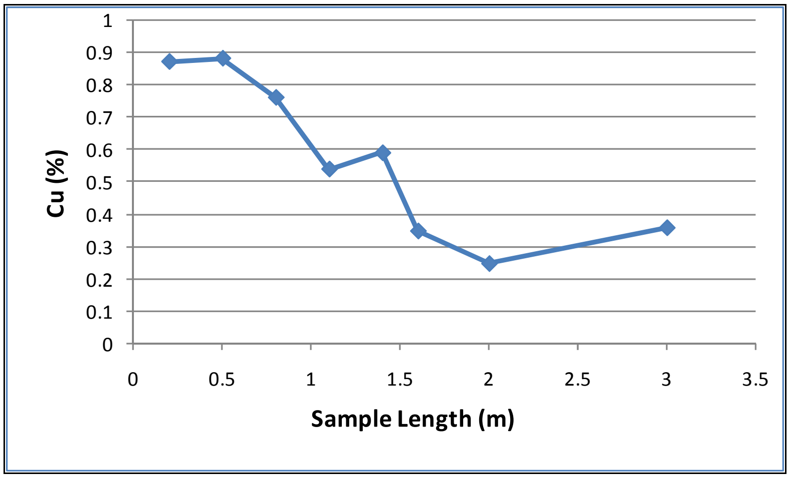



**Figure 16.17: Ridgetop Grade Variation with the Sample Length**

### 16.4.4 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 16.9. Note that no variogram models were designed for Ag grades. The Ag was estimated by the inversed distance squared method.

**Table 16.9: Ridgetop Cu Exponential Variogram Models**

| Zone | Nugget C0 | Sill C1 and C2 | Gemcom Rotations (RRR rule) | | | Ranges a1, a2 | | |
|---|---|---|---|---|---|---|---|---|
| | | | around Z | around Y | around Z | X-Rot | Y-Rot | Z-Rot |
| 80* | 0.05 | 0.75 | 50 | 24 | -48 | 60 | 30 | 15 |
| | | 0.20 | | | | 600 | 100 | 20 |
| 90* | 0.05 | 0.75 | 50 | 24 | -48 | 60 | 30 | 15 |
| | | 0.20 | | | | 600 | 100 | 20 |
| 100 | 0.05 | 0.75 | 50 | 24 | -48 | 60 | 30 | 15 |
| | | 0.20 | | | | 600 | 100 | 20 |
| 110 | 0.10 | 0.60 | 50 | 24 | -22 | 50 | 80 | 15 |
| | | 0.30 | | | | 160 | 800 | 60 |
| 120 | 0.25 | 0.50 | 50 | 24 | -48 | 70 | 40 | 10 |
| | | 0.25 | | | | 200 | 100 | 12 |
| 140 | 0.05 | 0.75 | 50 | 24 | -48 | 70 | 50 | 15 |
| | | 0.20 | | | | 600 | 300 | 20 |
| 160 | 0.10 | 0.50 | 50 | 24 | -48 | 60 | 40 | 12 |
| | | 0.40 | | | | 200 | 140 | 60 |

*Variogram models assigned from Domain 100

### 16.4.5 Resource Estimation Methodology

The geometrical parameters of the block model are summarised in Table 16.10.

**Table 16.10: Specifications for the Ridgetop Block Model**

| Description | Easting (X) | Northing (Y) | Elevation (Z) |
|---|---|---|---|
| Block Model Origin | 384,650 | 6,943,200 | 1000 |
| Parent Block Dimension | 10 | 10 | 3 |
| Number of Blocks | 90 | 90 | 135 |
| Minimum Sub- Block Dimension | No Sub-block | | |
| 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 40x30x15 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.

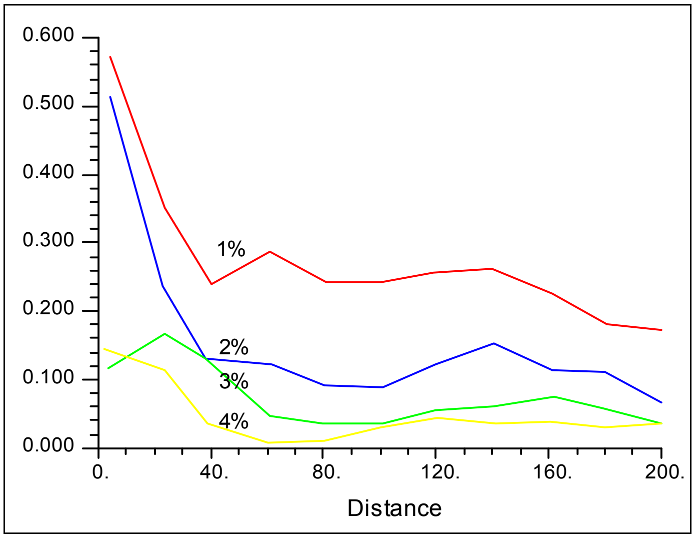



**Figure 16.18: Ridgetop - Continuity of High Grade Assays at Different Thresholds**

### 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 16.11).

**Table 16.11: Ridgetop Estimation parameters**

| Parameters | 80 | 90 | 100 | 110 | 120 | 140 | 160 |
|---|---|---|---|---|---|---|---|
| Rotated X (m) | 60 | 60 | 60 | 35 | 60 | 60 | 60 |
| Rotated Y (m) | 35 | 35 | 35 | 60 | 35 | 35 | 35 |
| Rotated Z (m) | 20 | 20 | 20 | 20 | 20 | 20 | 20 |
| Min data | 4 | 4 | 4 | 4 | 4 | 4 | 4 |
| Max data | 16 | 16 | 16 | 16 | 16 | 16 | 16 |
| Max number of samples per dh | 4 | 4 | 4 | 4 | 4 | 4 | 4 |
| Minimum number of octants | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Minimum number of holes | 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, 14 very high SG values were excluded. Block specific gravity values were estimated by the ID2 method. At least eight samples within a 100 x 100 x 25 m radius were needed to estimate a block.

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

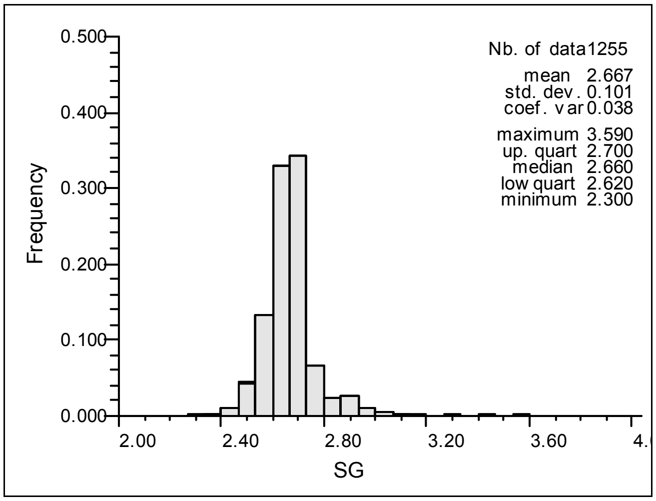



**Figure 16.19: Ridgetop Distribution of SG Values in the Mineralized Domains**

## 16.4.6 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.

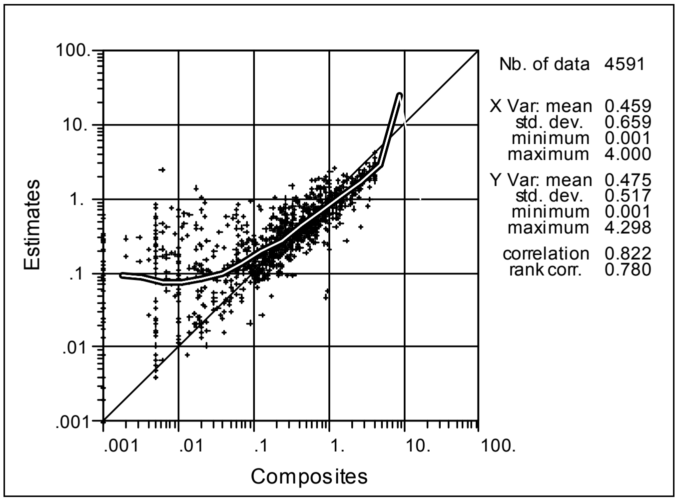



**Figure 16.20: Ridgetop Comparison of Cu Block Estimates with Composite Assay Data Contained Within the Blocks**

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.

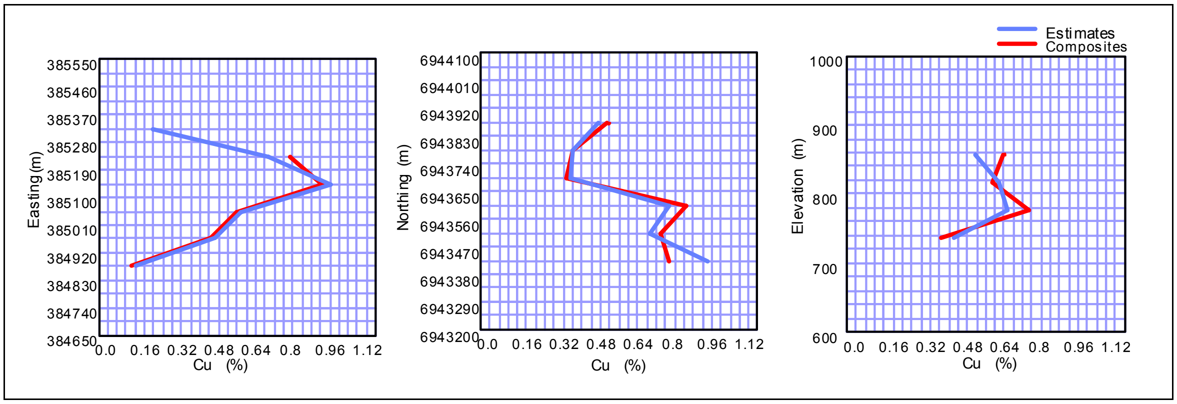



**Figure 16.21: Ridgetop Declustered Average Cu Composite Grades Compared to Cu Block Estimates in the 140 zone**

## 16.4.7 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;
- 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 if informed from at least 12 composites from three or more separate drill holes within a search ellipse of the same orientation as used for estimating the blocks, but at reduced size of 30 x 20 x 15 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 if informed from at least 8 composites from two or more separate drill holes within a search ellipse of the same orientation as used for estimating the blocks, but a reduced size of 45 x 30 x 20 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.

### 16.4.8 Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 16.12 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 16.12: Ridgetop Sensitivity Analysis of Global Tonnage and Grades in the Ridgetop Deposit at Various Cu Cut-off Grades**

| Classification | Cut-Off (Cu%) | Tonnes (000s)* | Copper (%) | Gold (g/t) | Silver (g/t) | Contained Cu (000s lbs)* | Contained Gold (000s oz)* | Contained Ag (000s oz)* |
|---|---|---|---|---|---|---|---|---|
| Measured (M) | >2.0 | 51 | 2.33 | 0.76 | 4.43 | 2,606 | 1 | 7 |
| | >1.5 | 198 | 1.87 | 0.63 | 3.64 | 8,173 | 4 | 23 |
| | >1.0 | 569 | 1.44 | 0.45 | 2.95 | 18,036 | 8 | 54 |
| | >0.5 | 1,568 | 0.98 | 0.26 | 2.12 | 33,719 | 13 | 107 |
| | >0.4 | 1,848 | 0.90 | 0.24 | 1.97 | 36,500 | 14 | 117 |
| | >0.3 | 2,138 | 0.82 | 0.21 | 1.83 | 38,749 | 15 | 126 |
| | >0.2 | 2,449 | 0.75 | 0.19 | 1.68 | 40,466 | 15 | 133 |
| | >0.1 | 2,810 | 0.67 | 0.17 | 1.53 | 41,644 | 15 | 138 |
| Indicated (I) | >2.0 | 142 | 2.52 | 1.27 | 11.47 | 7,893 | 6 | 52 |
| | >1.5 | 358 | 2.03 | 0.93 | 8.35 | 16,030 | 11 | 96 |
| | >1.0 | 758 | 1.60 | 0.66 | 6.07 | 26,813 | 16 | 148 |
| | >0.5 | 2,355 | 0.98 | 0.33 | 3.30 | 50,926 | 25 | 250 |
| | >0.4 | 3,043 | 0.86 | 0.28 | 2.84 | 57,694 | 27 | 278 |
| | >0.3 | 4,140 | 0.72 | 0.22 | 2.34 | 66,058 | 30 | 311 |
| | >0.2 | 5,857 | 0.58 | 0.17 | 1.85 | 75,397 | 32 | 348 |
| | >0.1 | 7,379 | 0.50 | 0.14 | 1.56 | 80,522 | 34 | 370 |
| Sub-total (M+I)** | >2.0 | 193 | 2.47 | 1.14 | 9.62 | 10,499 | 7 | 60 |
| | >1.5 | 556 | 1.98 | 0.82 | 6.67 | 24,203 | 15 | 119 |
| | >1.0 | 1,327 | 1.53 | 0.57 | 4.73 | 44,849 | 24 | 202 |
| | >0.5 | 3,923 | 0.98 | 0.30 | 2.83 | 84,645 | 38 | 357 |
| | >0.4 | 4,891 | 0.87 | 0.26 | 2.51 | 94,194 | 41 | 395 |
| | >0.3 | 6,279 | 0.76 | 0.22 | 2.16 | 104,806 | 44 | 437 |
| | >0.2 | 8,306 | 0.63 | 0.18 | 1.80 | 115,863 | 47 | 480 |
| | >0.1 | 10,189 | 0.54 | 0.15 | 1.55 | 122,167 | 49 | 508 |
| Inferred | >2.0 | 18 | 2.36 | 0.76 | 5.27 | 924 | 0 | 3 |
| | >1.5 | 59 | 1.91 | 0.63 | 5.13 | 2,498 | 1 | 10 |
| | >1.0 | 208 | 1.38 | 0.50 | 4.15 | 6,359 | 3 | 28 |
| | >0.5 | 686 | 0.90 | 0.26 | 2.38 | 13,644 | 6 | 53 |
| | >0.4 | 919 | 0.79 | 0.22 | 2.06 | 15,949 | 7 | 61 |
| | >0.3 | 1,265 | 0.67 | 0.18 | 1.75 | 18,607 | 7 | 71 |
| | >0.2 | 1,747 | 0.55 | 0.15 | 1.47 | 21,214 | 8 | 83 |
| | >0.1 | 2,458 | 0.44 | 0.12 | 1.18 | 23,599 | 9 | 93 |

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

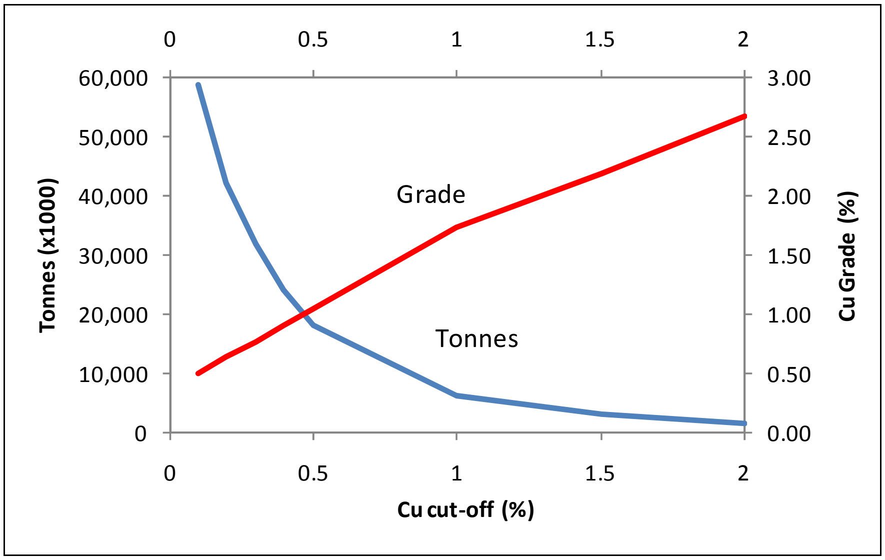



**Figure 16.22: Ridgetop Cu Grade Tonnage Curve for Measured and Indicated Resources**

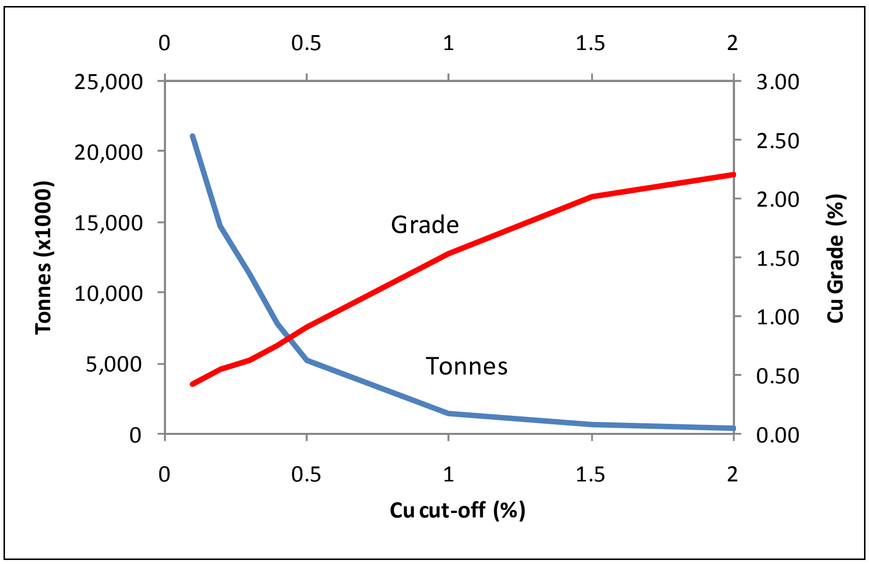



**Figure 16.23: Ridgetop - Cu Grade Tonnage Curve for Inferred Resources**

### 16.4.9 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 16.13 presents the mineral resource statement for the Ridgetop deposits.

**Table 16.13: Mineral Resource Statement at 0.5% Cu Cut-off for the Ridgetop Deposit, SRK Consulting June 9, 2009**

| Classification | Tonnes (Kt)* | Copper (%) | Gold (g/t) | Silver (g/t) | Contained Cu (K lbs)* | Contained Gold (K oz)* | Contained Ag (K oz)* |
|---|---|---|---|---|---|---|---|
| Measured (M) | 1,568 | 0.98 | 0.26 | 2.12 | 33,719 | 13 | 107 |
| Indicated (I) | 2,355 | 0.98 | 0.33 | 3.30 | 50,926 | 25 | 250 |
| **Sub-total (M+I)**** | **3,923** | **0.98** | **0.30** | **2.83** | **84,645** | **38** | **357** |
| **Inferred** | **686** | **0.90** | **0.26** | **2.38** | **13,644** | **6** | **53** |

**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 16.14) 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 drillholes 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 16.14: 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 |

### 16.5.1 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.

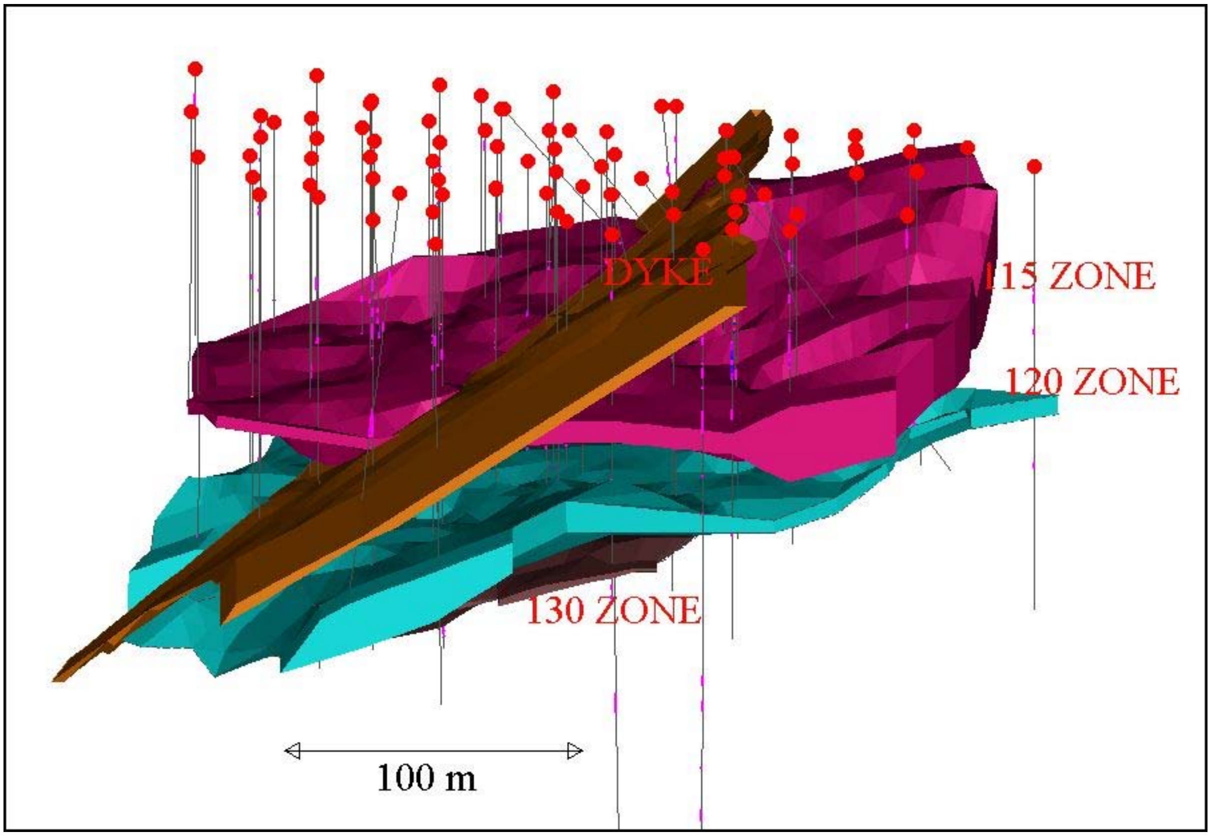



**Figure 16.24: View from the North of the Modelled Minto North Mineralized Domains**

### 16.5.2 Data

The drill hole database was supplied in electronic format by MintoEx. This included collars, downhole surveys, lithology data and assay data (i.e. Au g/t, Cu%, Ag g/t, SG with downhole 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 16.15 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 16.15: 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** |

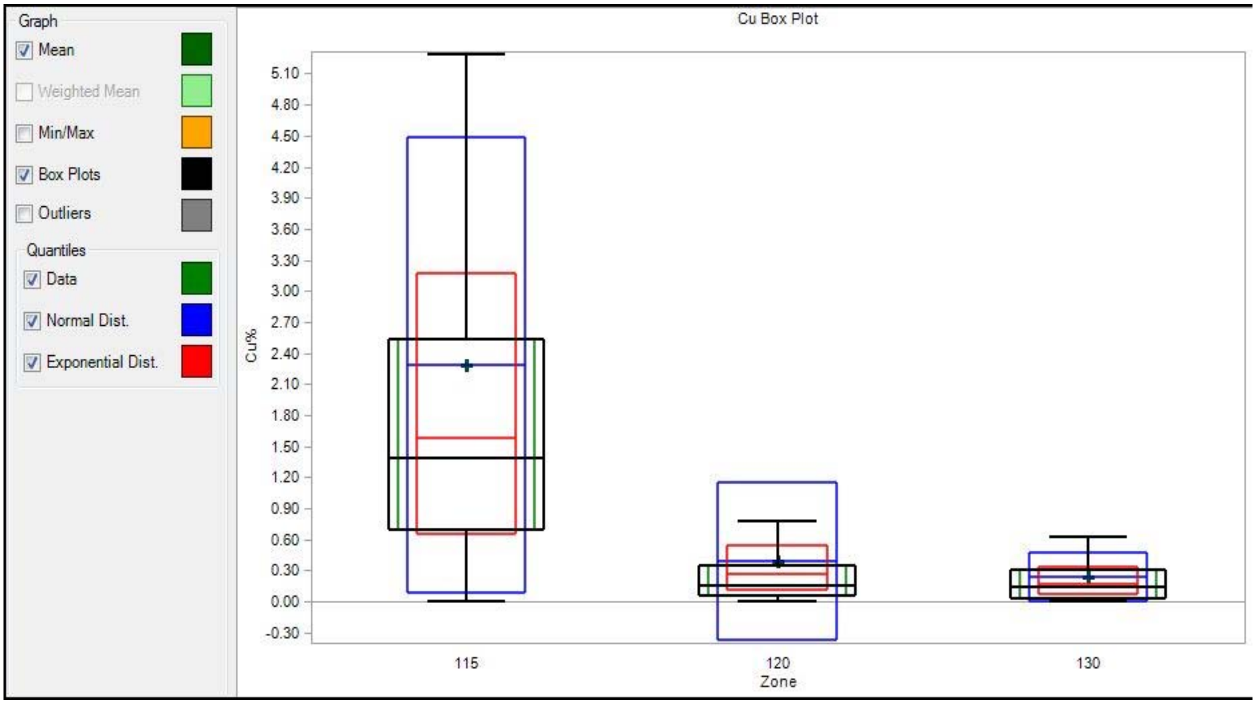



**Figure 16.25: Minto North - Basic statistics of Cu assay grades in the mineralized zones**

## 16.5.3 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 16.16 and Figure 16.26 shows 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 16.16: 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** |

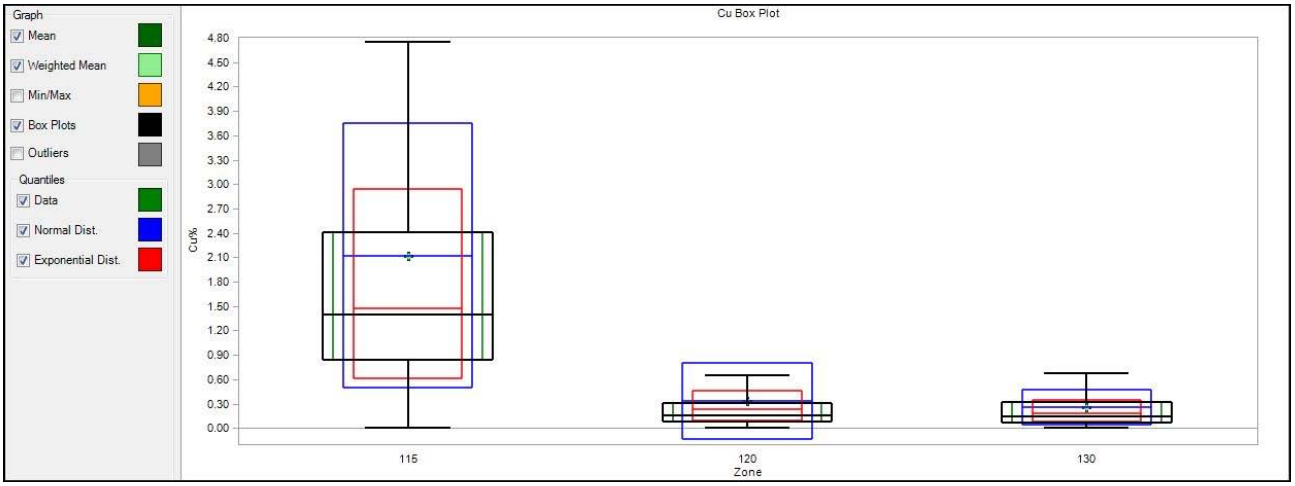



**Figure 16.26: Minto North – Basic Statistics of Cu Composite grades in the mineralized zones**

### 16.5.4 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.

### 16.5.5 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 100x100x25 m radius were needed to estimate a block. Values greater than 3.3 were limited to a 20 m radius in influence.

### 16.5.6 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 16.17. Note that the rotations of the angles are given according to the GSLIB convention used by MineSight™ Compass.

**Table 16.17: 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.

### 16.5.7 Block Model Definition

The Block Model used for calculating the resources was defined according to the limits specified in Table 16.18. 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 drillholes) which are at approximately 15 to 20 m centers and a proposed 3 m bench height.

**Table 16.18: Specifications for the Minto North Block Model**

| Description | Easting (X) | Northing (Y) | Elevation (Z) |
|---|---|---|---|
| Block Model Origin | 384,000 | 6,945,750 | 750 |
| Block Dimension | 10 | 10 | 3 |
| Number of Blocks | 60 | 50 | 80 |
| Rotation | 0 | 0 | 0 |

### 16.5.8 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 16.19 summarizes the search ellipse dimensions for the estimation passes.

**Table 16.19: 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 Drillhole |
|---|---|---|---|---|---|---|---|---|---|
| 1 | 70 | 60 | 10 | 165 | 0 | 0 | 4 | 16 | 4 |
| 2 | 40 | 30 | 7 | 165 | 0 | 0 | 4 | 16 | 4 |

### 16.5.9 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.

### 16.5.10 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 drillhole;
- 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 drillholes which was 3 however in cases where the minimum number of drillholes 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 drillholes, average distance to block from composite, number of composites used along with relative error, were evaluated to insure 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 drillholes was in most cases 2 however in cases where the minimum number of drillholes 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 drillholes, 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.

### 16.5.11 Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 16.19 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.

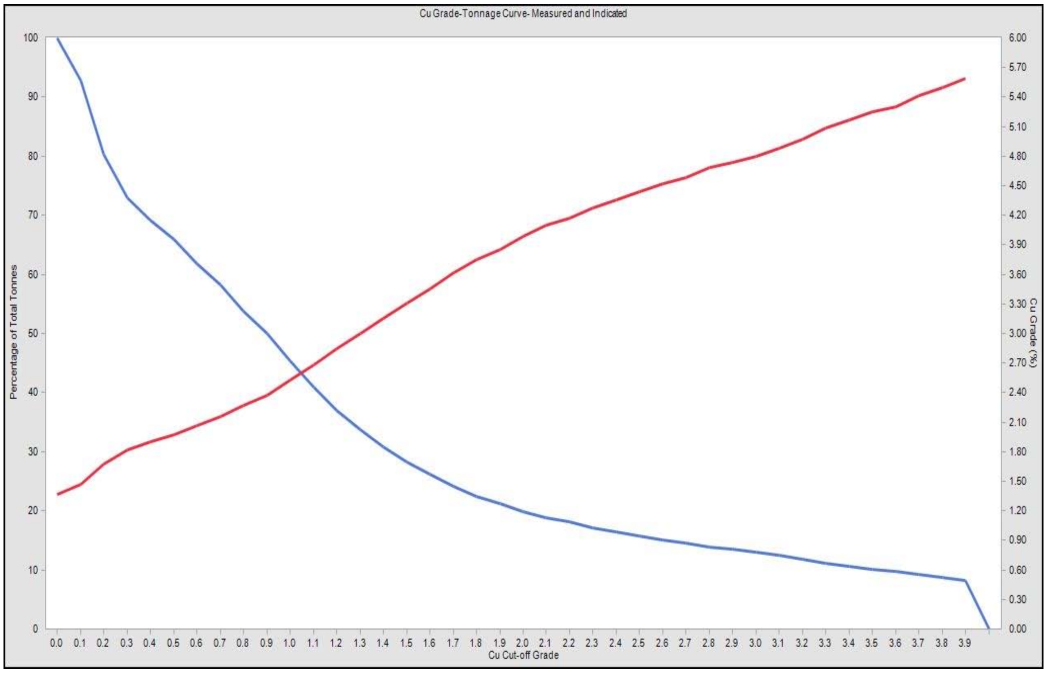



**Figure 16.27: Minto North - Cu Grade Tonnage Curve for Measured and Indicated Resources**

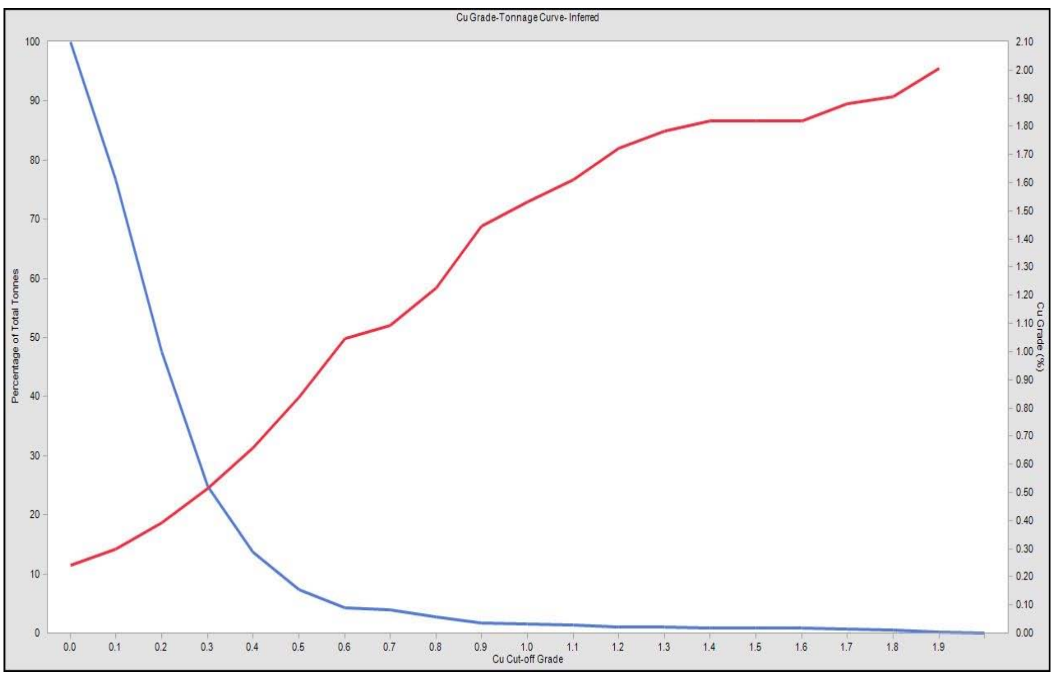



**Figure 16.28: Minto North - Cu Grade Tonnage Curve for Inferred Resources**

### 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 16.20 presents the mineral resource statement for the Minto North deposit.

**Table 16.20: 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** |

Table 16.21 presents combined mineral resource at a 0.5% Cu cut-off for Area 2/118, Ridgetop, and Minto North Deposits.

**Table 16.21: Combined Mineral Resource Statement at 0.5% Cu Cut-off for Area 2/118, Ridgetop, and Minto North Deposits, 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) | 10,348 | 1.37 | 0.55 | 4.57 | 311,887 | 183 | 1,519 |
| Indicated (I) | 13,920 | 0.94 | 0.30 | 3.39 | 287,179 | 136 | 1,519 |
| **Sub-total (M+I)**** | **24,267** | **1.12** | **0.41** | **3.89** | **599,066** | **319** | **3,038** |
| **Additional Inferred** | **5,827** | **0.91** | **0.25** | **2.93** | **116,520** | **46** | **548** |

*Excludes Minto Main deposit mineral resource

## 16.6 Mineral Reserves

### 16.6.1 Net Smelter Model

The 3D resource models were used as the basis for deriving the economic pit limit for the Phase IV 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 to the year-end 2009. 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 16.22 below.

### Table 16.22: NSR Parameters

| | | | | |
|---|---|---|---|---|
| **Metal Prices** | | | | **Comments** |
| Metal prices (US$) | US | $2.00 | /lb Cu | |
| | US | $300 | /oz Au | as per Silver Wheaton agreement |
| | US | $2.90 | /oz Ag | as per Silver Wheaton agreement |
| **Exchange Rate** | | | | **Comments** |
| US Dollars/Canadian Dollars | | 0.83 | | Estimate |
| **Grade Factor** | | | | **Comments** |
| Dilution | | 4.70% | % waste rock in mill feed | Minto North |
| | | 8.00% | | Area 2/118 |
| | | 10.30% | | 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 | 42.00% | | |
| | 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 MRI Trading contract (confidential) | | | | |
| **Treatment and Refining** | | | | **Comments** |
| Cu conc. treatment | US | $60.00 | /dmt | MRI Trading Contract |
| Cu refining | US | $0.06 | /lb Cu | MRI Trading Contract |
| Au refining | US | $6.00 | /oz Au | MRI Trading Contract |
| Ag refining | US | $0.40 | /oz Ag | MRI Trading Contract |
| **Freight and Marketing** | | | | **Comments** |
| Freight & Marketing (all inclusive) | US | $134.26 | /wmt | includes trucking; shipping; port charges; insurance |
| Freight & Marketing (all inclusive) | US | $145.93 | /dmt | |
| **Royalty** | | | | **Comments** |
| Royalty charge | | - | of Net Value | Payable to Selkirk First Nations (confidential cooperation agreement) |

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

### 16.6.2 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.

#### Optimization Parameters and Results

The geotechnical parameters, dilution/recovery, mining, milling and G&A costs (based on an assumed mill throughput of 1.37 MTPA) are summarized in Table 16.23. The estimated projected topography as of the end of Main Pit mining was used as the starting surface for the pit optimization and was based on the 2010 Budget schedule compiled by MintoEx in October 2009. 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 IV.

### Table 16.23: Pit Optimization Parameters

| Item | Unit | Value | | |
|---|---|---|---|---|
| | | North | Ridgetop | Area2/118 |
| ***Operating Costs*** | | | | |
| Waste mining Cost | C$/waste tonne | 2.11 | | |
| Ore Mining Cost | C$/ore tonne | 2.11 | | |
| Processing and G&A Cost | C$/milled tonne | 23.09 | | |
| ***Pit Slope Angles*** | | | | |
| Overburden | inter-ramp ° | 30° | 30° | 30° |
| Rock | inter-ramp ° | 52° | 47°west, 53°east | 47°west, 53°east |
| Dilution | % | 4.7% | 10.3% | 8.0% |
| Mining recovery | % | 100% | 100% | 100% |
| Strip ratio (est.) | t:t | 7.9 | 6.8 | 8.8 |
| Internal NSR cut-off | C$/t | 24.18 | 25.47 | 24.94 |
| External NSR cut-off (est.) | C$/t | 42.95 | 41.93 | 45.62 |
| Processing rate | t/day milled | 3,750 | | |
| Processing rate | t/yr milled | 1,368,750 | | |

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;
- Specific gravity.

The results of the Whittle™ pit optimization evaluation for varying revenue factors are summarized in Tables 16.24 through 16.26, as well as Figures 16.29 through to 16.31, 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 16.24: Ridgetop Pit Optimization Results

| Final Pit | Revenue Factor | Mine Life (yr) | Ore Diluted (tonnes) | Diluted Grades | | | | Contained Metal | | | Waste (tonnes) | Strip Ratio | Total (tonnes) | Total CF (C$) | NPV Best $ disc | NPV Worst $ disc |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | Cu (%) | Au (g/t) | Ag (g/t) | NSR (C$/t) | Cu (Mlbs) | Au (koz) | Ag (koz) | | | | | | |
| 10 | 0.56 | 0.1 | 68,705 | 1.55 | 0.03 | 1.92 | 57.67 | 2 | 0 | 4 | 118,034 | 1.72 | 186,739 | 1,981,786 | 1,972,327 | 1,972,327 |
| 11 | 0.58 | 0.1 | 134,566 | 1.76 | 0.52 | 6.33 | 69.05 | 5 | 2 | 27 | 831,436 | 6.18 | 966,002 | 4,147,033 | 4,108,356 | 4,108,356 |
| 12 | 0.60 | 0.1 | 143,209 | 1.74 | 0.51 | 6.28 | 68.20 | 5 | 2 | 29 | 862,973 | 6.03 | 1,006,182 | 4,336,819 | 4,293,787 | 4,293,787 |
| 13 | 0.62 | 0.1 | 154,673 | 1.71 | 0.50 | 6.10 | 66.95 | 6 | 2 | 30 | 897,292 | 5.80 | 1,051,964 | 4,563,901 | 4,515,010 | 4,515,010 |
| 14 | 0.64 | 0.1 | 166,049 | 1.69 | 0.50 | 6.10 | 66.15 | 6 | 3 | 33 | 953,705 | 5.74 | 1,119,755 | 4,786,626 | 4,731,599 | 4,731,599 |
| 15 | 0.66 | 0.1 | 172,063 | 1.67 | 0.50 | 6.06 | 65.57 | 6 | 3 | 33 | 976,250 | 5.67 | 1,148,313 | 4,886,577 | 4,828,379 | 4,828,379 |
| 16 | 0.68 | 0.1 | 194,158 | 1.66 | 0.54 | 6.20 | 65.50 | 7 | 3 | 39 | 1,177,013 | 6.06 | 1,371,171 | 5,341,059 | 5,269,335 | 5,269,335 |
| 17 | 0.70 | 0.2 | 233,565 | 1.63 | 0.58 | 6.27 | 64.71 | 8 | 4 | 47 | 1,498,641 | 6.42 | 1,732,206 | 6,066,704 | 5,968,834 | 5,968,834 |
| 18 | 0.72 | 0.2 | 238,356 | 1.62 | 0.58 | 6.21 | 64.31 | 9 | 4 | 48 | 1,512,999 | 6.35 | 1,751,355 | 6,129,010 | 6,028,124 | 6,028,124 |
| 19 | 0.74 | 0.2 | 278,182 | 1.58 | 0.60 | 6.15 | 62.74 | 10 | 5 | 55 | 1,777,533 | 6.39 | 2,055,715 | 6,692,001 | 6,563,620 | 6,563,620 |
| 20 | 0.76 | 0.2 | 293,534 | 1.55 | 0.58 | 5.95 | 61.66 | 10 | 5 | 56 | 1,825,990 | 6.22 | 2,119,524 | 6,850,296 | 6,711,700 | 6,711,700 |
| 21 | 0.78 | 0.2 | 305,556 | 1.54 | 0.58 | 5.91 | 61.15 | 10 | 6 | 58 | 1,898,137 | 6.21 | 2,203,693 | 6,980,644 | 6,833,688 | 6,833,688 |
| 22 | 0.80 | 0.2 | 318,281 | 1.53 | 0.58 | 5.85 | 60.73 | 11 | 6 | 60 | 1,989,966 | 6.25 | 2,308,247 | 7,110,571 | 6,954,714 | 6,954,714 |
| 23 | 0.82 | 0.2 | 329,287 | 1.51 | 0.57 | 5.77 | 60.09 | 11 | 6 | 61 | 2,033,076 | 6.17 | 2,362,363 | 7,198,464 | 7,035,287 | 7,035,287 |
| 24 | 0.84 | 0.3 | 353,487 | 1.48 | 0.54 | 5.49 | 58.66 | 12 | 6 | 62 | 2,117,231 | 5.99 | 2,470,718 | 7,361,375 | 7,182,391 | 7,182,391 |
| 25 | 0.86 | 0.3 | 368,634 | 1.46 | 0.53 | 5.41 | 58.11 | 12 | 6 | 64 | 2,213,088 | 6.00 | 2,581,722 | 7,460,941 | 7,271,862 | 7,271,862 |
| 26 | 0.88 | 0.3 | 374,663 | 1.45 | 0.53 | 5.36 | 57.71 | 12 | 6 | 65 | 2,224,944 | 5.94 | 2,599,606 | 7,485,321 | 7,292,563 | 7,292,563 |
| 27 | 0.90 | 0.3 | 388,586 | 1.44 | 0.52 | 5.29 | 57.16 | 12 | 7 | 66 | 2,306,227 | 5.93 | 2,694,813 | 7,551,138 | 7,349,556 | 7,349,556 |
| 28 | 0.92 | 0.3 | 395,340 | 1.43 | 0.52 | 5.25 | 56.87 | 12 | 7 | 67 | 2,343,174 | 5.93 | 2,738,514 | 7,576,838 | 7,371,103 | 7,371,103 |
| 29 | 0.94 | 0.9 | 1,229,688 | 1.18 | 0.33 | 3.07 | 46.23 | 32 | 13 | 121 | 7,530,162 | 6.12 | 8,759,850 | 9,974,310 | 9,155,785 | 9,155,785 |
| 30 | 0.96 | 0.9 | 1,254,120 | 1.18 | 0.33 | 3.05 | 46.07 | 33 | 13 | 123 | 7,656,734 | 6.11 | 8,910,855 | 10,021,619 | 9,183,574 | 9,183,574 |
| 31 | 0.98 | 1.0 | 1,386,512 | 1.16 | 0.33 | 2.97 | 45.34 | 35 | 15 | 133 | 8,434,479 | 6.08 | 9,820,991 | 10,131,734 | 9,210,181 | 9,210,213 |
| 32 | 1.00 | 1.1 | 1,441,437 | 1.15 | 0.33 | 2.96 | 45.06 | 37 | 15 | 137 | 8,754,695 | 6.07 | 10,196,132 | 10,159,940 | 9,234,193 | 9,228,759 |
| 33 | 1.02 | 1.1 | 1,513,015 | 1.14 | 0.33 | 2.91 | 44.61 | 38 | 16 | 141 | 9,111,218 | 6.02 | 10,624,233 | 10,146,033 | 9,219,603 | 9,196,845 |
| 34 | 1.04 | 1.2 | 1,583,647 | 1.13 | 0.33 | 2.87 | 44.25 | 39 | 17 | 146 | 9,517,822 | 6.01 | 11,101,469 | 10,081,895 | 9,160,186 | 9,103,895 |
| 35 | 1.06 | 1.2 | 1,607,179 | 1.13 | 0.33 | 2.86 | 44.09 | 40 | 17 | 148 | 9,625,664 | 5.99 | 11,232,843 | 10,043,422 | 9,125,224 | 9,055,565 |
| 36 | 1.08 | 1.2 | 1,688,333 | 1.12 | 0.33 | 2.86 | 43.89 | 42 | 18 | 155 | 10,294,830 | 6.10 | 11,983,163 | 9,827,468 | 8,931,487 | 8,808,072 |
| 37 | 1.10 | 1.2 | 1,698,049 | 1.12 | 0.33 | 2.86 | 43.82 | 42 | 18 | 156 | 10,338,939 | 6.09 | 12,036,989 | 9,799,118 | 8,906,221 | 8,774,578 |
| 38 | 1.12 | 1.3 | 1,734,685 | 1.11 | 0.33 | 2.84 | 43.59 | 43 | 18 | 158 | 10,537,414 | 6.07 | 12,272,099 | 9,666,890 | 8,788,811 | 8,629,049 |
| 39 | 1.14 | 1.3 | 1,747,585 | 1.11 | 0.33 | 2.83 | 43.48 | 43 | 19 | 159 | 10,591,465 | 6.06 | 12,339,050 | 9,599,082 | 8,728,798 | 8,559,021 |
| 40 | 1.16 | 1.3 | 1,774,836 | 1.11 | 0.33 | 2.82 | 43.38 | 43 | 19 | 161 | 10,818,951 | 6.10 | 12,593,787 | 9,436,842 | 8,585,586 | 8,388,965 |
| 41 | 1.18 | 1.3 | 1,794,663 | 1.10 | 0.33 | 2.81 | 43.22 | 44 | 19 | 162 | 10,915,834 | 6.08 | 12,710,497 | 9,306,197 | 8,470,608 | 8,252,059 |
| 42 | 1.20 | 1.7 | 2,373,441 | 1.12 | 0.38 | 3.31 | 44.09 | 58 | 29 | 252 | 19,436,506 | 8.19 | 21,809,947 | 3,811,677 | 3,826,782 | 2,308,072 |
| 43 | 1.22 | 1.7 | 2,381,916 | 1.12 | 0.38 | 3.30 | 44.02 | 59 | 29 | 252 | 19,477,048 | 8.18 | 21,858,965 | 3,733,689 | 3,763,657 | 2,226,975 |
| 44 | 1.24 | 1.7 | 2,391,079 | 1.12 | 0.38 | 3.30 | 44.01 | 59 | 29 | 253 | 19,594,208 | 8.19 | 21,985,286 | 3,638,100 | 3,685,952 | 2,124,035 |
| 45 | 1.26 | 1.8 | 2,488,048 | 1.11 | 0.37 | 3.27 | 43.67 | 61 | 30 | 261 | 20,510,691 | 8.24 | 22,998,739 | 2,683,154 | 2,917,442 | 1,127,502 |
| 46 | 1.28 | 1.8 | 2,507,377 | 1.10 | 0.37 | 3.26 | 43.59 | 61 | 30 | 263 | 20,683,880 | 8.25 | 23,191,257 | 2,476,118 | 2,751,509 | 917,689 |
| 47 | 1.30 | 1.8 | 2,524,812 | 1.10 | 0.37 | 3.25 | 43.48 | 61 | 30 | 264 | 20,795,393 | 8.24 | 23,320,206 | 2,280,687 | 2,594,946 | 726,671 |
| 48 | 1.32 | 1.8 | 2,532,108 | 1.10 | 0.37 | 3.25 | 43.45 | 61 | 30 | 264 | 20,862,654 | 8.24 | 23,394,762 | 2,178,830 | 2,512,714 | 628,536 |
| 49 | 1.34 | 1.9 | 2,535,816 | 1.10 | 0.37 | 3.25 | 43.43 | 62 | 30 | 265 | 20,918,514 | 8.25 | 23,454,330 | 2,096,183 | 2,445,072 | 550,085 |

### Table 16.25: Area 2/118 Pit Optimization Results

| Final | Revenue | Mine | Ore Diluted | Diluted Grades | | | | Contained Metal | | | Waste | Strip | Total | Total CF | NPV Best | NPV Worst |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Pit | Factor | Life | (tonnes) | Cu (%) | Au (g/t) | Ag (g/t) | NSR (C$/t) | Cu (Mlbs) | Au (koz) | Ag (koz) | (tonnes) | Ratio | (tonnes) | (C$) | $ disc | $ disc |
| 10 | 0.58 | 0.0 | 10,900 | 1.70 | 0.09 | 1.80 | 63.52 | 0 | 0 | 1 | 20,366 | 1.87 | 31,266 | 374,739 | 374,455 | 374,455 |
| 11 | 0.60 | 0.0 | 23,725 | 1.77 | 0.24 | 3.30 | 67.05 | 1 | 0 | 3 | 119,559 | 5.04 | 143,283 | 740,567 | 739,344 | 739,344 |
| 12 | 0.62 | 0.0 | 26,764 | 1.79 | 0.27 | 3.58 | 68.06 | 1 | 0 | 3 | 150,661 | 5.63 | 177,426 | 829,312 | 827,768 | 827,768 |
| 13 | 0.64 | 0.0 | 27,432 | 1.80 | 0.27 | 3.66 | 68.50 | 1 | 0 | 3 | 159,642 | 5.82 | 187,074 | 850,894 | 849,270 | 849,270 |
| 14 | 0.66 | 0.0 | 39,835 | 1.72 | 0.29 | 5.35 | 65.84 | 2 | 0 | 7 | 241,218 | 6.06 | 281,053 | 1,109,738 | 1,106,664 | 1,106,664 |
| 15 | 0.68 | 1.5 | 2,003,753 | 1.51 | 0.59 | 5.50 | 60.34 | 67 | 38 | 354 | 15,559,358 | 7.77 | 17,563,111 | 37,582,149 | 33,132,106 | 33,132,106 |
| 16 | 0.70 | 1.6 | 2,140,686 | 1.51 | 0.59 | 5.42 | 60.06 | 71 | 40 | 373 | 16,510,768 | 7.71 | 18,651,455 | 39,790,077 | 34,817,335 | 34,699,238 |
| 17 | 0.72 | 1.6 | 2,168,184 | 1.50 | 0.59 | 5.40 | 59.93 | 72 | 41 | 377 | 16,651,354 | 7.68 | 18,819,537 | 40,170,392 | 35,097,404 | 34,958,287 |
| 18 | 0.74 | 1.7 | 2,268,326 | 1.50 | 0.58 | 5.35 | 59.59 | 75 | 42 | 390 | 17,294,975 | 7.62 | 19,563,300 | 41,526,640 | 36,082,984 | 35,850,906 |
| 19 | 0.76 | 1.7 | 2,281,692 | 1.49 | 0.58 | 5.34 | 59.49 | 75 | 42 | 391 | 17,335,032 | 7.60 | 19,616,723 | 41,669,751 | 36,181,060 | 35,941,477 |
| 20 | 0.78 | 1.8 | 2,406,399 | 1.48 | 0.57 | 5.23 | 58.90 | 78 | 44 | 405 | 18,081,189 | 7.51 | 20,487,588 | 42,932,963 | 37,025,374 | 36,663,107 |
| 21 | 0.80 | 3.0 | 4,046,313 | 1.47 | 0.56 | 4.96 | 58.34 | 131 | 73 | 645 | 33,910,140 | 8.38 | 37,956,453 | 62,531,400 | 49,698,727 | 45,650,966 |
| 22 | 0.82 | 3.0 | 4,087,254 | 1.46 | 0.56 | 4.95 | 58.23 | 132 | 73 | 650 | 34,167,825 | 8.36 | 38,255,078 | 62,894,343 | 49,884,835 | 45,699,093 |
| 23 | 0.84 | 3.0 | 4,102,778 | 1.46 | 0.56 | 4.95 | 58.23 | 132 | 73 | 652 | 34,345,054 | 8.37 | 38,447,833 | 63,047,204 | 49,966,514 | 45,720,871 |
| 24 | 0.86 | 3.2 | 4,359,484 | 1.44 | 0.55 | 4.88 | 57.45 | 139 | 77 | 684 | 35,948,907 | 8.25 | 40,308,391 | 64,746,324 | 51,213,352 | 46,133,396 |
| 25 | 0.88 | 3.2 | 4,376,223 | 1.44 | 0.55 | 4.87 | 57.37 | 139 | 77 | 685 | 35,996,985 | 8.23 | 40,373,208 | 64,820,068 | 51,266,265 | 46,114,454 |
| 26 | 0.90 | 3.2 | 4,388,644 | 1.44 | 0.55 | 4.87 | 57.32 | 139 | 77 | 687 | 36,060,152 | 8.22 | 40,448,796 | 64,876,463 | 51,306,682 | 46,104,901 |
| 27 | 0.92 | 3.2 | 4,411,901 | 1.44 | 0.54 | 4.86 | 57.26 | 140 | 77 | 689 | 36,244,010 | 8.22 | 40,655,911 | 64,970,186 | 51,373,436 | 46,075,496 |
| 28 | 0.94 | 3.2 | 4,443,665 | 1.44 | 0.54 | 4.85 | 57.15 | 141 | 78 | 693 | 36,460,299 | 8.21 | 40,903,964 | 65,058,006 | 51,434,763 | 45,995,361 |
| 29 | 0.96 | 3.3 | 4,479,095 | 1.43 | 0.54 | 4.84 | 57.06 | 142 | 78 | 697 | 36,761,013 | 8.21 | 41,240,108 | 65,139,534 | 51,490,813 | 45,878,882 |
| 30 | 0.98 | 3.3 | 4,496,567 | 1.43 | 0.54 | 4.83 | 56.98 | 142 | 78 | 699 | 36,850,006 | 8.20 | 41,346,573 | 65,159,307 | 51,503,410 | 45,814,298 |
| 31 | 1.00 | 3.4 | 4,590,729 | 1.42 | 0.53 | 4.78 | 56.56 | 144 | 79 | 705 | 37,329,371 | 8.13 | 41,920,101 | 65,198,181 | 51,521,564 | 45,266,518 |
| 32 | 1.02 | 3.4 | 4,613,709 | 1.42 | 0.53 | 4.77 | 56.46 | 144 | 79 | 708 | 37,465,631 | 8.12 | 42,079,340 | 65,188,756 | 51,512,232 | 45,127,946 |
| 33 | 1.04 | 3.4 | 4,646,007 | 1.42 | 0.53 | 4.76 | 56.37 | 145 | 79 | 711 | 37,759,804 | 8.13 | 42,405,811 | 65,150,530 | 51,481,086 | 44,907,506 |
| 34 | 1.06 | 3.4 | 4,680,645 | 1.42 | 0.53 | 4.75 | 56.27 | 146 | 80 | 715 | 38,070,893 | 8.13 | 42,751,537 | 65,077,770 | 51,424,901 | 44,652,726 |
| 35 | 1.08 | 3.4 | 4,698,330 | 1.41 | 0.53 | 4.75 | 56.19 | 146 | 80 | 717 | 38,195,137 | 8.13 | 42,893,466 | 65,029,422 | 51,388,241 | 44,502,870 |
| 36 | 1.10 | 3.4 | 4,702,132 | 1.41 | 0.53 | 4.75 | 56.17 | 147 | 80 | 717 | 38,210,565 | 8.13 | 42,912,697 | 65,018,826 | 51,380,225 | 44,472,855 |
| 37 | 1.12 | 3.5 | 4,769,694 | 1.41 | 0.53 | 4.73 | 55.97 | 148 | 81 | 725 | 38,903,262 | 8.16 | 43,672,956 | 64,689,332 | 51,137,184 | 43,867,156 |
| 38 | 1.14 | 3.6 | 4,907,761 | 1.39 | 0.52 | 4.68 | 55.43 | 151 | 82 | 739 | 39,960,983 | 8.14 | 44,868,745 | 64,037,748 | 50,662,954 | 42,441,178 |
| 39 | 1.16 | 3.6 | 4,924,440 | 1.39 | 0.52 | 4.68 | 55.35 | 151 | 82 | 740 | 40,049,787 | 8.13 | 44,974,227 | 63,952,271 | 50,601,475 | 42,260,908 |
| 40 | 1.18 | 3.6 | 4,935,475 | 1.39 | 0.52 | 4.67 | 55.29 | 151 | 82 | 741 | 40,105,357 | 8.13 | 45,040,832 | 63,890,025 | 50,556,840 | 42,134,223 |
| 41 | 1.20 | 3.6 | 4,982,232 | 1.39 | 0.52 | 4.66 | 55.12 | 152 | 83 | 746 | 40,526,264 | 8.13 | 45,508,496 | 63,548,636 | 50,313,588 | 41,538,784 |
| 42 | 1.22 | 3.7 | 5,005,393 | 1.39 | 0.52 | 4.65 | 55.03 | 153 | 83 | 748 | 40,746,659 | 8.14 | 45,752,052 | 63,334,881 | 50,162,169 | 41,198,450 |
| 43 | 1.24 | 3.7 | 5,009,810 | 1.38 | 0.52 | 4.65 | 55.02 | 153 | 83 | 749 | 40,796,275 | 8.14 | 45,806,085 | 63,287,895 | 50,128,958 | 41,130,353 |
| 44 | 1.26 | 3.7 | 5,090,870 | 1.38 | 0.51 | 4.61 | 54.66 | 154 | 84 | 755 | 41,433,950 | 8.14 | 46,524,820 | 62,560,053 | 49,617,407 | 39,947,164 |
| 45 | 1.28 | 3.8 | 5,172,200 | 1.37 | 0.51 | 4.59 | 54.38 | 156 | 85 | 763 | 42,367,645 | 8.19 | 47,539,846 | 61,542,955 | 48,909,850 | 38,552,210 |
| 46 | 1.30 | 3.9 | 5,296,069 | 1.36 | 0.50 | 4.53 | 53.84 | 158 | 85 | 771 | 43,306,390 | 8.18 | 48,602,459 | 60,324,448 | 48,076,181 | 36,585,149 |
| 47 | 1.32 | 3.9 | 5,321,206 | 1.35 | 0.50 | 4.52 | 53.79 | 159 | 86 | 774 | 43,695,256 | 8.21 | 49,016,463 | 59,939,606 | 47,813,794 | 36,099,314 |
| 48 | 1.34 | 3.9 | 5,332,723 | 1.35 | 0.50 | 4.52 | 53.77 | 159 | 86 | 775 | 43,911,617 | 8.23 | 49,244,340 | 59,704,377 | 47,653,250 | 35,834,258 |

### Table 16.26: Minto North Pit Optimization Results

| Final Pit | Revenue | Mine | Ore Diluted | Diluted Grades | | | | Contained Metal | | | Waste | Strip | Total | Total CF | NPV Best | NPV Worst |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Factor | Life | (tonnes) | Cu (%) | Au (g/t) | Ag (g/t) | NSR (C$/t) | Cu (Mlbs) | Au (koz) | Ag (koz) | (tonnes) | Ratio | (tonnes) | (C$) | $ disc | $ disc |
| 10 | 0.52 | 0.7 | 998,629 | 3.01 | 1.77 | 11.29 | 124.08 | 66 | 57 | 362 | 9,467,126 | 9.48 | 10,465,755 | 78,771,802 | 74,470,636 | 74,470,636 |
| 11 | 0.54 | 0.7 | 1,018,144 | 2.98 | 1.75 | 11.17 | 123.02 | 67 | 57 | 366 | 9,574,700 | 9.40 | 10,592,845 | 79,390,039 | 74,972,806 | 74,972,806 |
| 12 | 0.56 | 0.8 | 1,029,496 | 2.96 | 1.73 | 11.11 | 122.30 | 67 | 57 | 368 | 9,610,445 | 9.34 | 10,639,941 | 79,684,823 | 75,203,174 | 75,203,174 |
| 13 | 0.58 | 0.8 | 1,069,745 | 2.91 | 1.69 | 10.85 | 119.87 | 69 | 58 | 373 | 9,773,965 | 9.14 | 10,843,710 | 80,645,268 | 75,937,550 | 75,937,550 |
| 14 | 0.60 | 0.8 | 1,071,229 | 2.91 | 1.69 | 10.85 | 119.88 | 69 | 58 | 374 | 9,811,577 | 9.16 | 10,882,806 | 80,726,686 | 76,007,873 | 76,007,873 |
| 15 | 0.62 | 0.8 | 1,072,258 | 2.91 | 1.69 | 10.84 | 119.81 | 69 | 58 | 374 | 9,812,972 | 9.15 | 10,885,229 | 80,744,741 | 76,020,477 | 76,020,477 |
| 16 | 0.64 | 0.8 | 1,081,082 | 2.89 | 1.68 | 10.78 | 119.21 | 69 | 58 | 375 | 9,828,823 | 9.09 | 10,909,905 | 80,890,799 | 76,120,211 | 76,120,211 |
| 17 | 0.66 | 0.8 | 1,084,792 | 2.89 | 1.67 | 10.76 | 118.97 | 69 | 58 | 375 | 9,841,659 | 9.07 | 10,926,451 | 80,954,234 | 76,164,016 | 76,164,016 |
| 18 | 0.68 | 0.9 | 1,206,955 | 2.71 | 1.54 | 9.99 | 111.36 | 72 | 60 | 388 | 10,123,148 | 8.39 | 11,330,102 | 82,633,754 | 77,211,971 | 77,211,971 |
| 19 | 0.70 | 0.9 | 1,209,299 | 2.70 | 1.54 | 9.98 | 111.22 | 72 | 60 | 388 | 10,125,554 | 8.37 | 11,334,853 | 82,661,803 | 77,227,999 | 77,227,999 |
| 20 | 0.72 | 1.0 | 1,313,241 | 2.58 | 1.44 | 9.44 | 105.87 | 75 | 61 | 399 | 10,432,390 | 7.94 | 11,745,631 | 83,922,559 | 77,948,981 | 77,948,981 |
| 21 | 0.74 | 1.0 | 1,391,458 | 2.49 | 1.37 | 9.08 | 102.23 | 76 | 61 | 406 | 10,633,515 | 7.64 | 12,024,973 | 84,746,362 | 78,468,141 | 78,466,760 |
| 22 | 0.76 | 1.0 | 1,421,529 | 2.47 | 1.35 | 8.97 | 101.19 | 77 | 62 | 410 | 10,848,744 | 7.63 | 12,270,273 | 85,129,063 | 78,820,500 | 78,810,407 |
| 23 | 0.78 | 1.1 | 1,462,662 | 2.43 | 1.32 | 8.80 | 99.46 | 78 | 62 | 414 | 10,963,223 | 7.50 | 12,425,885 | 85,489,247 | 79,150,144 | 79,111,520 |
| 24 | 0.80 | 1.1 | 1,487,707 | 2.41 | 1.31 | 8.73 | 98.61 | 79 | 63 | 417 | 11,125,861 | 7.48 | 12,613,569 | 85,740,448 | 79,379,442 | 79,312,899 |
| 25 | 0.82 | 1.1 | 1,497,708 | 2.40 | 1.30 | 8.69 | 98.22 | 79 | 63 | 418 | 11,161,354 | 7.45 | 12,659,062 | 85,814,691 | 79,446,863 | 79,366,611 |
| 26 | 0.84 | 1.1 | 1,506,228 | 2.39 | 1.30 | 8.66 | 97.90 | 79 | 63 | 419 | 11,198,177 | 7.43 | 12,704,404 | 85,873,798 | 79,500,434 | 79,407,366 |
| 27 | 0.86 | 1.1 | 1,507,362 | 2.39 | 1.29 | 8.66 | 97.87 | 79 | 63 | 420 | 11,211,021 | 7.44 | 12,718,382 | 85,883,369 | 79,509,126 | 79,414,302 |
| 28 | 0.88 | 1.1 | 1,562,536 | 2.34 | 1.26 | 8.48 | 95.89 | 81 | 63 | 426 | 11,497,973 | 7.36 | 13,060,509 | 86,191,938 | 79,786,789 | 79,577,034 |
| 29 | 0.90 | 1.2 | 1,588,303 | 2.32 | 1.24 | 8.38 | 94.91 | 81 | 63 | 428 | 11,577,494 | 7.29 | 13,165,797 | 86,287,984 | 79,871,912 | 79,591,412 |
| 30 | 0.92 | 1.2 | 1,589,732 | 2.32 | 1.24 | 8.38 | 94.88 | 81 | 63 | 428 | 11,597,469 | 7.30 | 13,187,201 | 86,295,357 | 79,878,497 | 79,594,906 |
| 31 | 0.94 | 1.2 | 1,595,523 | 2.31 | 1.24 | 8.37 | 94.68 | 81 | 64 | 429 | 11,630,865 | 7.29 | 13,226,388 | 86,311,548 | 79,892,658 | 79,591,588 |
| 32 | 0.96 | 1.2 | 1,622,558 | 2.29 | 1.22 | 8.28 | 93.71 | 82 | 64 | 432 | 11,755,689 | 7.25 | 13,378,247 | 86,360,219 | 79,934,073 | 79,547,101 |
| 33 | 0.98 | 1.2 | 1,628,129 | 2.29 | 1.22 | 8.26 | 93.51 | 82 | 64 | 432 | 11,775,831 | 7.23 | 13,403,960 | 86,366,293 | 79,938,982 | 79,533,369 |
| 34 | 1.00 | 1.2 | 1,635,655 | 2.28 | 1.22 | 8.27 | 93.35 | 82 | 64 | 435 | 11,896,434 | 7.27 | 13,532,089 | 86,371,286 | 79,942,678 | 79,511,373 |
| 35 | 1.02 | 1.2 | 1,643,736 | 2.28 | 1.21 | 8.24 | 93.07 | 83 | 64 | 436 | 11,938,102 | 7.26 | 13,581,839 | 86,367,131 | 79,937,966 | 79,477,830 |
| 36 | 1.04 | 1.2 | 1,698,502 | 2.23 | 1.18 | 8.07 | 91.23 | 84 | 64 | 441 | 12,249,546 | 7.21 | 13,948,048 | 86,305,285 | 79,875,518 | 79,195,537 |
| 37 | 1.06 | 1.2 | 1,704,060 | 2.23 | 1.17 | 8.05 | 91.07 | 84 | 64 | 441 | 12,299,414 | 7.22 | 14,003,474 | 86,295,024 | 79,865,581 | 79,160,829 |
| 38 | 1.08 | 1.3 | 1,722,599 | 2.21 | 1.16 | 7.99 | 90.43 | 84 | 64 | 443 | 12,373,005 | 7.18 | 14,095,604 | 86,253,211 | 79,825,590 | 79,037,507 |
| 39 | 1.10 | 1.3 | 1,726,851 | 2.21 | 1.16 | 7.98 | 90.29 | 84 | 64 | 443 | 12,397,761 | 7.18 | 14,124,612 | 86,239,632 | 79,812,810 | 79,006,115 |
| 40 | 1.12 | 1.3 | 1,745,656 | 2.20 | 1.15 | 7.92 | 89.71 | 85 | 65 | 445 | 12,534,699 | 7.18 | 14,280,355 | 86,162,088 | 79,740,504 | 78,848,681 |
| 41 | 1.14 | 1.3 | 1,774,313 | 2.18 | 1.14 | 7.84 | 88.87 | 85 | 65 | 447 | 12,780,316 | 7.20 | 14,554,628 | 86,012,754 | 79,602,403 | 78,575,933 |
| 42 | 1.16 | 1.3 | 1,778,755 | 2.17 | 1.13 | 7.82 | 88.74 | 85 | 65 | 447 | 12,813,391 | 7.20 | 14,592,146 | 85,988,650 | 79,580,170 | 78,531,319 |
| 43 | 1.18 | 1.3 | 1,780,511 | 2.17 | 1.13 | 7.82 | 88.68 | 85 | 65 | 448 | 12,820,237 | 7.20 | 14,600,748 | 85,979,710 | 79,571,917 | 78,514,164 |
| 44 | 1.20 | 1.3 | 1,780,696 | 2.17 | 1.13 | 7.82 | 88.68 | 85 | 65 | 448 | 12,822,462 | 7.20 | 14,603,159 | 85,975,971 | 79,568,517 | 78,509,767 |
| 45 | 1.22 | 1.3 | 1,781,128 | 2.17 | 1.13 | 7.82 | 88.67 | 85 | 65 | 448 | 12,830,139 | 7.20 | 14,611,266 | 85,971,192 | 79,564,153 | 78,503,154 |
| 46 | 1.24 | 1.3 | 1,781,361 | 2.17 | 1.13 | 7.82 | 88.66 | 85 | 65 | 448 | 12,833,958 | 7.20 | 14,615,319 | 85,968,553 | 79,561,744 | 78,499,527 |
| 47 | 1.26 | 1.3 | 1,781,443 | 2.17 | 1.13 | 7.82 | 88.66 | 85 | 65 | 448 | 12,834,684 | 7.20 | 14,616,127 | 85,967,851 | 79,561,101 | 78,498,461 |
| 48 | 1.28 | 1.3 | 1,793,702 | 2.16 | 1.13 | 7.78 | 88.26 | 86 | 65 | 449 | 12,920,106 | 7.20 | 14,713,808 | 85,855,233 | 79,458,149 | 78,331,099 |
| 49 | 1.30 | 1.3 | 1,793,721 | 2.16 | 1.13 | 7.78 | 88.26 | 86 | 65 | 449 | 12,920,898 | 7.20 | 14,714,618 | 85,854,741 | 79,457,703 | 78,330,605 |

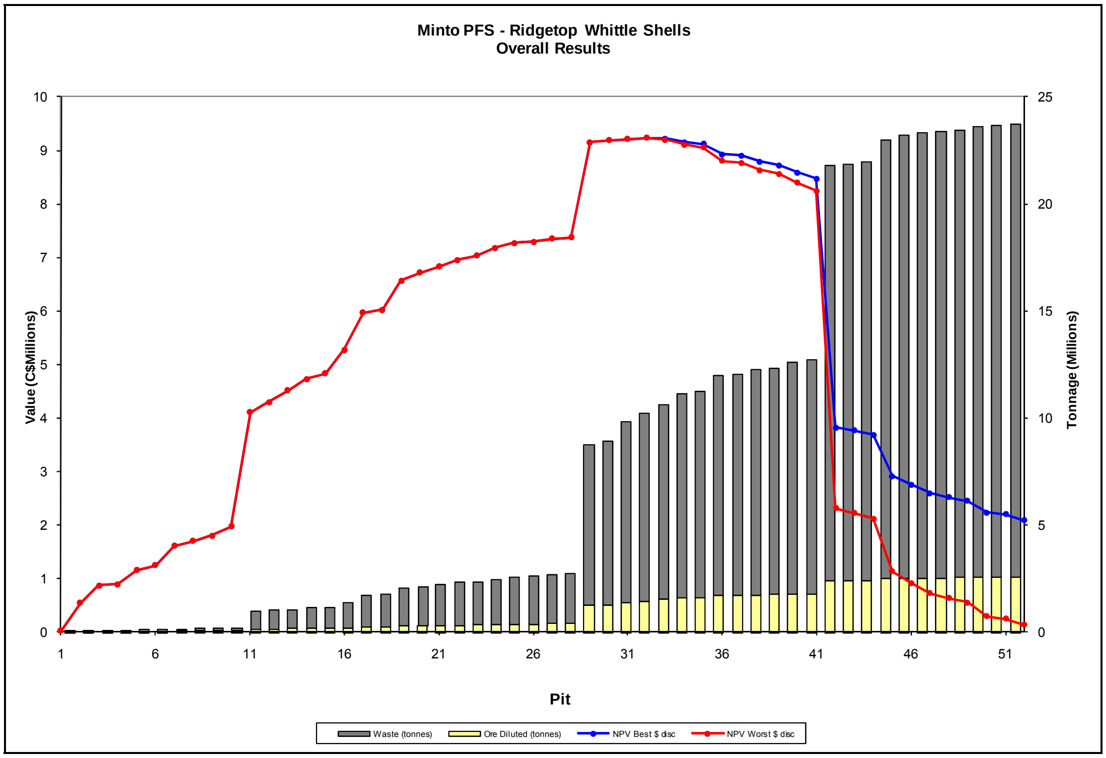



**Figure 16.29: Ridgetop Optimization Results**

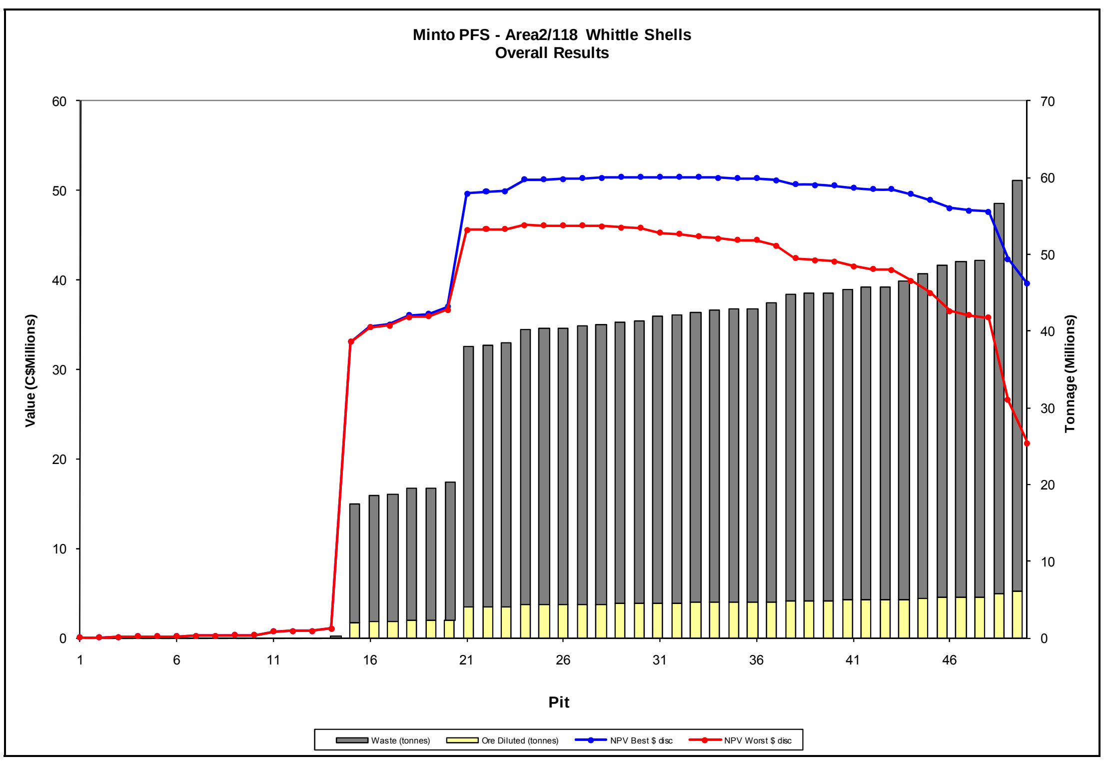
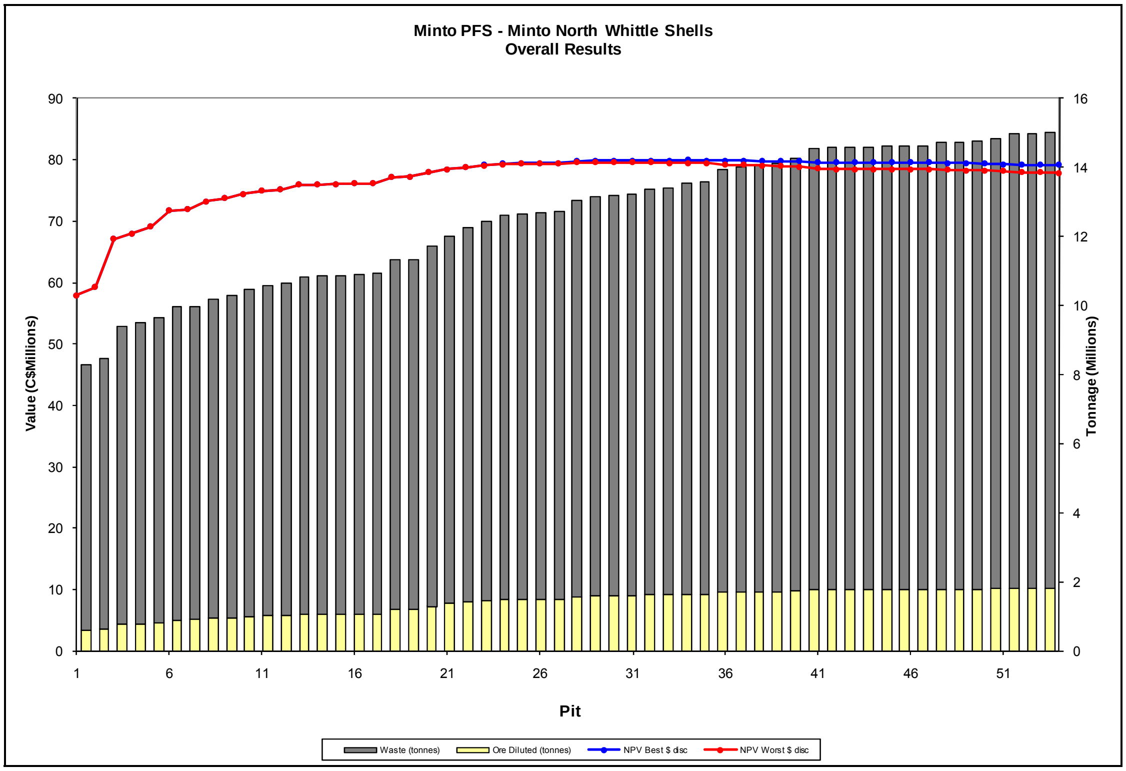



**Figure 16.30: Area 2/118 Optimization Results**




## Figure 16.31: Minto North 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 IV pits. These detailed pit designs take into consideration, minimum mining widths, access ramps, and detailed bench configurations as summarized in Table 16.27 below.

## Table 16.27: Detailed Pit Design Parameters

| Design Parameter | Unit | North | Area2/118 | Ridgetop |
|---|---|---|---|---|
| Overburden angle | ° | 30 | 30 | 30 |
| Inter-ramp angle | ° | 52 | 47 west, 53 east | 53 east |
| Ramp width | m | 25 | 25 | 25 |
| Ramp grade | % | 10 | 10 | 10 |
| Bench height | m | 9 | 9 | 9 |
| Bench face angle | ° | 72 | 64 west, 73 east | 73 |
| Bench configuration | single/double | Double | Double | Double |
| Berm width | m | 8 | 8 | 8 |

Sub-out maximum depth 6.0 m

Single lane ramp width 15 m @10%

### 16.6.3 Reserves

The mineral reserves estimate for the detailed pit designs are summarized in Table 16.28 below. The mineral reserve for Main Pit includes the ore stockpile balance predicted for the end of 2009 as well as proposed mining from 2010 going forward. The various estimated copper cut-off grades used within the planned pits are as noted. The reserves are further summarized by stockpile/grade bin in Table 16.29.

**Table 16.28: PFS Mineral Reserve Estimates**

| Deposit | Reserve Class | Tonnes ('000s) | Cut-off Grade (%Cu equiv.) | Diluted grade | | | Contained Metal | | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | Cu (%) | Au (g/t ) | Ag (g/t ) | Cu (Mlb) | Au (oz) | Ag (oz) |
| Main Pit | Proven | 3,920 | 0.62 | 1.64 | 0.58 | 6.51 | 142 | 72 | 820 |
| | Probable | 206 | 0.62 | 1.20 | 0.45 | 5.25 | 5 | 3 | 35 |
| | Sub-total | 4,126 | 0.62 | 1.62 | 0.57 | 6.45 | 147 | 75 | 855 |
| North Pit | Proven | 1,346 | 0.55 | 2.50 | 1.37 | 9.04 | 74 | 59 | 391 |
| | Probable | 3 | 0.55 | 2.91 | 1.07 | 13.11 | 0 | 0 | 1 |
| | Sub-total | 1,349 | 0.55 | 2.50 | 1.37 | 9.05 | 74 | 60 | 393 |
| Ridgetop Pit | Proven | 802 | 0.58 | 1.17 | 0.31 | 2.33 | 21 | 8 | 60 |
| | Probable | 522 | 0.58 | 1.39 | 0.50 | 4.90 | 16 | 8 | 82 |
| | Sub-total | 1,324 | 0.58 | 1.26 | 0.38 | 3.34 | 37 | 16 | 142 |
| Area 2/118 Pit | Proven | 3,707 | 0.56 | 1.56 | 0.59 | 5.36 | 127 | 71 | 639 |
| | Probable | 387 | 0.56 | 1.09 | 0.19 | 2.79 | 9 | 2 | 35 |
| | Sub-total | 4,094 | 0.56 | 1.51 | 0.56 | 5.12 | 137 | 73 | 674 |
| **Total** | **Proven** | **9,775** | **0.58** | **1.69** | **0.67** | **6.08** | **364** | **211** | **1,911** |
| | **Probable** | **1,118** | **0.58** | **1.25** | **0.38** | **4.26** | **31** | **14** | **153** |
| | **Total** | **10,893** | **0.58** | **1.64** | **0.64** | **5.89** | **395** | **224** | **2,064** |

Within these detailed pit designs there a total of 49 kt of inferred mineral resources at a copper grade of 1.0%. Additional in-fill drilling has commenced to attempt to convert these inferred resources to higher classifications for reduction of strip ratios. 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.

### Table 16.29: Mineral Reserves by Grade Bin

| Grade Bin<br>Stockpile Destination | cut-off to 1.0% Cu<br>1 | 1.0% to 2.0% Cu<br>2 | 2.0% to 4.0% Cu<br>3 | >4.0% Cu<br>4 | Total |
|---|---|---|---|---|---|
| **Tonnes (kt)** | | | | | |
| Main Pit | 1,082 | 2,038 | 931 | 75 | **4,126** |
| Area 2/118 | 1,491 | 1,915 | 657 | 31 | **4,094** |
| North | 222 | 576 | 323 | 228 | **1,349** |
| Ridgetop | 682 | 551 | 91 | 0 | **1,324** |
| **Total tonnes** | **3,477** | **5,081** | **2,002** | **334** | **10,893** |
| **Grades** | | | | | |
| **Cu%** | | | | | |
| Main Pit | 0.82 | 1.43 | 2.69 | 4.90 | **1.62** |
| Area 2/118 | 0.83 | 1.59 | 2.67 | 4.95 | **1.51** |
| North | 0.88 | 1.48 | 3.05 | 5.86 | **2.50** |
| Ridgetop | 0.86 | 1.50 | 2.76 | - | **1.26** |
| **Total Cu %** | **0.84** | **1.50** | **2.74** | **5.56** | **1.64** |
| **Au (g/t)** | | | | | |
| Main Pit | 0.24 | 0.51 | 0.99 | 1.52 | **0.57** |
| Area 2/118 | 0.21 | 0.59 | 1.17 | 1.98 | **0.56** |
| North | 0.26 | 0.68 | 2.14 | 3.09 | **1.37** |
| Ridgetop | 0.20 | 0.47 | 1.18 | - | **0.38** |
| **Total Au (g/t)** | **0.22** | **0.56** | **1.24** | **2.64** | **0.64** |
| **Ag (g/t)** | | | | | |
| Main Pit | 3.07 | 5.53 | 11.04 | 23.01 | **6.44** |
| Area 2/118 | 2.47 | 5.34 | 9.68 | 22.30 | **5.12** |
| North | 2.67 | 4.48 | 12.64 | 21.73 | **9.05** |
| Ridgetop | 2.00 | 3.72 | 11.09 | - | **3.34** |
| **Total Ag (g/t)** | **2.57** | **5.14** | **10.85** | **22.07** | **5.89** |

## 16.6.4 Cut-off Grade Calculation

Table 16.30 summarizes the cut-off grade calculations for the various deposits in Phase IV. 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 16.30: Copper Cut-off Grade Estimate

| Parameter | Unit | Area 2/118 | | Ridgetop | | Minto North | |
|---|---|---|---|---|---|---|---|
| | | Res COG | Incr. COG | Res. COG | Incr. COG | Res. COG | Incr. COG |
| **Revenue, Smelting and TC/RC/Transport** | | | | | | | |
| Cu price | US$/lb Cu | 2.00 | 2.00 | 2.00 | 2.00 | 2.00 | 2.00 |
| Exchange rate | C$/US$ | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 |
| Cu price | C$/lb Cu | 2.41 | 2.41 | 2.41 | 2.41 | 2.41 | 2.41 |
| Payable copper | % | 96.75 | 96.75 | 96.75 | 96.75 | 96.75 | 96.75 |
| TC/RC/Transport | C$/lb Cu payable | 0.34 | 0.34 | 0.34 | 0.34 | 0.34 | 0.34 |
| NSR (Cu only) | C$/lb Cu payable | 1.99 | 1.99 | 1.99 | 1.99 | 1.99 | 1.99 |
| **Opex estimates** | | | | | | | |
| Mining cost | C$/t mined | 2.11 | 0.00 | 2.11 | 0.00 | 2.11 | 0.00 |
| Strip Ratio | t:t | 8.80 | 0.00 | 6.80 | 0.00 | 7.90 | 0.00 |
| Mining Cost | C$/t milled | 20.68 | 0.00 | 16.46 | 0.00 | 18.78 | 0.00 |
| Processing and G&A cost | C$/t milled | 23.09 | 23.09 | 23.09 | 23.09 | 23.09 | 23.09 |
| Site Cost | C$/tonne milled | 43.77 | 23.09 | 39.55 | 23.09 | 41.87 | 23.09 |
| **Recovery and Dilution** | | | | | | | |
| Recovered Cu grade | %Cu | 1.00 | 0.53 | 0.90 | 0.53 | 0.95 | 0.53 |
| Process Recovery | average % | 92% | 92% | 92% | 92% | 92% | 92% |
| Plant feed Cu grade | diluted %Cu | 1.08 | 0.57 | 0.98 | 0.57 | 1.04 | 0.57 |
| Dilution | % | 8% | 8% | 10.3% | 10.3% | 4.7% | 4.7% |
| **Cut-off Grade** | | | | | | | |
| In-situ cut-off Cu grade (Cu only) | %Cu | 1.18 | 0.62 | 1.09 | 0.64 | 1.09 | 0.60 |
| By-product contribution (est.) | % of Cu value | 10% | 10% | 10% | 10% | 10% | 10% |
| In-situ cut-off Cu grade (inc. by-product value) | %Cu | 1.07 | 0.56 | 0.99 | 0.58 | 0.99 | 0.55 |

# 17 Other Relevant Data and Information

## 17.1 Underground Mining Potential at Minto

### 17.1.1 Introduction

Exploration on the Minto project has historically been focused on finding near-surface deposits conducive to open pit mining. In the course of exploration, several deeper deposits have been discovered that may provide an opportunity to add mill feed material using underground mining methods.

Both deep penetrating geophysical surveys (Titan-24) and core drilling have provided some preliminary definition of deposits below 150 m in depth that are likely to be too deep for open pit mining unless a step-change in open pit mining and processing costs are achieved that allow pits to be driven deeper. These deposits and targets may be amenable to underground exploitation.

The known deposits that may have the grade, continuity and volume to be considered potentially mineable from underground are described here.

Underground mining can generally be accomplished with a significantly reduced surficial footprint, as compared to open pit mining. This results in better control of potential environmental impacts. Closure and reclamation of an underground mine is therefore not as extensive as what is required for open pit mines.

### 17.1.2 Areas with Known Resources

Mineral resources that may have a potential for underground mining are located in Area 118 and Area 2. Further work will need to be done in order to classify these mineral resources as mineral reserves.

#### Area 118

Two significant deposits and several smaller ones with underground mining potential are located west of the proposed Area 2 Pit at depths of roughly 200 m to 300 m below surface.

The measured and indicated mineral resources at Area 118 that could conceptually be mined using underground mining are as shown in Table 17.1. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

#### Area 2

The lower-most lens of the proposed Area 2 Pit has the potential to be mined using underground methods. The resource information for the lens is shown in Table 17.1. Some of these mineral resources are presently included in the Area 2 Pit mineral reserves estimate. An OP/UG cross-over study must be done before UG mining should be considered to determine if the lower lens is best mined by underground or open pit.

**Table 17.1: Potentially UG Mineable Mineralization in Area 118 and Area 2 (Measured and Indicated Resources only)**

| Deposit* | Cut-off Value ($/t NSR) | Insitu Tonnes (Kt) | Diluted Grade (10%) | | | Thickness (Min. 3 m) | | Benching % |
|---|---|---|---|---|---|---|---|---|
| | | | Cu (%) | Au (g/t) | Ag (g/t) | Max Thickness (m) | Avg Thickness (m) | |
| **Area 118** | | | | | | | | |
| 1 | 60 | 519,000 | 2.26 | 0.92 | 8.99 | 17.0 | 7.4 | 46 |
| | 75 | 381,000 | 2.54 | 1.03 | 10.16 | 17.0 | 7.5 | 45 |
| | 90 | 305,000 | 2.74 | 1.13 | 10.89 | 14.0 | 7.3 | 43 |
| 2 | 60 | 488,000 | 1.95 | 0.81 | 9.41 | 15.5 | 7.5 | 41 |
| | 75 | 339,000 | 2.17 | 0.91 | 11.16 | 15.0 | 6.5 | 31 |
| | 90 | 193,000 | 2.46 | 1.05 | 14.57 | 13.0 | 6.4 | 31 |
| 3 | 60 | 120,000 | 1.72 | 0.65 | 6.13 | 19.0 | 7.1 | 44 |
| | 75 | 63,000 | 2.00 | 0.79 | 7.29 | 15.0 | 5.8 | 28 |
| | 90 | 17,000 | 2.60 | 1.20 | 11.72 | 9.0 | 5.4 | 9 |
| 4 | 60 | 118,000 | 1.49 | 0.57 | 5.60 | 14.5 | 6.0 | 29 |
| | 75 | - | - | - | - | - | - | - |
| | 90 | - | - | - | - | - | - | - |
| 5 | 60 | 95,000 | 1.52 | 0.41 | 5.68 | 10.0 | 5.5 | 16 |
| | 75 | - | - | - | - | - | - | - |
| | 90 | - | - | - | - | - | - | - |
| **Area 118 Total** | **60** | **1,340,000** | **1.98** | **0.79** | **8.32** | **15.9** | **7.1** | **40** |
| | **75** | **783,000** | **2.34** | **0.96** | **10.4** | **16.0** | **6.9** | **38** |
| | **90** | **515,000** | **2.63** | **1.10** | **12.27** | **13.5** | **6.9** | **37** |
| **Area 2** | | | | | | | | |
| **(Area 2 Total)** | **60** | **1,814,000** | **1.83** | **0.75** | **6.01** | **33.0** | **11.2** | **64** |
| | **75** | **1,186,000** | **2.04** | **0.87** | **6.75** | **30.0** | **8.8** | **53** |
| | **90** | **555,000** | **2.34** | **1.07** | **7.76** | **27.5** | **7.3** | **42** |

** See Figures 17.3 deposit locations*

## 17.1.3 Underground Targets for Further Exploration

Exploration targets at a depth below surface where underground mining methods could potentially be considered have been identified. Further exploration work, including drilling, must be completed before any mineral resource estimation can be done on these targets.

### Minto East

Three recently completed diamond drill holes, following up a strong geophysical anomaly, intersected mineralization at a depth of about 300 m below surface at the southeast corner of the Minto Main Pit. The significant drill intercepts are shown in Table 17.2.

**Table 17.2: Select Minto East Drilling Results**

| Hole ID | From (m) | To (m) | Interval (m)* | Copper (%) |
|---|---|---|---|---|
| 09SWC-584 | 302.0 | 315.6 | 13.6 | 3.45 |
| *including* | *308.0* | *314.0* | *6.0* | *4.14* |
| 09SWC-586 | 279.8 | 306.8 | 27.0 | 2.75 |
| *including* | *284.3* | *293.3* | *9.0* | *3.7* |
| 07SWC-176* | 291.9 | 303.6 | 11.7 | 2.95 |
| *including* | *296.2* | *302.2* | *6.0* | *3.97* |

Details of this drilling program, quality assurance and quality control programs and other relevant information are discussed elsewhere in this report. Minto East does not have a mineral resource estimate and requires more drilling.

### Copper Keel

A deposit to the south east of the proposed Area 2 Pit is located at 240 m depth and contains intersections as shown in Table 17.3.

**Table 17.3: Copper Keel Select Drilling Results**

| Hole ID | From (m) | To (m) | Interval (m)* | Copper (%) |
|---|---|---|---|---|
| 09SWC-395 | 241.2 | 245.5 | 4.3 | 3.12 |
| 07SWC-243 | 68.2 | 72.3 | 4.1 | 3.10 |
| 07SWC-241 | 88.2 | 90.3 | 2.1 | 2.84 |

Details of this drilling program, quality assurance and quality control programs and other relevant information are discussed elsewhere in this report. Copper Keel does not currently have a mineral resource estimate and requires more drilling.

## 17.1.4 Underground Mining Context

The context or physical characteristics of each mineral deposit determine the appropriate mining method(s) that can be applied. The deposits in the 118 area are better known than the other underground targets and therefore have been used to determine a possible mining method and plan. The characteristics of the 118 deposit are shown in Table 17.4. Due to the similar geometries and nature of mineralization encountered in all of the known deposits at Minto, it is likely that the other underground targets at Minto will have somewhat similar characteristics.

**Table 17.4: 118 Deposit Context**

| Parameters | Unit | Value | Comment |
|---|---|---|---|
| Depth below surface | m | 150-200 | |
| Dip | deg. | 10-30 | |
| Thickness | m | 3-20 | |
| Size (aerial) | m | 50x100 | |
| Production Capacity | t/vm | 6,000 | Approximate tonnes per vertical metre |
| Mineral Value | $/t NSR | 90 | Approximate only |
| Mineralization | Mineralized zones are visually and geochemically obvious due to density of visible sulphides and the degree of foliation. | | |
| Continuity | The two 118 area zones appear to be continuous over tens of metres, similar to other mineralized zones at Minto. | | |
| 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. | | |
| Hydrogeology | Not well defined, but tightness of the rock infers that there will not likely not be hydrogeological issues. | | |
| Constraints | There are no known constraints such as heat, radiation, groundwater or rock stress. | | |

### 17.1.5 Underground Mining Method Selection

The conclusion of the preliminary mining method review was that room and pillar (RAP) mining would be appropriate. 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. RAP allows development ramps to be placed within the mineralized zone, saving development costs. The strong, massive nature of the Minto rock and shallow depth of the deposits mean that fairly high extraction ratios (70% to 85%) could reasonably be expected.

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.

### 17.1.6 Description of Room and Pillar Mining

Room and pillar 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. 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.

At Minto, many of the mineralized zones are thicker than can be mined in a single pass. In these areas, a hanging wall cut will be made first, the back supported and then the bottom cut or bench taken out. This means that the back only needs to be supported (rock bolted) once and will 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. The estimated benching percentages are included in Table 17.1.

### 17.1.7 Conceptual Mine Design

#### Mine Access

The proposed access to the UG prospects at Minto is via a 15% gradient 5 mH x 5 mW decline driven from a portal location south of the Area 2 Pit (see Figure 17.1). The proposed portal location was chosen in an area of minimal overburden, part way up the ridge south of Area 2. The ramp was designed to access, Area 2/118 deep mineralization and can be extended to the Minto East prospect. Its position also allows for access development to the Copper Keel prospect. The ramp passes through the main Area 118 mineralized zones and also passes near the bottom of the Area 2 Pit and could be used to access resources at or near the bottom of the planned pit.

Ventilation raises are planned at strategic locations off of the main ramp and provide return airways to surface and secondary egress.

Development metres and tonnes are summarized in Table 17.5. As the ramp passes through the mineral resource, development tonnes have been categorized as either Waste Tonnes or Measured and Indicated Tonnes.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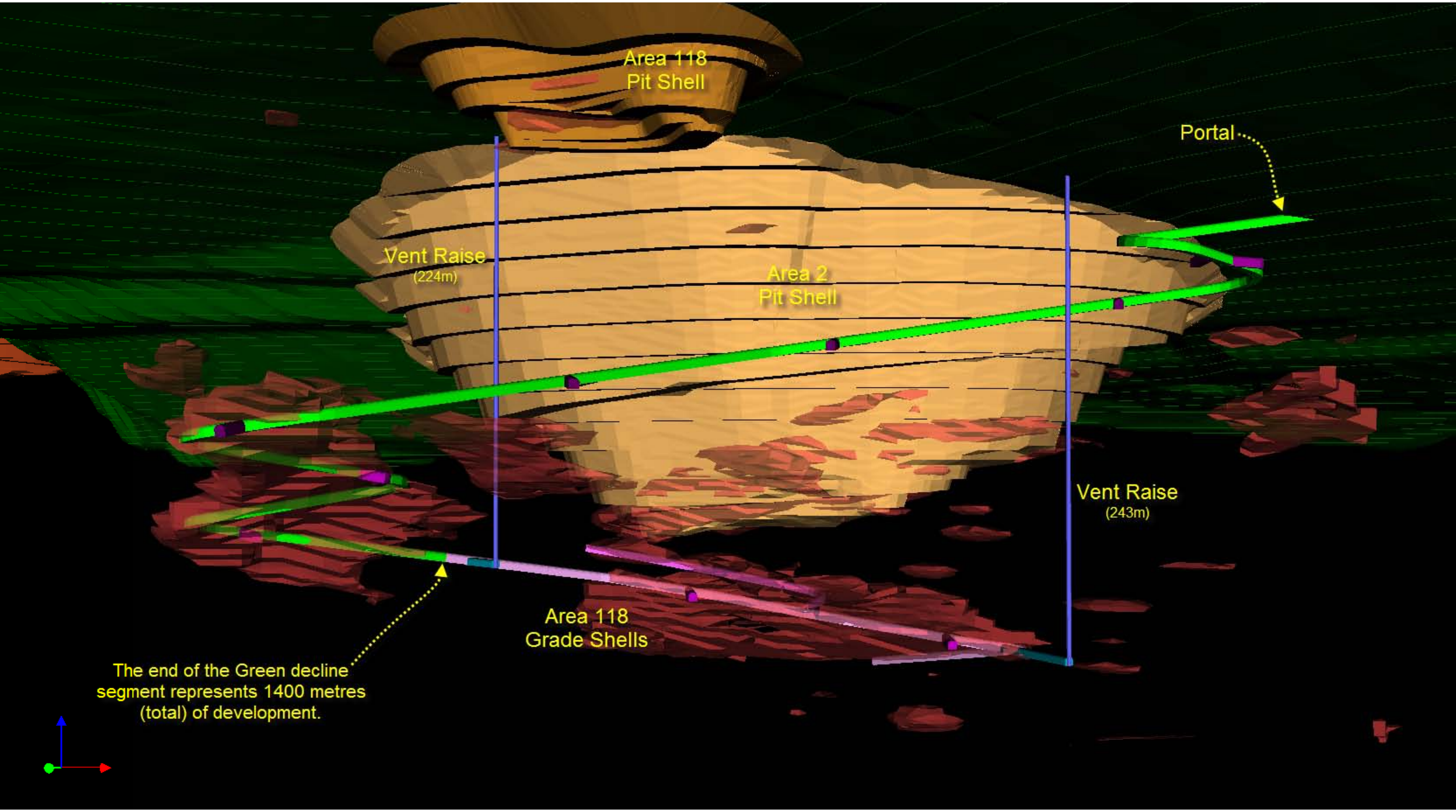
Vancouver B.C.

**Minto Explorations Ltd.**

Minto Waste Dumps and Pits

| | | | |
|---|---|---|---|
| SRK JOB NO.: 2CM022.001 | - | DATE: Oct. 20, 2009 | APPROVED: MDR |
| FILE NAME: SRK_Minto_Pit_Dump_Plan_Oct09.dwg | | FIGURE: 1 | |



Area 118
Pit Shell
Portal
Vent Raise
(224m)
Area 2
Pit Shell
Vent Raise
(243m)
Area 118
Grade Shells
The end of the Green decline
segment represents 1400 metres
(total) of development.

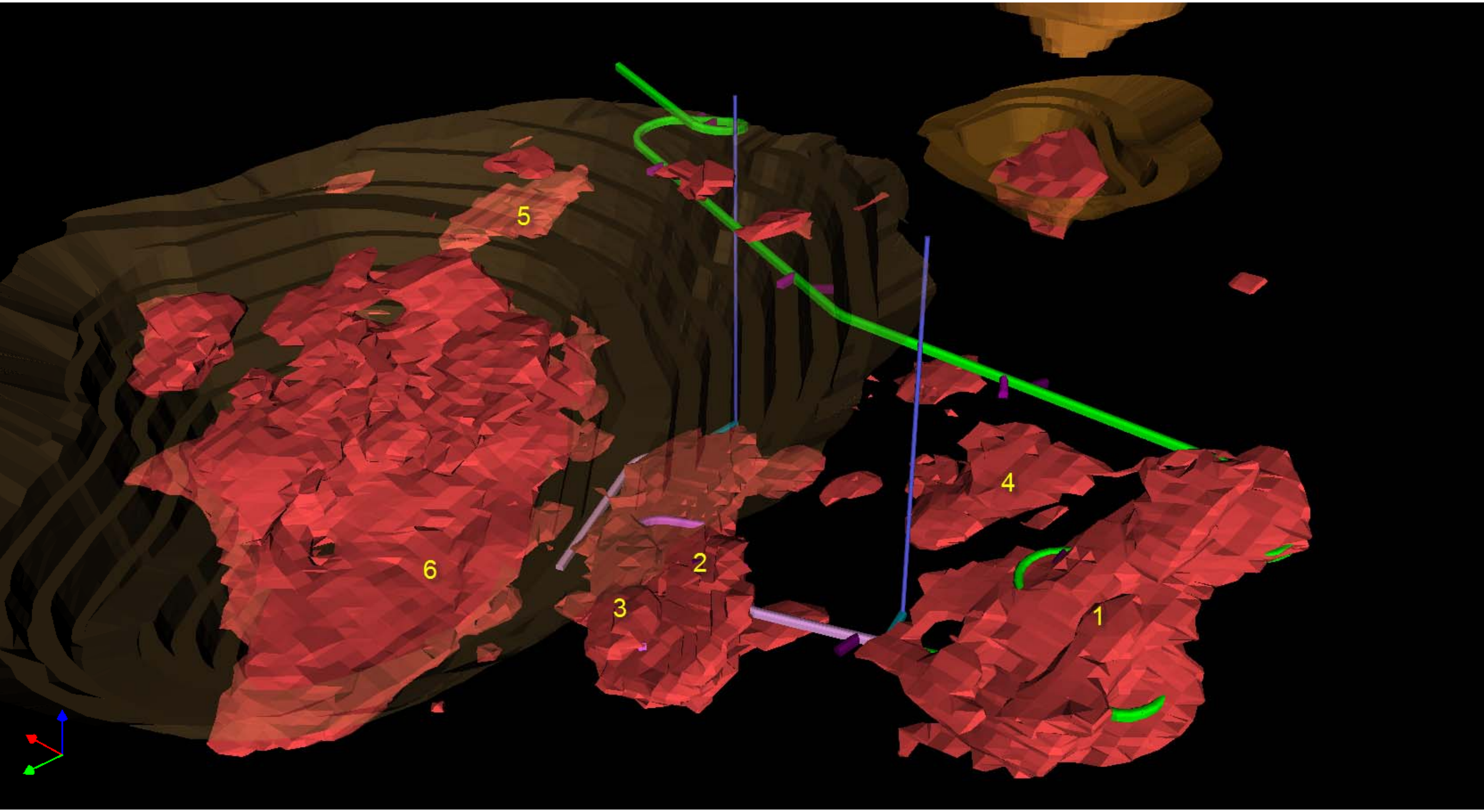

5
4
6
2
3
1

**Table 17.5: Development Lengths for Area 118**

| Description | Ramp Development (m) | Raise Development (m) | Waste Tonnes (t) | Tonnes M+I (t) |
|---|---|---|---|---|
| Exploration Decline | 1,650 | 0 | 67,759 | 24,998 |
| Sump | 165 | 0 | 6,537 | 2,585 |
| Remuck | 150 | 0 | 7,441 | 791 |
| Vent Access | 90 | 0 | 3,184 | 873 |
| Vent Raise | 0 | 467 | 6,014 | 0 |
| Production Decline | 197 | 0 | 2,046 | 8,130 |
| **Total** | **2,251** | **467** | **92,980** | **37,378** |

### Stoping

Room and pillar mining would take place off of the “in-ore” ramp with appropriate pillars around the ramp to ensure its long term stability. Room and pillar sizes have been reviewed to determine the appropriate dimensions based on dip and thickness of the deposits. The preliminary geotechnical recommendations are detailed in Section 18.2.

### Waste and Water Management

Waste rock from the initial mine access ramp would be hauled to surface and missed with open pit waste on the currently planned waste dumps. The volume of waste rock from the potential underground excavations is so small that the addition of this material to the open pit dumps is insignificant. Once stoping commences, waste rock from on-going ramp development could be put back into mined out rooms and not brought to surface. It is estimated that the total waste rock brought to surface from the underground excavations would be less than 0.2 mt or less than 0.3% of the total waste being mined in the LOM open pit plan.

Ground water encountered by an underground mine would be re-cycled underground as drilling with excess water pumps to the mill process water pond.

### Potential Production Rate

Based on the presence of several deposits in the 118 area, and the lack of constraining backfill, it is likely an underground scenario could achieve 1,000 to 2,000 tpd. The tonnage in a large development end of 5 mW x 5 mH x 4.3 mL would yield about 300 tonnes meaning that 1,000 tpd of production would only require the blasting of a little over three large advances per day. The potential addition of more underground deposits (from prospects such as Minto East and Copper Keel) could increase the possible tonnes per day.

### Operating Cost Estimate

Operating costs for a room and pillar method would likely be in the $25 to $40/t range depending on many factors including, but not limited to, production rate, mineralized zone thickness, dip, extent, continuity and rock strength.

### Cut-off Grade (COG)

If a $30/t average mining cost is selected and a processing and G&A cost of $23.09/t is used (See Section 24) then the estimated underground cut-off grade it would be about 1.5% copper in situ. See Table 17.6 for details. This equates to an approximate cut-off NSR value of $60.00/t. See Table 17.1 for the potentially mineable deposits in Area 118.

### Table 17.6: Underground COG Estimate

| Parameter | Unit | UG COG |
|---|---|---|
| **Revenue, smelting & refining** | | |
| Cu price | US$/lb Cu | 2.00 |
| Exchange rate | C$/US$ | 1.10 |
| Cu price | C$/lb Cu | 2.20 |
| Payable metal | % Cu | 97% |
| TC/RC/Transport | C$/lb Cu payable | 0.29 |
| NSR (Cu only) | C$/lb Cu payable | 1.84 |
| **Opex estimates** | | |
| Mining Cost | C$/t milled | 30.00 |
| Processing and G&A cost | C$/t milled | 23.09 |
| Site Cost | C$/tonne milled | 53.09 |
| **Recovery and Dilution** | | |
| Recovered Cu grade | %Cu | 1.31 |
| Process Recovery | average | 92% |
| Plant feed Cu grade | diluted %Cu | 1.42 |
| Dilution | | 10% |
| **Cut-off Grade** | | |
| In-situ cut-off Cu grade (Cu only) | %Cu | 1.6 |
| By-product contribution | % of Cu value | 7% |
| In-situ cut-off Cu grade (inc. by-product value) | %Cu | 1.5 |

### Capital Cost Estimate

SRK estimates that an initial exploration decline to the lowest deposit is 118 will cost approximately $11 m, including leased equipment, infrastructure. A breakdown of the conceptual exploration capital cost estimate is shown in Table 17.7.

**Table 17.7: Conceptual 118 Exploration Development Capital Cost (excluding contingency and exploration drilling)**

| Equipment Type | Cost (K$) |
|---|---|
| **Mobile Equipment Lease** | **1,395** |
| Portal face cut and support | 25 |
| Raise development (467 m @ $3,000/m) | 1,401 |
| Ramp and development (2,251 m @ $3,345/m) | 7,529 |
| **Subtotal Capital Development** | **8,955** |
| **Electrical (portal power line and transformers)** | **235** |
| **Ventilation (fans, air heater and propane tank)** | **195** |
| **Pumping and water control** | **48** |
| **Ground Preparation** | **47** |
| **Misc. (mine rescue, safety, small equipment)** | **200** |
| **Indirect Costs (freight, EP, etc.)** | **255** |
| **TOTAL CAPITAL using Leased Equipment** | **11,330** |

Should the underground mining potential of the Area2/118, Copper Keel and Minto East zones be shown to be favourable, it is estimated that an additional $8 M to $10 M would be required for additional mining equipment, infrastructure and mine development to exploit these zones.

# 18 Mining Operations

## 18.1 Mine Plan

Mine planning for the Phase IV 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 2010 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 IV deposits and used as the basis for the detailed ultimate pit designs for Area 2, 118, North and Ridgetop, along with associated pit phasing. Waste dump were then designed to account for the material produced in each mining phase.

Table 18.1 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 18.2 further summarizes the Minto pits by material types.

### Table 18.1: Open Pit Design

| Pits | Diluted Ore (Kt) | Waste (Kt) | Total material (Kt) | Strip ratio (t:t) | Ore grade | | | Contained Metal | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Cu (%) | Au (g/t) | Ag (g/t) | Cu (Mlbs) | Au (koz) | Ag (koz) |
| Main Ore Stockpile | 873 | NA | 873 | NA | 1.45 | 0.51 | 5.39 | 28 | 14 | 151 |
| Main | 3,253 | 14,530 | 17,784 | 4.5 | 1.66 | 0.58 | 6.73 | 119 | 61 | 703 |
| Minto North | 1,349 | 10,626 | 11,975 | 7.9 | 2.50 | 1.37 | 9.05 | 74 | 60 | 393 |
| Ridgetop | 1,324 | 9,011 | 10,335 | 6.8 | 1.26 | 0.38 | 3.34 | 37 | 16 | 142 |
| 118 | 88 | 639 | 727 | 7.3 | 1.32 | 0.27 | 3.93 | 3 | 1 | 11 |
| Area2 | 4,006 | 35,578 | 39,584 | 8.9 | 1.52 | 0.56 | 5.15 | 134 | 72 | 663 |
| **Total** | **10,894** | **70,384** | **81,278** | **7.0** | **1.64** | **0.64** | **5.89** | **395** | **224** | **2,064** |

*strip ratio does not include Main Ore Stockpile starting balance

**Table 18.2: Material by Type**

| Pit | Rock (kt) | Overburden (kt) | Sulphide ore (kt) | Total Material (kt) |
|---|---|---|---|---|
| Main Ore Stockpile | NA | NA | 873 | 873 |
| Main | 6,269 | 8,261 | 3,253 | 17,784 |
| Minto North | 8,882 | 1,745 | 1,349 | 11,975 |
| Ridgetop | 7,374 | 1,637 | 1,324 | 10,335 |
| 118 | 442 | 197 | 88 | 727 |
| Area 2 | 30,532 | 5,040 | 4,006 | 39,579 |
| **Total** | **53,498** | **16,881** | **10,894** | **81,273** |

## 18.2 Mine Design

### 18.2.1 Geotechnical Test work and Conclusions

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 following section is intended to summarize the evaluation; the complete report is included in Appendix B Geotechnical Evaluation and should be referenced for additional detail.

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;
- Recommendation of optimal pit slope angles and pit architecture for mine design purposes; and,
- Recommendation of room and pillar dimensions as well as ground support requirements for the alternative underground development of Area 118.

As commissioned, the work reported herein 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 18.3.

### Table 18.3: 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 18.4.

### Table 18.4: 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, yielding compressive strengths ($\sigma_1$) ranging between 213.8 and 294 .1 mPa with a mean value of 262.1 MPa under confining pressures ($\sigma_3$) ranging between 6.9 and 20.7 MPa, with a mean value of 13.8 MPa.

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.

### *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 do 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 18.5.

**Table 18.5: 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 |

## *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 drillhole data, the Area 2 weathered domain is adjudged to extend to depths of approximately 50 to 100 m below the current ground surface.

### *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 approximately16 m. The depth of bedrock weathering at Area 118 is generally to about 30 to 60 m below the current ground surface.

### *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.

### *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.

### *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.

**Table 18.6: Interramp Slope Stability Modelling Results**

| Deposit | Sector | Height (m) | Mean FS | 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 of the 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 drillholes 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.

Major faults at similar orientations are also anticipated through the Area 118 underground mining areas and development.

### *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 modeled 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 porewater 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, backbreak 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, pit slope design recommendations for each of the Minto areas are summarized in 18.7.

**Table 18.7: Summary of Pit Slope Design Recommendations**

| Deposit Area | Sector(s) | Max. Slope Height (m) | Inter-ramp Angle (°) | Bench Face Angle (°) | Bench Height (m) | Berm Width (m) | Stepout 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.

## Area 118 Underground Pillar Assessment

In addition to the small open pit at Area 118 previously discussed, underground mining is also planned for Area 118. Based on the geotechnical data previously described, pillar strengths were evaluated in order to recommend suitable pillar dimensions for room and pillar mining. Based on estimates of ore deposit depth and thickness variability, pillar heights of 5 m, 10 m and 15 m were assessed and ore depths, and respective overburden stresses, of 150 m, 200 m and 250 m were considered.

In-situ Rock Mass Rating (IRMR) and Rock Mass Strength (RMS) values were evaluated for the ore zone as well as materials above and below the ore zone in geotechnical drillholes C09-01 and C09-02. An average IRMR and RMS of 55 and 60 mPa, respectively, were conservatively estimated for pillar, roof and floor materials. Using Laubscher's (1990) method, the IRMR of 55 was reduced to a Mining Rock Mass Rating (MRMR) of 47 and the 60 mPa RMS to a Design Rock Mass Strength (DRMS) of 51 mPa by applying appropriate reductions for joint orientation, blasting and water.

Based on empirical data presented by Ouchi (2004), assuming a RMR value of 55, the maximum unsupported span distance was estimated to be 6 m for all pillar height/deposit depth combinations considered. Subsequently, the tributary area method was used to estimate minimum pillar dimensions required to support 6 m x 6 m or, if required, lesser, roof spans based on pillar height and overburden stresses. The resultant recommended room and pillar dimensions and extraction ratios are summarized below in Table 18.8.

**Table 18.8: Room and Pillar Size Recommendations**

| Depth (m) | Pillar Height (m) | Pillar Dimensions (m) | Room Dimensions (m) | Extraction Ratio |
|---|---|---|---|---|
| 150 | 5 | 4x4 | 6x6 | 84% |
| 150 | 10 | 5x5 | 6x6 | 79% |
| 150 | 15 | 6x6 | 6x6 | 75% |
| 200 | 5 | 4.5x4.5 | 6x6 | 82% |
| 200 | 10 | 6x6 | 6x6 | 75% |
| 200 | 15 | 7.5x7.5 | 6x6 | 69% |
| 250 | 5 | 5x5 | 6x6 | 79% |
| 250 | 10 | 7x7 | 6x6 | 71% |
| 250 | 15 | 8x8 | 5x5 | 62% |

Based on geotechnical conditions previously described, ground support requirements for development such as the 5 m x 5 m declines were estimated as follows:

- Pattern bolting with 2.4 m long bolts at a 2 m spacing within and between rings; and,
- Welded wire mesh in back and top of walls.

## Recommendations for Additional 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 drillholes 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 portion of Area 118 will also require additional geotechnical drilling for rock mass characterization at the feasibility and design levels. The Area 118 and Ridgetop open pits most likely will not require additional geotechnical drilling unless major changes are made to the current plans.

## 18.3 Mine Operation

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 2011 and correlates with the completion of mining in the Main Pit. The owner-operator mining cost unit rate used in the Whittle optimization was $2.11 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 1.4 Mtpa. Mining costs were developed with the assistance of an experienced mining contractor familiar with the area and similar sized operations.

### Mine Equipment

The major mining equipment requirements are indicated in Table 18.9 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 40,000 tpd total material, which will be sufficient for the proposed life-of-mine plan.

**Table 18.9: Mine Equipment**

| No. of units | Equipment Type |
|---|---|
| 2 | Hitachi EX1900 Front Shovel |
| 8 | Cat 777F Haul Truck |
| 1 | Cat 992G Loader |
| 1 | Cat 365CL Excavator |
| 3 | Cat D9T Dozer |
| 2 | Cat 16 m Grader |
| 2 | Atlas Copco PV235 Drill |
| 1 | Atlas Copco D9-11 Drill |
| 1 | Cat 777C Water Truck |
| 1 | Cat 777B w/trailer |

### 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 shovels 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 additional equipment listed in Table 18.9 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.

#### 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).

## 18.4 Production Schedule

### 18.4.1 Mine Sequence/Phasing

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, while providing the required mill feed production per period. The overall site plan final configuration is illustrated in Figure 18.2 below.

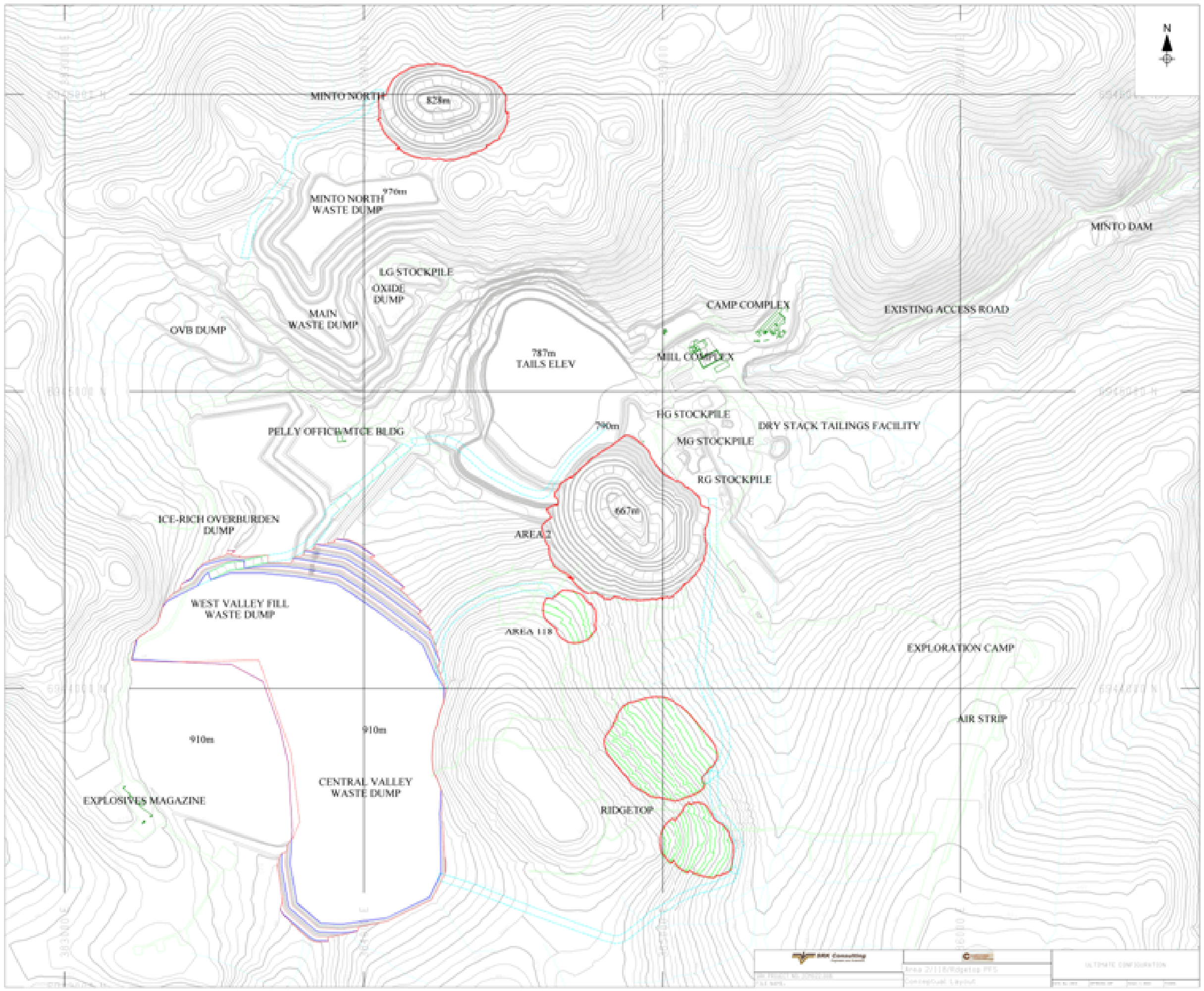



**Figure 18.1: Overall Site Plan Final Configuration**

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 IV pits to be backfilled into the Main Pit, thereby, eliminating the current need of drying the tailings and significantly reduce overall costs.

Main Pit will be followed by Minto North, then Ridgetop, 118 and ends with the Area 2 deposit. During the initial pre-stripping of the Phase IV 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 three stages. The stage tonnages and associated grades are summarized in Table 18.9, while a breakdown of material types is summarized in Table 18.10.

## Table 18.10: 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 (Mlbs) | Au (koz) | Ag (koz) |
| Main Ore Stockpile | 873 | NA | 873 | NA | 1.45 | 0.51 | 5.39 | 28 | 14 | 151 |
| Main | 3,253 | 14,530 | 17,784 | 4.5 | 1.66 | 0.58 | 6.73 | 119 | 61 | 703 |
| Subtotal Main only | 4,127 | 14,530 | 18,657 | 4.5 | 1.62 | 0.57 | 6.44 | 147 | 75 | 855 |
| Minto North | 1,349 | 10,626 | 11,975 | 7.9 | 2.50 | 1.37 | 9.05 | 74 | 60 | 393 |
| Ridgetop South | 231 | 2,227 | 2,457 | 9.6 | 1.66 | 0.95 | 8.66 | 8 | 7 | 64 |
| Ridgetop North | 1,093 | 6,785 | 7,878 | 6.2 | 1.17 | 0.26 | 2.22 | 28 | 9 | 78 |
| Subtotal Ridgetop only | 1,324 | 9,011 | 10,335 | 6.8 | 1.26 | 0.38 | 3.34 | 37 | 16 | 142 |
| 118 | 88 | 639 | 727 | 7.3 | 1.32 | 0.27 | 3.93 | 3 | 1 | 11 |
| Area2 - Phase1 | 1,440 | 13,175 | 14,615 | 9.1 | 1.42 | 0.52 | 5.43 | 45 | 24 | 251 |
| Area2 - Phase2 | 1,768 | 11,120 | 12,887 | 6.3 | 1.50 | 0.54 | 4.71 | 58 | 31 | 268 |
| Area2 - Phase3 | 798 | 11,283 | 12,081 | 14.1 | 1.72 | 0.69 | 5.61 | 30 | 18 | 144 |
| Subtotal Area2 only | 4,006 | 35,578 | 39,584 | 8.9 | 1.52 | 0.56 | 5.15 | 134 | 72 | 663 |
| | | | | | | | | | | |
| **Grand total** | **10,894** | **70,384** | **81,278** | **7.0** | **1.64** | **0.64** | **5.89** | **395** | **224** | **2,064** |

*strip ratio does not include Main Ore Stockpile starting balance

### Table 18.11: Material Types

| Phase | Rock (kt) | Overburden (kt) | Sulphide ore (kt) | Total Material (kt) |
|---|---|---|---|---|
| Main* | 6,269 | 8,261 | 4,127 | 18,657 |
| Minto North | 8,882 | 1,745 | 1,349 | 11,975 |
| Ridgetop South | 1,792 | 434 | 231 | 2,458 |
| Ridgetop North | 5,582 | 1,203 | 1,093 | 7,878 |
| 118 | 442 | 197 | 88 | 727 |
| Area 2 – Phase 1 | 11,179 | 1,996 | 1,440 | 14,615 |
| Area 2 – Phase 2 | 9,567 | 1,552 | 1,768 | 12,887 |
| Area 2 – Phase 3 | 9,786 | 1,492 | 798 | 12,076 |
| **Grand total** | **53,498** | **16,881** | **10,894** | **81,273** |

Note: Main Pit includes ore stockpile start balance

Figure 18.3 further summarizes the stage designs for each of the deposits (illustrating mineralized rock and waste tonnages, and copper grade.

The pit stages were based on the detailed pit designs created. The pit waste for each of the individual deposits will be placed into the valley fill waste dumps south west of the final pit limits. All process plant feed rock will be hauled to the appropriate ROM ore stockpiles.

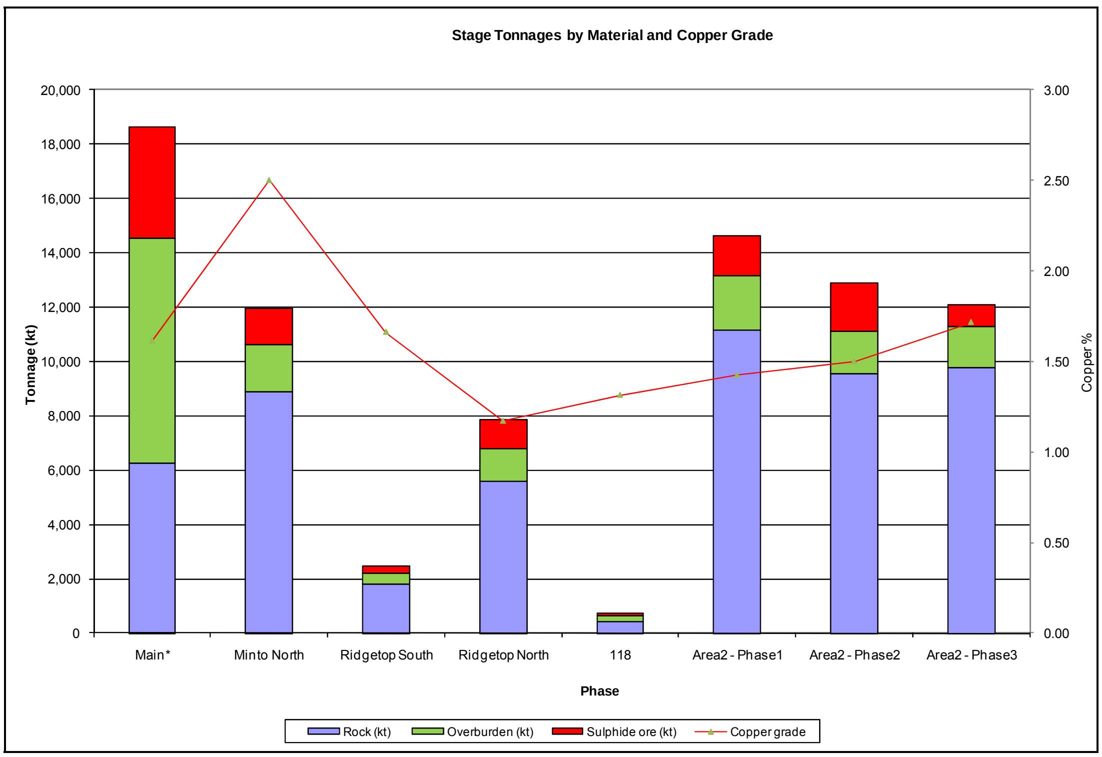



**Figure 18.2: Stage Summary**

## 18.4.2 Open Pit Mine Production Schedule

The production schedule for the Minto deposits was developed with the aid of MineSight™ software, and incorporated the deposits at Main, North, Ridgetop, 118 and Area 2 mentioned above.

The planned ramp up in plant throughput is as follows: Q1 and Q2 2010 at 3,000 tpd; Q3, Q4 2010 and Q1 2011 at 3,475 tpd and; Q2 2011 and beyond at 3,750 tpd. Completion of Main Pit will be carried out first and is scheduled to be completed by 2011, followed by the Phase IV pits. The maximum amount of planned total material to be moved is approximately 35,000 t/day. The average total mining rate is planned to be 27,500 t/day. Only measured and indicated resources were used in the LOM plan.

Table 18.12 below is a summary of total material movement by year for the open pit mine production schedule, with Table 18.13 summarizing the process schedule.

### Table 18.12: Mine Production Schedule – Minto Deposits

| Parameter | Units | Year | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Total | 2010 | 2011 | 2012 | 2013 | 2014 | 2015 | 2016 | 2017 |
| | | | Main Pit | | Phase IV Pits | | | | | |
| **Mining** | | | | | | | | | | |
| Ore | Mt | **10.0** | 2.0 | 1.3 | 0.3 | 1.4 | 1.2 | 1.4 | 1.3 | 1.1 |
| Overburden | Mt | **16.9** | 4.9 | 3.4 | 2.3 | 1.2 | 1.6 | 1.0 | 1.9 | 0.7 |
| Waste Rock | Mt | **53.5** | 3.3 | 3.0 | 7.1 | 6.0 | 8.6 | 7.9 | 9.7 | 8.0 |
| Total Waste | Mt | **70.4** | 8.2 | 6.3 | 9.4 | 7.2 | 10.2 | 8.9 | 11.6 | 8.6 |
| Total Material | Mt | **80.4** | 10.2 | 7.6 | 9.7 | 8.6 | 11.4 | 10.3 | 12.9 | 9.8 |
| Strip ratio | Wt:Ot | **7.0** | 4.1 | 5.0 | 33.2 | 5.1 | 8.6 | 6.3 | 8.7 | 7.6 |
| Daily production | Kt/day | **27.5** | 27.8 | 20.9 | 26.4 | 23.5 | 31.1 | 28.3 | 35.3 | 26.8 |
| Mined Cu grade | % | **1.66** | 1.71 | 1.59 | 1.20 | 2.43 | 1.28 | 1.42 | 1.42 | 1.80 |
| Mined Au grade | g/t | **0.65** | 0.52 | 0.67 | 0.50 | 1.24 | 0.43 | 0.51 | 0.51 | 0.73 |
| Mined Ag grade | g/t | **5.93** | 7.04 | 6.23 | 2.27 | 8.71 | 3.76 | 5.23 | 4.48 | 6.00 |
| Mined Contained Cu | Mlbs | **367** | 74 | 45 | 7 | 75 | 33 | 44 | 42 | 45 |
| Mined Contained Au | Koz | **210** | 33 | 28 | 5 | 56 | 16 | 23 | 22 | 27 |
| Mined Contained Ag | Koz | **1,912** | 447 | 257 | 21 | 394 | 143 | 238 | 192 | 221 |

### Table 18.13: Processing Production Schedule – Minto Deposits

| Parameter | Units | Year | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Total | 2010 | 2011 | 2012 | 2013 | 2014 | 2015 | 2016 | 2017 | 2018 |
| | | | Main Pit | | Phase IV Pits | | | | | | |
| **Processing** | | | | | | | | | | | |
| Processed Ore | Mt | **10.9** | 1.2 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 0.1 |
| Process rate | dmt/day | **3,704** | 3,334 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 |
| Proc. Cu grade | % | **1.64** | 2.33 | 1.68 | 1.10 | 2.47 | 1.22 | 1.44 | 1.40 | 1.64 | 0.81 |
| Proc. Au grade | g/t | **0.64** | 0.80 | 0.67 | 0.35 | 1.27 | 0.40 | 0.52 | 0.50 | 0.65 | 0.25 |
| Proc. Ag grade | g/t | **5.89** | 9.84 | 6.48 | 3.64 | 8.88 | 3.66 | 5.32 | 4.44 | 5.52 | 2.67 |
| **Recovery** | | | | | | | | | | | |
| Copper | % | **92.8** | 94.0 | 94.0 | 93.6 | 92.0 | 92.3 | 92.0 | 92.0 | 92.4 | 92.0 |
| Gold | % | **73.8** | 80.0 | 80.0 | 77.9 | 70.0 | 71.3 | 70.0 | 70.2 | 71.8 | 70.0 |
| Silver | % | **81.3** | 86.7 | 86.7 | 84.9 | 78.0 | 79.1 | 78.0 | 78.2 | 79.6 | 78.0 |
| **Metal in Concentrates** | | | | | | | | | | | |
| Copper | Mlbs | **366** | 59 | 48 | 31 | 69 | 34 | 40 | 39 | 46 | 1 |
| Gold | oz | **164,814** | 24,961 | 23,470 | 12,163 | 39,168 | 12,529 | 16,028 | 15,594 | 20,407 | 494 |
| Silver | oz | **1,684,688** | 333,701 | 247,310 | 136,463 | 304,882 | 127,345 | 182,541 | 153,122 | 193,450 | 5,874 |

### Table 18.14: Production Schedule – Minto Deposits

| Parameter | Units | Year | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Total | 2010 | 2011 | 2012 | 2013 | 2014 | 2015 | 2016 | 2017 | 2018 |
| | | | Main Pit | | Phase IV Pits | | | | | | |
| **Mining** | | | | | | | | | | | |
| Ore | Mt | **10.0** | 2.0 | 1.3 | 0.3 | 1.4 | 1.2 | 1.4 | 1.3 | 1.1 | - |
| Overburden | Mt | **16.9** | 4.9 | 3.4 | 2.3 | 1.2 | 1.6 | 1.0 | 1.9 | 0.7 | |
| Waste Rock | Mt | **53.5** | 3.3 | 3.0 | 7.1 | 6.0 | 8.6 | 7.9 | 9.7 | 8.0 | |
| Total Waste | Mt | **70.4** | 8.2 | 6.3 | 9.4 | 7.2 | 10.2 | 8.9 | 11.6 | 8.6 | - |
| Total Material | Mt | **80.4** | 10.2 | 7.6 | 9.7 | 8.6 | 11.4 | 10.3 | 12.9 | 9.8 | - |
| Strip ratio | Wt:Ot | **7.0** | 4.1 | 5.0 | 33.2 | 5.1 | 8.6 | 6.3 | 8.7 | 7.6 | - |
| Daily production | Kt/day | **27.5** | 27.8 | 20.9 | 26.4 | 23.5 | 31.1 | 28.3 | 35.3 | 26.8 | - |
| Mined Cu grade | % | **1.66** | 1.71 | 1.59 | 1.20 | 2.43 | 1.28 | 1.42 | 1.42 | 1.80 | - |
| Mined Au grade | g/t | **0.65** | 0.52 | 0.67 | 0.50 | 1.24 | 0.43 | 0.51 | 0.51 | 0.73 | - |
| Mined Ag grade | g/t | **5.93** | 7.04 | 6.23 | 2.27 | 8.71 | 3.76 | 5.23 | 4.48 | 6.00 | - |
| Mined Contained Cu | Mlbs | **367** | 74 | 45 | 7 | 75 | 33 | 44 | 42 | 45 | - |
| Mined Contained Au | Koz | **210** | 33 | 28 | 5 | 56 | 16 | 23 | 22 | 27 | - |
| Mined Contained Ag | Koz | **1,912** | 447 | 257 | 21 | 394 | 143 | 238 | 192 | 221 | - |
| **Processing** | | | | | | | | | | | |
| Processed Ore | Mt | **10.9** | 1.2 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 0.1 |
| Process rate | dmt/day | **3,704** | 3,334 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 |
| Proc. Cu grade | % | **1.64** | 2.33 | 1.68 | 1.10 | 2.47 | 1.22 | 1.44 | 1.40 | 1.64 | 0.81 |
| Proc. Au grade | g/t | **0.64** | 0.80 | 0.67 | 0.35 | 1.27 | 0.40 | 0.52 | 0.50 | 0.65 | 0.25 |
| Proc. Ag grade | g/t | **5.89** | 9.84 | 6.48 | 3.64 | 8.88 | 3.66 | 5.32 | 4.44 | 5.52 | 2.67 |

With an assumed schedule start date of January 2010, the Minto open pit mine will produce a further of 10.9 million tonnes (Mt) of mill feed (includes Main Pit stockpile balance at start of schedule) and 70.4 Mt of waste rock over an 18-year mine operating life (yielding an overall strip ratio of 7.0:1 (t:t). The mine schedule focuses on achieving the required plant feed production rate, mining of higher grade material early in schedule, while balancing grade and strip ratios. The ROM stockpiles are used in the schedule in order to smooth out mill head grades and provide required mill feed during initial pre-stripping of Phase IV pits. Figures 18.4 through to 18.6 summarize pit tonnages and grade by period, as well as annual mined benches from each stage.

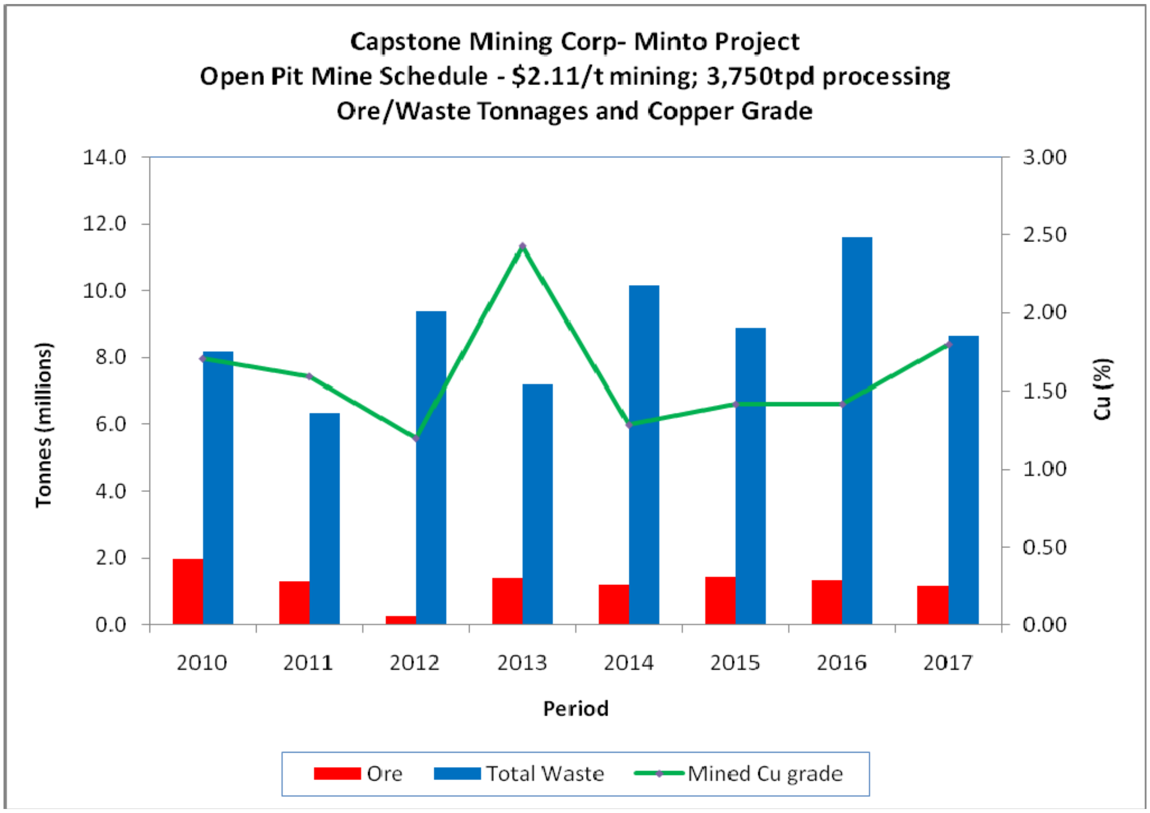



**Figure 18.3: Period Tonnages and Copper Grade**

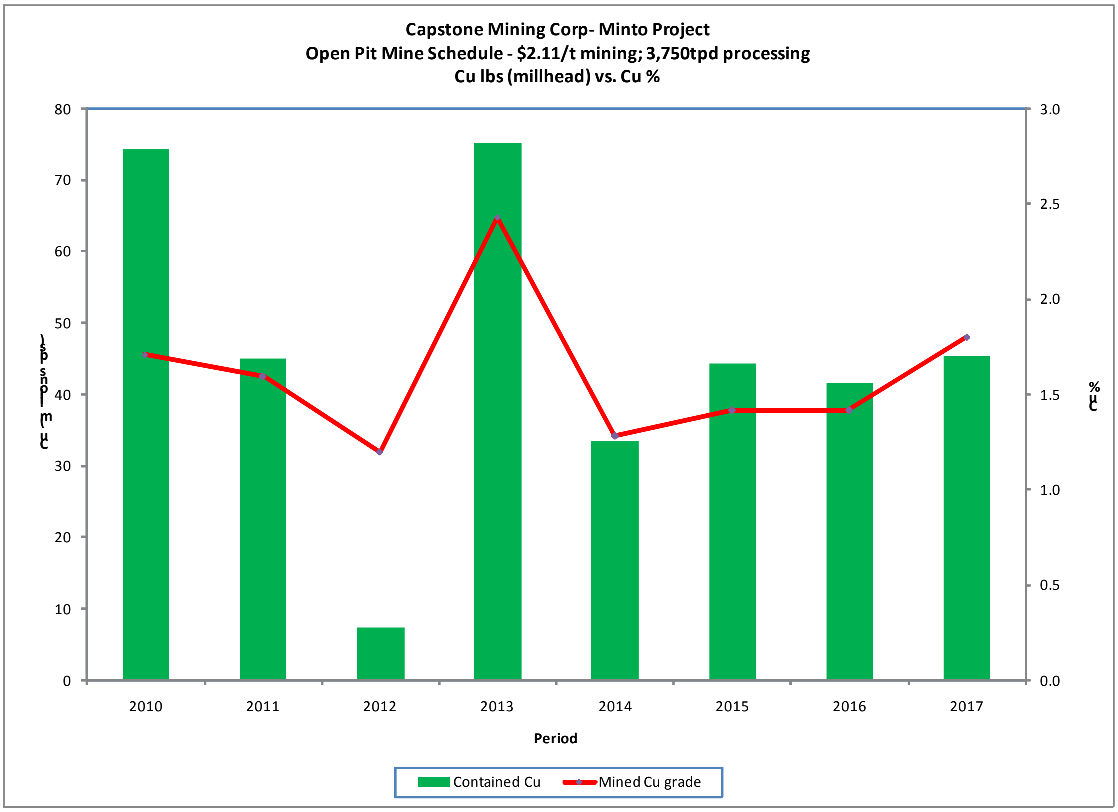



Figure 18.4: Mined Contained Copper and Grades

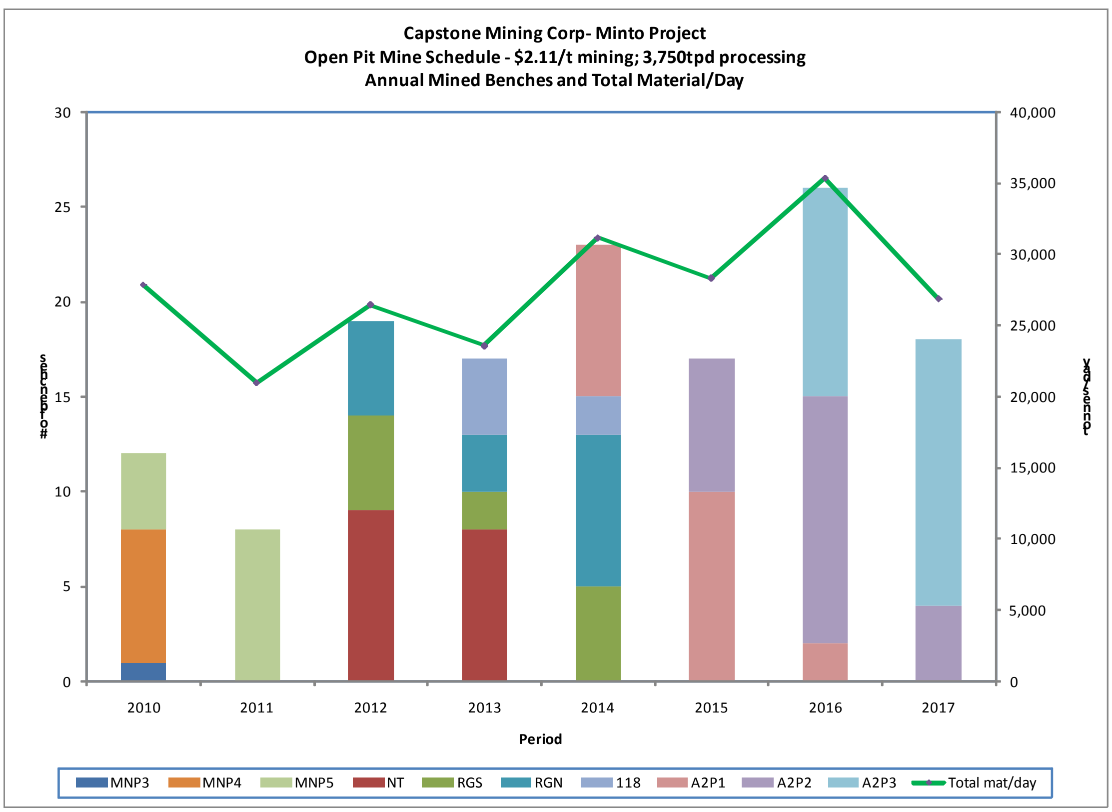



Figure 18.5: Annual Mined Benches

To further illustrate the progression of mining of the Minto deposit, Table 18.15 provides the pit and stage bottom elevation at the end of each period, while Figures 18.7 through to 18.12 provide the status of the pit configuration, dump advance, as well as the tailings management facility, at the end of years 2012 through to the end of mining in 2017, respectively.

### Table 18.15: Bottom Bench Elevations by Stage and Period

| Mine Phase | 2010 | 2011 | 2012 | 2013 | 2014 | 2015 | 2016 | 2017 |
|---|---|---|---|---|---|---|---|---|
| MNP3 | 724 | | - | - | - | - | - | - |
| MNP4 | 712 | | - | - | - | - | - | - |
| MNP5 | 790 | 712 | - | - | - | - | - | - |
| NT | - | - | 900 | 828 | - | - | - | - |
| RGS | - | - | 892 | 874 | 829 | - | - | - |
| RGN | - | - | 883 | 856 | 784 | - | - | - |
| A2118 | | - | - | 883 | 865 | - | - | - |
| A2P1 | - | - | - | - | 811 | 721 | 703 | - |
| A2P2 | - | - | - | - | - | 829 | 712 | 676 |
| A2P3 | - | - | - | - | - | - | 793 | 667 |

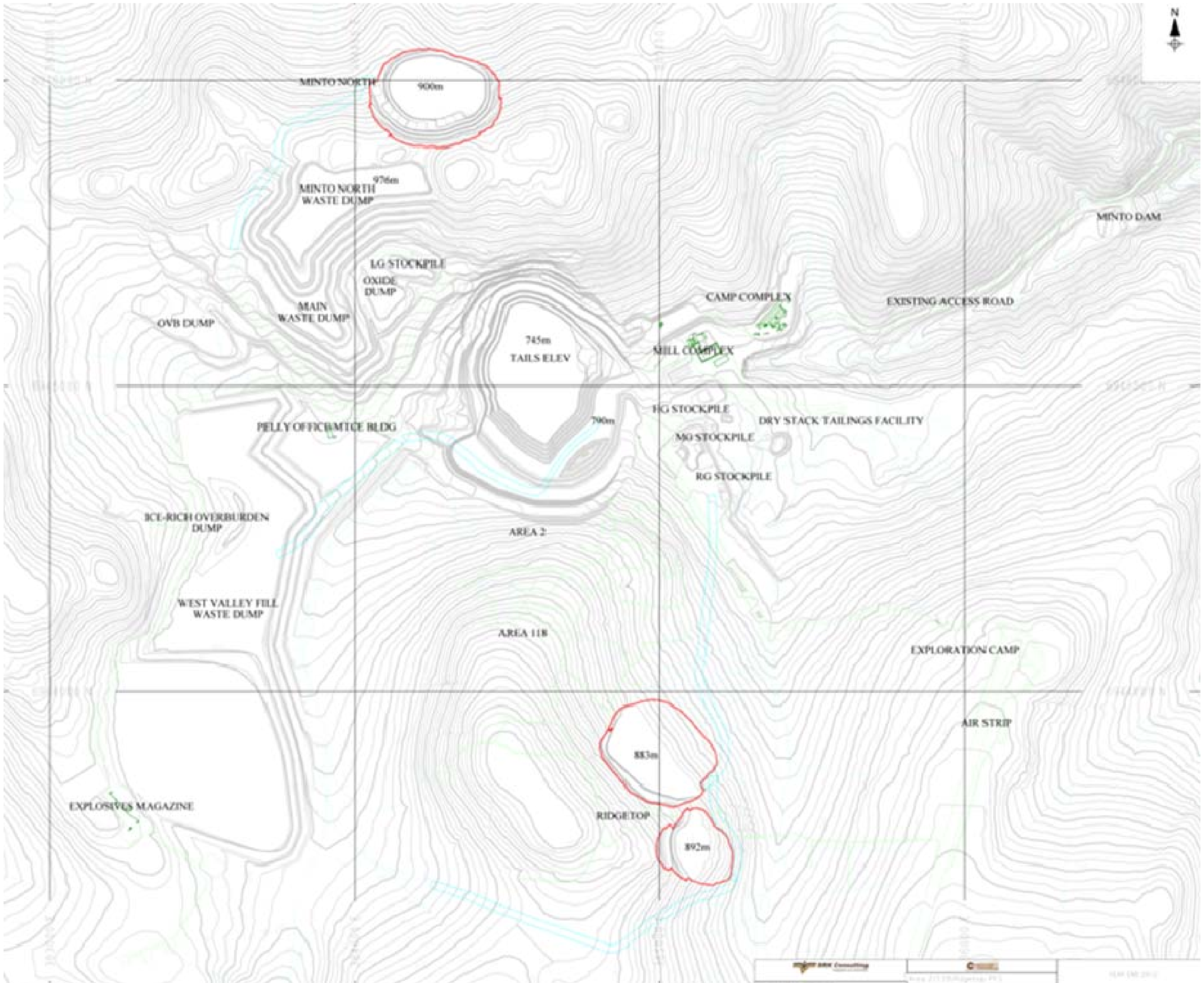



**Figure 18.6: End of Year 2012**

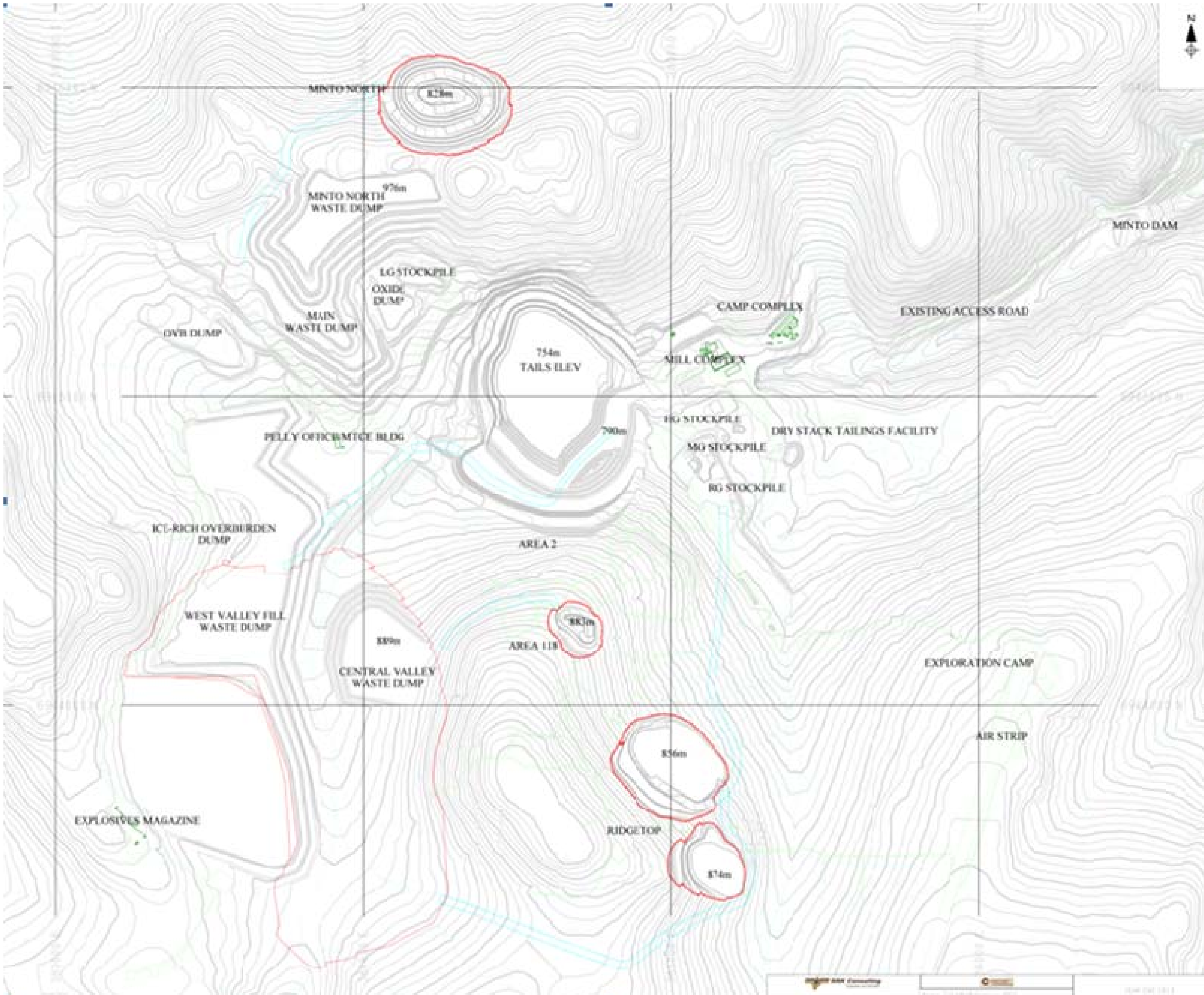



**Figure 18.7: End of Year 2013**

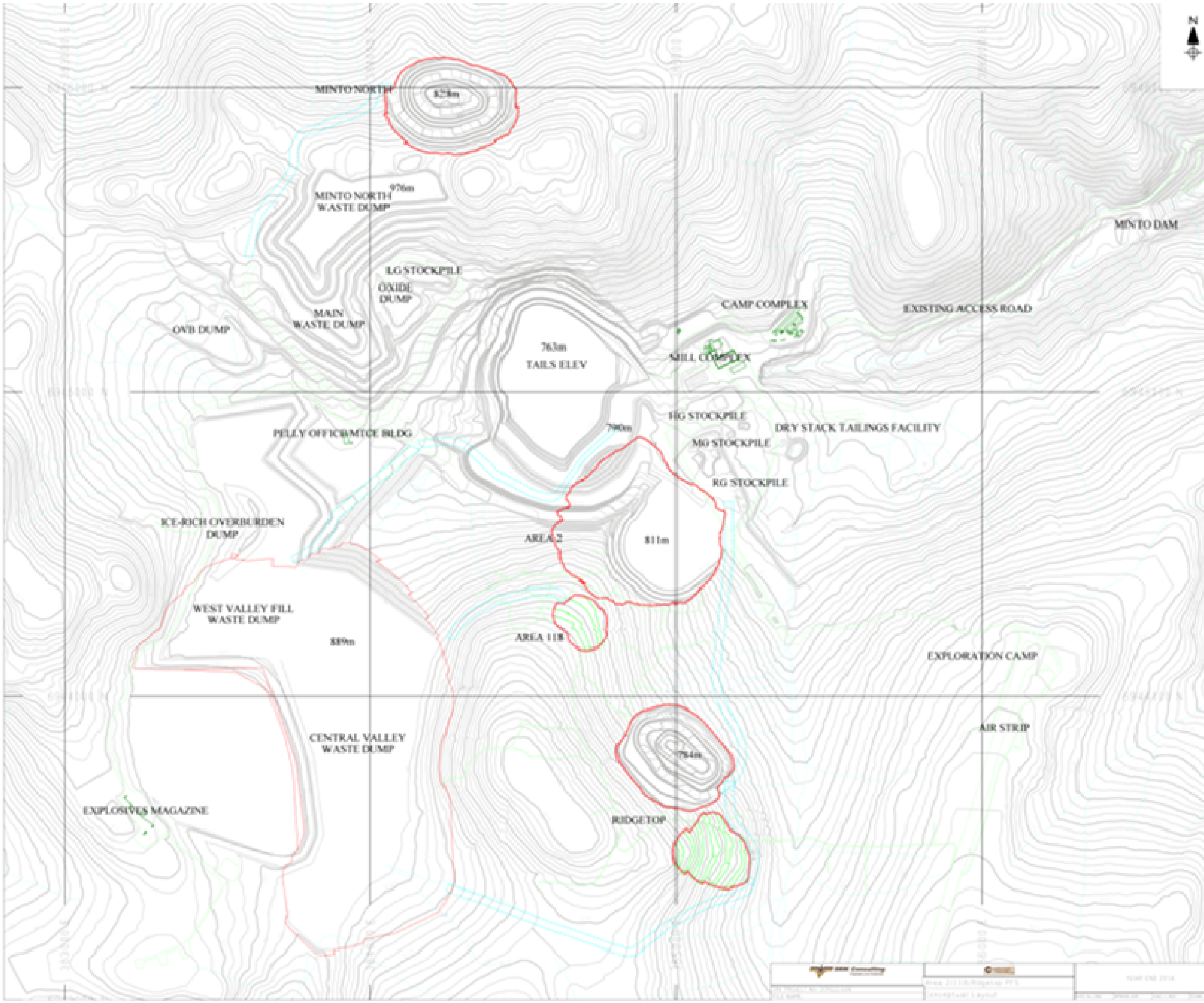



**Figure 18.8: End of Year 2014**

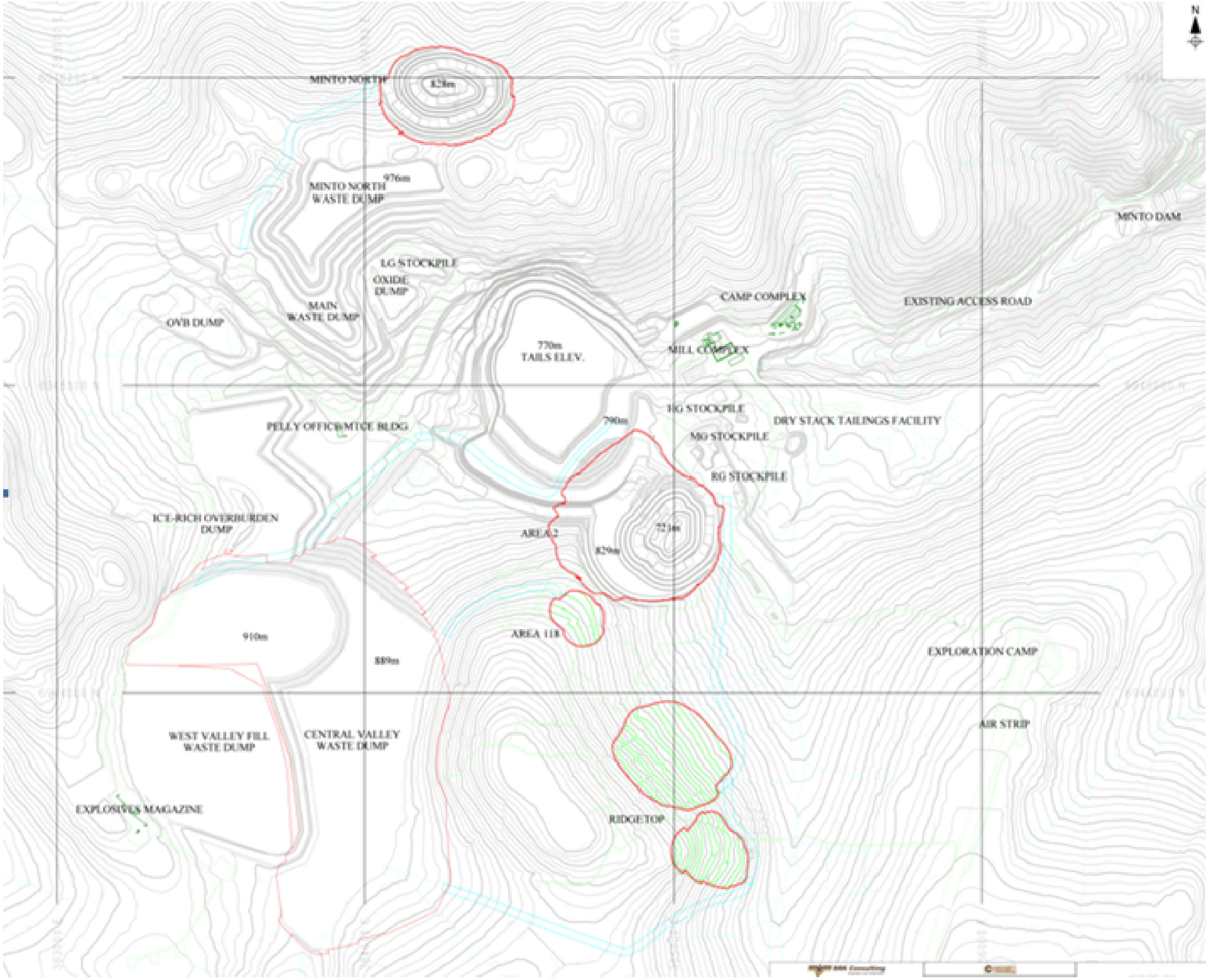



**Figure 18.9: End of Year 2015**

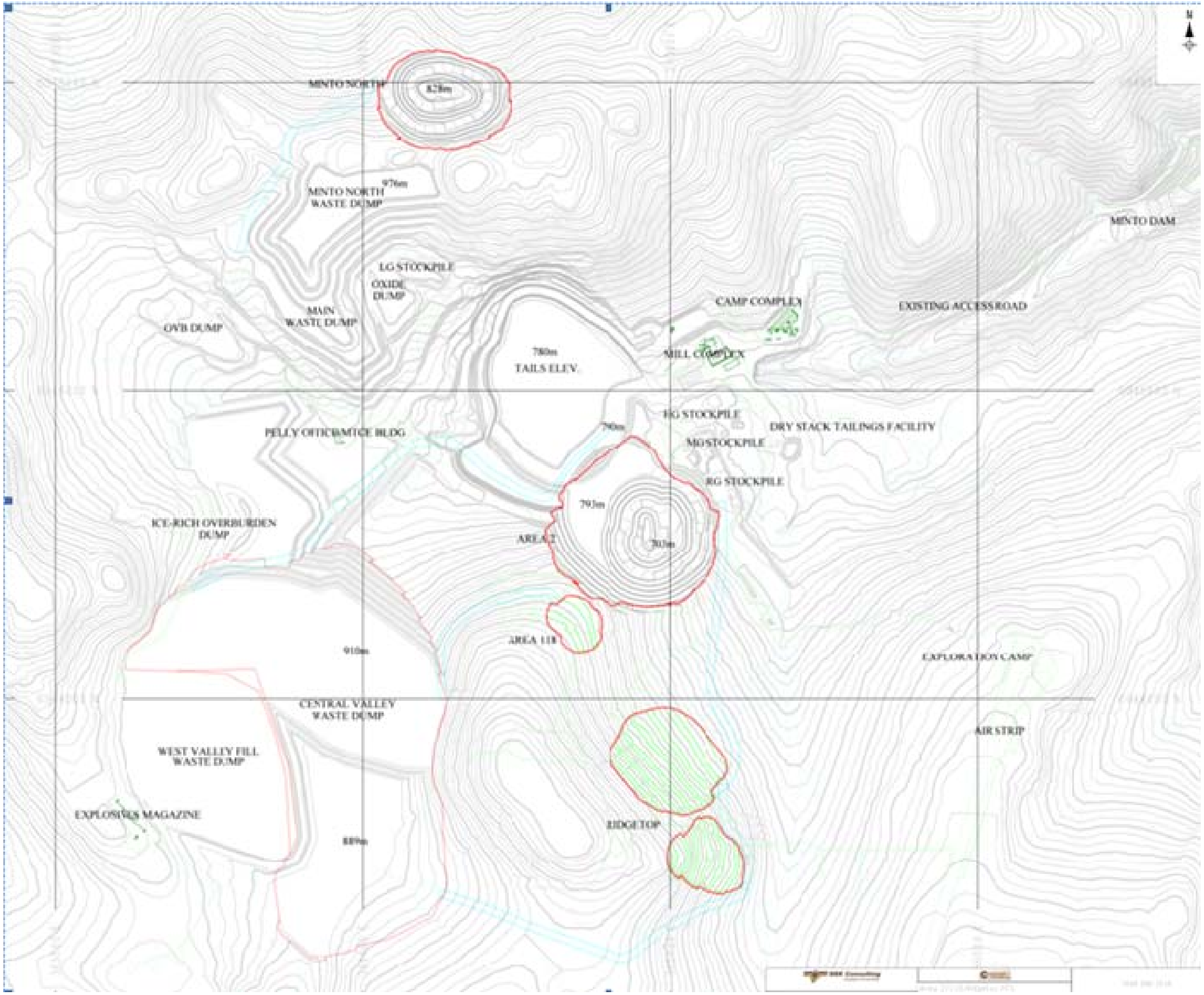



**Figure 18.10: End of Year 2016**

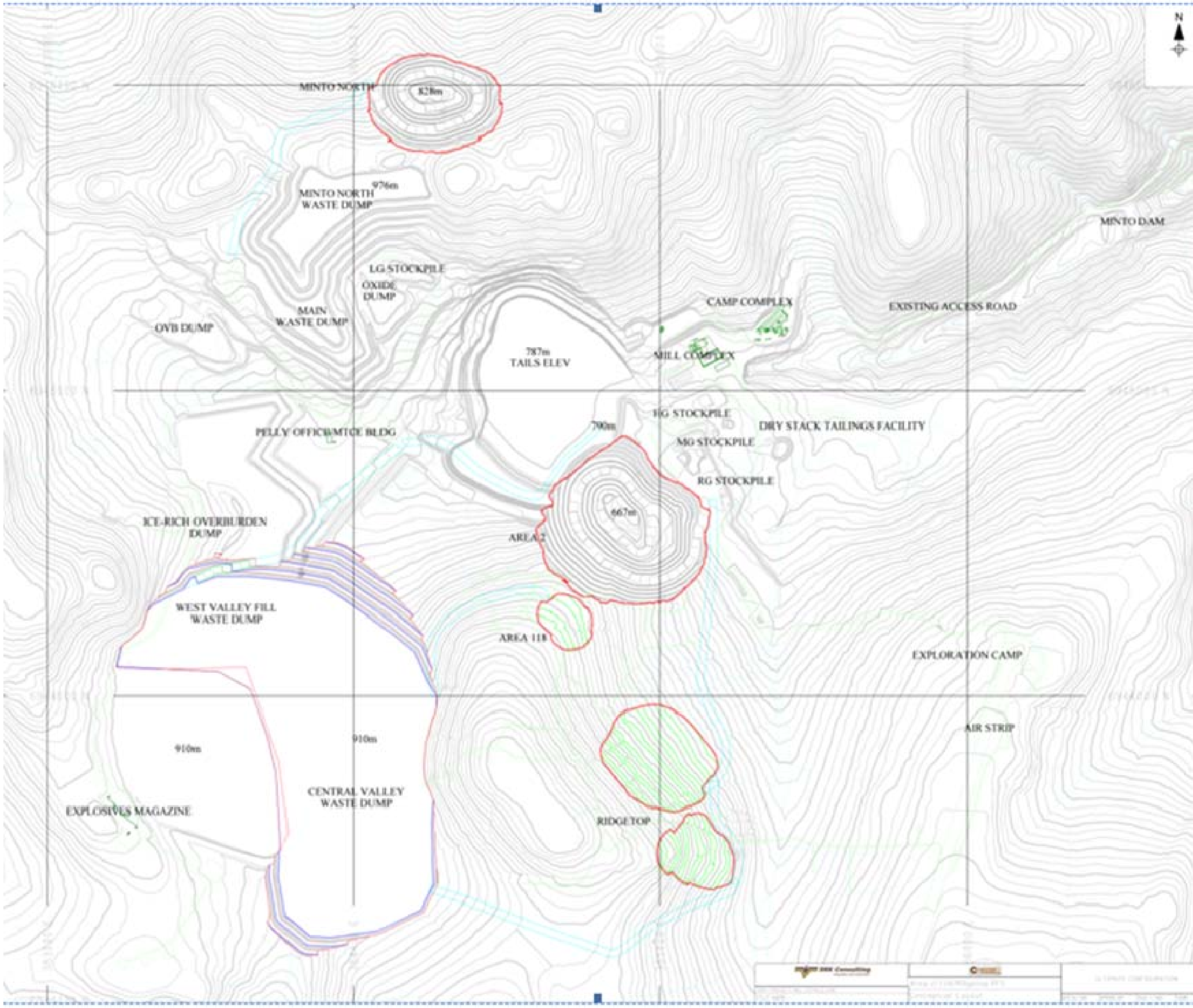



**Figure 18.11: End of Year 2017**

**Pit Development**

2010: Mining of Main Pit continues as per the 2010 Budget (Phase 3 and 5 completed). Total of 8.2 Mt of waste mined in period along with 2.0 Mt of ore, at a mill head copper grade of 1.71%. Processing rate increased to 3,475 tpd by end of year.

2011: Mining of the Main Pit is completed by year end. A total of 1.3 Mt of plant feed is mined in the year. Mill head grade for the year averages 1.59% Cu. 6.3 Mt of waste produced for a mined strip ratio of 5:1 (waste: ore). Stockpile 2 (regular grade) depleted. Transition to an owner-operated fleet.

2012: Transition stage from Main Pit to Phase IV pits, where pre-stripping of North, and Ridgetop commences. Buttress is constructed in Main Pit and tailings converted from dry-stack method to thickened tailings and deposition back into the Main Pit commences. Only 280kt of ore mined and the balance of the mill feed required is drawn from existing stockpile inventories. The waste produced for the year totals 9.4 Mt. Mill head grade averages 1.2% Cu.

2013: Minto North pit completed, and mining continues at Ridgetop. Pre-stripping of 118 pit started. Average mill head grade increases to 2.43% Cu. The waste produced over the period totals 7.2 Mt for a 5.1:1 strip ratio (1.4 Mt plant feed).

2014: Mining in both stages of Ridgetop completed, as well as 118 pit. Area2 phase 1 started with total of 1.2 Mt of plant feed mined in the period. Plant head grade is 1.28% Cu. Total waste tonnage is 10.2 Mt at an average strip ratio of 8.6:1.

2015: Second pushback in Area 2 commenced as phase 1 nears completion. A total of 8.9 mt of waste mined and 1.4 Mt ore at a head grade of 1.42% Cu.

2016: Area 2 stage 1 completed. Final pushback commenced. Mill feed head grade maintained at 1.42% Cu. Strip ratio at 8.7:1 due to stripping of final stage of Area 2 Pit.

2017: Ultimate limits of Area 2 reached with 1.1 Mt of ore mined with a mill head grade of 1.80% Cu and 8.6 Mt of waste produced for a mined strip ratio of 7.6:1.

### 18.4.3 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 18.16 below and their locations are noted on the site plan in the report.

**Table 18.16: 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 18.17 details the various predicted 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. As illustrated by the lack of year end inventories, the higher grade ores are fed to the mill as they are exposed in the pits in order to maintain the ore production at the highest possible head grade while mining. The lower grade stockpiles are depleted gradually once mining is completed in Main Pit (beyond 2011) and used to smooth the mill feed during the initial pre-strip period of the Phase IV pits.

**Table 18.17: Ore Stockpile Balance at Year End**

| Stockpile | Units | Year | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | 2010 | 2011 | 2012 | 2013 | 2014 | 2015 | 2016 | 2017 |
| Stockpile 1 (>0cutoff,<1.0%Cu) | Kt | 865 | 1,082 | 450 | 487 | 301 | 351 | 312 | 88 |
| | % Cu | 0.82 | 0.82 | 0.82 | 0.82 | 0.82 | 0.82 | 0.82 | 0.81 |
| | Au g/t | 0.21 | 0.24 | 0.24 | 0.24 | 0.24 | 0.24 | 0.24 | 0.25 |
| Stockpile 2 (>1.0%Cu,<2.0%Cu) | Kt | 763 | 458 | | | | | | |
| | % Cu | 1.44 | 1.44 | | | | | | |
| | Au g/t | 0.42 | 0.42 | | | | | | |
| Stockpile 3 (>2.0%Cu,<3.0%Cu) | Kt | | | | | | | | |
| | % Cu | | | | | | | | |
| | Au g/t | | | | | | | | |
| Stockpile 4 (>4.0%Cu) | Kt | | | | | | | | |
| | % Cu | | | | | | | | |
| | Au g/t | | | | | | | | |
| **Total Stockpiles** | **Kt** | **1,629** | **1,540** | **450** | **487** | **301** | **351** | **312** | **88** |
| | **% Cu** | **1.11** | **1.00** | **0.82** | **0.82** | **0.82** | **0.82** | **0.82** | **0.81** |
| | **Au g/t** | **0.31** | **0.29** | **0.24** | **0.24** | **0.24** | **0.24** | **0.24** | **0.25** |

## 18.5 Waste Management

### 18.5.1 Waste Rock Dump Designs

#### West Valley Fill Dump

The waste rock material, and low grade oxide, generated from the Main Pit 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 ultimate crest height at the 910 m elevation with 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.

The overburden generated from the mining of the Main Pit will also be placed on the West Valley Fill dump and will be concentrated on the north west portion of the dump in order to allow for the material to be accessible for reclamation purposes.

### Central Valley Fill Dump

The waste rock material generated from the mining of Area 2, 118 and Ridgetop pits will be placed in the Central Valley Fill dump located adjacent to and on top of the West Valley dump. The proposed dump is planned to have an ultimate crest height also at the 910 m elevation with 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.

The majority of the overburden generated from the mining of the pits will also be placed on the upper reaches of the Central Valley Fill dump and will be concentrated on the southern portion of the dump in order to allow for the material to be accessible for reclamation purposes. This will also allow for containment of the material should it be required in terms of geotechnical stability.

### 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 ultimate crest height at the 979 m elevation with 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.

### Backfill Dumps

Both Ridgetop pits as well as the 118 pits will be backfilled with waste generated from subsequent mining. Overburden material will then be placed as a cap on the backfilled pits.

## 18.5.2 Capacities and Sequence

Table 18.18 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 18.18: Waste Quantities by Stage**

| Zone | Overburden | Rock | Total Waste | | | |
|---|---|---|---|---|---|---|
| | (kt) | (kt) | (kt) | (kbcm) | SG (t/m3) | (km3) |
| Main Pit | 8,261.2 | 6,269.0 | 14,530.2 | 6,496.3 | 2.24 | 8,445.1 |
| Subtotal Minto North | 1,744.7 | 8,881.5 | 10,626.2 | 4,162.5 | 2.55 | 5,411.2 |
| Subtotal Area2/118 | 5,237.2 | 30,974.2 | 36,211.4 | 14,287.2 | 2.53 | 18,573.3 |
| Subtotal Ridgetop | 1,637.5 | 7,373.6 | 9,011.1 | 3,617.2 | 2.49 | 4,702.3 |
| **Grand Total Waste** | **8,619.4** | **47,229.3** | **55,848.6** | **22,066.8** | **2.53** | **28,686.9** |

*Note 1.3 swell factor used (m3/bcm)

Table 18.19 below further summarizes, in detail, all waste dumps and ore stockpile tonnages for all stages of this pre-feasibility study.

### Table 18.19: Waste Dump and Stockpile Summary

| Tonnage (kt) | Stage | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Dump/ stockpile | Main | North | Ridgetop South | Ridgetop North | 118 | Area 2 Phase1 | Area 2 Phase2 | Area 2 Phase3 | Grand Total |
| West VF dump | 6,269 | | | | | | | | 6,269 |
| OVB dump | 8,261 | 1,745 | 434 | 1,203 | 197 | 1,996 | 1,552 | 1,492 | 16,881 |
| North dump | | 8,882 | | | | | | | 8,882 |
| Central VF dump | | | 1,792 | 5,582 | 442 | 11,179 | 9,567 | 9,786 | 38,348 |
| Stockpile 1 | 783 | 222 | 71 | 611 | 42 | 624 | 640 | 185 | 3,178 |
| Stockpile 2 | 1,600 | 576 | 106 | 445 | 34 | 615 | 858 | 408 | 4,643 |
| Stockpile 3 | 794 | 323 | 54 | 37 | 12 | 178 | 262 | 204 | 1,866 |
| Stockpile 4 | 75 | 228 | 0 | 0 | 0 | 23 | 8 | 0 | 334 |
| **Grand Total** | **17,783** | **11,975** | **2,457** | **7,878** | **727** | **14,615** | **12,887** | **12,076** | **80,399** |

## 18.6 In-pit Tailings Disposal - Conceptual design

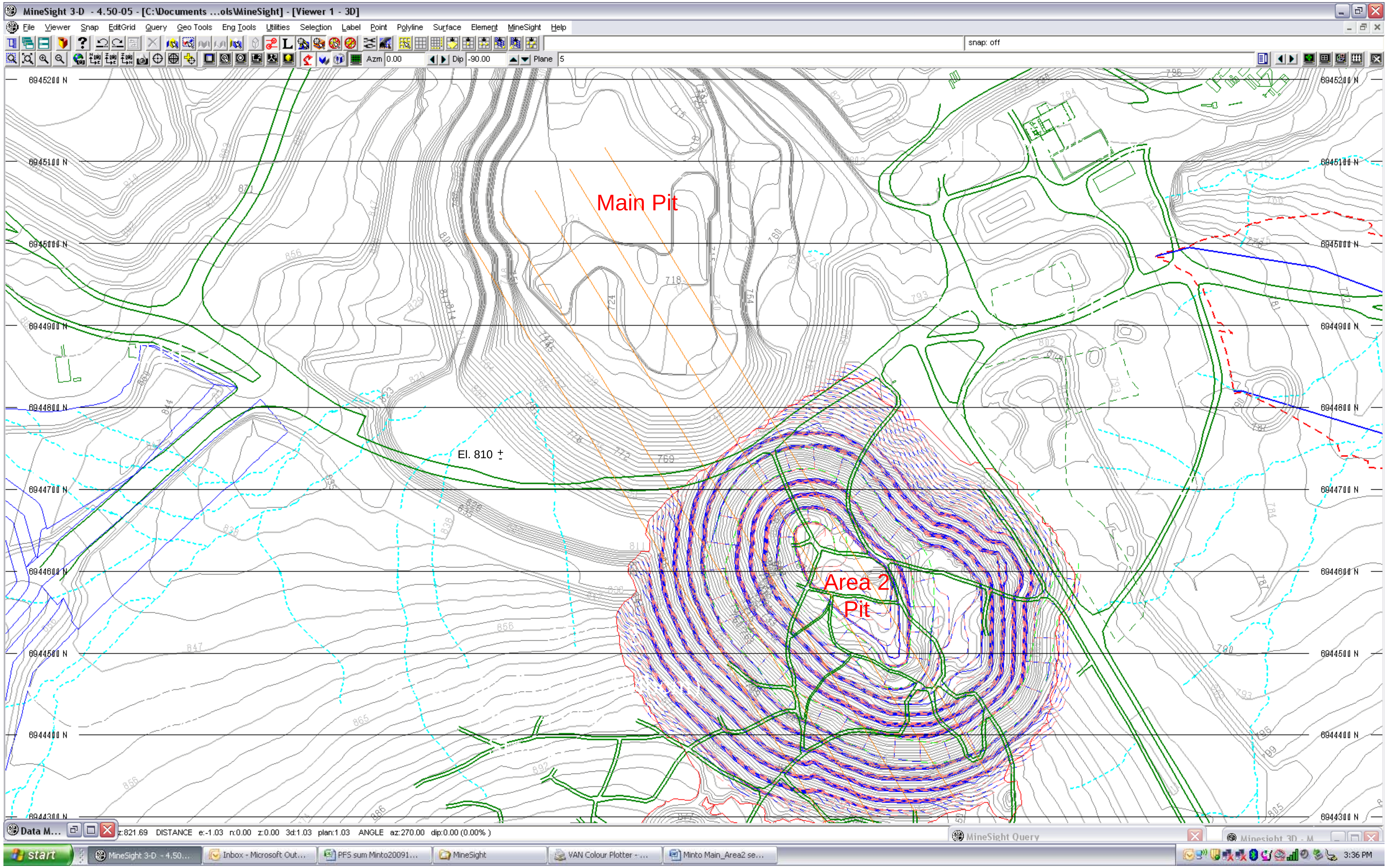
SRK has developed this conceptual design for in-pit sub-aqueous tailings disposal in the Main Pit using a spreadsheet-based tailings solids occupation and surface water balance model based on available topography and climate data. Using an assumed tailings dry density after deposition of 1.12 t/m$^3$ (70 lbs/ft$^3$) and ignoring volume losses for ore concentrate, it was determined that the volume occupation of 6.7 million tonnes of tailings (~6,000,000 m$^3$) would exceed the available storage capacity of the Main Pit (5,000,000 m$^3$) following development of the Area 2 Pit (i.e. requiring tailings disposal higher than the residual ridge crest between the two pits at approximately elevation 766 m amsl). A conceptual divider embankment was designed to increase the storage capacity of the Main Pit, a stage curve was developed based on the revised pit configuration (with divider embankment), and a detailed tailings solids volume occupation and surface water balance model was developed to evaluate the effects of monthly precipitation on in-pit tailings disposal.

An additional 1.0 million tonnes of low-grade ore stockpiled during the mine life will be processed following the completion of mining in the Area 2 Pit. Although it is our understanding that the tailings derived from processing this ore may be deposited in the Area 2 Pit, for the purposes of this conceptual design, it was assumed that all tailings, or a total of 7.7 million tonnes, be deposited in the Main Pit during a mine life of approximately 5.8 years beyond completion of mining in the Minto Main Pit. If the additional 1.0 million tonnes of tailings are in fact deposited in the Area 2 Pit, the final embankment height can be lowered by approximately 6 m, as described in further detail below.

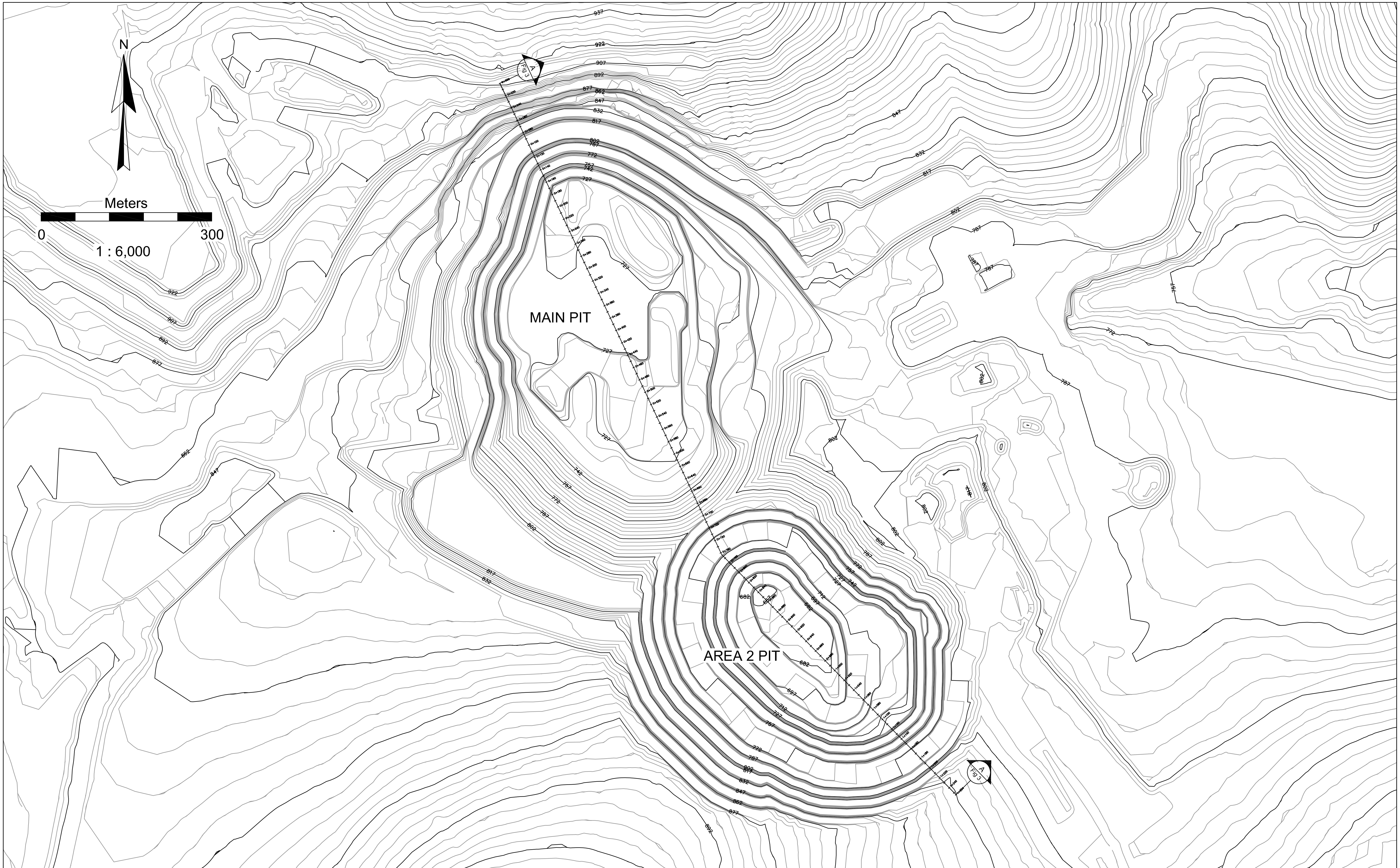
### 18.6.1 Basis of Conceptual Design

The conceptual design described herein is based on available site and project-specific data, including existing site topography, hydrological reports and technical memoranda/spreadsheets, meteorological data, and mine operational data including planned tailings solids deposition rates, typical tailings solids' particle size distribution, slurry characteristics, operational pumpback requirements, regional and local seismicity, and typical freshet runoff volumes.

Existing site topography of the Main Pit and proposed Area 2 Pit was provided by Minto and is depicted on Figure 18.12. This figure shows the Main Pit wall and base configurations and the currently planned Area 2 Pit existing ground surface and planned post-mining pit topography. In addition, the site topography shown on Figure 18.13 (3 m contour intervals) was used to determine stage-area-capacity characteristics for evaluation of conceptual storage options.




| | | |
|---|---|---|
| SRK Consulting<br>Engineers and Scientists<br>VANCOUVER | MINTO COPPER MINE | Site Topography and Existing Road Layout at the Area 2 Pit |
| Job No: 2CM022.06<br>Filename: Fgiure 18.13_RoadLayout_20091208.ppt | Minto Phase IV PSF Technical Report<br>Tailings Disposal Water Management | Date: December 2006 · Approved: · Figure: 18.12 |




| | | | | | |
|---|---|---|---|---|---|
| DESIGN: DB | DRAWN: ML | REVIEWED: - | PREPARED FOR: **MINTO COPPER MINE** | DRAWING TITLE: **SITE TOPOGRAPHY AND SECTION LOCATION** | |
| CHECKED: - | APPROVED: DB | DATE: 12/03/2009 | **MINTO PHASE IV PSF TECHNICAL REPORT TAILINGS DISPOSAL WATER MANAGEMENT** | DRAWING NO. **FIGURE 18.13** | REVISION NO. **A** |
| FILE NAME: Fig18.14-18_Topography_ML_rbb_20091203.dwg | | | | SRK JOB NO. **2CM022.06** | |

J:\01_SITES\Minto\!ACAD\Received 15Dec09\Fig18.14-18_Topography_ML_rbb_20091203.dwg

Climate and hydrological data were obtained from:

- Memorandum CCL-MC1 - Minto Copper Mine – Site Hydrology Update, dated October 6, 2006, Clearwater Consultants Ltd.
- Final Draft Memorandum CCL-MC3 – Minto Copper Project - Water Balance Model, dated March 28, 2008, Clearwater Consultants Ltd
- Memorandum CCL-MC4 - Minto Mine – Water Balance Update, dated July, 2009, Clearwater Consultants Ltd.

Table 18.20 contains meteorological data and the assumptions used for hydrologic calculations, including average monthly precipitation, evaporation, winter precipitation, sublimation as a percentage of winter precipitation, the sublimation rate, and runoff and freshet coefficients utilized.

## TABLE 18.20 Water Balance Spreadsheet for 1.12 t/m$^3$ Dry Density

### Assumptions and Input Parameters

| | | | |
|---|---|---|---|
| Average Deposition Dry Density of Tailings = | 1.12 | tonnes/m$^3$ | |
| Tailings Specific Gravity = | 2.7 | | |
| Process Water Inflow Rate = | 150 | m$^3$/hr | |
| Reclaim Water Outflow Rate = | 150 | m$^3$/hr | |
| Upgradient Catchment Areas = | 4,291,000 | m$^2$ | |
| Upgradient Catchment Runoff Coefficient = | 100% | (reporting to pit) | |
| Pit Sidewall and Pool Surface Runoff Coefficient = | 100% | (reporting to pool) | |
| Pit Base Area = | 966 | m$^2$ (elev. 718 m) | |
| Pit Perimeter Area = | 180,178 | m$^2$ (elev. 800 m) | |
| April Freshet Coefficient = | 70% | | |
| May Freshet Coefficient = | 30% | | |
| Sublimation = | 15% | (as % of winter precipitation total) | |
| Sublimation Rate = | 22.2 | mm per snow season (October-April) | |
| Upgradient Catchment Sublimation = | 95,260 | m$^3$ | |
| Pit Sidewall Sublimation = | 0 | m$^3$ (not currently considered) | |

### Tailings Solids Occupation and Surface Water Balance

| Row | A Year | B Month | C Month # | D Days/Mo. | E Precip. (mm) | F Evap. (mm) | G INFLOWS: Ore Process Rate (tonnes/day) | H Tailings Solids (m$^3$) | I Process Water (m$^3$) | J Precipitation (m$^3$): Upgradient Catchment | K Pit Sidewall | L Pool Surface | M Total Monthly Precip. Inflow | N Total Water Inflows (m$^3$) | O LOSSES/OUTFLOWS: Evaporation from Pool (m$^3$) | P Reclaim Water (m$^3$) | Q Entrainment in Solids (m$^3$) | R Total Outflows (m$^3$) | S INVENTORY: Beginning of Month Solids (m$^3$) | T Beginning of Month Water (m$^3$) | U Change in Storage (Inflows - Outflows) (m$^3$) | V End of the Month Solids (m$^3$) | W Solids (tonnes) | X Solids elev. (m) | Y Water (m$^3$) | Z Water (m$^2$) | AA Water elev. (m) | AB Water Pumped from Pit (m$^3$) | AC Pumping Rate (m$^3$/hr) | AD Water Depth Over Solids |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 14 | 1 | Oct | 1 | 31 | 29.0 | 0 | 3000 | 83,036 | 111,600 | 124,439 | 5,197 | 28 | 28 | 111,628 | 0 | 111,600 | 48,591 | 160,191 | 0 | 242,582 | -48,563 | 83,036 | 93,000 | 728.37 | 194,019 | 59,793 | 732.39 | 0 | 0 | 4 |
| 15 | | Nov | 2 | 30 | 27.0 | 0 | 3000 | 80,357 | 108,000 | 115,857 | 3,250 | 1,614 | 1,614 | 109,614 | 0 | 108,000 | 47,024 | 155,024 | 83,036 | 194,019 | -45,409 | 163,393 | 183,000 | 730.47 | 148,610 | 60,277 | 732.98 | 0 | 0 | 3 |
| 16 | | Dec | 3 | 31 | 23.6 | 0 | 3000 | 83,036 | 111,600 | 101,268 | 2,830 | 1,423 | 1,423 | 113,023 | 0 | 111,600 | 48,591 | 160,191 | 163,393 | 148,610 | -47,169 | 246,429 | 276,000 | 731.87 | 101,441 | 62,073 | 733.53 | 0 | 0 | 2 |
| 17 | 2 | Jan | 4 | 31 | 21.9 | 0 | 3000 | 83,036 | 111,600 | 93,973 | 2,587 | 1,359 | 1,359 | 112,959 | 0 | 111,600 | 48,591 | 160,191 | 246,429 | 101,441 | -47,232 | 329,464 | 369,000 | 733.25 | 54,209 | 63,902 | 734.08 | 0 | 0 | 1 |
| 18 | | Feb | 5 | 28 | 16.4 | 0 | 3000 | 75,000 | 100,800 | 70,372 | 1,907 | 1,048 | 1,048 | 101,848 | 0 | 100,800 | 43,889 | 144,689 | 329,464 | 54,209 | -42,841 | 404,464 | 453,000 | 734.40 | 11,368 | 65,546 | 734.57 | 0 | 0 | 0 |
| 19 | | Mar | 6 | 31 | 13.7 | 0 | 3000 | 83,036 | 111,600 | 58,787 | 1,570 | 898 | 898 | 112,498 | 0 | 111,600 | 48,591 | 160,191 | 404,464 | 11,368 | -47,693 | 487,500 | 546,000 | 735.67 | 0 | 69,208 | 735.67 | 0 | 0 | 0 |
| 20 | | Apr | 7 | 30 | 16.4 | 12 | 3000 | 80,357 | 108,000 | 70,372 | 1,820 | 1,135 | 392,413 | 500,413 | 830 | 108,000 | 47,024 | 155,854 | 487,500 | 0 | 344,559 | 567,857 | 636,000 | 736.82 | 344,559 | 75,325 | 741.53 | 0 | 0 | 5 |
| 21 | | May | 8 | 31 | 24.2 | 83 | 3000 | 83,036 | 111,600 | 103,842 | 2,537 | 1,823 | 275,893 | 387,493 | 6,252 | 111,600 | 48,591 | 166,443 | 567,857 | 344,559 | 221,050 | 650,893 | 729,000 | 737.98 | 565,609 | 79,205 | 745.47 | 0 | 0 | 7 |
| 22 | | Jun | 9 | 30 | 40.0 | 119 | 3000 | 80,357 | 108,000 | 171,640 | 4,039 | 3,168 | 178,847 | 286,847 | 9,425 | 108,000 | 47,024 | 164,449 | 650,893 | 565,609 | 122,398 | 731,250 | 819,000 | 739.10 | 688,007 | 82,466 | 747.99 | 0 | 0 | 9 |
| 23 | | Jul | 10 | 31 | 57.7 | 112 | 3475 | 96,183 | 111,600 | 247,591 | 5,638 | 4,758 | 257,987 | 369,587 | 9,236 | 111,600 | 56,285 | 177,121 | 731,250 | 688,007 | 192,466 | 827,433 | 926,725 | 740.39 | 795,553 | 86,150 | 750.39 | 84,919 | 114 | 10 |
| 24 | | Aug | 11 | 31 | 41.7 | 80 | 3475 | 96,183 | 111,600 | 178,935 | 3,921 | 3,592 | 186,448 | 298,048 | 6,892 | 111,600 | 56,285 | 174,777 | 827,433 | 795,553 | 123,271 | 923,616 | 1,034,450 | 741.68 | 812,562 | 88,425 | 751.68 | 106,262 | 143 | 10 |
| 25 | | Sep | 12 | 30 | 30.1 | 24 | 3475 | 93,080 | 108,000 | 129,159 | 2,762 | 2,662 | 134,582 | 242,582 | 2,122 | 108,000 | 54,469 | 164,591 | 923,616 | 812,562 | 77,991 | 1,016,696 | 1,138,700 | 742.90 | 829,089 | 90,813 | 752.90 | 61,464 | 85 | 10 |
| 26 | | Oct | 13 | 31 | 29.0 | 0 | 3475 | 96,183 | 111,600 | 124,439 | 2,592 | 2,634 | 2,634 | 114,234 | 0 | 111,600 | 56,285 | 167,885 | 1,016,696 | 829,089 | -53,651 | 1,112,879 | 1,246,425 | 744.15 | 775,438 | 91,739 | 753.37 | 0 | 0 | 9 |
| 27 | | Nov | 14 | 30 | 27.0 | 0 | 3475 | 93,080 | 108,000 | 115,857 | 2,388 | 2,477 | 2,477 | 110,477 | 0 | 108,000 | 54,469 | 162,469 | 1,112,879 | 775,438 | -51,992 | 1,205,960 | 1,350,675 | 745.34 | 723,445 | 92,634 | 753.83 | 0 | 0 | 8 |
| 28 | | Dec | 15 | 31 | 23.6 | 0 | 3475 | 96,183 | 111,600 | 101,268 | 2,066 | 2,186 | 2,186 | 113,786 | 0 | 111,600 | 56,285 | 167,885 | 1,205,960 | 723,445 | -54,099 | 1,302,143 | 1,458,400 | 746.53 | 669,347 | 93,345 | 754.28 | 0 | 0 | 8 |
| 29 | 3 | Jan | 16 | 31 | 21.9 | 0 | 3475 | 96,183 | 111,600 | 93,973 | 1,902 | 2,044 | 2,044 | 113,644 | 0 | 111,600 | 56,285 | 167,885 | 1,302,143 | 669,347 | -54,241 | 1,398,326 | 1,566,125 | 747.73 | 615,106 | 93,938 | 754.72 | 0 | 0 | 7 |
| 30 | | Feb | 17 | 28 | 16.4 | 0 | 3475 | 86,875 | 100,800 | 70,372 | 1,414 | 1,541 | 1,541 | 102,341 | 0 | 100,800 | 50,838 | 151,638 | 1,398,326 | 615,106 | -49,297 | 1,485,201 | 1,663,425 | 748.77 | 565,809 | 94,469 | 755.12 | 0 | 0 | 6 |
| 31 | | Mar | 18 | 31 | 13.7 | 0 | 3475 | 96,183 | 111,600 | 58,787 | 1,174 | 1,294 | 1,294 | 112,894 | 0 | 111,600 | 56,285 | 167,885 | 1,485,201 | 565,809 | -54,991 | 1,581,384 | 1,771,150 | 749.90 | 510,818 | 95,051 | 755.55 | 0 | 0 | 6 |
| 32 | | Apr | 19 | 30 | 16.4 | 12 | 3750 | 100,446 | 108,000 | 70,372 | 1,396 | 1,559 | 388,477 | 496,477 | 1,141 | 108,000 | 58,780 | 167,920 | 1,581,384 | 510,818 | 328,556 | 1,681,830 | 1,883,650 | 751.08 | 839,374 | 107,874 | 759.84 | 0 | 0 | 8.8 |
| 33 | | May | 20 | 31 | 24.2 | 83 | 3750 | 103,795 | 111,600 | 103,842 | 1,750 | 2,611 | 274,024 | 385,624 | 8,954 | 111,600 | 60,739 | 181,293 | 1,681,830 | 839,374 | 204,332 | 1,785,625 | 1,999,900 | 752.23 | 998,691 | 111,277 | 762.23 | 45,015 | 61 | 10 |
| 34 | | Jun | 21 | 30 | 40.0 | 119 | 3750 | 100,446 | 108,000 | 171,640 | 2,756 | 4,451 | 178,847 | 286,847 | 13,242 | 108,000 | 58,780 | 180,022 | 1,785,625 | 998,691 | 106,825 | 1,886,071 | 2,112,400 | 753.35 | 1,022,697 | 112,673 | 763.35 | 82,819 | 115 | 10 |
| 35 | | Jul | 22 | 31 | 57.7 | 112 | 3750 | 103,795 | 111,600 | 247,591 | 3,895 | 6,501 | 257,987 | 369,587 | 12,619 | 111,600 | 60,739 | 184,959 | 1,886,071 | 1,022,697 | 184,628 | 1,989,866 | 2,228,650 | 754.48 | 1,047,497 | 114,097 | 764.48 | 159,829 | 215 | 10 |
| 36 | | Aug | 23 | 31 | 41.7 | 80 | 3750 | 103,795 | 111,600 | 178,935 | 2,756 | 4,758 | 186,448 | 298,048 | 9,128 | 111,600 | 60,739 | 181,467 | 1,989,866 | 1,047,497 | 116,581 | 2,093,661 | 2,344,900 | 755.57 | 1,068,422 | 115,478 | 765.57 | 95,656 | 129 | 10 |
| 37 | | Sep | 24 | 30 | 30.1 | 24 | 3750 | 100,446 | 108,000 | 129,159 | 1,947 | 3,476 | 134,582 | 242,582 | 2,771 | 108,000 | 58,780 | 169,551 | 2,093,661 | 1,068,422 | 73,031 | 2,194,107 | 2,457,400 | 756.63 | 1,091,046 | 116,813 | 766.63 | 50,408 | 70 | 10 |
| 38 | | Oct | 25 | 31 | 29.0 | 0 | 3750 | 103,795 | 111,600 | 124,439 | 1,838 | 3,388 | 3,388 | 114,988 | 0 | 111,600 | 60,739 | 172,339 | 2,194,107 | 1,091,046 | -57,352 | 2,297,902 | 2,573,650 | 757.66 | 1,033,694 | 117,311 | 767.02 | 0 | 0 | 9 |
| 39 | | Nov | 26 | 30 | 27.0 | 0 | 3750 | 100,446 | 108,000 | 115,857 | 1,697 | 3,167 | 3,167 | 111,167 | 0 | 108,000 | 58,780 | 166,780 | 2,297,902 | 1,033,694 | -55,612 | 2,398,348 | 2,686,150 | 758.64 | 978,082 | 117,791 | 767.40 | 0 | 0 | 9 |
| 40 | | Dec | 27 | 31 | 23.6 | 0 | 3750 | 103,795 | 111,600 | 101,268 | 1,472 | 2,780 | 2,780 | 114,380 | 0 | 111,600 | 60,739 | 172,339 | 2,398,348 | 978,082 | -57,959 | 2,502,143 | 2,802,400 | 759.65 | 920,123 | 118,282 | 767.79 | 0 | 0 | 8 |
| 41 | 4 | Jan | 28 | 31 | 21.9 | 0 | 3750 | 103,795 | 111,600 | 93,973 | 1,356 | 2,590 | 2,590 | 114,190 | 0 | 111,600 | 60,739 | 172,339 | 2,502,143 | 920,123 | -58,149 | 2,605,938 | 2,918,650 | 760.62 | 861,974 | 118,770 | 768.18 | 0 | 0 | 8 |
| 42 | | Feb | 29 | 29 | 16.4 | 0 | 3750 | 97,098 | 104,400 | 70,372 | 1,007 | 1,948 | 1,948 | 106,348 | 0 | 104,400 | 56,820 | 161,220 | 2,605,938 | 861,974 | -54,873 | 2,703,036 | 3,027,400 | 761.50 | 807,101 | 119,223 | 768.53 | 0 | 0 | 7 |
| 43 | | Mar | 30 | 31 | 13.7 | 0 | 3750 | 103,795 | 111,600 | 58,787 | 835 | 1,633 | 1,633 | 113,233 | 0 | 111,600 | 60,739 | 172,339 | 2,703,036 | 807,101 | -59,106 | 2,806,830 | 3,143,650 | 762.44 | 747,996 | 119,701 | 768.91 | 0 | 0 | 6 |
| 44 | | Apr | 31 | 30 | 16.4 | 12 | 3750 | 100,446 | 108,000 | 70,372 | 992 | 1,963 | 386,266 | 494,266 | 1,436 | 108,000 | 58,780 | 168,216 | 2,806,830 | 747,996 | 326,050 | 2,907,277 | 3,256,150 | 763.34 | 1,074,046 | 125,261 | 772.39 | 0 | 0 | 9 |
| 45 | | May | 32 | 31 | 24.2 | 83 | 3750 | 103,795 | 111,600 | 103,842 | 1,329 | 3,031 | 272,904 | 384,504 | 10,397 | 111,600 | 60,739 | 182,736 | 2,907,277 | 1,074,046 | 201,768 | 3,011,071 | 3,372,400 | 764.25 | 1,205,966 | 126,276 | 774.25 | 69,848 | 94 | 10 |
| 46 | | Jun | 33 | 30 | 40.0 | 119 | 3750 | 100,446 | 108,000 | 171,640 | 2,076 | 5,131 | 178,847 | 286,847 | 15,265 | 108,000 | 58,780 | 182,045 | 3,011,071 | 1,205,966 | 104,802 | 3,111,518 | 3,484,900 | 765.13 | 1,217,936 | 129,687 | 775.13 | 92,832 | 129 | 10 |
| 47 | | Jul | 34 | 31 | 57.7 | 112 | 3750 | 103,795 | 111,600 | 247,591 | 2,913 | 7,483 | 257,987 | 369,587 | 14,525 | 111,600 | 60,739 | 186,864 | 3,111,518 | 1,217,936 | 182,723 | 3,215,313 | 3,601,150 | 766.03 | 1,233,778 | 131,024 | 776.03 | 166,881 | 224 | 10 |
| 48 | | Aug | 35 | 31 | 41.7 | 80 | 3750 | 103,795 | 111,600 | 178,935 | 2,050 | 5,464 | 186,448 | 298,048 | 10,482 | 111,600 | 60,739 | 182,821 | 3,215,313 | 1,233,778 | 115,227 | 3,319,107 | 3,717,400 | 766.91 | 1,245,919 | 132,319 | 776.91 | 103,086 | 139 | 10 |
| 49 | | Sep | 36 | 30 | 30.1 | 24 | 3750 | 100,446 | 108,000 | 129,159 | 1,441 | 3,983 | 134,582 | 242,582 | 3,176 | 108,000 | 58,780 | 169,955 | 3,319,107 | 1,245,919 | 72,627 | 3,419,554 | 3,829,900 | 767.77 | 1,257,668 | 133,572 | 777.77 | 60,878 | 85 | 10 |
| 50 | | Oct | 37 | 31 | 29.0 | 0 | 3750 | 103,795 | 111,600 | 124,439 | 1,352 | 3,874 | 3,874 | 115,474 | 0 | 111,600 | 60,739 | 172,339 | 3,419,554 | 1,257,668 | -56,865 | 3,523,348 | 3,946,150 | 768.65 | 1,200,803 | 134,202 | 778.12 | 0 | 0 | 9 |
| 51 | | Nov | 38 | 30 | 27.0 | 0 | 3750 | 100,446 | 108,000 | 115,857 | 1,241 | 3,623 | 3,623 | 111,623 | 0 | 108,000 | 58,780 | 166,780 | 3,523,348 | 1,200,803 | -55,156 | 3,623,795 | 4,058,650 | 769.48 | 1,145,646 | 135,004 | 778.45 | 0 | 0 | 9 |
| 52 | | Dec | 39 | 31 | 23.6 | 0 | 3750 | 103,795 | 111,600 | 101,268 | 1,066 | 3,186 | 3,186 | 114,786 | 0 | 111,600 | 60,739 | 172,339 | 3,623,795 | 1,145,646 | -57,553 | 3,727,589 | 4,174,900 | 770.33 | 1,088,093 | 135,823 | 778.78 | 0 | 0 | 8 |
| 53 | 5 | Jan | 40 | 31 | 21.9 | 0 | 3750 | 103,795 | 111,600 | 93,973 | 971 | 2,975 | 2,975 | 114,575 | 0 | 111,600 | 60,739 | 172,339 | 3,727,589 | 1,088,093 | -57,765 | 3,831,384 | 4,291,150 | 771.18 | 1,030,329 | 136,639 | 779.12 | 0 | 0 | 8 |
| 54 | | Feb | 41 | 28 | 16.4 | 0 | 3750 | 93,750 | 100,800 | 70,372 | 714 | 2,241 | 2,241 | 103,041 | 0 | 100,800 | 54,861 | 155,661 | 3,831,384 | 1,030,329 | -52,620 | 3,925,134 | 4,396,150 | 771.95 | 977,709 | 137,367 | 779.42 | 0 | 0 | 7 |
| 55 | | Mar | 42 | 31 | 13.7 | 0 | 3750 | 103,795 | 111,600 | 58,787 | 587 | 1,882 | 1,882 | 113,482 | 0 | 111,600 | 60,739 | 172,339 | 3,925,134 | 977,709 | -58,857 | 4,028,929 | 4,512,400 | 772.77 | 918,851 | 138,164 | 779.74 | 0 | 0 | 7 |
| 56 | | Apr | 43 | 30 | 16.4 | 12 | 3750 | 100,446 | 108,000 | 70,372 | 689 | 2,266 | 384,765 | 492,765 | 1,658 | 108,000 | 58,780 | 168,438 | 4,028,929 | 918,851 | 324,328 | 4,129,375 | 4,624,900 | 773.56 | 1,243,179 | 148,843 | 782.70 | 0 | 0 | 9 |
| 57 | | May | 44 | 31 | 24.2 | 83 | 3750 | 103,795 | 111,600 | 103,842 | 758 | 3,602 | 272,131 | 383,731 | 12,354 | 111,600 | 60,739 | 184,693 | 4,129,375 | 1,243,179 | 199,038 | 4,233,170 | 4,741,150 | 774.37 | 1,389,893 | 155,245 | 784.37 | 52,324 | 70 | 10 |
| 58 | | Jun | 45 | 30 | 40.0 | 119 | 3750 | 100,446 | 108,000 | 171,640 | 997 | 6,210 | 178,847 | 286,847 | 18,474 | 108,000 | 58,780 | 185,254 | 4,233,170 | 1,389,893 | 101,593 | 4,333,616 | 4,853,650 | 775.16 | 1,412,691 | 156,498 | 785.16 | 78,795 | 109 | 10 |
| 59 | | Jul | 46 | 31 | 57.7 | 112 | 3750 | 103,795 | 111,600 | 247,591 | 1,366 | 9,030 | 257,987 | 369,587 | 17,528 | 111,600 | 60,739 | 189,867 | 4,333,616 | 1,412,691 | 179,720 | 4,437,411 | 4,969,900 | 775.95 | 1,432,658 | 157,757 | 785.95 | 159,753 | 215 | 10 |
| 60 | | Aug | 47 | 31 | 41.7 | 80 | 3750 | 103,795 | 111,600 | 178,935 | 935 | 6,578 | 186,448 | 298,048 | 12,621 | 111,600 | 60,739 | 184,960 | 4,437,411 | 1,432,658 | 113,088 | 4,541,205 | 5,086,150 | 776.73 | 1,452,624 | 159,016 | 786.73 | 93,122 | 125 | 10 |
| 61 | | Sep | 48 | 30 | 30.1 | 24 | 3750 | 100,446 | 108,000 | 129,159 | 637 | 4,786 | 134,582 | 242,582 | 3,816 | 108,000 | 58,780 | 170,596 | 4,541,205 | 1,452,624 | 71,986 | 4,641,652 | 5,198,650 | 777.50 | 1,474,308 | 160,224 | 787.50 | 50,303 | 70 | 10 |
| 62 | | Oct | 49 | 31 | 29.0 | 0 | 3750 | 103,795 | 111,600 | 124,439 | 579 | 4,646 | 4,646 | 116,246 | 0 | 111,600 | 60,739 | 172,339 | 4,641,652 | 1,474,308 | -56,093 | 4,745,446 | 5,314,900 | 778.27 | 1,418,215 | 160,688 | 787.79 | 0 | 0 | 10 |
| 63 | | Nov | 50 | 30 | 27.0 | 0 | 3750 | 100,446 | 108,000 | 115,857 | 526 | 4,339 | 4,339 | 112,339 | 0 | 108,000 | 58,780 | 166,780 | 4,745,446 | 1,418,215 | -54,441 | 4,845,893 | 5,427,400 | 779.00 | 1,363,774 | 161,136 | 788.08 | 0 | 0 | 9 |
| 64 | | Dec | 51 | 31 | 23.6 | 0 | 3750 | 103,795 | 111,600 | 101,268 | 449 | 3,803 | 3,803 | 115,403 | 0 | 111,600 | 60,739 | 172,339 | 4,845,893 | 1,363,774 | -56,936 | 4,949,688 | 5,543,650 | 779.76 | 1,306,838 | 161,593 | 788.36 | 0 | 0 | 9 |
| 65 | 6 | Jan | 52 | 31 | 21.9 | 0 | 3750 | 103,795 | 111,600 | 93,973 | 407 | 3,539 | 3,539 | 115,139 | 0 | 111,600 | 60,739 | 172,339 | 4,949,688 | 1,306,838 | -57,200 | 5,053,482 | 5,659,900 | 780.51 | 1,249,638 | 162,046 | 788.65 | 0 | 0 | 8 |
| 66 | | Feb | 53 | 28 | 16.4 | 0 | 3750 | 93,750 | 100,800 | 70,372 | 297 | 2,658 | 2,658 | 103,458 | 0 | 100,800 | 54,861 | 155,661 | 5,053,482 | 1,249,638 | -52,204 | 5,147,232 | 5,764,900 | 781.18 | 1,197,434 | 162,451 | 788.91 | 0 | 0 | 8 |
| 67 | | Mar | 54 | 31 | 13.7 | 0 | 3750 | 103,795 | 111,600 | 58,787 | 243 | 2,226 | 2,226 | 113,826 | 0 | 111,600 | 60,739 | 172,339 | 5,147,232 | 1,197,434 | -58,514 | 5,251,027 | 5,881,150 | 781.88 | 1,138,921 | 162,892 | 789.19 | 0 | 0 | 7 |
| 68 | | Apr | 55 | 30 | 16.4 | 12 | 3750 | 100,446 | 108,000 | 70,372 | 283 | 2,671 | 382,486 | 490,486 | 1,955 | 108,000 | 58,780 | 168,734 | 5,251,027 | 1,138,921 | 321,752 | 5,351,473 | 5,993,650 | 782.56 | 1,460,673 | 167,081 | 791.75 | 0 | 0 | 9 |
| 69 | | May | 56 | 31 | 24.2 | 83 | 3750 | 103,795 | 111,600 | 103,842 | 317 | 4,043 | 270,980 | 382,580 | 13,868 | 111,600 | 60,739 | 186,207 | 5,351,473 | 1,460,673 | 196,374 | 5,455,268 | 6,109,900 | 783.26 | 1,610,696 | 169,601 | 793.26 | 46,350 | 62 | 10 |
| 70 | | Jun | 57 | 30 | 40.0 | 119 | 3750 | 100,446 | 108,000 | 171,640 | 423 | 6,784 | 178,847 | 286,847 | 20,183 | 108,000 | 58,780 | 186,962 | 5,455,268 | 1,610,696 | 99,885 | 5,555,714 | 6,222,400 | 783.94 | 1,626,786 | 170,716 | 793.94 | 83,795 | 116 | 10 |
| 71 | | Jul | 58 | 31 | 57.7 | 112 | 3750 | 103,795 | 111,600 | 247,591 | 546 | 9,850 | 257,987 | 369,587 | 19,120 | 111,600 | 60,739 | 191,459 | 5,555,714 | 1,626,786 | 178,128 | 5,659,509 | 6,338,650 | 784.60 | 1,637,024 | 171,808 | 794.60 | 167,890 | 226 | 10 |

# TABLE 18.20 Water Balance Spreadsheet for 1.12 t/m$^3$ Dry Density

## Assumptions and Input Parameters

| | | | | | |
|---|---|---|---|---|---|
| Average Deposition Dry Density of Tailings = | 1.12 tonnes/m$^3$ | Upgradient Catchment Areas = | 4,291,000 m$^2$ | Pit Base Area = | 966 m$^2$ (elev. 718 m) |
| Tailings Specific Gravity = | 2.7 | Upgradient Catchment Runoff Coefficient = | 100% (reporting to pit) | Pit Perimeter Area = | 180,178 m$^2$ (elev. 800 m) |
| Process Water Inflow Rate = | 150 m$^3$/hr | Pit Sidewall and Pool Surface Runoff Coefficient = | 100% (reporting to pool) | April Freshet Coefficient = | 70% |
| Reclaim Water Outflow Rate = | 150 m$^3$/hr | | | May Freshet Coefficient = | 30% |

| | |
|---|---|
| Sublimation = | 15% (as % of winter precipitation total) |
| Sublimation Rate = | 22.2 mm per snow season (October-April) |
| Upgradient Catchment Sublimation = | 95,260 m$^3$ |
| Pit Sidewall Sublimation = | 0 m$^3$ (not currently considered) |

## Tailings Solids Occupation and Surface Water Balance

| Row | Year | Month | Month # | Days/Mo. | Precip. (mm) | Evap. (mm) | INFLOWS: Ore Process Rate (tonnes/day) | INFLOWS: Tailings Solids (m$^3$) | INFLOWS: Process Water (m$^3$) | Precipitation (m$^3$): Upgradient Catchment | Precipitation (m$^3$): Pit Sidewall | Precipitation (m$^3$): Pool Surface | Precipitation (m$^3$): Total Monthly Precip. Inflow | Total Water Inflows (m$^3$) | LOSSES/OUTFLOWS: Evaporation from Pool (m$^3$) | LOSSES/OUTFLOWS: Reclaim Water (m$^3$) | LOSSES/OUTFLOWS: Entrainment in Solids (m$^3$) | Total Outflows (m$^3$) | INVENTORY: Beginning of Month Solids (m$^3$) | INVENTORY: Beginning of Month Water (m$^3$) | Change in Storage (Inflows - Outflows) (m$^3$) | End of the Month Solids (m$^3$) | End of the Month Solids (tonnes) | End of the Month Solids elev. (m) | End of the Month Water (m$^3$) | End of the Month Water (m$^2$) | End of the Month Water elev. (m) | Water Pumped from Pit (m$^3$) | Pumping Rate (m$^3$/hr) | Water Depth Over Solids |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 72 | | Aug | 59 | 31 | 41.7 | 80 | 3750 | 103,795 | 111,600 | 178,935 | 349 | 7,164 | 186,448 | 298,048 | 13,745 | 111,600 | 60,739 | 186,084 | 5,659,509 | 1,637,024 | 111,964 | 5,763,304 | 6,454,900 | 785.27 | 1,646,667 | 172,893 | 795.27 | 102,322 | 138 | 10 |
| 73 | | Sep | 60 | 30 | 30.1 | 24 | 3750 | 100,446 | 108,000 | 129,159 | 219 | 5,204 | 134,582 | 242,582 | 4,149 | 108,000 | 58,780 | 170,929 | 5,763,304 | 1,646,667 | 71,653 | 5,863,750 | 6,567,400 | 785.91 | 1,655,999 | 173,944 | 795.91 | 62,322 | 87 | 10 |
| 74 | | Oct | 61 | 31 | 29.0 | 0 | 3750 | 103,795 | 111,600 | 124,439 | 181 | 5,044 | 5,044 | 116,644 | 0 | 111,600 | 60,739 | 172,339 | 5,863,750 | 1,655,999 | -55,695 | 5,967,545 | 6,683,650 | 786.57 | 1,600,304 | 174,424 | 796.18 | 0 | 0 | 10 |
| 75 | | Nov | 62 | 30 | 27.0 | 0 | 3750 | 100,446 | 108,000 | 115,857 | 155 | 4,709 | 4,709 | 112,709 | 0 | 108,000 | 58,780 | 166,780 | 5,967,545 | 1,600,304 | -54,070 | 6,067,991 | 6,796,150 | 787.20 | 1,546,234 | 174,896 | 796.44 | 0 | 0 | 9 |
| 76 | | Dec | 63 | 31 | 23.6 | 0 | 3750 | 103,795 | 111,600 | 101,268 | 125 | 4,128 | 4,128 | 115,728 | 0 | 111,600 | 60,739 | 172,339 | 6,067,991 | 1,546,234 | -56,612 | 6,171,786 | 6,912,400 | 787.84 | 1,489,622 | 175,376 | 796.71 | 0 | 0 | 9 |
| 77 | 7 | Jan | 64 | 31 | 21.9 | 0 | 3750 | 103,795 | 111,600 | 93,973 | 105 | 3,841 | 3,841 | 115,441 | 0 | 111,600 | 60,739 | 172,339 | 6,171,786 | 1,489,622 | -56,898 | 6,275,580 | 7,028,650 | 788.48 | 1,432,724 | 175,854 | 796.97 | 0 | 0 | 8 |
| 78 | | Feb | 65 | 28 | 16.4 | 0 | 3750 | 93,750 | 100,800 | 70,372 | 71 | 2,884 | 2,884 | 103,684 | 0 | 100,800 | 54,861 | 155,661 | 6,275,580 | 1,432,724 | -51,977 | 6,369,330 | 7,133,650 | 789.06 | 1,380,747 | 176,279 | 797.21 | 0 | 0 | 8 |
| 79 | | Mar | 66 | 31 | 13.7 | 0 | 3750 | 103,795 | 111,600 | 58,787 | 53 | 2,415 | 2,415 | 114,015 | 0 | 111,600 | 60,739 | 172,339 | 6,369,330 | 1,380,747 | -58,324 | 6,473,125 | 7,249,900 | 789.70 | 1,322,423 | 176,742 | 797.47 | 0 | 0 | 8 |
| 80 | | Apr | 67 | 30 | 16.4 | 12 | 3750 | 100,446 | 108,000 | 70,372 | 56 | 2,899 | 381,287 | 489,287 | 2,121 | 108,000 | 58,780 | 168,901 | 6,473,125 | 1,322,423 | 320,386 | 6,573,571 | 7,362,400 | 790.31 | 1,642,809 | 180,063 | 799.83 | 0 | 0 | 10 |
| 81 | | May | 68 | 31 | 24.2 | 83 | 3750 | 103,795 | 111,600 | 103,842 | 3 | 4,358 | 270,369 | 381,969 | 14,945 | 111,600 | 60,739 | 187,284 | 6,573,571 | 1,642,809 | 194,685 | 6,677,366 | 7,478,650 | 790.94 | 1,738,817 | 180,812 | 800.94 | 98,676 | 133 | 10 |
| 82 | | Jun | 69 | 30 | 40.0 | 119 | 3750 | 100,446 | 108,000 | 171,640 | -25 | 7,232 | 178,847 | 286,847 | 21,517 | 108,000 | 58,780 | 188,296 | 6,677,366 | 1,738,817 | 98,551 | 6,777,812 | 7,591,150 | 791.54 | 1,747,161 | 181,220 | 801.54 | 90,207 | 125 | 10 |
| 83 | | Jul | 70 | 31 | 57.7 | 112 | 3750 | 103,795 | 111,600 | 247,591 | -60 | 10,456 | 257,987 | 369,587 | 20,297 | 111,600 | 60,739 | 192,636 | 6,777,812 | 1,747,161 | 176,951 | 6,881,607 | 7,707,400 | 792.16 | 1,756,772 | 182,081 | 802.16 | 167,340 | 225 | 10 |
| 84 | | Aug | 71 | 31 | 41.7 | 80 | 3750 | 103,795 | 111,600 | 178,935 | -79 | 7,593 | 186,448 | 298,048 | 14,566 | 111,600 | 60,739 | 186,906 | 6,881,607 | 1,756,772 | 111,143 | 6,985,402 | 7,823,650 | 792.79 | 1,769,175 | 184,179 | 802.79 | 98,740 | 133 | 10 |
| 85 | | Sep | 72 | 30 | 30.1 | 24 | 3750 | 100,446 | 108,000 | 129,159 | -120 | 5,544 | 134,582 | 242,582 | 4,420 | 108,000 | 58,780 | 171,200 | 6,985,402 | 1,769,175 | 71,382 | 7,085,848 | 7,936,150 | 793.38 | 1,779,082 | 186,172 | 803.38 | 61,476 | 85 | 10 |

Notes:
1) Meterological data from Clear Water Consultants Ltd. memo CCL-MC1, October 2006.
2) Sublimation rate based on Hood et al. (1999); Pomeroy et al. (1996) and Zhang et al. (2004)
3) Freshet coefficient from Clearwater Consultants Ltd. Memo CCL-MC4, July 2009.

### 18.6.2 Site Geology, Hydrogeology and Geotechnical Conditions

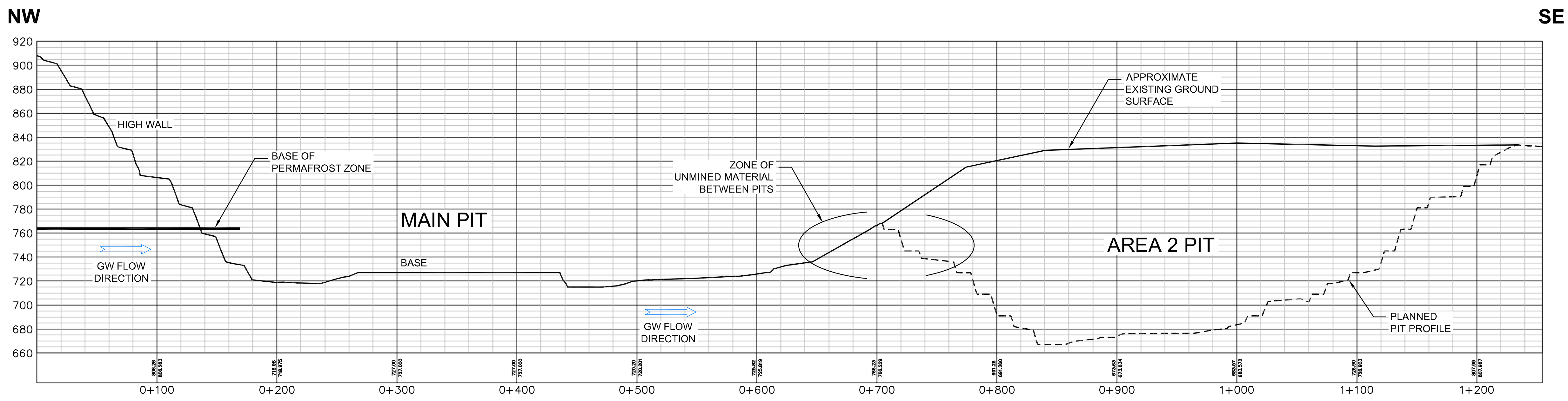
Site geology is described in Section 6. For the purposes of this conceptual design, it was assumed that the groundwater regime between and around the existing and proposed pit configurations can be controlled via embankment cut-off elements, tailings management (e.g. pre-sliming the base of the pit to decrease the bedrock permeability) and dewatering via wells installed in or downstream of the embankment. It was also assumed that during the freshet and the summer months (April through September), the ground surface is saturated and all precipitation falling as rain runs off.

Information regarding the location and extent of the Main Pit high wall soils and rock is provided in Section 18.2.1 and shown in cross-section on Figure 18.14. Information regarding the geotechnical characteristics of these materials is summarized in the following two SRK documents:

- Pit Slope Evaluation for Minto Mine Main Pit (July 2007); and,
- Pit Slope Evaluation for Minto Mine Main Pit South Wall (December 2007).

The existing tailings gradation is shown on Figure 18.15 within an envelope of typical copper tailings grinds (S.G. Vick, 1970), and falls within the coarser side of the envelope. The planned tailings are anticipated to have a nominally coarser grind (around a $P_{80}$ of 250 microns) than that of the existing tailings, and are assumed to classify as a silty, fine to medium sand according to the Unified Soil Classification System. With this gradation, the tailings would be amenable to effective cycloning techniques, if required.

Based on experience at other similar copper tailing projects, the average dry density of the deposited tailings is estimated to be 1.12 t/m$^3$ (70 pcf) for the anticipated sub-aqueous deposition conditions.




**SECTION A-A**

SCALE 1 : 4000

| | | | | | |
|---|---|---|---|---|---|
| DESIGN: DB | DRAWN: ML | REVIEWED: - | PREPARED FOR: **MINTO COPPER MINE** | DRAWING TITLE: **HYDROGEOLOGIC AND GEOTECHNICAL CROSS-SECTION** | |
| CHECKED: - | APPROVED: DB | DATE: 12/03/2009 | **MINTO PHASE IV PSF TECHNICAL REPORT TAILINGS DISPOSAL WATER MANAGEMENT** | DRAWING NO. **FIGURE 18.14** | REVISION NO. **A** |
| FILE NAME: Fig18.14-18_Topography_ML_rbb_20091203.dwg | | | | SRK JOB NO. **2CM022.06** | |

J:\01_SITES\Minto\!ACAD\Received 15Dec09\Fig18.14-18_Topography_ML_rbb_20091203.dwg

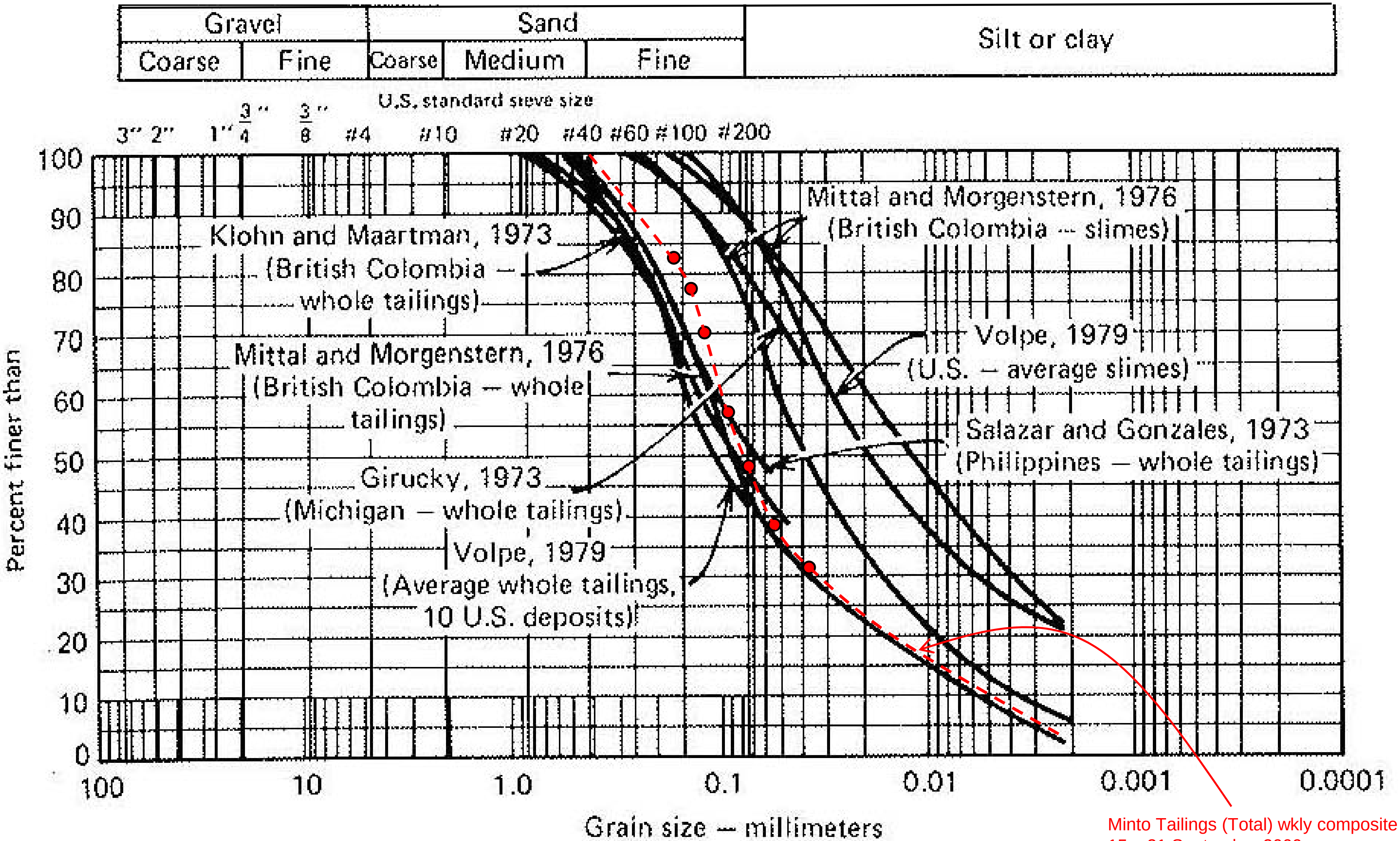



**Figure 1.13 Gradations of copper tailings.**

*From S. G. Vick, 1970*

| | | |
|---|---|---|
| SRK Consulting<br>Engineers and Scientists<br>VANCOUVER | MINTO COPPER MINE | **Gradations of CopperTailings** |
| Job No: 2CM022.066<br>Filename: Figure 18.13_CUTialilngs_200912108.ppt | Minto Phase IV Technical Report<br>Tailings Disposal Water Management | Date: December 2009 / Approved: / Figure: **18.15** |

### 18.6.3 Conceptual Design Criteria

The following conceptual design criteria apply:

- The facility must accommodate up to 7.7 million tonnes of mill tailings solids (allowing for 6.7 Mt of from milled reserves plus 1.0 Mt for ore in the existing low grade stockpile). The forecast daily and annual tailings rates are provided in Column G of Table 18.20.
- The design must provide a means of storing/managing the annual freshet volume, which is typically on the order of 700,000 cubic meters.
- A minimum of 2.0 m of dry freeboard must be provided to accommodate potential wave action (1.5 m) and winter ice formation (2 m);
- All required embankment construction will be performed using waste rock and/or overburden from pre-mining development of Area 2 Pit (i.e., at this time, no cycloning of tailings for embankment construction is envisioned);
- The design must allow for maintenance of a minimum of 150,000 $m^3$ of plant operating water inventory available on the impoundment, additional to dead storage for barge operation;
- Water will be recycled from the tailings impoundment back to the mill facilities using a barge-mounted pump and, during summer, this will require a minimum operating depth (dead storage) of about 2 meters to provide appropriate intake conditions that enable minimization of the potential for pumping suspended solids;
- The tailings thickener design underflow solid:water ratio ranges from 50:50 to 55:45 with a ratio of around 51:49 being used in the tailings solids occupation and surface water balance. At the full design rate of 3,750 tonnes per day, this represents approximately 150 $m^3$/hr of water pumped in the slurry to the impoundment, which is the target recycle pump rate; and
- The design must include measures to minimize contribution of both spillage and seepage of tailings impoundment waters into the Area 2 Pit.

### 18.6.4 Conceptual Design

The SRK design is based in part on the spreadsheet-based tailings solids occupation and surface water balance model developed to evaluate and quantify the potential storage characteristics of the Main Pit for tailings deposition and water management (refer to Table 18.20). The spreadsheet shows the anticipated impoundment behavior on a monthly basis, and provides data on both monthly solids (tailings) and water surface elevations, and a basis for determining critical elevations and balancing pump, treat and discharge requirements.

The conceptual design includes a constructed divider embankment composed of waste rock and/or overburden generated during Area 2 Pit development.

The embankment is configured with a 40 m wide crest and 2H:1V sideslopes for stability. The crest elevation was set at 805 m amsl with an assumed 2.5 m deep spillway.

With this configuration, the embankment will require approximately 2.1 million cubic meters of waste rock to construct. The water management strategy facilitates water treatment, implementation of phased construction and design and implementation of sub-aqueous deposition methods. The following figures illustrate the conceptual design.

- Figure 18.16 - (based on 7.7 Mt) provides the layout of tailings disposal facility elements included in the selected conceptual design and the location of these elements relative to the Main Pit topography and proposed Area 2 Pit, and includes:
  - main and saddle embankments;
  - seepage collection and recycling facilities;
  - slurry transport and deposition pipeline location(s);
  - recycling pump and barge location(s).
- Figure 18.17 - provides a cross-section through the conceptual tailings disposal facility elements.
- Figure 18.18 - provides elevation-area-storage relationships for the layout in Figure 18.17 for average dry densities of 1.12 t/m$^3$ and 1.25 t/m$^3$, with the latter considered an optimistic upper bound value. The key elevations for the 6.7 Mt scenario are also included on this figure.

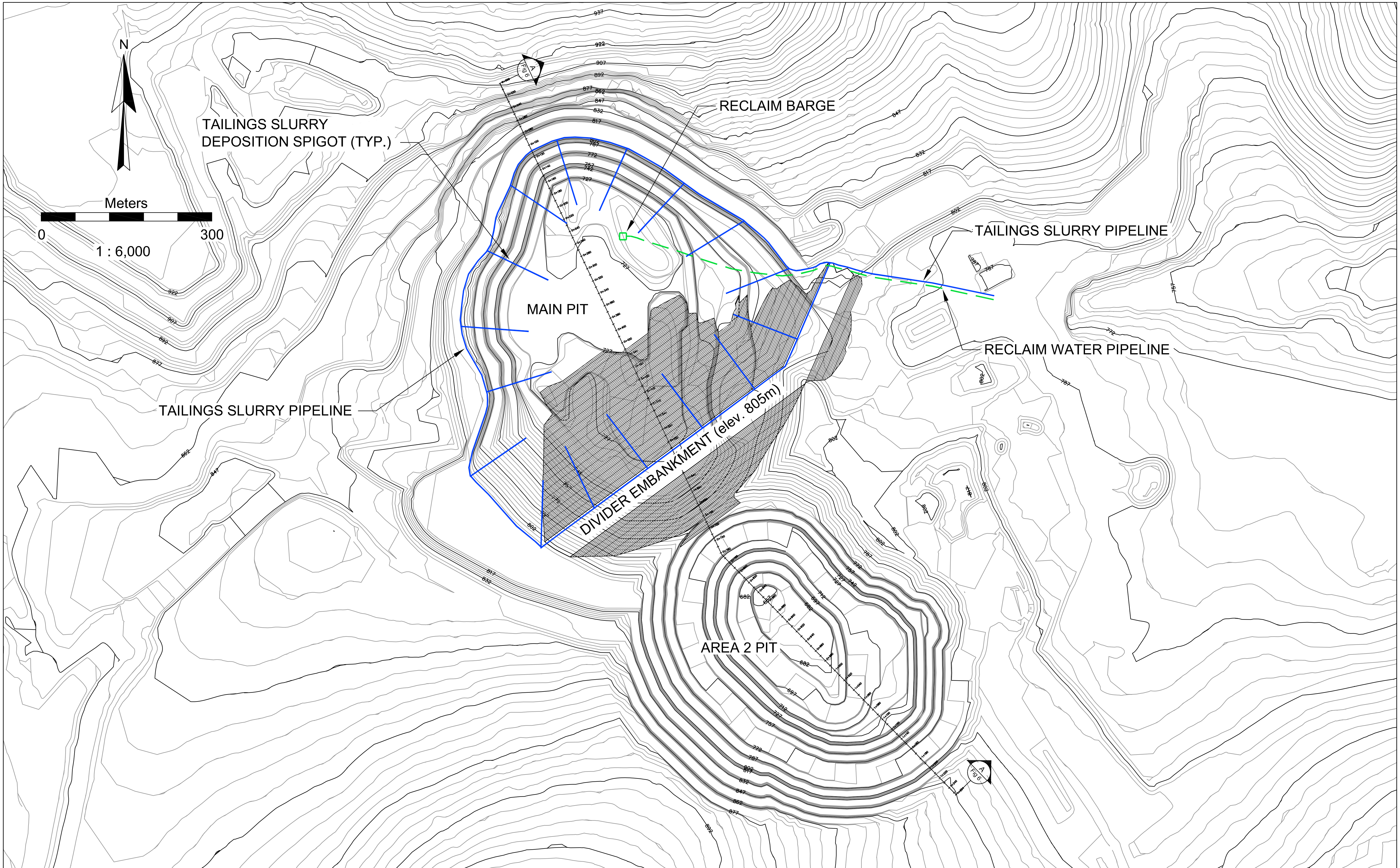
The following tables illustrate the findings of the conceptual design evaluation

- Table 18.20 – Water Balance Spreadsheet for 1.12 t/m$^3$ Dry Density
- Table 18.21 – Monthly and Annual Rate of Rise Characteristics
- The stage data provided in Figure 18.19 and utilized in the tailings occupation and surface water balance model account for the volume loss resulting from the conceptual embankment configuration, and indicate that the maximum potential tailings elevation will be ~ 792.5 m amsl, consistent with the values in Cells X83 and X84 (highlighted in yellow) in Table 18.20.

Pertinent conceptual design elements are summarized below:

- Required monthly pumping rates (in cubic meters per hour) to maintain the water depth at a maximum of 10m are shown in Table 18.20 in Column AC and indicate that pumping and potential treatment for discharge requirements range between 100 m$^3$/hour and 250 m$^3$/hour from May through September.
- From Table 18.21, monthly rates of rise will vary from 2m/month to around 0.6m/month as the available deposition area increases from the pit base to the final tailings elevation.
- During the early stages of tailings deposition, seepage from the Main Pit to the Area 2 Pit will be largely controlled by the permeability of the bedrock mass separating the two pits. In turn, the permeability of the rock mass will depend on structural features such as faults and joints, as well the frequency, continuity, orientation and infilling associated with these features. As the water level in the Main Pit rises above this rock mass, the permeability of the embankment within the Main Pit will also be a factor in the rate of seepage to the Area 2 Pit. Since this embankment is likely to be constructed mainly of waste rock and overburden, the permeability of the embankment could be one or more orders of magnitude more permeable than the inter-pit bedrock mass. At this stage, the following seepage control actions are assumed to be necessary:

  - Construction of a fine-grained overburden zone within the embankment.
  - Deposition of a tailings layer over the base of the Main Pit, using existing tailings, to decrease the pit base permeability prior to full scale production deposition.
  - Installation and operation of vertical dewatering wells to draw the phreatic levels within the embankment below the level of the inter-pit rock mass.
- In order to account for anticipated operational limitations due to winter conditions the operations will have to rely on consistent fixed point deposition during the winter months, and more flexible deposition rotation in the summer months. The winter limitations will also affect barge pumping and recycling operations.
- Deposition must be implemented in a manner that results in a uniform tailings topographic surface that minimizes “peak and valley formation” and results in optimum volume occupation and density for the conditions. This will require summertime filling in of valleys created by fixed winter deposition.
- From Table 18.21, the predicted annual rates of rise indicate that the embankment construction can be phased by building a “starter” embankment to a crest elevation of around 766m amsl, and then constructing raises, starting with a minimum raise of 10m during the second available construction season and followed by two annual 8m to 10m raises, with a final raise in the final year of operation of about 13m. The details of the last raise would change in the event the total volume of tailings is limited to 6.7 Mt, in which case the final elevation of the divider embankment could be lower by approximately 6m.
- The conceptual plan for tailings impoundment closure plan includes:
  - Utilization of the Area 2 Pit to manage water from the Main Pit tailings impoundment on an annual basis until cover installation is complete.
  - Phased construction of a waste rock cover on the surface of the tailings to form a surface topography that drains towards a spillway structure into the Area 2 Pit. The minimum thickness of the waste rock cover is estimated at 3 m.
  - Placement of 1 m of suitable growth media over the surface of the waste rock. Construction of a spillway that will be sized for an extreme event commensurate with the downstream risks as well as the ability of the pit to safely attenuate the design flood event. Details related to the sizing and location of the spillway, as well as the likely geotechnical conditions, have not yet been assessed.




| PREPARED FOR: | DRAWING TITLE: | |
|---|---|---|
| **MINTO COPPER MINE** | **CONCEPTUAL DESIGN LAYOUT** | |
| **MINTO PHASE IV PSF TECHNICAL REPORT TAILINGS DISPOSAL WATER MANAGEMENT** | DRAWING NO. **FIGURE 18.16** | REVISION NO. **A** |
| | SRK JOB NO. **2CM022.06** | |

| DESIGN: DB | DRAWN: ML | REVIEWED: - |
|---|---|---|
| CHECKED: - | APPROVED: DB | DATE: 12/03/2009 |
| FILE NAME: Fig18.14-18_Topography_ML_rbb_20091203.dwg | | |

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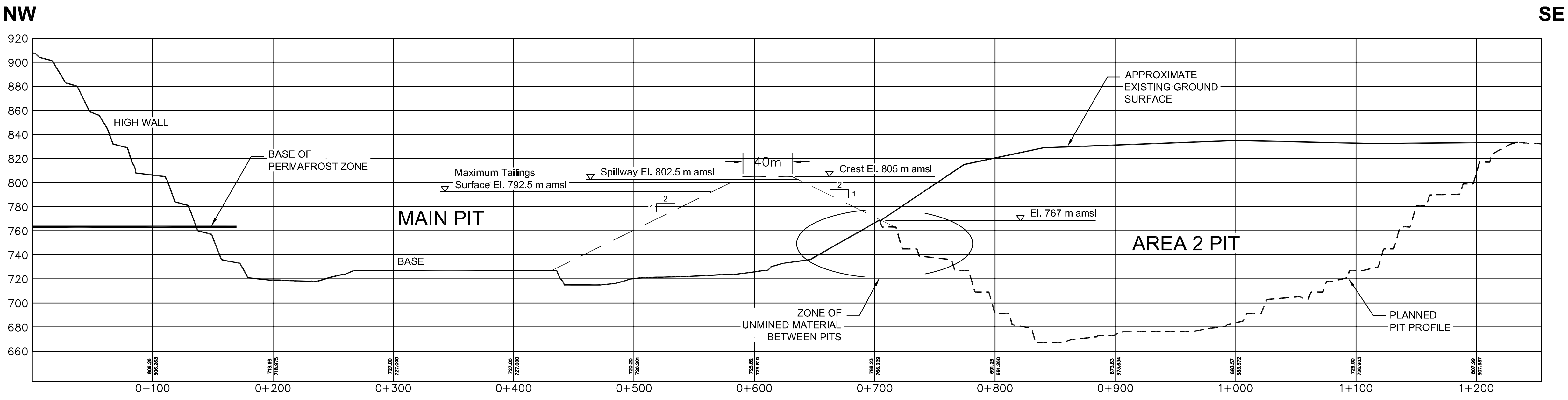


## SECTION A-A

SCALE 1 : 4000

| | | | | | |
|---|---|---|---|---|---|
| DESIGN: DB | DRAWN: ML | REVIEWED: - | PREPARED FOR: **MINTO COPPER MINE** | DRAWING TITLE: **CONCEPTUAL DESIGN CROSS-SECTION** | |
| CHECKED: - | APPROVED: DB | DATE: 12/03/2009 | **MINTO PHASE IV PSF TECHNICAL REPORT TAILINGS DISPOSAL WATER MANAGEMENT** | DRAWING NO. **FIGURE 18.17** | REVISION NO. **A** |
| FILE NAME: Fig18.14-18_Topography_ML_rbb_20091203.dwg | | | | SRK JOB NO. **2CM022.06** | |

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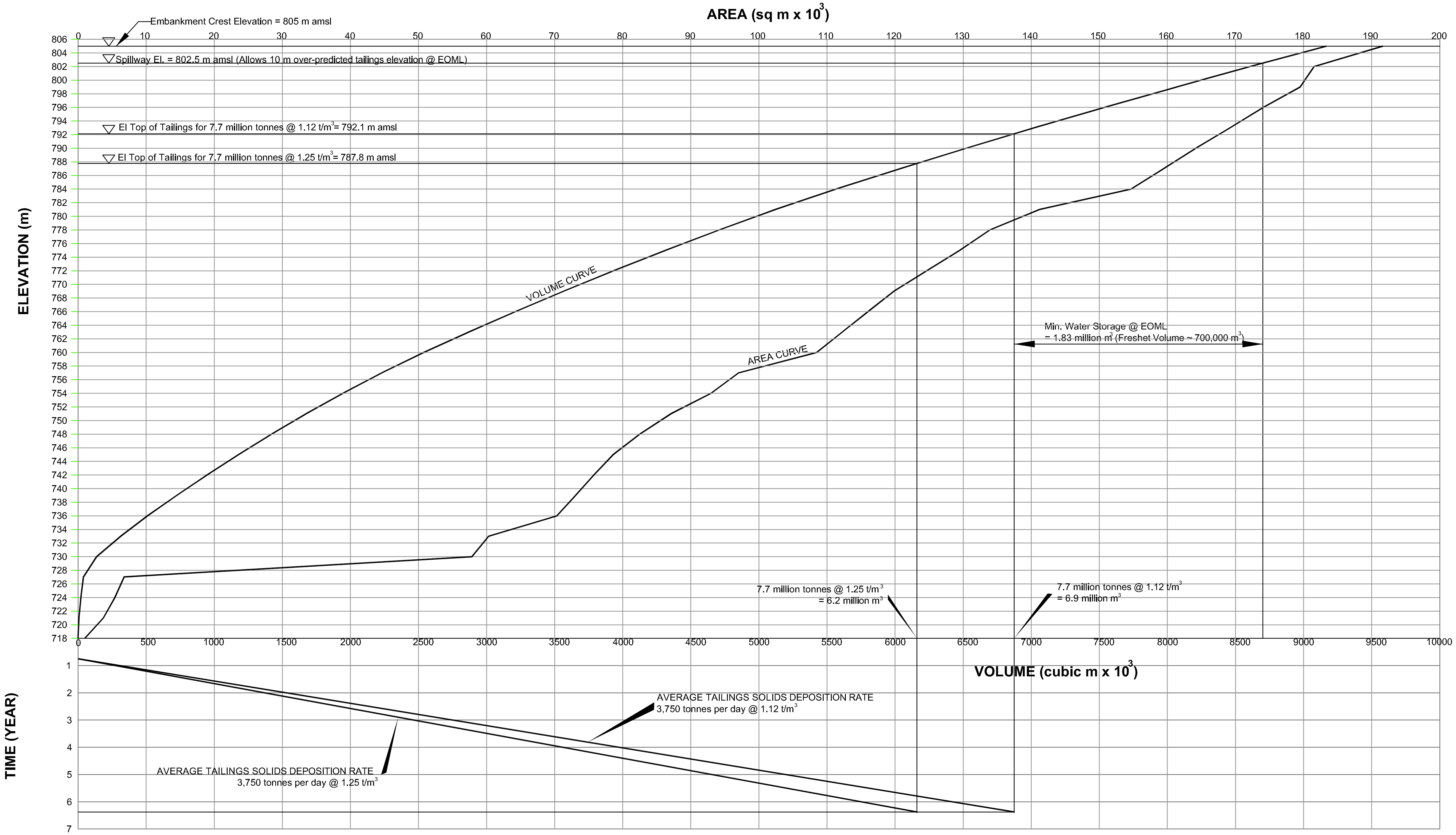



| | | | |
|---|---|---|---|
| DESIGN: BB | DRAWN: ML | REVIEWED: - | PREPARED FOR: **MINTO COPPER MINE** |
| CHECKED: - | APPROVED: BB | DATE: 12/03/2009 | **MINTO PHASE IV PSF TECHNICAL REPORT TAILINGS DISPOSAL WATER MANAGEMENT** |
| FILE NAME: Fig18.19_ML_20091203.dwg | | | |

| | |
|---|---|
| DRAWING TITLE: | **STAGE CURVE** |
| DRAWING NO. | **FIGURE 18.18** |
| SRK JOB NO. | **2CM022.06** |
| REVISION NO. | **A** |

J:\01_SITES\Minto\!ACAD\Received 15Dec09\Fig18.19_ML_20091203.dwg

## Table 18.21 Monthly and Annual Rate of Rise Characteristics

| Year | Month | Average Tailings Dry Density = 1.12 t/m$^3$ | | | Average Tailings Dry Density = 1.25 t/m$^3$ | | |
|---|---|---|---|---|---|---|---|
| | | Elev. (m) | Rate of Rise (m per month) | Rate of Rise (m per year) | Elev. (m) | Rate of Rise (m per month) | Rate of Rise (m per year) |
| 1 | Oct | 728.4 | 10.4 | 13.9 | 728.1 | 10.1 | 13.4 |
| | Nov | 730.5 | 2.1 | | 730.2 | 2.1 | |
| | Dec | 731.9 | 1.4 | | 731.4 | 1.3 | |
| 2 | Jan | 733.3 | 1.4 | 14.7 | 732.7 | 1.3 | 13.4 |
| | Feb | 734.4 | 1.1 | | 733.8 | 1.1 | |
| | Mar | 735.7 | 1.3 | | 734.9 | 1.1 | |
| | Apr | 736.8 | 1.2 | | 736.0 | 1.1 | |
| | May | 738.0 | 1.2 | | 737.0 | 1.0 | |
| | Jun | 739.1 | 1.1 | | 738.0 | 1.0 | |
| | Jul | 740.4 | 1.3 | | 739.2 | 1.2 | |
| | Aug | 741.7 | 1.3 | | 740.4 | 1.2 | |
| | Sep | 742.9 | 1.2 | | 741.5 | 1.1 | |
| | Oct | 744.1 | 1.2 | | 742.6 | 1.1 | |
| | Nov | 745.3 | 1.2 | | 743.7 | 1.1 | |
| | Dec | 746.5 | 1.2 | | 744.8 | 1.1 | |
| 3 | Jan | 747.7 | 1.2 | 13.1 | 745.9 | 1.1 | 12.3 |
| | Feb | 748.8 | 1.0 | | 746.9 | 1.0 | |
| | Mar | 749.9 | 1.1 | | 748.0 | 1.1 | |
| | Apr | 751.1 | 1.2 | | 749.0 | 1.1 | |
| | May | 752.2 | 1.2 | | 750.1 | 1.1 | |
| | Jun | 753.3 | 1.1 | | 751.2 | 1.1 | |
| | Jul | 754.5 | 1.1 | | 752.2 | 1.0 | |
| | Aug | 755.6 | 1.1 | | 753.2 | 1.0 | |
| | Sep | 756.6 | 1.1 | | 754.2 | 1.0 | |
| | Oct | 757.7 | 1.0 | | 755.2 | 1.0 | |
| | Nov | 758.6 | 1.0 | | 756.2 | 0.9 | |
| | Dec | 759.7 | 1.0 | | 757.1 | 1.0 | |
| 4 | Jan | 760.6 | 1.0 | 10.7 | 758.0 | 0.9 | 10.0 |
| | Feb | 761.5 | 0.9 | | 758.9 | 0.8 | |
| | Mar | 762.4 | 0.9 | | 759.8 | 0.9 | |
| | Apr | 763.3 | 0.9 | | 760.6 | 0.8 | |
| | May | 764.2 | 0.9 | | 761.5 | 0.8 | |
| | Jun | 765.1 | 0.9 | | 762.3 | 0.8 | |
| | Jul | 766.0 | 0.9 | | 763.1 | 0.8 | |
| | Aug | 766.9 | 0.9 | | 763.9 | 0.8 | |
| | Sep | 767.8 | 0.9 | | 764.7 | 0.8 | |
| | Oct | 768.6 | 0.9 | | 765.5 | 0.8 | |
| | Nov | 769.5 | 0.8 | | 766.3 | 0.8 | |
| | Dec | 770.3 | 0.8 | | 767.1 | 0.8 | |
| 5 | Jan | 771.2 | 0.8 | 9.4 | 767.9 | 0.8 | 8.8 |
| | Feb | 771.9 | 0.8 | | 768.6 | 0.7 | |
| | Mar | 772.8 | 0.8 | | 769.4 | 0.8 | |
| | Apr | 773.6 | 0.8 | | 770.1 | 0.7 | |
| | May | 774.4 | 0.8 | | 770.9 | 0.8 | |
| | Jun | 775.2 | 0.8 | | 771.6 | 0.7 | |
| | Jul | 775.9 | 0.8 | | 772.3 | 0.7 | |
| | Aug | 776.7 | 0.8 | | 773.1 | 0.7 | |
| | Sep | 777.5 | 0.8 | | 773.8 | 0.7 | |
| | Oct | 778.3 | 0.8 | | 774.5 | 0.7 | |
| | Nov | 779.0 | 0.7 | | 775.2 | 0.7 | |
| | Dec | 779.8 | 0.8 | | 775.9 | 0.7 | |
| 6 | Jan | 780.5 | 0.8 | 8.1 | 776.6 | 0.7 | 7.8 |
| | Feb | 781.2 | 0.7 | | 777.3 | 0.6 | |
| | Mar | 781.9 | 0.7 | | 778.0 | 0.7 | |
| | Apr | 782.6 | 0.7 | | 778.6 | 0.7 | |
| | May | 783.3 | 0.7 | | 779.3 | 0.7 | |
| | Jun | 783.9 | 0.7 | | 780.0 | 0.7 | |
| | Jul | 784.6 | 0.7 | | 780.6 | 0.7 | |
| | Aug | 785.3 | 0.7 | | 781.3 | 0.7 | |
| | Sep | 785.9 | 0.6 | | 781.9 | 0.6 | |
| | Oct | 786.6 | 0.7 | | 782.5 | 0.6 | |
| | Nov | 787.2 | 0.6 | | 783.1 | 0.6 | |
| | Dec | 787.8 | 0.6 | | 783.8 | 0.6 | |
| 7 | Jan | 788.5 | 0.6 | 5.5 | 784.4 | 0.6 | 5.2 |
| | Feb | 789.1 | 0.6 | | 784.9 | 0.5 | |
| | Mar | 789.7 | 0.6 | | 785.5 | 0.6 | |
| | Apr | 790.3 | 0.6 | | 786.1 | 0.6 | |
| | May | 790.9 | 0.6 | | 786.7 | 0.6 | |
| | Jun | 791.5 | 0.6 | | 787.2 | 0.6 | |
| | Jul | 792.2 | 0.6 | | 787.8 | 0.6 | |
| | Aug | 792.8 | 0.6 | | 788.4 | 0.6 | |
| | Sep | 793.4 | 0.6 | | 788.9 | 0.6 | |

# 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 | 41.0 | 16.0 |
| Area 118 | 198 | 70 | 2.0 | 0.6 | 95.0 | 77.0 | 39.6 | 9.4 |
| RTE | 168 | 66 | 1.1 | 0.3 | 88.0 | 55.7 | 39.0 | 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 | 73.0 | 39.5 | 15.8 |
| Main (South) Partially Oxidized | 223 | 76 | 1.7 | 0.6 | 84.0 | 64.7 | 36.9 | 10.5 |
| Average | 210 | 78 | 2.1 | 0.8 | 91.7 | 70.6 | 38.9 | 11.9 |
| Average (excluding Main (S) Partially Oxidized | 209 | 79 | 2.1 | 0.9 | 92.6 | 71.3 | 39.2 | 12.0 |

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 (+38% Cu), its very low contaminants 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 MRI Trading AG ("MRI"). 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 2007 to June 2010. The contract may be extended by mutual agreement one or more years.

This study assumes that treatment charges will be US$50.00/dmt of concentrate and refining charges will be US$ 0.05/lb payable copper through the life of the operation. These assumptions are based on the continuation of a general supply shortage of concentrate and in particular, high quality concentrates.

## 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 |
|---|---|---|
| 2010 | 12,609 tonnes Cu | US$ 2.19/lb Cu |
| 2011 | 8,312 tonnes Cu | US$ 2.26/lb Cu |

## 20.3 Precious Metal Price Contract

MintoEx sold its gold and silver production to Silverstone Resources in November 2008. Silverstone was subsequently bought out by Silver Wheaton who now owns the Minto mine precious metal stream. Silver Wheaton 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 |
|---|---|---|
| MRI Trading AG | Concentrate purchase | Agreement to purchase all Minto concentrates up to June 2010. 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. See Marketing Section |
| Silver Wheaton | 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$3.90/oz Ag. |
| Yukon Energy Corporation | Grid power | Minto agrees to purchase power from YEC, pay for a portion of the new transmission line and sell its 4 main existing diesel generators to YEC in exchange for YEC building the main new transmission line. |
| Dyno Nobel Canada Inc. | Explosive and accessory supply | Supply, storage, transportation and placement of explosives and accessories |
| Pelly Construction | Mining and mobile equipment supply | Pelly Construction currently performs all open pit mining functions at Minto and uses its ancillary equipment for various jobs on site. The mining costs shown in 2011 and beyond assume owner/operator mining, not contractor. |
| Canadian Lynden Transport Co. | Concentrate transport | Provides terms and conditions for the road transportation of concentrates to Skagway, AK. Valid until the 2$^{nd}$ Qtr, of 2014. |
| Great Northern Oil | Diesel supply | Transport and supply of diesel fuel. |
| Domco | Camp Services | Lodging and catering |

### Contract Mining

Minto and Pelly Construction have entered an agreement to the end of 2010 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 for over two years. 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 Government (YG) 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) has 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 (November 2009) entering the assessment process under YESAA again for water management and mining and milling rate amendments to the major authorizations.

**Table 22.1: Previous Environmental Assessments of the Minto Project**

| Activity | Period | Sources |
|---|---|---|
| Minto Explorations Ltd.<br>Minto Mine Development, Operation and Closure | 1996 to 1998 | Government and company reports, 1996.<br>DIAND EARP screening and Decision Report, Water Licence QZ96-006. |
| Minto Explorations Ltd.<br>Minto Mine Development, Operation and Closure<br>Cumulative Effects Assessment | 1999 | Company report on Cumulative Effects, 1999.<br>Quartz Mining Licence QLM-9902. |
| Minto Explorations Ltd.<br>Minto Mine Development, Operation and Closure<br>Licence Amendments – Expiry Extensions and Temporary Closure Modifications | 2004 to 2005 | Government and company reports, 2004.<br>YG AO Development Assessment Branch<br>YEAA Screening<br>Water Licence Amendment and Quartz Mining Licence QML-0001 |
| Minto Explorations Ltd.<br>Mining and Milling Rate Increase, Minto Project | 2008 | Company Reports, 2008.<br>YESAA DO Assessment<br>Quartz Mining Licence QML-0001 Amendment |

### 22.1.1 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, as discussed below.

#### Type B Water Use Licence

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 Licence

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.

### 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 (QLM-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, YG 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. YG 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 YG 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 YG 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 m3 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 m3 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 YG 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 m3 of water from the Minto Mine site provided it met amended discharge standards.

All of the above noted licences have an expiry date of June 30, 2016.

### 22.1.2 Assessment and Licensing for Phase IV

The expansion of the Minto Mine in the Phase IV 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 IV 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. Most information required for an assessment at the ExComm level is completed or in progress, and MintoEx is preparing a Project Proposal containing information sufficient for submission to the Executive Committee.

## 22.2 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, 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 recently 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.

## 22.3 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) 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 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. Currently an updated Environmental Conditions report is in preparation to support the Phase IV development that updates 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 (Unnamed Creek B ) 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, stream sediment and fisheries investigations were all conducted in the Unnamed Creek B drainage in 2009 by Access Consulting Group, and a summary report on the baseline conditions is being prepared for inclusion with the Environmental Conditions report for the environmental assessment project proposal. The Unnamed Creek B and Minto Creek Drainages are presented in Figure 22.1 below, with the approximate location of the Minto North Deposit.

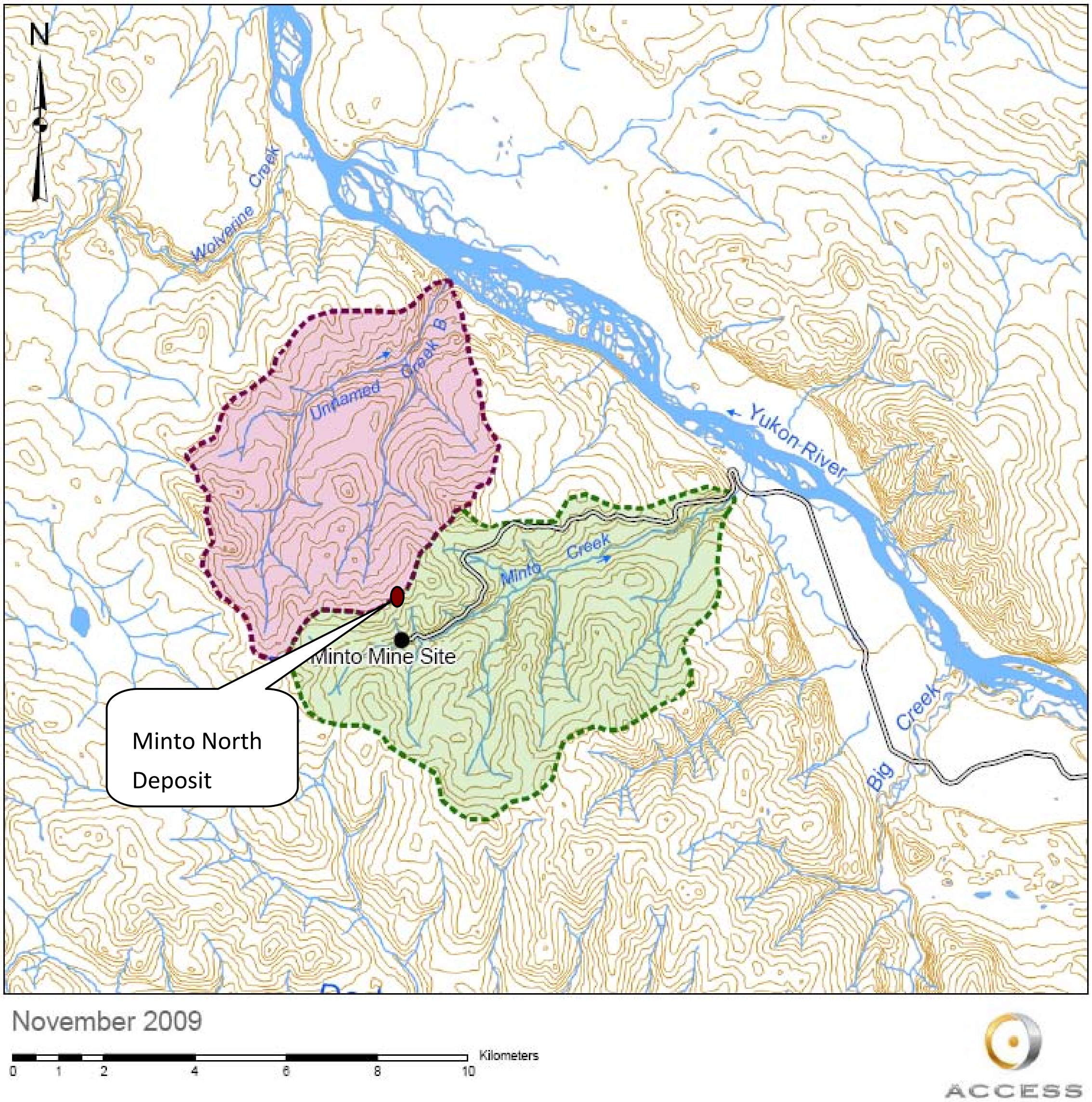



**Figure 22.1. Minto Creek and Unnamed Creek B Drainages relative to Minto Mine Site, Minto North Deposit and Yukon River**

Groundwater monitoring information is minimal at the Minto site, so augmentation of the groundwater monitoring network at the site is currently being planned. Monitoring wells are scheduled for installation around the existing site – below existing waste deposition areas and down gradient of the Main Pit – and around the Minto North deposit, with the intention of characterizing the groundwater conditions in all areas of current and proposed mining activities.

## 22.4 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 |
| Aluminum | 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.

Continued reliance on the Yukon Waters Act emergency provisions to manage water at the Minto Mine site is undesirable; however Minto Ex is unable to consistently meet the WUL QZ96-006 effluent discharge standards. MintoEx has therefore revised the site Water Management Plan and has submitted a YESAA 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.

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. Licence amendment approval is expected in 2010. 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 developments, the plan will require further reassessment during the Phase IV development planning process.

The critical consideration with respect to water management for Phase IV 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 practices for waste rock, tailings and wastewater will be scrutinized for the Phase IV development and these management plans must be fully integrated for the Phase IV development. The currently proposed water management plan identifies the Main Pit as a contingency storage location. As the mine planning progresses for Phase IV, this contingency storage requirement should be reassessed, as it may have implications on pit sequencing and/or waste deposition in the Main Pit.

## 22.5 Closure Planning

### 22.5.1 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 YG 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 YG based on approved closure cost estimates.

The first update to this DDRP was submitted in September 2009 and is currently undergoing government review. 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 are currently under review.

As required in Section 14.3 of QML-0001, this DDRP will be updated again in 2011.

### 22.5.2 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 YG.

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.

However, no substantial progressive reclamation was conducted on the site in the first two years of operations according to the proposed schedule in the 2007 DDRP, so Energy Mines and Resources, Mineral Resources Division, may be less willing to offset security requirements through future commitments to progressive reclamation.

### 22.5.3 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 a various trial plot locations around the site to optimize revegetation success of progressive and final reclamation seeding. Dry stack tailings cover design is being refined through ongoing research and proposed trials beginning in 2010.

### 22.5.4 Closure Planning for Phase IV Expansion

Closure philosophies and measures for the Phase IV mine plan will mirror those presented in the previously submitted and approved DDRPs. Although closure and reclamation concepts will be required for the Phase IV 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 on the existing QML schedule (every 2 years on the anniversary of the mill start up – August 1). Revisions to the DDRP reflecting the Phase IV mine plan would not be required until the amendments to the WUL and QML authorizing mining and milling activities in the Phase IV deposits are issued, as the DDRP applies to authorized mining activities and plans. Closure measures for the site following the completion of the Phase IV mine plan are expected to generally follow those currently authorized.

## 22.6 Metal Leaching/ Acid Rock Drainage Characterization

### 22.6.1 Introduction

The Phase 4 mine plan will introduce the following components to the presently-permitted facilities at the Minto Mine:

- Waste rock from the Area 2, Ridgetop, and Minto North open pits;
- Development rock from the access drift to the Area 118 underground;
- Tailings from processing ore from Area 2, Area 118, Ridgetop and Minto North.

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 4 operations. The results are presented in the following sections, and the Phase 4 results are compared with operational monitoring of Minto Pit tailings and waste rock.

## 22.6.2 Phase 4 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 was rock has been tested to determine ML/ARD characteristics. There will be little to no waste rock produced during underground mining of the Area 118 deposit. Development rock expected to be produced while driving the access ramp is expected to be unmineralized- confirmation testing may be undertaken either in advance of development, or as underground development proceeds.
- Ridgetop: The current understanding of the Ridgetop deposit geology is summarized in Section 16, 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.

  Twenty 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 waste were selected, along with two intervals of mineralized waste and two ore-grade intervals.
- Minto North: The current understanding of the Minto North deposit geology is summarized in Section 16.5.1. 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. The current understanding of the Minto North deposit geology is summarized in Section 16.

The drillhole IDs and From-To intervals for all the samples tested for ML/ARD characteristics are listed in Appendix C.

### 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.

### Results

Results of Phase 4 waste rock ABA testing and elemental determinations are compiled in Appendix C.

#### *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 4 pit. A plot of NP and AP values for all Phase 4 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.

### *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.

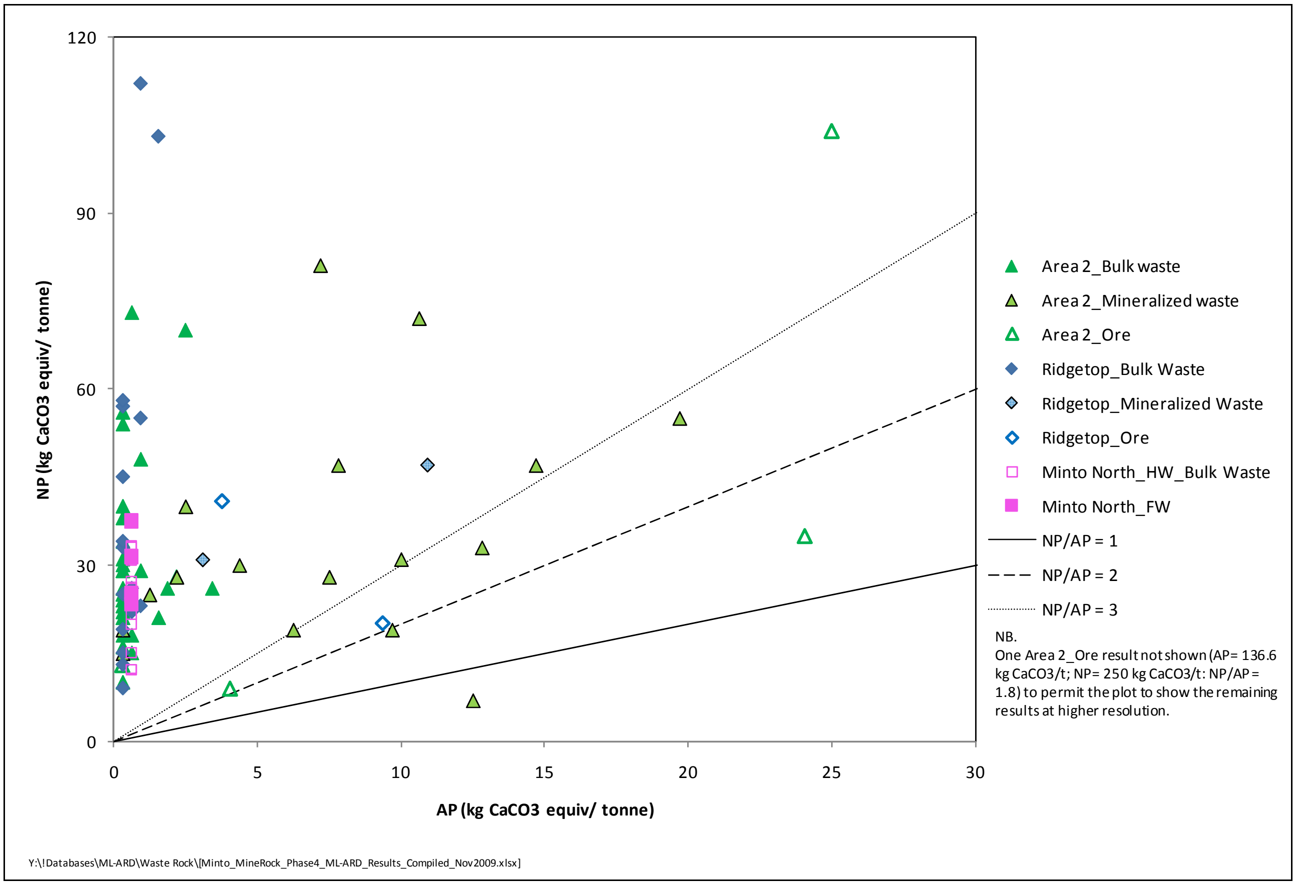



**Figure 22.2: NP/AP Results for Phase 4 Mine Rock Samples**

### *Elemental Content*

Elemental content of mine rock tested in Phase 4 was compared with crustal average concentrations of granitic rocks (Price, 1997), crustal average concentrations and Phase 4 elemental contents are included in Appendix C. A value of three times (3x) the crustal average concentration was used as a screen to determine whether Phase 4 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 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 in Section 22.6.5.

## 22.6.3 Phase 4 Tailings Characterization

### Sample Selection

Tailings samples were selected from residues from metallurgical testing of ores from the Area 2, Area 118, Ridgetop, and Minto North 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.

### 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, and Minto North: 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. Elemental analyses consisted of aqua regia digestion followed by determination of 36 elements by ICP-MS.

### Results

Results of Phase 4 tailings ABA testing and elemental determinations are compiled in Appendix C. Phase 4 tailings samples were assigned ARD classifications based on the categories described for waste rock in Section 22.6.2 (PAG, uncertain, or NPAG).

All Phase 4 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 4 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.

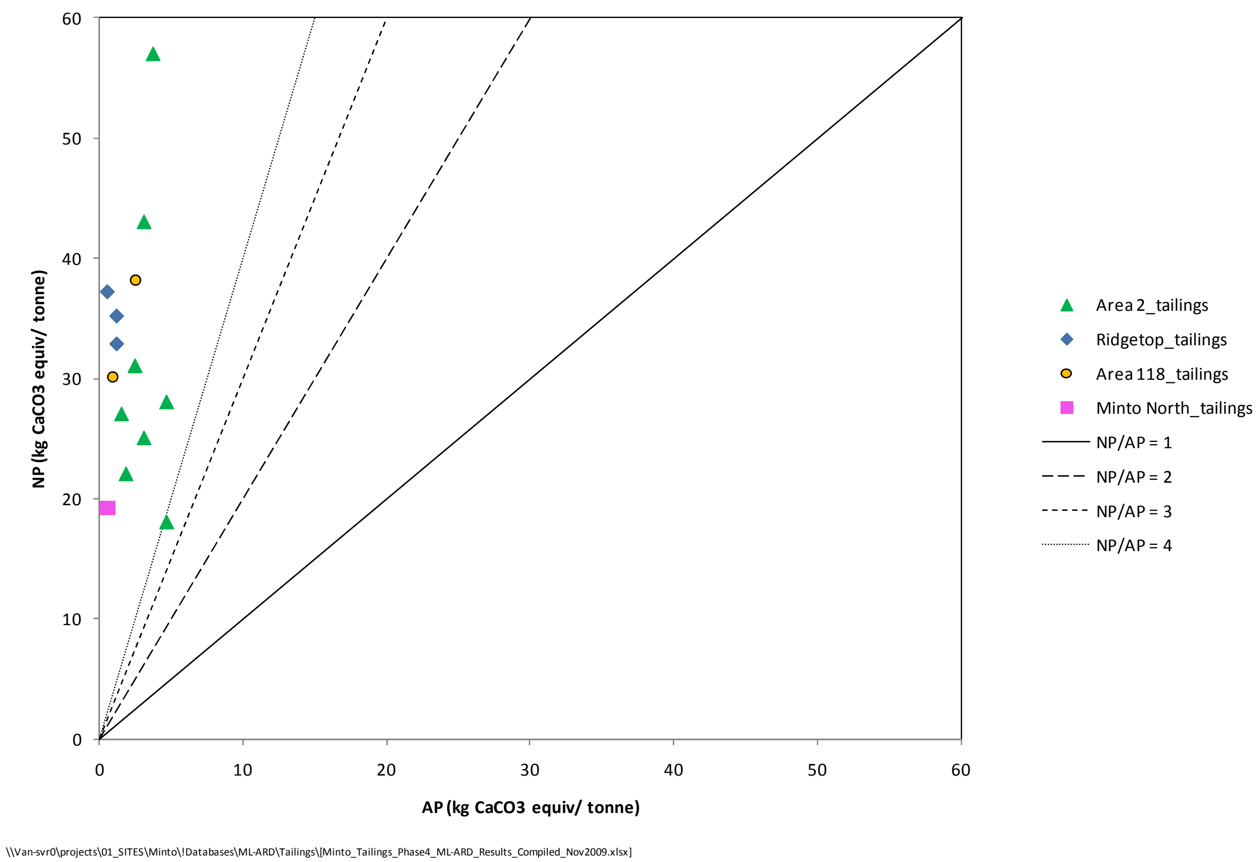



**Figure 22.3: NP/AP Results for Phase 4 Tailings Samples**

### 22.6.4 Ongoing Tests in Progress

Several field and laboratory testing programs were underway at the time of publication of this report to provide information on rates of neutral pH metal leaching from Minto tailings and waste rock materials. The results of these testing programs will be used as inputs to the environmental assessment process. Testing is being carried out either at the Minto Mine site or at SGS CEMI in Burnaby BC.

The following points summarize the testing programs that are underway.

- Barrel tests: Four on-site barrels were set up during 2009 each containing roughly 200 kg of mineralized waste collected from available Area 2 drill core. The contents of each barrel have a narrow range of sulphur concentrations, corresponding approximately to the $10^{th}$, $50^{th}$, $75^{th}$ and $90^{th}$ percentile sulphur concentrations within the Area 2 mineralized waste. Results of these tests will aid in evaluating how small quantities of mineralized waste rock will influence waste dump water quality.

- Humidity cell tests: four tests are underway using drill core of Area 2 waste rock with a range of copper and sulphur concentrations. The results of these humidity cell tests will inform assessments of long-term loadings from rock exposed in final pit walls.

- Modified MWMP tests: four modified Meteoric Water Mobility Procedure tests are underway on two waste rock samples from the existing Main Dump and on two additional samples from the existing Blue Dump. The purpose of these tests is to assess whether concentrations of copper or other parameters in waste dump drainage are likely to be controlled by geochemical equilibrium with solid minerals with the waste material.

- Subaqueous column testing of tailings: two tests are underway on composite samples of Ridgetop and Area 118 tailings. The purpose of these tests is allow estimation of water chemistry in tailings porewater and in surface water overlying tailings that will be deposited in the mined-out Minto Pit.

- Mineralogical characterization, consisting of quantitative x-ray diffraction and optical petrography, is being carried out on several of the samples used in the tests noted above. This characterization is being carried out to define the minerals that contribute neutralization potential to Minto wastes, to identify similarities and potential differences between the waste materials from the different deposits, and to provide an indication of the minerals that host the parameters of concern for leaching.

### 22.6.5 Implications of Phase 4 ML/ARD Characterization for Waste Management

Waste rock produced during mining of the Phase 4 pits is expected to have a substantial excess of NP. As such, prevention of ARD does not need to be considered in developing management plans for Phase 4 waste rock.

However, detailed characterization of mineralized waste and ore from within the Area 2 Pit shell has shown that a small proportion of mineralized waste will 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 4 pits as well. However, 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. Care should be taken during operations to ensure that mineralized waste is placed randomly with the bulk waste to avoid the creation of local acidic 'hot spots' within the larger neutral mass of waste rock.

Tailings produced during processing of ore from the Phase 4 pits 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 4 tailings.

Neutral pH leaching of copper from mine rock and tailings is expected to continue to present the greatest challenge for the mine during operations. Exposed ore stockpiles, mine faces, waste dumps, and tailings together provide a large amount of surface area for weathering reactions and subsequent leaching of copper, and the mine will need to continue to be diligent in applying the necessary controls to ensure water leaving the mine site meets discharge criteria. Based on the lack of elevated concentrations of other elements in the results of the Phase 4 characterization discussed above, neutral pH leaching of other elements is unlikely to occur at environmentally-significant concentrations.

SRK has not carried out a site water quality prediction for either the operational period or for the post-closure conditions. It is expected that a site water quality prediction will be a component of the Phase 4 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 declining balance basis for the mine capital assets and mill capital assets. It, however, does not have loss carryover provision. Taxes are paid at rates that increase as income increases.

The opening balances for the tax pools for both taxes are included in the cash flow model.

Hedging gains and losses are not taken into account for taxation.

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.

# 24 Cost Estimation

## 24.1 Operating Cost Estimate

### 24.1.1 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 2011.The operating costs for the owner-operated scenario are presented in Q3-2009 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 24.1 by mining category.

**Table 24.1: Open Pit Operating Cost Estimate - by Category**

| Cost Category | Cost/Tonne Mined |
|---|---|
| Operating Labour | $0.46 |
| Maintenance Labour | $0.18 |
| Service and Support Equipment | $0.06 |
| Supervision | $0.11 |
| Non-energy Consumables | $0.66 |
| Fuel | $0.65 |
| Leases, Outside Services, Misc. | $0.01 |
| **Total Open Pit Operating Cost** | **$2.11** |

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.

## 24.2 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 actual Minto plant operating costs for the first half of 2009.

A summary of the average operating costs per tonne of ore treated for the Project is outlined in Table 24.2. The costs were divided into the key cost centres and all figures are as of the fourth quarter 2009 (calendar year).

**Table 24.2: Estimated Average Operating Costs ($/t)**

| Summary | Current Plant (2747 tpd) | Plant Upgrade (3750 tpd) |
|---|---|---|
| Labour | 5.91 | 4.40 |
| Power | 3.18 | 2.23 |
| Reagents and Consumables | 4.86 | 4.09 |
| Contract Secondary Crushing | 2.37 | 0.00 |
| Other Maintenance Materials | 0.61 | 0.44 |
| Assay and Met Lab | 1.23 | 0.90 |
| Loader feeding Jaw Crusher | 0.90 | 0.66 |
| Re-handle of fine crushed material | 0.60 | 0.00 |
| Re-handle on coarse ore stockpile | 0.08 | 0.06 |
| Tails filtration and dry stacking | 3.49 | 0.00 |
| **TOTAL $/t** | **23.21** | **12.79** |

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 current Minto process plant based on an annualised throughput of 1,002,711 tonnes was $23.21 /t. The calculated operating cost for the plant upgrade based on an annualised throughput of 1,368,837 tonnes was $12.79 /t. The reduction in operating cost ($/t) is primarily due to:

- Bypass the current tailings filtration and paste disposal system and direct disposal of the tailings thickened slurry from the existing tailings thickener into the Minto Main Pit. This results in a cost saving of $ 4.51 /t (including power saving);
- Replace the contract secondary crushing system with an installed site operated secondary crusher. This results in a cost saving of around $2.38 million per year or $2.57 /t; and
- Increased annual plant throughput.

### 24.2.1 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 24.3.

**Table 24.3: Derivation of Plant Operating Costs**

| Cost Category | Source Of Cost Data |
|---|---|
| Labour | Manning schedules and rates provided by MintoEx. |
| Power | Consumption from load estimate and power unit rate from MintoEx. |
| Reagents | Consumptions from MintoEx 2009 budget; unit prices from actuals as reported by MintoEx for first half 2009. |
| Consumables | Consumptions based on actuals as reported by MintoEx for first half 2009 and Ausenco experience; unit prices from actuals as reported by MintoEx. |
| Maintenance Materials | Based on actuals as reported by MintoEx for first half 2009 and benchmarked against similar operations in the Ausenco database. |
| Contract Secondary Crushing | Cost per tonne as supplied by MintoEx |
| Tailings Filtration | Based on actuals as reported by MintoEx for first half 2009. |
| Assay and Metallurgical Laboratory | Based on actuals as reported by MintoEx for first half 2009. |

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.

### 24.2.2 Plant Operating Cost Estimate Inclusions

Included in the operating cost estimate are:

- Labour for supervision, management and reporting of onsite organisational 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.

### 24.2.3 Labour

Labour costs for the plant were provided by MintoEx. These were verified against the 2009 half year actual plant labour costs. 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 24.4.

**Table 24.4: Site Labour Cost Summary**

| Labour ($M/year) | Current Plant (2747 tpd) | Plant Upgrade (3750 tpd) |
|---|---|---|
| Mill Operations | 3.45 | 3.45 |
| Mill Maintenance | 2.45 | 2.45 |
| Mill Administration | 0.03 | 0.03 |
| Water Treatment Plant | - | 0.10 |
| TOTAL $M/y | 5.92 | 6.02 |
| **TOTAL $/t** | **5.91** | **4.40** |

For the purposes of estimating overall operating costs labour for the plant was not adjusted for the upgrade, despite the removal of the tailings filtration plant.

### 24.2.4 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 2009 budget unit power rate plus 5% to provide a value of $0.105 /kWh.

The power requirements for the current plant were developed from the electrical load list. The load study on which the power costs were based calculates a specific power draw given the installed equipment power (excluding installed standby equipment) and a utility factor to allow for intermittently running equipment. Power consumption was derived from the specific power draw and plant operating hours.

The electrical load list was determined for the plant upgrade based on new and redundant equipment. The same methodology was used to calculate the power requirement for the plant upgrade.

The plant power consumption is expected to vary 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 24.5.

**Table 24.5: Process Plant Power Cost Summary**

| Power Costs ($/t) | Current Plant (2747 t/d) | Plant Upgrade (3750 t/d) |
|---|---|---|
| Mill Building | 0.05 | 0.04 |
| Crushing | 0.10 | 0.21 |
| Grinding | 1.68 | 1.27 |
| Flotation | 0.34 | 0.59 |
| Concentrate Handling | 0.04 | 0.03 |
| Tailings Disposal | 1.02 | 0.12 |
| Reagents | 0.01 | 0.01 |
| Plant Services and Reclaim Water | 0.09 | 0.09 |
| **TOTAL $/t** | **$3.34** | **$2.35** |
| **TOTAL $M/y** | **$3.35** | **$3.21** |

### 24.2.5 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 etc.

The Minto cost centres assigned to maintenance tools/miscellaneous costs are shown in Table 24.6.

### Table 24.6: 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 for both the current plant and the upgraded plant 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 have been replaced with a new supplier and replacement frequency and costs are expected to reduce;
- Original pipework around the milling area was not rubber lined. Pipework was replaced with rubber lined pipes which will reduce the frequency of change-outs;
- Original installed pumps had a high failure rate. Subsequently various pumps have been upgraded and standby tailings pumps installed under operating cost budgets.

Therefore the final cost of $1.71 million per year used for maintenance materials was based on the Minto half year 2009 actual costs minus 15% to bring the cost in-line with the expected cost based on benchmarked plants.

Exclusions from these costs are:

- Crusher wear components, mill liners and lifters, and other components included in reagents and consumables (included in Section 24.2.7).
- Maintenance labour costs (included in the Section on labour costs).
- Sustaining capital costs.

### 24.2.6 Reagents and Consumables

Reagent consumptions rates used for the estimate were based on the Minto 2009 budget. Exceptions to this were:

- The flotation tailings thickener flocculant consumption rate was reduced from 0.042 kg/t to 0.037 kg/t. It is expected that the flocculant consumption rate will reduce once auto-dilution modifications are completed on the thickener feedwell during 4th quarter 2009; and
- Reagents for the proposed water treatment plant were estimated based on unit costs per cubic meter of water treated as supplied by MintoEx. The total amount of water treated per year was estimated at 360,000 $m^3$.

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 average actual unit rates for the first half of 2009. The rates are ex-works and do not include freight and logistics, handling and taxes.

The average LOM consumptions and the unit costs are presented in Table 24.7.

**Table 24.7: Reagent Consumptions and Unit Costs**

| Reagent | Unit $/kg | Consumption kg/t Feed | Current Plant $M/y | Plant Upgrade $M/y |
|---|---|---|---|---|
| Flotation Collector (PAX) | 2.81 | 0.07 | 0.18 | 0.25 |
| Flotation Frother (MIBC) | 3.89 | 0.03 | 0.12 | 0.17 |
| Nitric Acid (kg) | 1.33 | 0.04 | 0.06 | - |
| Diesel for tails re-handle and stacking (litres) | 0.80 | 0.64 | 0.52 | - |
| Diesel for the Mill (litres) | 0.80 | 0.66 | 0.53 | 0.53 |
| Tailings Flocculant (AE4270) | 4.83 | 0.04 | 0.18 | 0.21 |
| Concentrate Flocculant (AE4330) | 5.61 | 0.001 | 0.005 | 0.01 |
| TMT/Sulphide for WTP | | | - | 0.05 |
| Floc for WTP | | | - | 0.01 |
| Aluminex for WTP | | | - | 0.18 |
| **TOTAL $M/y** | | | **1.59** | **1.41** |
| **TOTAL $/t** | | | **1.59** | **1.03** |

### 24.2.7 Plant Consumables

Plant consumables include major items, such as crusher and mill liners and grinding media. Consumption rates were estimated from the half year 2009 actual consumption rates as well as benchmarking. Unit costs are ex-works and do not include freight and logistics, handling and taxes. Consumption rates and unit costs are summarised in Table 24.8.

### Table 24.8: Crusher and Mill Liner Consumption

| Item | Unit Cost ($M per set) | Current Plant Consumption | Plant Upgrade Consumption |
|---|---|---|---|
| SAG Mill Liners | 0.150 | 1.0 set per year | 1.0 set per year |
| Ball Mill Liners | 0.050 | 2.0 sets per year | 2.0 sets per year |
| Jaw Crusher Liners | 0.025 | 2.0 sets per year | 2.7 sets per year |
| Secondary Crusher Liners | 0.068 | - | 4.0 sets per year |
| Regrind Mill Liners | 0.068 | - | 1.0 set per year |

The cost of secondary crusher liners for the contract secondary crushing plant were 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 24.9.

### Table 24.9: Crusher and Mill Liner Costs

| Item | Current Plant Cost ($M) | Plant Upgrade ($M) |
|---|---|---|
| SAG Mill Liners | 0.150 | 0.150 |
| Ball Mill Liners | 0.100 | 0.100 |
| Jaw Crusher Liners | 0.050 | 0.068 |
| Secondary Crusher Liners | - | 0.274 |
| Regrind Mill Liners | - | 0.023 |
| **TOTAL $M/y** | **0.300** | **0.592** |
| **TOTAL $/t** | **0.299** | **0.433** |

Details of the grinding media and consumption rates for the SAG, ball and regrind mills are detailed in Table 24.10. The SAG and ball mill media consumption rates were estimated using the actual consumption rates for the half year 2009. The regrind mill media consumption rate was estimated by benchmarking.

### Table 24.10: 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 | 1.23 | 0.34 | 0.34 |
| Ball Mill | 75 mm | Forged | 1.16 | 0.32 | 0.32 |
| Ball Mill | 50 mm | Forged | 1.29 | 0.38 | 0.38 |
| Regrind Mill | 12 mm | Forged | 1.16 | - | 0.09 |

Table 24.11 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 24.11: Grinding Media Costs

| Item | Current Plant Cost ($M) | Plant Upgrade ($M) |
|---|---|---|
| SAG Mill Balls | 0.41 | 0.57 |
| Ball Mill Balls (75 mm) | 0.37 | 0.50 |
| Ball Mill Balls (50 mm) | 0.49 | 0.66 |
| Regrind Mill Media | - | 0.14 |
| **TOTAL $M/y** | **1.27** | **1.87** |
| **TOTAL $/t** | **1.26** | **1.36** |

## 24.2.8 Contract Secondary Crushing

The operating cost for the contract crushing plant was calculated based on a $3.95 /t of crushed ore estimation provided by MintoEx with 60% of the SAG mill feed being secondary crushed. The cost for the contract secondary crushing was used solely to establish the plant baseline operating cost. Contract secondary crushing is no longer required for the plant upgrade.

### Table 24.12: Contract Secondary Crushing Costs

| Item | Current Plant Cost ($M) | Plant Upgrade ($M) |
|---|---|---|
| Secondary Contract Crushing | 2.38 | - |
| Ore re-handle | 0.60 | - |
| **TOTAL $M/y** | **2.98** | - |
| **TOTAL $/t** | **2.97** | - |

## 24.2.9 Tails Filtration and Dry Stacking

The operating cost for the current plant tails filtration and dry stacking operation was estimated at $3.57 /t. The estimate was based on the half year 2009 actual cost as supplied by MintoEx.

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.

### 24.2.10 Assay and Metallurgical Laboratory

The operating cost for the Assay and Metallurgical Laboratory was estimated based on the half year 2009 actual cost as supplied by MintoEx. A fixed cost of $1.24 million per year was used for the operating cost calculations for both plant throughput scenarios.

## 24.3 Capital Cost Estimate

### 24.3.1 Open Pit Mine

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 open pit equipment capital costs (based on new equipment) required to achieve the target processing rate of 1.4 Mtpa is summarized in Table 24.13 below.

**Table 24.13: Open Pit Equipment Capital Cost Summary**

| Item | Unit | # units | Total |
|---|---|---|---|
| **Primary ($C)** | | | |
| Atlas Copco PV235 Crawler-Mounted Drill | $M | 2 | 2.9 |
| Atlas Copco D9-11 Crawler-Mounted Drill | $M | 1 | 0.6 |
| Hitachi EX1900 Front Shovel | $M | 2 | 6.3 |
| Caterpillar 992G Wheel Loader | $M | 1 | 2.1 |
| Caterpillar 777F, 100-ton Haul Truck | $M | 8 | 13.2 |
| Caterpillar D9T Dozer | $M | 3 | 2.9 |
| Caterpillar 16 m Grader | $M | 2 | 1.5 |
| Caterpillar 834H Rubber-tire Dozer | $M | 1 | 1.0 |
| Caterpillar 777C Water Truck | $M | 1 | 0.8 |
| Subtotal Primary | $M | | 31.2 |
| **Ancillary ($C)** | | | |
| Caterpillar 365CL Excavator | $M | 1 | 0.4 |
| Caterpillar 988H Wheel Loader | $M | 1 | 1.0 |
| Caterpillar IT38H /Cat 216B | $M | 1 | 0.4 |
| Caterpillar 777B w/trailer | $M | 1 | 0.8 |
| Subtotal Ancillary | $M | | 2.5 |
| **Total ($C)** | **$M** | | **33.7** |

# 24.4 Process Plant Capital Cost Estimation

## 24.4.1 General

The estimate was developed by applying factors to mechanical equipment supply costs. The factors were derived from data generated from previous projects for plants of similar type, materials, complexity and location.. The estimate is presented in Canadian dollars (C$) and has an overall accuracy of ±25% as of the fourth quarter 2009.

Indirect costs have been estimated based on a factor of the total direct costs established from previous projects. Table 24.14 shows a summary of the costs which exclude any escalation or foreign currency fluctuations and are current day costs only.

**Table 24.14: Cost Breakdown Summary**

| Facility | C$M |
|---|---|
| Process Plant | 6.90 |
| Spares & First Fills | 0.22 |
| Temporary Construction Facilities | 0.30 |
| EPCM | 1.68 |
| **Total** | **9.10** |

### 24.4.2 Scope of Estimate

This estimate is based on the following inclusions and exclusions:

Included in the capital cost estimate:

- Mechanical equipment costs for new process plant equipment that has not been purchased by MintoEx;
- Tailings delivery pipeline and deposition cyclones;
- Freight allowance;
- EPCM, EPCM contractors fee & commissioning costs;
- Allowance for vendor representatives;
- Infrastructure buildings as noted below;
- Allowance for capital spares, first fills and initial consumables;
- Temporary 10 person construction camp; and
- Temporary construction facilities.
- 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 such as the S4800 secondary cone crusher;
- Owner's costs for the project;
- Access roads to the plant;
- 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;
- Operating costs; and
- On-going, future or deferred capital costs.

### 24.4.3 Capital Cost Methodology

The following is a brief methodology for the determination of capital cost estimates for the Minto process plant upgrade and related ancillary infrastructure.

The Minto Phase IV plant upgrade capital cost estimate was derived by factoring the mechanical equipment costs, which are defined in the pre-feasibility study mechanical equipment list (Appendix D). Equipment costs for the major equipment were sourced directly from vendors.

All other mechanical equipment costs were based on either quotations or purchase orders from previous projects. The cost estimates for all other disciplines were factored from the mechanical equipment list using factors developed from the Ausenco database of projects.

### 24.4.4 Detailed Cost Estimate Build-up

The estimated capital cost for the Minto plant upgrade and non-mining infrastructure has been produced as follows.

- The mechanical equipment and installation costs for the secondary crushing circuit and gravity gold concentrator were not included as these have been purchased by MintoEx and will not form part of the project budget;
- Major mechanical equipment budget costs were sourced directly from suppliers. This includes the Vertimill, flotation cells and the flotation concentrate thickener overflow water clarifier;
- Other mechanical equipment costs for smaller equipment such as pumps and the regrind cyclones have been selected from similar installations or estimates and modified as required to suit the Minto plant requirements; and
- The equipment was selected on an area by area basis to suit the respective process requirements of each area.

Plant Site Development for the new flotation/regrind mill building includes bulk earthworks and drainage. This was included as a provisional cost item due to the lack of geotechnical data for the specific site area. The PC sum used has been established from previous detailed estimates and completed projects with typically expected geotechnical conditions. Factors for undertaking earthworks in rock have not been included in the estimate pricing.

Concrete was estimated from preliminary material takeoffs generated from the general arrangement drawings. An all inclusive rate of $2,000 was used for footings and pedestals and $1,500 for slabs and bund walls.

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 meters of high density poly-ethylene pipe. The deposition costs were estimated based on the use of five cyclones (10 inch inlet diameter cyclones).

EPCM costs were included in the estimate based on a factor 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.

Infrastructure building costs for the new flotation/regrind mill building were included based on preliminary quantity takeoffs from the general arrangement drawings. Rates have been used based on data from previous other similar projects.

A 10 man temporary construction camp was factored into the estimate. This assumes that the existing Minto camp messing, water supply and sewage treatment facilities are able to meet the increased demand during the construction period. The cost includes mobilisation, de-mobilisation, rental costs per unit, and shower/ablution facilities.

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.

Mobile equipment required during the construction phase for the plant upgrade has been included in the site mechanical installation costs. The factored estimate is based on previous detailed estimates and completed projects.

An allowance was made for temporary construction facilities and is based upon Ausenco's previous experience with projects of a similar size and location.

EPCM, Start Up and Commissioning Costs are also included in the estimate. An allowance has been included for two Senior Commissioning Process Engineers, one Senior Electrical Engineer and one Senior Mechanical engineer to be on site for a total of two weeks to commission the installation.

### 24.4.5 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 30$^{th}$ of October 2009.

The estimate is expressed in Canadian Dollars.

For reference, the currency conversions rates used during the estimate preparation are:

- C$ 1.00 = US$ 0.90

#### 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.

### 24.4.6 Contingency

Contingency is not included in the PFS capital cost estimates, however, a 10% contingency was applied in the cash flow analysis for mine equipment, construction facilities and EPCM.

### 24.4.7 Owner's Costs

Owner's costs have been excluded from this estimate.

### 24.4.8 Project Fee

A project fee of 3% of the direct costs was included.

A Fee is a notional allowance considered chargeable by any reputable Engineer or Project Managers as profit and takes in to account the type of project, project location, project value and project risk.

This allowance also considers the Engineer/Project Managers liabilities for such items as process guarantees, liquidated damages, indemnity insurance and other such liabilities.

In most cases the Fee is calculated as a percentage of the overall cost of the project or in some cases may be negotiated as a fixed sum depending on the extent of risk and liability the project owners are prepared to accept.

### 24.4.9 Escalation

Escalation provision past the fourth quarter 2009 was not included in the estimate.

# 25 Economic Analyses

## 25.1 Assumptions

A financial model was compiled by SRK based on the Minto Main LOM operations plan and the 2010 Minto Mine budget. The model includes taxation but excludes financing costs (debt principal and interest). Net annual cash flows were calculated by considering net smelter return from the payable Cu, Au and Ag metals, and then deducting the operating costs, capital costs and applicable taxes.

The metal prices in the analyses were estimated from the existing Minto sales contracts as shown in Table 25.1 and non-contract copper price assumptions shown in Table 25.2. Gold and silver prices are based on Minto's contract with Silver Wheaton, in which all gold and silver is sold at fixed prices of US$300/oz and US$3.90/oz respectively.

All three cases used the same mineral reserve, cost and production parameters. A summary of the average operating costs by area, as well as the capital costs are summarized in Tables 25.3 and 25.4 below.

### Table 25.1: Summary of Forward Sales Contract Metal Pricing for All Cases

| Metal | Units | 2010 | 2011 | 2012+ |
|---|---|---|---|---|
| Copper | US$/lb | 2.19 | 2.26 | |
| Gold | US$/oz | 300 | 300 | 300 |
| Silver | US$/oz | 3.90 | 3.90 | 3.90 |

### Table 25.2: Non-contract Copper Pricing Assumptions for All Cases

| Case | Units | 2010-2018 |
|---|---|---|
| 1 | US$/lb | 2.25 |
| 2 | US$/lb | 2.60 |
| 3 | US$/lb | 3.00 |

**Table 25.3: Summary of Operating Costs by Major Area for All Cases**

| Area | C$/t |
|---|---|
| Mining (C$/t moved) | 2.31 |
| Mining (C$/t ore) | 17.02 |
| Processing | 13.90 |
| General, administration, camp | 11.94 |
| **Total** | **42.86** |

**Table 25.4: Summary of Capital Costs for All Cases**

| Area | C$ millions |
|---|---|
| Plant Expansion | 9.1 |
| Open pit minig equipment | 33.7 |
| **Sub-total** | **42.8** |
| Sustaining Capital | 5.4 |
| **Life-of-mine capital** | **48.2** |

The other main economic factors used in the cash flow analysis were:

- C$:US$ exchange rate of: 1.10:1;
- A discount rate of 7.5%;
- Variable metal pricing;
- Closure allowance of $20M;
- Nominal 2009 dollars; and
- No inflation.

Costs, revenues and taxes were calculated for each period in which they occurred rather than at the actual date of payment.

## 25.2 Economic Results

The LOM financial results common to all three cases are shown in Table 25.5. Financial results specific to each case are shown in Table 25.6.

It must be noted that the net present value ("NPV") calculations in the financial model were done using 2010 as the starting year and do not take into account approximately $150 m in capital spent for initial plant and mine construction nor the revenue derived from operations from 2007-2009. This methodology only looks at the project going forward from the beginning of 2010 and, therefore, shows high returns.

Case 1 assumed a constant copper price of US$2.25/lb, equivalent to the long-term price forecast by many financial institutions. The results show the NPV at a 7.5% discount rate ("$NPV_{7.5\%}$") to be $160 m after tax and $199 m before tax. The total amount of tax paid is $53 m (undiscounted). Table 25.3 shows the NPV results at various discount rates. Case 1 clearly produces a very robust outcome at a copper price of $2.25/lb. The cash operating cost for Case 1 is $1.31/lb Cu not including by-product (Au and Ag) credits and $1.16/lb Cu with by-product credits.

### Case 1

#### Table 25.5: Discount Factors and Related Net Present Values for Case 1

| Discount Rate | NPV Pre-tax cash flow ($M) | NPV After tax cash flow ($M) |
|---|---|---|
| 0% | $252 | $200 |
| 7.5% | $199 | $160 |
| 10% | $186 | $150 |
| 15% | $163 | $132 |

### Case 2

Case 2 utilizes an assumed constant unhedged copper price of US$2.60/lb, with the NPV results as shown in Table 25.6.

#### Table 25.6: Discount Factors and Related Net Present Values for Case 2

| Discount Rate | NPV Pre-tax cash flow ($M) | NPV After tax cash flow ($M) |
|---|---|---|
| 0% | $370 | $274 |
| 7.5% | $291 | $218 |
| 10% | $270 | $203 |
| 15% | $236 | $179 |

### Case 3

Case 2 utilizes an assumed constant unhedged copper price of US$3.00/lb, with the NPV results as shown in Table 25.7.

#### Table 25.7: Discount Factors and Related Net Present Values for Case 3

| Discount Rate | NPV Pre-tax cash flow ($M) | NPV After tax cash flow ($M) |
|---|---|---|
| 0% | $505 | $356 |
| 7.5% | $395 | $281 |
| 10% | $366 | $262 |
| 15% | $320 | $231 |

**Table 25.8: Summary of Economic Model Results for All Three Cases**

| | | YEAR | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Item** | **Unit** | **2010** | **2011** | **2012** | **2013** | **2014** | **2015** | **2016** | **2017** | **2018** | **Total/ Ave.** |
| **MINING** | | | | | | | | | | | |
| Waste mined | Ktonnes | 8,180,907 | 6,346,855 | 9,370,518 | 7,180,870 | 10,170,193 | 8,900,076 | 11,584,697 | 8,642,263 | - | **61,734,115** |
| Ore mined | Ktonnes | 1,972,466 | 1,280,207 | 281,983 | 1,405,934 | 1,183,230 | 1,418,557 | 1,333,193 | 1,144,464 | - | **8,875,570** |
| Total mined | Ktonnes | 10,153,373 | 7,627,062 | 9,652,501 | 8,586,804 | 11,353,423 | 10,318,633 | 12,917,889 | 9,786,727 | | **70,609,685** |
| **MILLING** | | | | | | | | | | | |
| Mill Feed | Ktonnes | 1,216,900 | 1,368,750 | 1,372,500 | 1,368,750 | 1,368,750 | 1,368,750 | 1,372,500 | 1,368,750 | 87,668 | **9,436,900** |
| Mill Feed Rate | t/d | 3,334 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,750 | 3,746 | **3,698** |
| Copper millhead grade | % Cu | 2.33 | 1.68 | 1.10 | 2.47 | 1.22 | 1.44 | 1.40 | 1.64 | 0.81 | **1.64** |
| Gold millhead grade | g/t Au | 0.80 | 0.67 | 0.35 | 1.27 | 0.40 | 0.52 | 0.50 | 0.65 | 0.25 | **0.64** |
| Silver millhead grade | g/t Ag | 9.84 | 6.48 | 3.64 | 8.88 | 3.66 | 5.32 | 4.44 | 5.52 | 2.67 | **5.9** |
| Copper recovery to cons | % | 94% | 94% | 94% | 92% | 92% | 92% | 92% | 92% | 92% | **93%** |
| Gold recovery to cons | % | 80% | 80% | 78% | 70% | 71% | 70% | 70% | 72% | 70% | **74%** |
| Silver recovery cons | % | 87% | 87% | 85% | 78% | 79% | 78% | 78% | 80% | 78% | **81%** |
| Copper in cons | Mlb | 59 | 48 | 31 | 69 | 34 | 40 | 39 | 46 | 1 | **366** |
| Copper in cons | tonnes | 26,598 | 21,648 | 14,158 | 31,097 | 15,412 | 18,111 | 17,697 | 20,730 | 655 | **166,105** |
| Gold in cons | Koz | 24,961 | 23,470 | 12,163 | 39,168 | 12,529 | 16,028 | 15,594 | 20,407 | 494 | **164,814** |
| Silver in cons | Koz | 333,701 | 247,310 | 136,463 | 304,882 | 127,345 | 182,541 | 153,122 | 193,450 | 5,874 | **1,684,688** |
| Concentrate tonnes | dmt | 63,328 | 51,543 | 34,383 | 81,833 | 40,029 | 47,661 | 46,460 | 53,533 | 1,723 | **420,494** |
| Concentrate grade | % Cu | 42% | 42% | 41% | 38% | 39% | 38% | 38% | 39% | 38% | **40%** |
| **NET SMELTER RETURN** | | | | | | | | | | | |
| Payable copper | Mlb | 57 | 46 | 30 | 66 | 33 | 39 | 38 | 44 | 1 | **354** |
| Payable copper | tonnes | 25,733 | 20,944 | 13,698 | 30,086 | 14,911 | 17,523 | 17,122 | 20,056 | 634 | **160,706** |
| Payable gold | Koz | 24 | 23 | 12 | 38 | 12 | 16 | 15 | 20 | 0 | **160** |
| Payable silver | Koz | 273 | 198 | 103 | 226 | 89 | 137 | 108 | 142 | 4 | **1,279** |
| Exchange rate | C$/US$ | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | **1.10** |
| Transport cost | $M | 9.3 | 7.6 | 5.1 | 12.0 | 5.9 | 7.0 | 6.8 | 7.9 | 0.3 | **61.9** |
| TC/RC | $M | 6.3 | 5.1 | 3.3 | 7.7 | 3.8 | 4.5 | 4.3 | 5.1 | 0.2 | **40.2** |
| **UNIT OPERATING COSTS** | | | | | | | | | | | |
| Mining | $/t mined | 3.35 | 2.53 | 2.11 | 2.11 | 2.11 | 2.11 | 2.11 | 2.11 | 2.11 | **2.31** |
| | $/t milled | 27.95 | 14.10 | 14.84 | 13.24 | 17.50 | 15.91 | 19.86 | 15.09 | - | **17.02** |
| Milling cost | $/t milled | 18.56 | 16.50 | 12.79 | 12.79 | 12.79 | 12.79 | 12.79 | 12.79 | 12.79 | **13.90** |
| Camp services | $/t milled | 2.60 | - | - | - | - | - | - | - | - | **0.29** |
| Site services (power, barge, road) | $/t milled | 5.28 | 3.30 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | **3.10** |
| Technical services | $/t milled | 2.43 | 0.43 | 0.36 | 0.36 | 0.36 | 0.36 | 0.36 | 0.36 | 0.36 | **0.60** |
| Administration | $/t milled | 8.99 | 8.63 | 7.19 | 7.19 | 7.19 | 7.19 | 7.19 | 7.19 | 7.19 | **7.57** |
| Total OPEX (ex royalty) | $/t milled | 65.81 | 42.96 | 37.93 | 36.33 | 40.59 | 39.00 | 42.95 | 38.18 | 23.09 | **42.86** |
| Royalties (0.5%) | $/t milled | 0.54 | 0.40 | 0.25 | 0.57 | 0.27 | 0.32 | 0.32 | 0.37 | 0.18 | **0.38** |
| Opex (inc. royalties) | $/t milled | 66.35 | 43.36 | 38.18 | 36.90 | 40.87 | 39.32 | 43.26 | 38.55 | 23.27 | **42.86** |
| Unit On-site OPEX (inc. royalties) | US$/lb Cu payable | 1.29 | 1.17 | 1.58 | 0.69 | 1.55 | 1.27 | 1.43 | 1.08 | 1.33 | **1.20** |
| Unit Off-site OPEX | US$/lb Cu payable | 0.27 | 0.27 | 0.28 | 0.30 | 0.29 | 0.30 | 0.30 | 0.29 | 0.30 | **0.29** |
| Unit By-product Credit | US$/lb Cu payable | 0.15 | 0.16 | 0.13 | 0.19 | 0.12 | 0.13 | 0.13 | 0.15 | 0.11 | **0.15** |
| Unit OPEX net by-product credits | US$/lb Cu payable | **1.42** | **1.28** | **1.73** | **0.81** | **1.72** | **1.43** | **1.59** | **1.23** | **1.51** | 1.34 |
| Total Lease, Interest and AEIDA | $/lb Cu | 0.03 | 0.07 | 0.12 | 0.10 | 0.17 | 0.11 | 0.10 | 0.08 | - | **0.09** |
| **CAPITAL COSTS** | | | | | | | | | | | |
| Initial and sustaining | $M | 24.4 | 22.3 | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 | - | - | **48.2** |
| Closure allowance | $M | | | | | | | | | 20.0 | **20.0** |

### Table 25.9: Economic Results by Case (undiscounted cash flow)

| Item | Unit | YEAR 2010 | 2011 | 2012 | 2013 | 2014 | 2015 | 2016 | 2017 | 2018 | Total/ Ave. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **CASE 1** | | | | | | | | | | | |
| Copper Price (inc. hedging) | US$/lb | 2.22 | 2.25 | 2.25 | 2.25 | 2.25 | 2.25 | 2.25 | 2.25 | 2.25 | **2.25** |
| Gold price (inc. hedging) | US$/oz | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | **300.00** |
| Silver price (inc. hedging) | US$/oz | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | **3.90** |
| Smelter revenue | $M | 147.7 | 122.8 | 79.1 | 177.7 | 85.7 | 101.3 | 98.9 | 116.6 | 3.6 | **933** |
| NSR | $'M | 130.6 | 108.9 | 69.8 | 155.9 | 75.1 | 88.7 | 86.6 | 102.3 | 3.2 | **821.1** |
| NET OPERATING INCOME | $M | 48.3 | 46.1 | 13.7 | 98.8 | 13.3 | 30.3 | 23.1 | 45.8 | 1.1 | **320.6** |
| **TAXES** | | | | | | | | | | | |
| Taxes paid (recovered) | $M | 2.5 | 1.5 | 0.0 | 20.3 | 4.9 | 8.7 | 6.5 | 14.4 | (6.3) | **52.6** |
| **CASH FLOW** | | | | | | | | | | | |
| Pre-Tax cash Flow | $M | 24 | 24 | 13 | 99 | 13 | 30 | 23 | 46 | (19) | **252** |
| After tax cash flow | $M | 22 | 22 | 13 | 78 | 8 | 21 | 16 | 31 | (13) | **200** |
| **CASE 2** | | | | | | | | | | | |
| Copper Price (inc. hedging) | US$/lb | 2.40 | 2.47 | 2.60 | 2.60 | 2.60 | 2.60 | 2.60 | 2.60 | 2.60 | **2.55** |
| Gold price (inc. hedging) | US$/oz | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | **300.00** |
| Silver price (inc. hedging) | US$/oz | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | **3.90** |
| Smelter revenue | $M | 158.9 | 133.6 | 90.7 | 203.2 | 98.3 | 116.2 | 113.4 | 133.6 | 4.2 | **1,052** |
| NSR | $M | 141.7 | 119.6 | 81.5 | 181.5 | 87.7 | 103.6 | 101.1 | 119.4 | 3.7 | **939.8** |
| NET OPERATING INCOME | $M | 59.4 | 56.8 | 25.2 | 124.2 | 25.9 | 45.1 | 37.6 | 62.8 | 1.7 | **438.6** |
| **TAXES** | | | | | | | | | | | |
| Taxes paid (recovered) | $M | 5.1 | 3.5 | 0.0 | 39.5 | 8.7 | 13.9 | 11.3 | 20.7 | (6.2) | **96.5** |
| **CASH FLOW** | | | | | | | | | | | |
| Pre-Tax cash Flow | $M | 35 | 35 | 25 | 124 | 26 | 45 | 37 | 63 | (18) | **370** |
| After tax cash flow | $M | 30 | 31 | 25 | 84 | 17 | 31 | 26 | 42 | (12) | **274** |
| **CASE 3** | | | | | | | | | | | |
| Copper Price (inc. hedging) | US$/lb | 2.60 | 2.71 | 3.00 | 3.00 | 3.00 | 3.00 | 3.00 | 3.00 | 3.00 | **2.90** |
| Gold price (inc. hedging) | US$/oz | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | **300.00** |
| Silver price (inc. hedging) | US$/oz | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | **3.90** |
| Smelter revenue | $M | 171.6 | 145.8 | 104.0 | 232.4 | 112.8 | 133.2 | 130.0 | 153.1 | 4.8 | **1,188** |
| NSR | $M | 154.5 | 131.9 | 94.8 | 210.6 | 102.2 | 120.6 | 117.7 | 138.8 | 4.3 | **1,075.4** |
| NET OPERATING INCOME | $M | 72.0 | 69.0 | 38.5 | 153.3 | 40.3 | 62.1 | 54.1 | 82.1 | 2.3 | **573.6** |
| **TAXES** | | | | | | | | | | | |
| Taxes paid (recovered) | $M | 8.1 | 5.8 | 5.9 | 58.4 | 11.9 | 20.1 | 17.3 | 28.1 | (6.0) | **149.7** |
| **CASH FLOW** | | | | | | | | | | | |
| Pre-Tax cash Flow | $M | 48 | 47 | 38 | 153 | 40 | 62 | 54 | 82 | (18) | **505** |
| After tax cash flow | $M | 40 | 41 | 32 | 95 | 28 | 42 | 37 | 54 | (12) | **356** |

## 25.3 Sensitivities

The project was evaluated for sensitivity to the operating costs, capital costs, grade and metal price. All sensitivities were assessed for the range of -20% to +20% with the resulting $NPV_{7.5\%}$ value shown with the base case. Figures 25.1 to 25.3 show the graphical results of the sensitivity analysis.

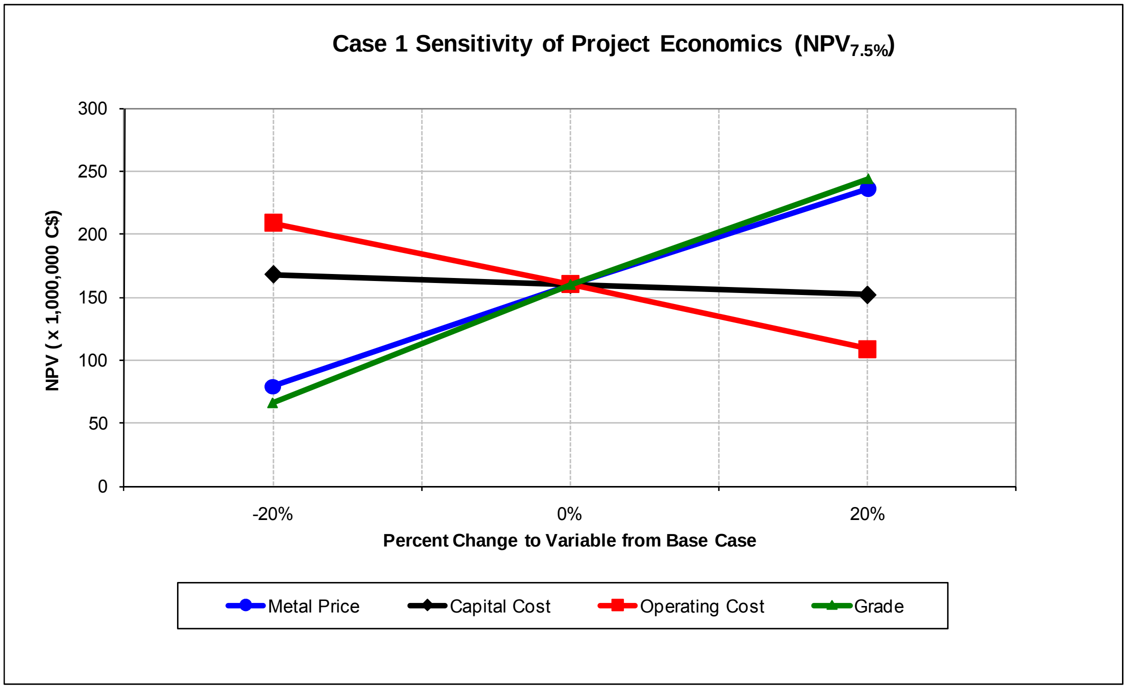
All sensitivities were done as mutually exclusive variations. A combination of variable changes was not conducted nor was an analysis of the probability of any variations.

Both the pre-tax and after taxation cash flow models show the project is most sensitive to changes to the Cu grade. This sensitivity is somewhat mitigated in the mine plan by the significant use of stockpiles to allow the early extraction 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 1 a 20% drop in Cu grade yields a $94 M (59%) drop in after-tax $NVP_{7.5\%}$. Diligent grade control practices will be important in achieving undiluted mill feed, especially in Area 2 where the mineralized zones are smaller and more numerous than is found in the Main Pit.

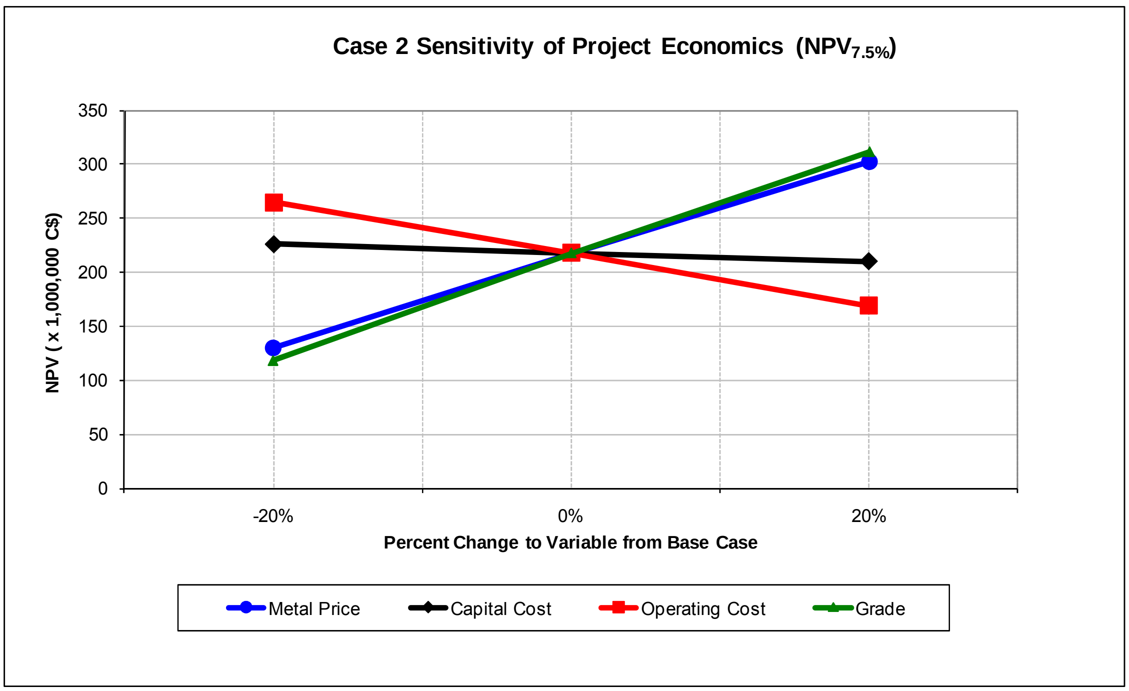
Metal prices demonstrate the second greatest sensitivity. In Minto's case, the metal prices are buffered somewhat by the fact that a portion of its copper production is hedged until late 2011 (with gold and silver fixed throughout) so a reduction or increase in the market price has a tempered affect on the NPV. Even with this forward sale arrangement, a 20% decrease or increase in Cu price changes the after-tax $NPV_{7.5\%}$ by approximately 50%.

A 20% reduction in OPEX yields a $49 M (31%) increase in after-tax $NPV_{7.5\%}$. Some of Minto's operating expenses including , TCs and RCs and concentrate transport are covered by contracts and, therefore, offer some protection from variances in the next several years. On the other hand, a 20% increase in OPEX yields a $50 m (31%) decrease in after-tax $NPV_{7.5\%}$. The mining OPEX used in this report is based on an owner-operated fleet and presents a significant change from the current contract mining scenario, both operational and in terms of predicted costs.

As most of the capital expenses have already been incurred, the project has a limited sensitivity to CAPEX.




## Figure 25.1: Case 1 Sensitivities




## Figure 25.2: Case 2 Sensitivities

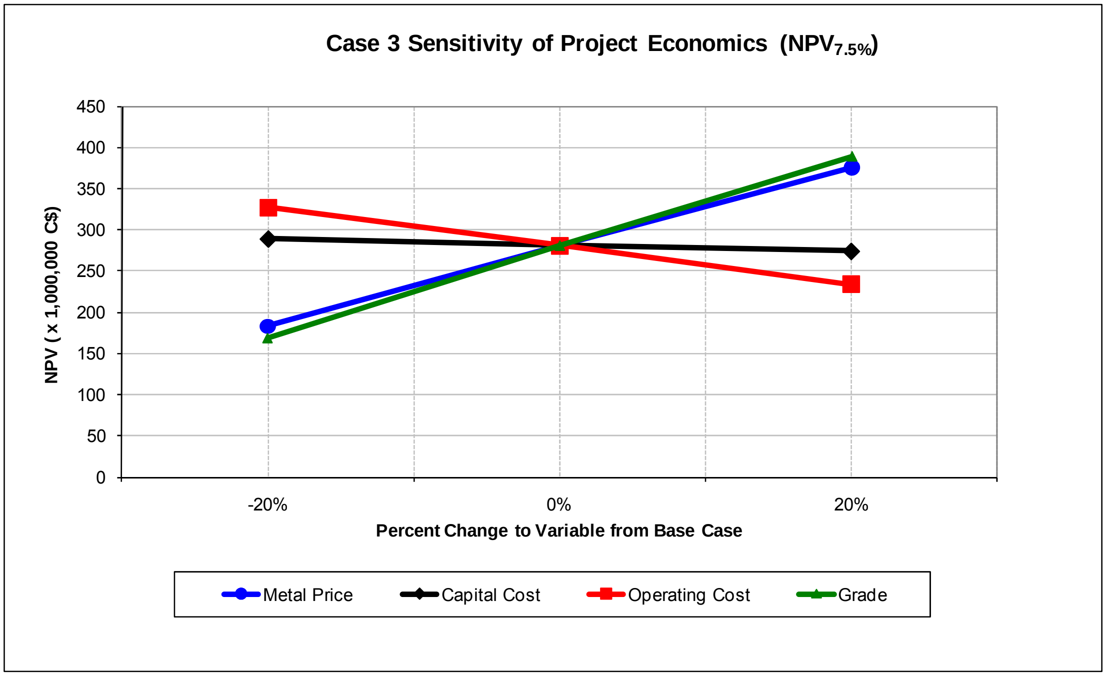



**Figure 25.3: Case 3 Sensitivities**

### 25.3.1 Comments

#### Metal Price

Regardless of which tax or pre-tax model is selected, changes to the metal grades and prices each contribute to make the most impact on the cash flow for the project. Considering that some of the mine production has hedged prices and the current strong metal price trend, any negative effect on NPV resulting from a drop in the price of Cu, Au and Ag seems to have a low near-term probability.

#### Grade

Changes to the grade of Cu represent the project's greatest economic vulnerability and the variable that can also be affected by internal efforts associated with mining and milling operations. With the high profile and mandate related to the management of the stockpiled ore, and the concerted effort to optimize recovery grades, the mitigation of the grade risk has been woven into the project strategy.

#### Tonnage

Changes to tonnage are only expected should grade control and waste mining require modification. The mine plan that has been modelled is the result of stringent efforts and continued monitoring of mining operations can alleviate risks associated with tonnages. The potential conversion of inferred resources to higher classifications could have a positive impact on overall results

#### Foreign Exchange

The effect of foreign exchange can have a significant negative or positive impact on the project cash flow, mainly because Minto's revenue contracts are fixed in US dollars while operating costs, taxes, and capital are in Canadian dollars. The magnitude of this impact is measured from the 1.10 C$:US$ average exchange rate used in the economic model and the actual exchange rate affecting the project.

## 25.4 Payback

The payback on all capital spending shown in this report will be within 2010 due to large cash flows and relatively minimal capital expenditures planned. This assumes all previous capital spent on the initial project construction are sunk costs.

## 25.5 Mine Life

With the current mineral reserve estimates used in the LOM plan, the mine operation will end in the first quarter of 2018. Open pit mining is estimated to finish in the last quarter of 2017 and the mill will continue to run for a short period on stockpiled material after mining ceases.

It is SRK's opinion that there is potential for the mine to extend its life if additional resources can be turned into reserves. There is no guarantee that this will happen, however, Minto has experienced a high level of success with its exploration drilling and there are still several open pit and underground exploration targets that require further drilling.

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 allowing more resources to be converted to reserves. It must be noted that, as per CIM guidelines, resources can only be converted to reserves if they are supported by at least a preliminary feasibility study.

# 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.

### 26.1.1 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 cone 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.

### 26.1.2 Crushed Ore Stockpile and Reclaim Risks and Opportunities

Previously around 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.

### 26.1.3 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; and

- 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.

### 26.1.4 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.

### 26.1.5 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).

### 26.1.6 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.

During years 2010 and 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.

### 26.1.7 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

### 26.3.1 Conclusions

- The Minto deposit, encompassing Main Pit and Phase IV pits (Area 2, North, 118 and Ridgetop), 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 IV deposits are estimated to be economic to exploit 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 IV pits.
- Based on test work conducted to date, the Phase IV waste rock does not appear to have any ARD issues.

### 26.3.2 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 affect on project return.
- Grade control: The Phase IV 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.

### 26.3.3 Opportunities

The most significant opportunities that should be investigated are listed below:

- 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. An 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.
- Underground mine potential.
- Relatively quick access to high grade zones
- Minimum footprint and environmental requirements for permitting and closure
- Provide overall versatility on ore extraction and throughput requirements
- Exploration target potential.
- Large Open Pit Potential.

#### *Introduction*

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 and lower operating costs were used to understand these potential pit limits.

Table 26.3 below compares the parameters that were modified for this large open pit option versus those used for the remainder of this report.

**Table 26.3: Open Pit Optimization Parameters**

| Item | Unit | PFS | Large Pit |
|---|---|---|---|
| Metal Price | US$/lb Cu | 2.00 | 3.00 |
| Mining Cost | C$/mined tonne | 2.11 | 1.80 |
| Processing and G&A Cost | C$/milled tonne | 23.09 | 18.00 |
| Processing rate | t/day milled | 3,750 | 7,500 |

A revised NSR model was created based on the copper price of $3.00/lb 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. 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.

### *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 (for measured and indicated material).

## Table 26.4: Large Pit Results

| Area2/118/Ridge/North Pits | Diluted | Waste | Total material | Strip ratio | Diluted grade | | | Contained Metal | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | ktonnes | ktonnes | ktonnes | (t:t) | Cu % | Au (g/t) | Ag (g/t) | Cu (Mlbs) | Au (koz) | Ag (koz) |
| **PFS** | | | | | | | | | | |
| Minto North Pit | 1,349 | 10,626 | 11,975 | 7.9 | 2.50 | 1.37 | 9.05 | 74 | 60 | 393 |
| Ridgetop Pit | 1,324 | 9,011 | 10,335 | 6.8 | 1.26 | 0.38 | 3.34 | 37 | 16 | 142 |
| Area2/118 Pit | 4,094 | 36,217 | 40,310 | 8.8 | 1.51 | 0.56 | 5.12 | 136 | 73 | 674 |
| **PFS Total** | **6,767** | **55,854** | **62,621** | **8.3** | **1.66** | **0.68** | **5.56** | **248** | **149** | **1209** |
| **$3.00 Cu + 7500tpd** | | | | | | | | | | |
| Minto North Pit | 1,994 | 13,827 | 15,820 | 6.9 | 2.01 | 1.03 | 7.22 | 88 | 66 | 463 |
| Ridgetop Pit | 5,172 | 21,097 | 26,269 | 4.1 | 0.77 | 0.23 | 2.22 | 88 | 38 | 369 |
| Area2/118 Pit | 22,244 | 157,249 | 179,493 | 7.1 | 0.83 | 0.27 | 2.83 | 407 | 193 | 2,024 |
| **$3.00 Cu + 7500tpd** | **29,410** | **192,173** | **221,583** | **6.5** | **0.90** | **0.31** | **3.02** | **583** | **297** | **2856** |

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 the report. The pits above also contain 2,950 kt of inferred material at a copper grade of 0.54% Cu, which, if with further drilling, are converted into measured or indicated resource category, would improve the strip ratios. Figure 26.1 below further provides a plan view of the resulting pit limits versus the PFS designs and illustrates the impact of the potential foot print of the pits.

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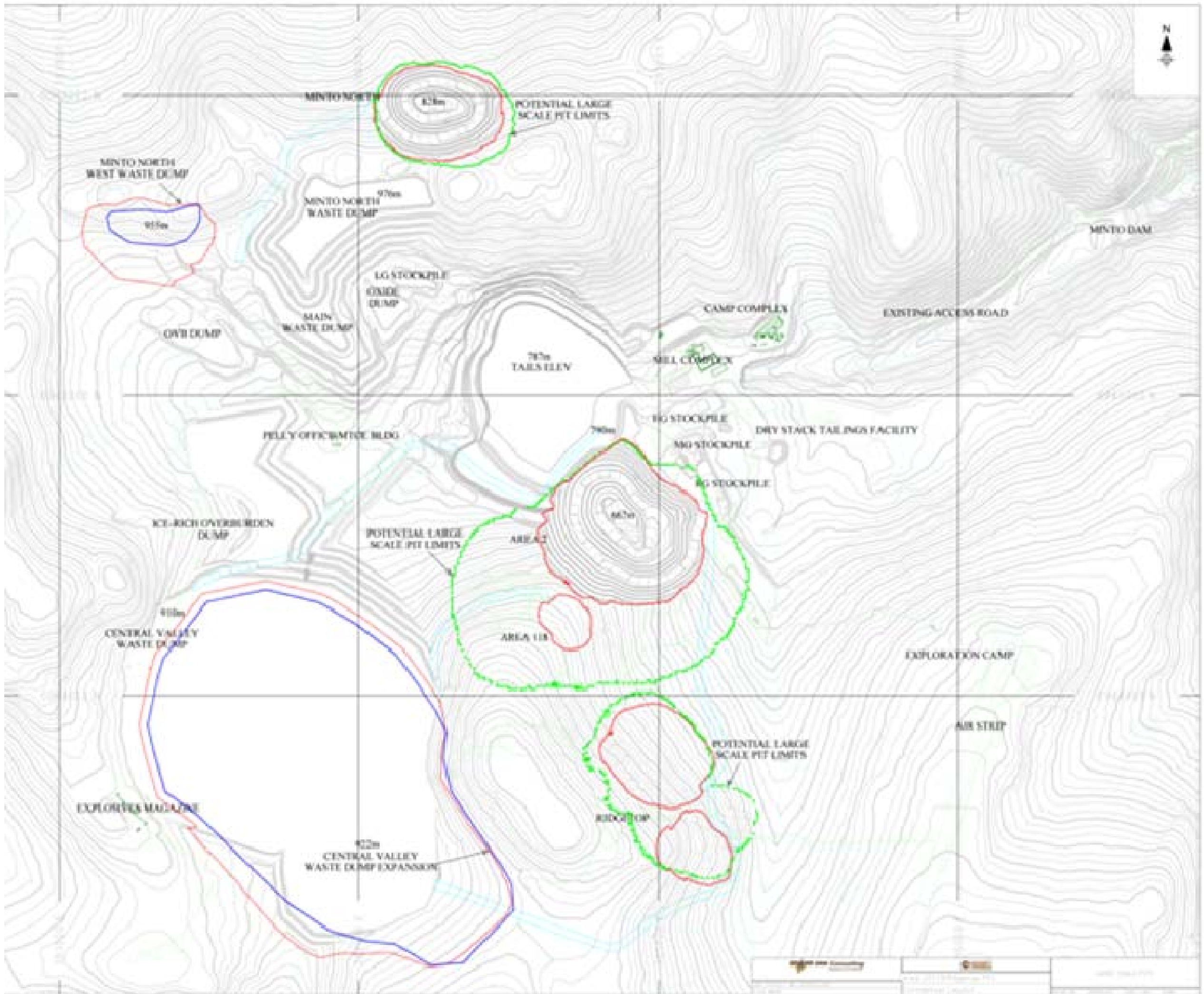


**Figure 26.1: Large Scale Pit Extents**

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 increased waste produced for Minto North pit could be handled by creating another dump further west of the proposed North dump outlined in this report (see Figure 17.3). The increased waste storage capacity required for Ridgetop, Area 2 and118 pits could be mitigated somewhat by additional lifts on the proposed Central Valley Dump (potentially could provide further 25 mt of capacity), however other large dump locations would need to be identified in order to handle the remaining 110 mt of potential waste material.

In terms of tailings disposal, additional storage capacity could be provided by placing the thickened tailings in the mined out Ridgetop pit as well as in the eastern, lower portion of Area 2. This has the potential of providing approximately half of the additional tailings storage required. Further studies would need to identify additional tailings storage options to make up this potential shortfall in storage capacity.

## 26.4 In-pit Tailings Disposal – Conclusions, Risks and Opportunities

### 26.4.1 Conclusions

- In-pit tailings disposal methods can be used to store the entire volume of tailings associated with the development of the Area 2 Pit in the Main Pit.
- In order to achieve the fulfill the storage requirements, a divider embankment comprised of approximately 2.1 million cubic metres of waste rock and/or overburden would have to be constructed within the limits of the Main Pit.
- The divider embankment may 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, and followed by up to three stages of annual raises, each of which would be approximately 10 m thick.
- 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 perimeter 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 pit using a floating barge located in the northeast quadrant of the pit 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.

### 26.4.2 Risks

- Storage of water and tailings behind the divider embankment in the Main Pit could lead to unexpected developments which significantly reduce the safety of personnel working in the Area 2 Pit and subsequently require the implementation of potentially expensive remediation methods.
- 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.

### 26.4.3 Opportunities

- The water collected in the wells which are likely to be installed in the divider embankment 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 Main Pit.
- Once the use of the Main Pit 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.

### 27.1.1 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.

### 27.1.2 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 $300 k.

## 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 underground mining potential are under-explored;
- Optimization studies should be conducted to smooth out the mill-feed grade profile and the mining schedule (in particular, the transition period from the completion of the Main Pit to the commencement of the Phase IV pits);
- More work should be conducted on the underground potential of Area 118 with the objective of estimating an underground mineral reserve;
- An open pit/underground cross-over study should be completed for the lower lens of the Area 2 mineral resource to determine the best mining method (open pit or underground) for this lens.

## 27.3 Additional Characterization for In-pit Tailings Disposal

### 27.3.1 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)

### 27.3.2 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)

### 27.3.3 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)

### 27.3.4 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

All illustrations are included in the report.

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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** | |
|---|---|
| $^oC$ | degree Celsius |
| $^oF$ | 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, 2009.

| Qualified Person | Signature | Date | Responsible Section |
|---|---|---|---|
| Cam Scott, P.Eng | | December 15, 2009 | 18.6, 26.4, and 27.3 |
| Clint Donkin, AusIMM | | December 15, 2009 | 15,19,24.2,24.4,26.1 and 27.1 |
| Dino Pilotto, P.Eng | | December 15, 2009 | 16.6, 18.1, 18.3 to 18.5, 24.1, 24.3, 26.3 and 27.2 |
| Gordon Doerksen, P.Eng | | December 15, 2009 | Executive Summary, 1 to 4, 14, 17,20,21,23, all parts of 24 and 25, 26 and 27 not claimed by other QP's, and 28 to 31 |
| Garth Kirkham, P.Geoph. | | December 15, 2009 | 12,13 |
| Mike Levy, PE | | December 15, 2009 | 18.2 |
| Wayne Barnett, P.Eng | | December 15, 2009 | 6,7,8,9,10,11 and 16 |

## CERTIFICATE OF QUALIFIED PERSON

**Cameron C. Scott, P. Eng.**
1722 W. 58th Avenue
Vancouver, BC V6P 1W9

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 IV, Pre-Feasibility Technical Report*" dated 15 December 2009.

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 18.6, 26.4 and 27.3 of "*Minto Phase IV, Pre-Feasibility Technical Report*".

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 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.

"Signed and sealed"

ORIGINAL SIGNED AND
SEALED

Cameron C. Scott, P.Eng.

Dated: 15 December 2009

**CERTIFICATE OF QUALIFIED PERSON**

***Clinton Donkin, MAusIMM***
Ausenco Minerals Canada Inc.
Suite 605
375 Water Street
Vancouver B.C.
Canada V6B 5C6

I, Clinton Donkin, MAusIMM certify that I am a Senior Metallurgist for Ausenco Minerals Canada Inc. Suite 605, 375 Water St. Vancouver, British Columbia, V6B 5C6, Canada.

This certificate applies to the technical report titled "*Minto Phase IV, Pre-Feasibility Technical Report*" dated December 15, 2009.

I am a member of the Australian Institute of Mining and Metallurgy (AusIMM). I graduated with a Bachelor of Mineral Process Engineering from the University of Queensland in 2002.

I have practiced my profession continuously since January, 2003 and have been involved in processing plant operations, design and commissioning.

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 Project site.

I am responsible for Sections 15, 19, 24.2, 24.4, 26.1 and 27.1 of "*Minto Phase IV, Pre-Feasibility Technical Report*".

I am independent of Minto Explorations Ltd. as independence is described by Section 1.4 of NI 43-101.

I have not had 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 SEALED

__________________________________

Clinton Donkin, MAusIMM

Dated: December 15, 2009

## CERTIFICATE OF QUALIFIED PERSON

***Dino Pilotto, P.Eng.***
#205-2100 Airport Drive, Saskatoon, SK S7T 0C8

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 IV, Pre-Feasibility Technical Report*" dated December 15, 2009.

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 twenty one years experience in open pit and underground mining operations encompassing technical, production and management roles. I have experience with a variety of commodities at locations in North America, South America, and Africa. I have three years experience as a consultant, conducting and managing all levels of technical studies and reviews.

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 most recently on September 21-22, 2009.

I am responsible for Sections 16.6, 18.1, 18.3-18.5, 24.1, 24.3, 26.3 and 27.2 of "*Minto Phase IV, Pre-Feasibility Technical Report*".

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 prior to the undertaking of the PFS described in this Technical Report, in terms of providing mine planning services for the Minto Main pit; involved in preparation of the Area 2 Pre-feasibility Study dated November 30, 2007; involved in preparation of Technical Report Minto Mine, Yukon dated June 30, 2008; and involved in preparation of Minto Mine Technical Report, Yukon for Silverstone Resources dated December 15, 2008.

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 SEALED

Dino Pilotto, P.Eng.

Dated: December 15, 2009

SRK Consulting (Canada) Inc.
Suite 2200 – 1066 West Hastings Street
Vancouver, B.C. V6E 3X2
Canada

vancouver@srk.com
www.srk.com

**Tel:** **604.681.4196**
**Fax:** **604.687.5532**

# 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 IV Pre-feasibility Technical Report" dated December 15, 2009 ("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 not claimed by other QPs, 25 and 28 to 31 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 SEALED

Gordon Doerksen, P.Eng. Dated: December 15, 2009



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Tucson 520.544.3688

## CERTIFICATE OF QUALIFIED PERSON

***Garth D. Kirkham, P.Geo.***
6331 Palace Place
Burnaby, BC V5E 1Z6

I, Garth David Kirkham, am a Professional Geoscientist, employed as a President and Principal Consultant with Kirkham Geosystems Ltd.

This certificate applies to the technical report titled "*Minto Phase IV, Pre-Feasibility Technical Report*" dated December 15, 2009.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC), the Association of Engineers, Geologists and Geophysicists of Alberta (APEGGA) and the Northwest Territories and Nunavut Association of Professional Engineers, Geologists and Geophysicists of (NAPEGG). I graduated with a B.Sc. from the University of Alberta in 1983. I have practiced my profession continuously since 1983 and have been involved in many poly-metallic projects which involved authoring NI43-101 Reports including :

- Morrison Deposit, BC, Canada
- Adi Nefas Polymetalic Deposit, Eritrea
- Debarwa Polymetalic Deposit, Eritrea
- Tahuehueto Project, Durango, Mexico
- Atlas Moly Deposit, Mexico
- Kutcho Creek Deposit, BC, Canada

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 Project site.

I am responsible for Sections 12, 13 and the Minto North Resource Estimation listed in Section 16 of "*Minto Phase IV, Pre-Feasibility Technical Report*".

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 February 2009 doing resource estimation studies.

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.

"Signed and sealed"

ORIGINAL SIGNED AND SEALED

Garth David Kirkham, P.Geo.

Dated: December 15, 2009

**CERTIFICATE OF QUALIFIED PERSON**

*Michael E. Levy, P.E. P.G.*
7175 West Jefferson Avenue, Suite 3000
Lakewood, Colorado 80235

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 IV, Pre-Feasibility Technical Report*" dated December 15, 2009.

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 Project site on several occasions with the most recent visit being on September 21 and 22, 2009.

I am responsible for Section 18.2 of "*Minto Phase IV, Pre-Feasibility Technical Report*".

I am independent of Capstone Mining Corp. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Minto Project since December 2006 conducting several geotechnical pit slope evaluations for the Main, Area 2, Area 118, Ridgetop and Minto North deposits.

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 SEALED

Dated: December 15, 2009

**CERTIFICATE OF QUALIFIED PERSON**

***Wayne Peter Barnett, PhD, Pr.Sci.Nat.***
837 Old Lillooet Road, North Vancouver, Canada

I, Wayne Peter Barnett, am a Professional Geologist, employed as a Senior Consulting Geologist with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "*Minto Phase IV, Pre-Feasibility Technical Report*" dated 15th December 2009.

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 practiced my profession continuously since 1997 and have been involved in:

- Geology and geotechnical engineering at three mining operations over a period of 8 years,
- Exploration geology for two years for De Beers,
- Consulting structural geology and geological modelling for two years in De Beers,
- Consulting geologist for SRK Consulting for nearly two years.

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 from the 4th to 6th March, 2009.

I am responsible for Sections 6 to 11, and Section 16 of "*Minto Phase IV, Pre-Feasibility Technical Report*".

I am independent of Minto Exploration Ltd.. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Minto Project since October 23rd 2008 doing geological modelling of the deposits.

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 SEALED

______________________________

Wayne Barnett, PhD, Pr.Sci.Nat.

Dated: 15th December, 2009

# Appendix A

# Mineral Resources/Geology

# APPENDIX A1

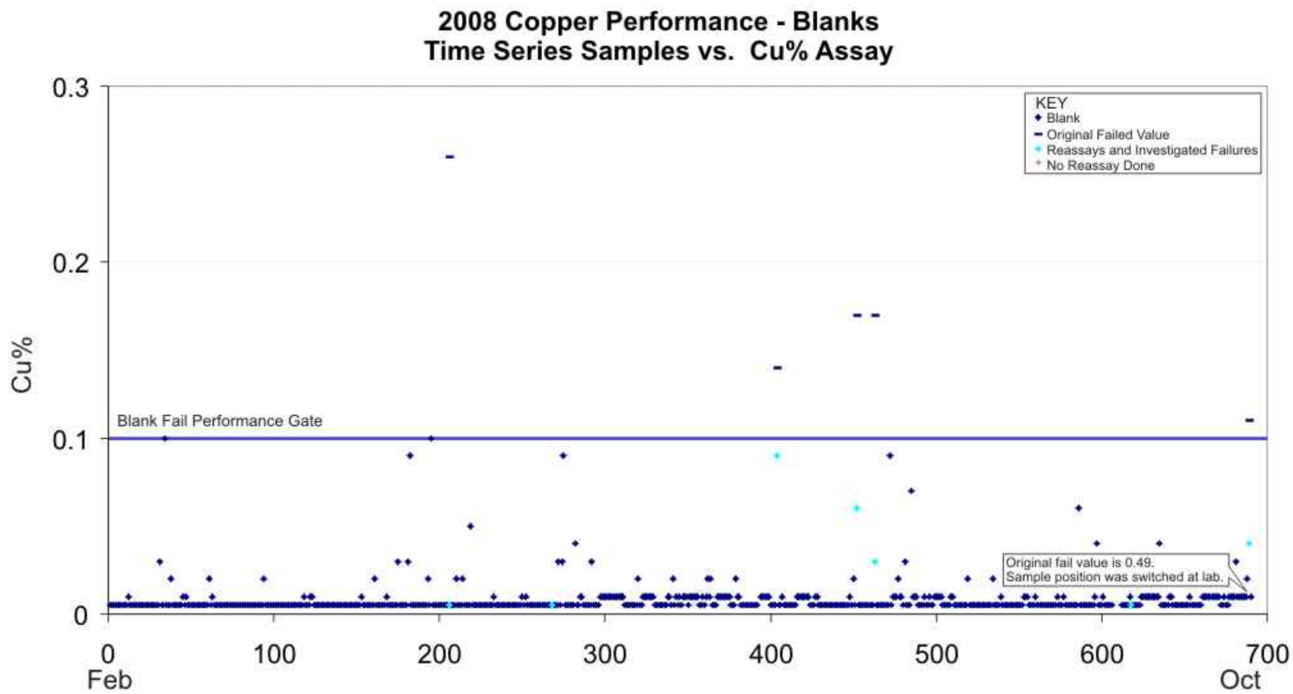
## Analytical Quality Control Data

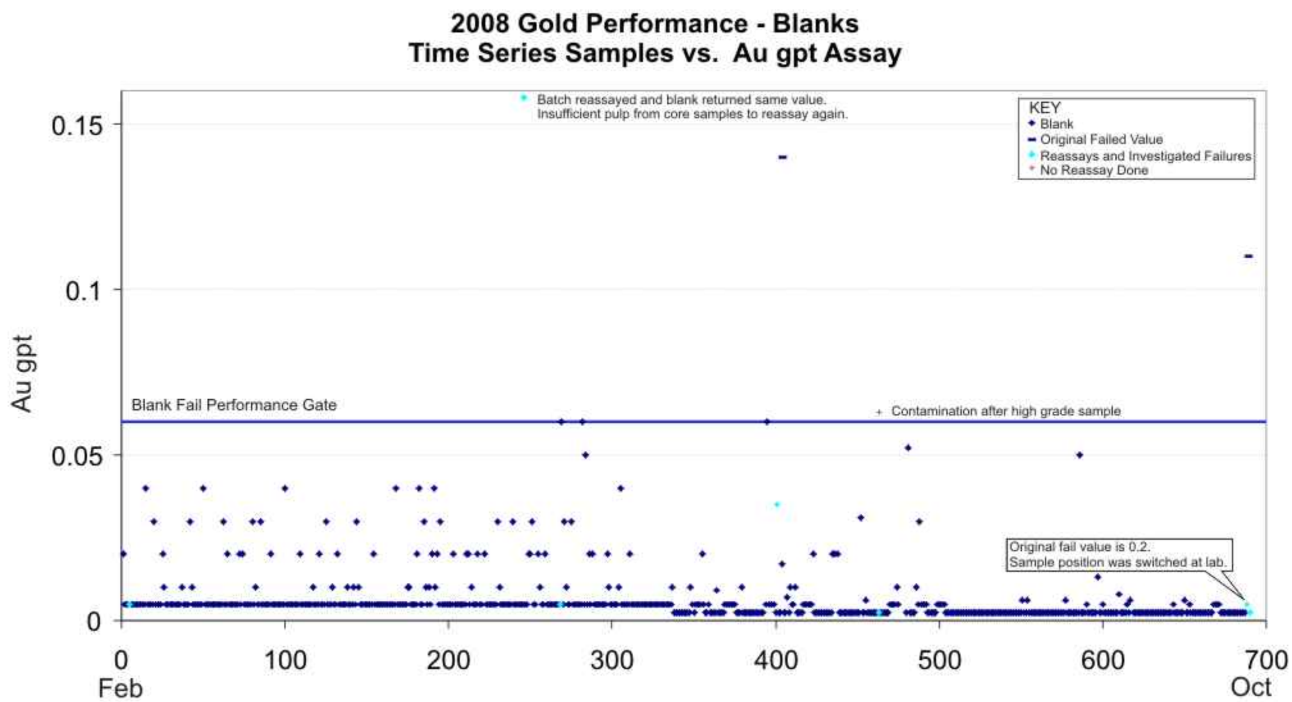
## Time Series for Control Samples
## and
## Duplicate Pair Comparison Charts

**Table 1: SRM Included in 2008 and 2009 MintoEx Borehole Samples**

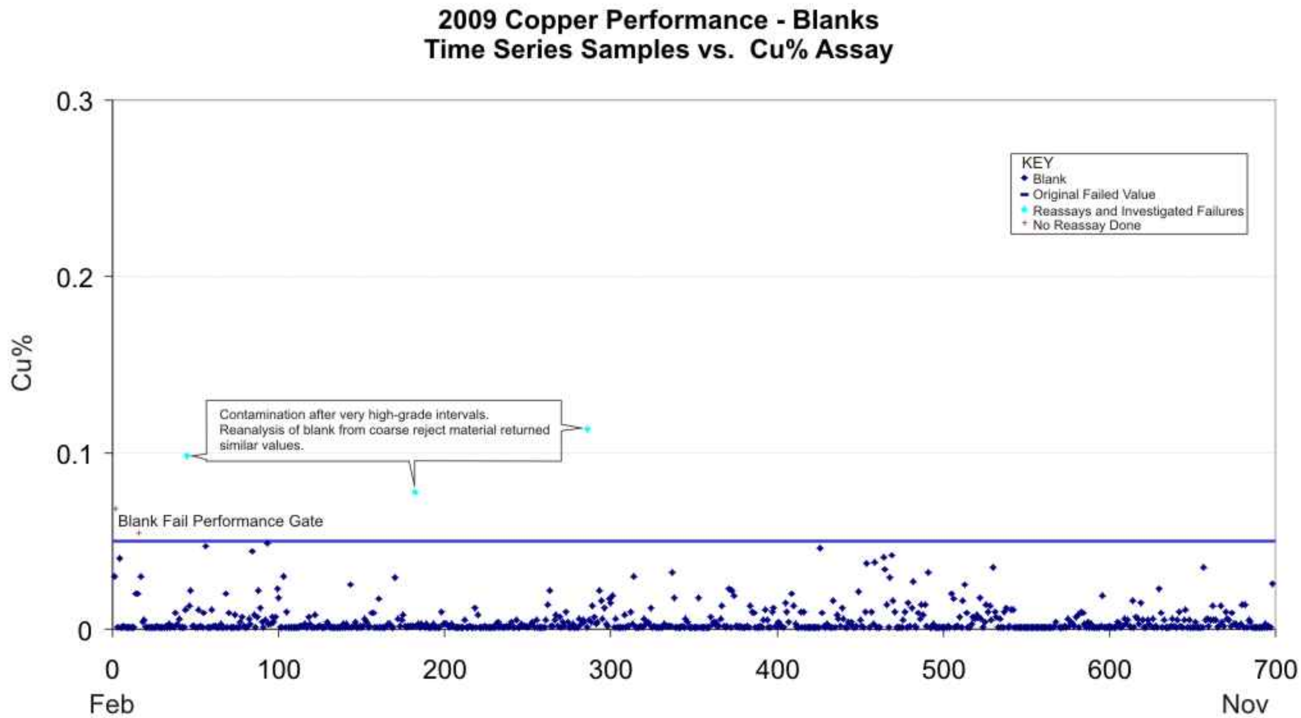
| SRM Sample Name | Source | Mean Grade | Upper Limits +2STD/+3STD | Lower Limits -2STD/-3STD | No. Samples Inserted | Dates |
|---|---|---|---|---|---|---|
| Copper, Gold and Silver | | | | | | |
| SRM-3 | Custom | 5.34 % Cu<br>2.23 g/t Au<br>18.40 g/t Ag | 5.78/6.00% Cu<br>2.40/2.49 g/t Au<br>19.35/19.83 g/t Ag | 4.90/4.68% Cu<br>2.06/1.98 g/t Au<br>17.45/16.98 g/t Ag | 24 | July to Nov 2009 |
| SRM-2 | Custom | 2.81 % Cu<br>1.57 g/t Au<br>9.40 g/t Ag | 2.96/3.03% Cu<br>1.71/1.78 g/t Au<br>10.40/10.90 g/t Ag | 2.70/2.60% Cu<br>1.43/1.36 g/t Au<br>8.4/7.9 g/t Ag | 27 | July to Nov 2009 |
| SRM-1 | Custom | 1.14 % Cu<br>0.43 g/t Au<br>3.90 g/t Ag | 1.22/1.26% Cu<br>0.46/0.48 g/t Au<br>4.13/4.25 g/t Ag | 1.06/1.02% Cu<br>0.39/0.37 g/t Au<br>3.67/3.56 g/t Ag | 27 | July to Nov 2009 |
| Copper and Gold | | | | | | |
| CGS-11 | CDN | 0.68 % Cu<br>0.73 g/t Au | 0.71/0.72% Cu<br>0.80/0.83 g/t Au | 0.66/0.64% Cu<br>0.66/0.63 g/t Au | 156+123 | Feb to Oct 2008<br>Feb to Nov 2009 |
| CM-2 | CDN | 1.01 % Cu<br>1.42 g/t Au | 1.05/1.07% Cu<br>1.55/1.62 g/t Au | 0.97/0.95% Cu<br>1.29/1.23 g/t Au | 99+117 | April to Oct 2008<br>Feb to July 2009 |
| CGS-21 | CDN | 1.30 % Cu<br>0.99 g/t Au | 1.38/1.43% Cu<br>1.08/1.13 g/t Au | 1.22/1.17% Cu<br>0.90/0.86 g/t Au | 7 | April to May 2009 |
| CGS-18 | CDN | 0.32 % Cu<br>0.3 g/t Au | 0.335/0.343% Cu<br>0.34/0.36g/t Au | 0.303/0.295% Cu<br>0.26/0.24g/t Au | 120+190 | May to Oct 2008<br>March to Nov 2009 |
| CGS-17 | CDN | 2.36 % Cu<br>2.43 g/t Au | 2.47/2.53% Cu<br>2.77/2.94g/t Au | 2.25/2.2% Cu<br>2.09/1.92g/t Au provisional | 56+12 | June to Oct 2008<br>April to May 2009 |
| CGS-15 | CDN | 0.45 % Cu<br>0.57 g/t Au | 0.47/0.48% Cu<br>0.63/0.66g/t Au | 0.43/0.42% Cu<br>0.51/0.48g/t Au | 177+191 | March to Oct 2008<br>Feb to Nov 2009 |
| CGS-12 | CDN | 0.27 % Cu<br>0.29 g/t Au | 0.28/0.29% Cu<br>0.33/0.35g/t Au | 0.25/0.24% Cu<br>0.25/0.23g/t Au | 31 | Feb to April 2008 |
| CGS-10 | CDN | 1.55 % Cu<br>1.73 % Au | 1.62/1.66% Cu<br>1.88/1.96% Au | 1.48/1.45% Cu<br>1.58/1.51% Au | 24 | Feb to April 2008 |
| CM-3 | CDN | 0.55 % Cu<br>0.46 g/t Au | 0.57/0.58% Cu<br>0.52/0.55g/t Au | 0.53/0.52% Cu<br>0.40/0.37g/t Au | 27 | Feb to May 2008 |
| Copper | | | | | | |
| SRM-95 | ASL | 2.59% Cu | 2.72/2.785 % Cu | 2.46/2.395 % Cu | 12+3 | April to Oct 2008<br>March to April 2009 |

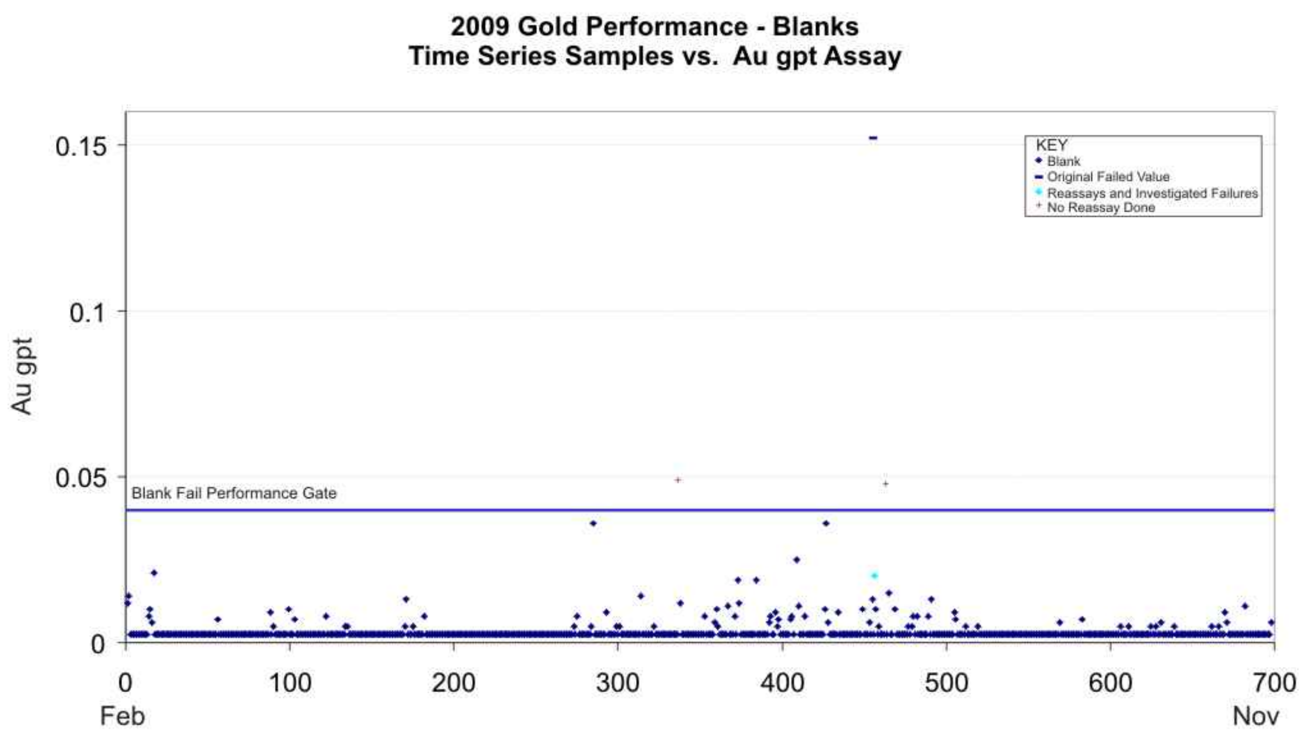
Assay Results for Blanks Inserted with 2008 Samples



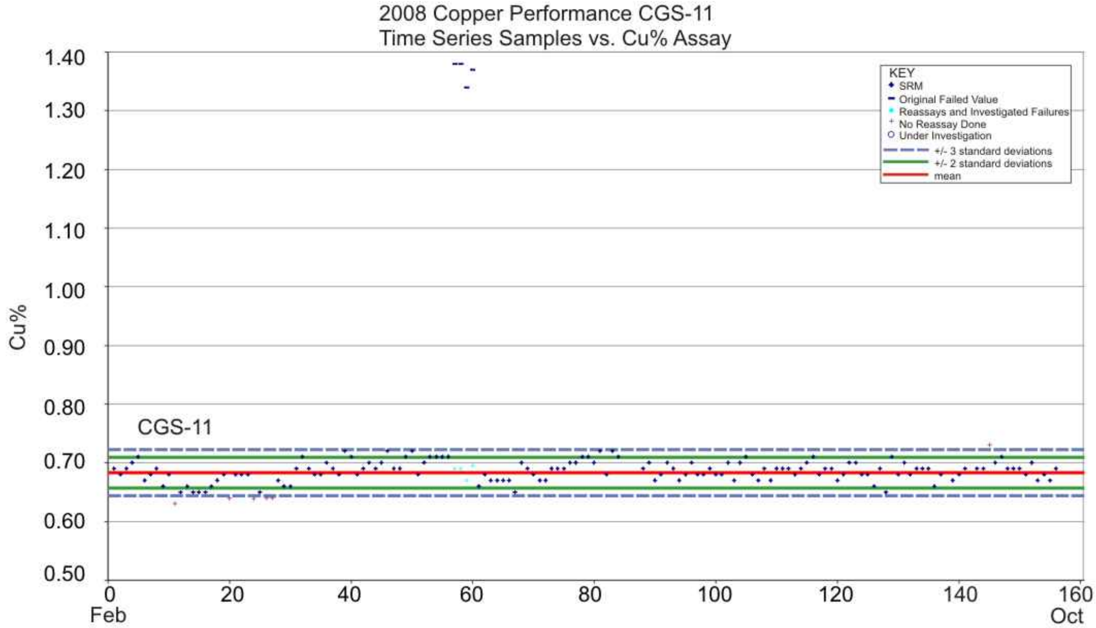



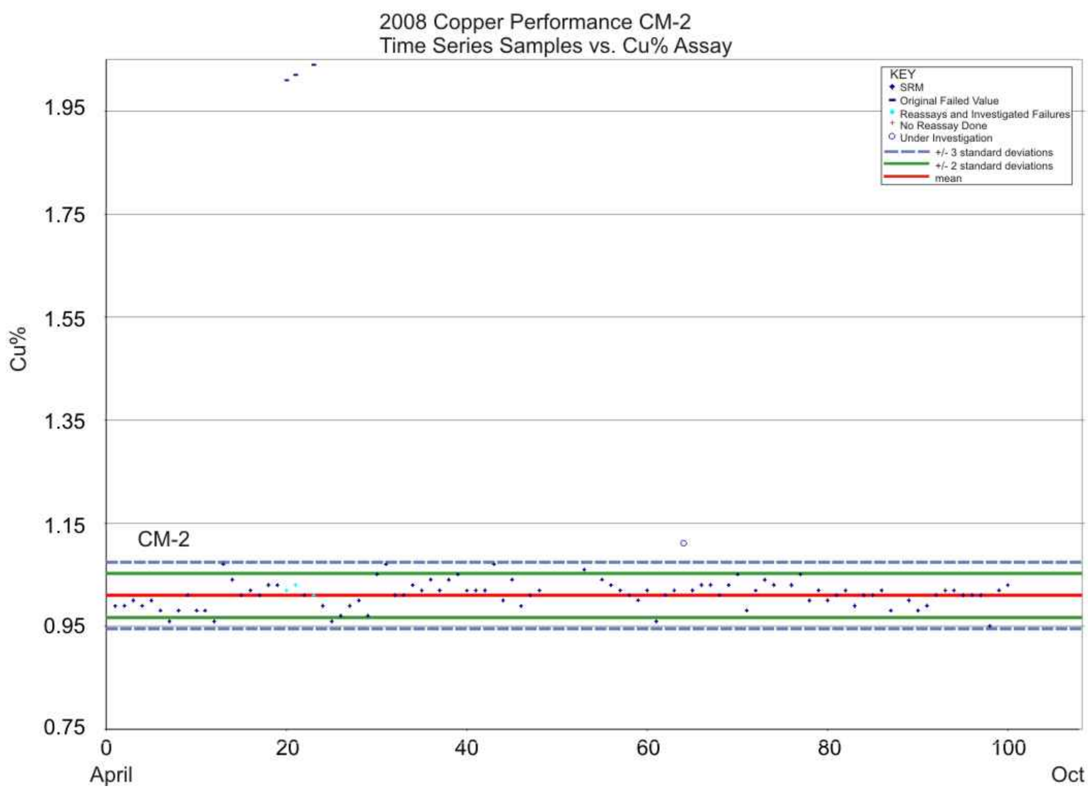
Assay Results for Blanks Inserted with 2009 Samples

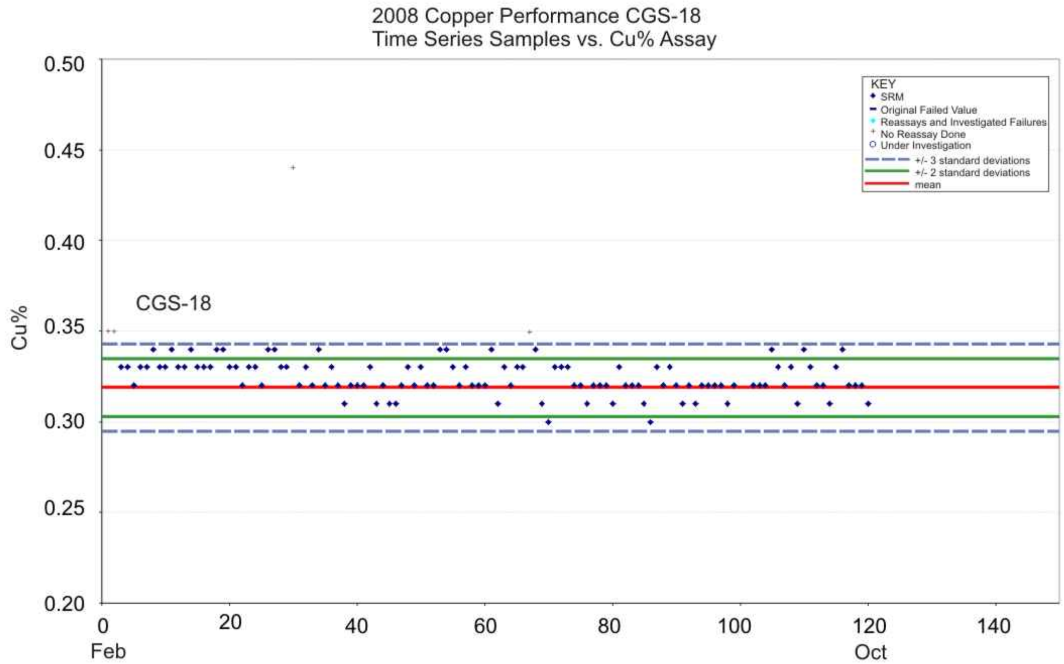


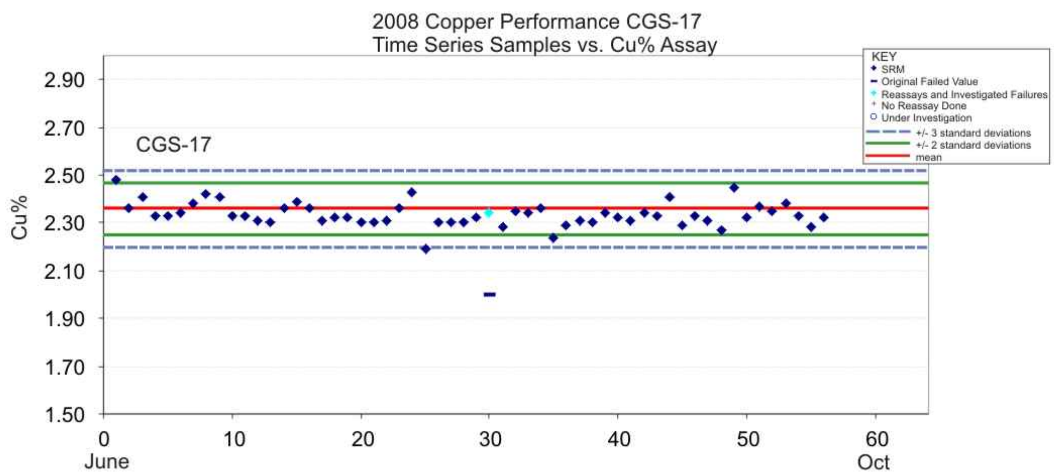



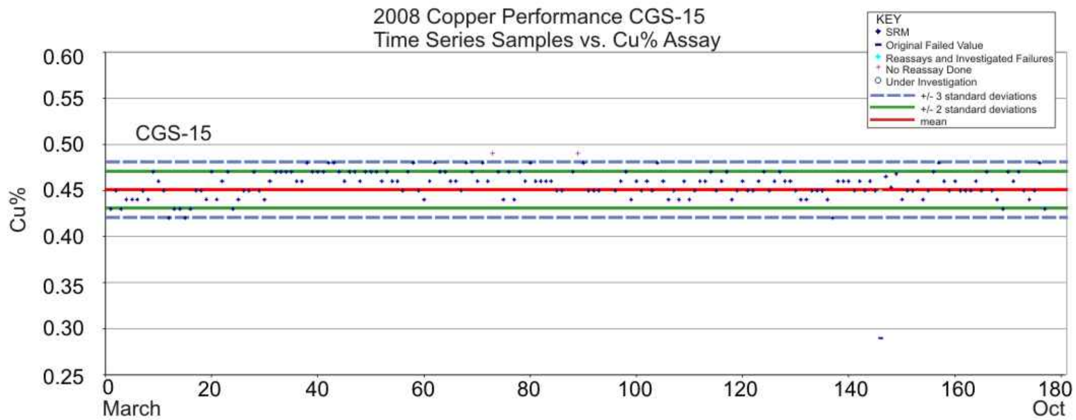
Copper Assay Results for Purchased SRM Inserted with 2008 Samples



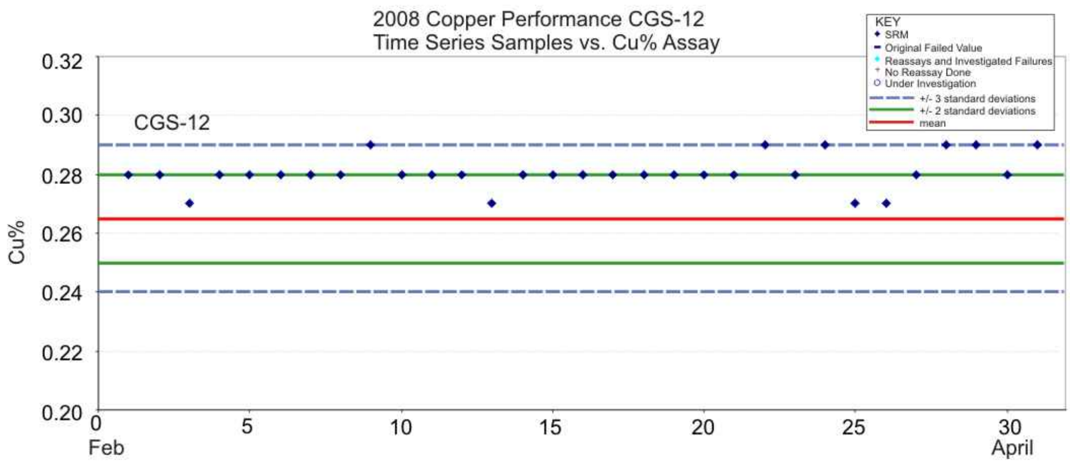




2008 Copper Performance CGS-18
Time Series Samples vs. Cu% Assay
KEY
SRM
Original Failed Value
Reassays and Investigated Failures
No Reassay Done
Under Investigation
+/- 3 standard deviations
+/- 2 standard deviations
mean
CGS-18
Cu%
0.50
0.45
0.40
0.35
0.30
0.25
0.20
0
20
40
60
80
100
120
140
Feb
Oct

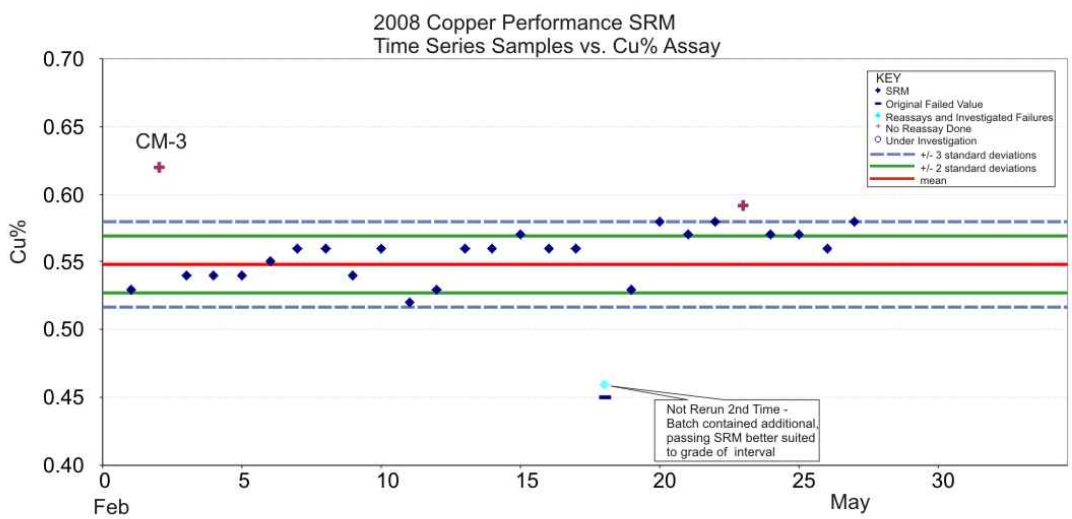

2008 Copper Performance CGS-17
Time Series Samples vs. Cu% Assay
KEY
SRM
Original Failed Value
Reassays and Investigated Failures
No Reassay Done
Under Investigation
+/- 3 standard deviations
+/- 2 standard deviations
mean
CGS-17
Cu%
2.90
2.70
2.50
2.30
2.10
1.90
1.70
1.50
0
10
20
30
40
50
60
June
Oct



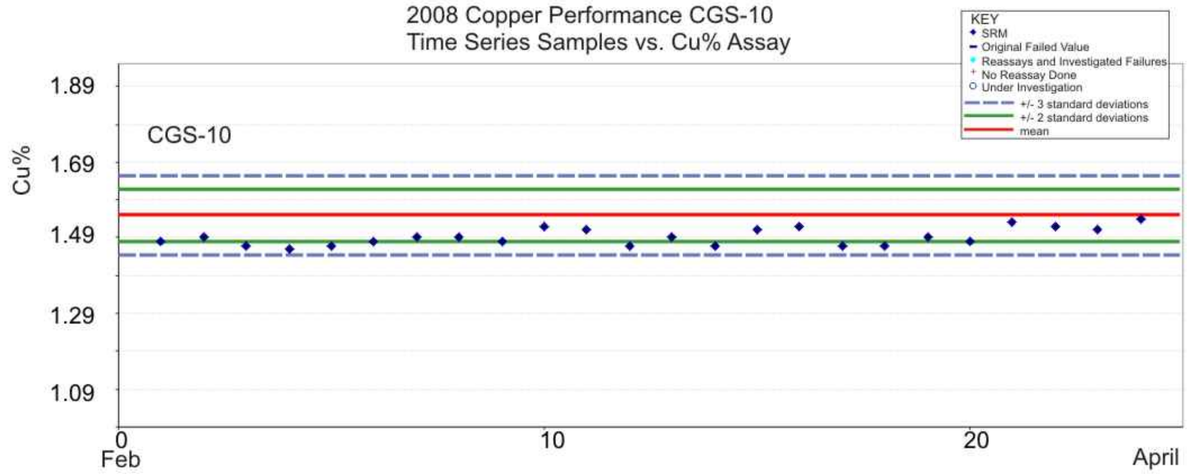
2008 Copper Performance CGS-15
Time Series Samples vs. Cu% Assay
KEY
SRM
Original Failed Value
Reassays and Investigated Failures
No Reassay Done
Under Investigation
+/- 3 standard deviations
+/- 2 standard deviations
mean
CGS-15
Cu%
March
Oct

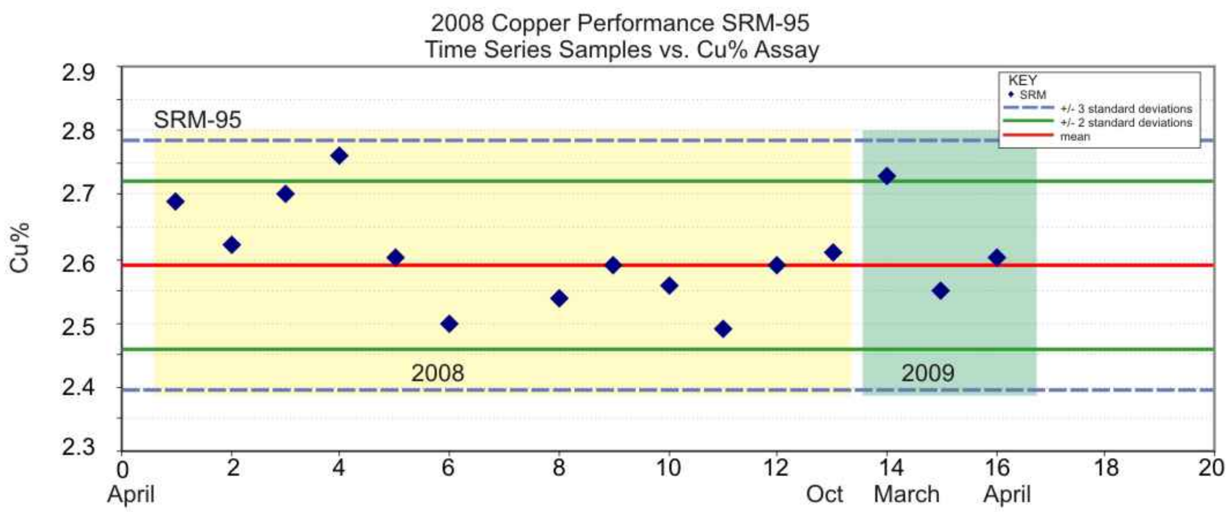
2008 Copper Performance CGS-12
Time Series Samples vs. Cu% Assay
KEY
SRM
Original Failed Value
Reassays and Investigated Failures
No Reassay Done
Under Investigation
+/- 3 standard deviations
+/- 2 standard deviations
mean
CGS-12
Cu%
Feb
April

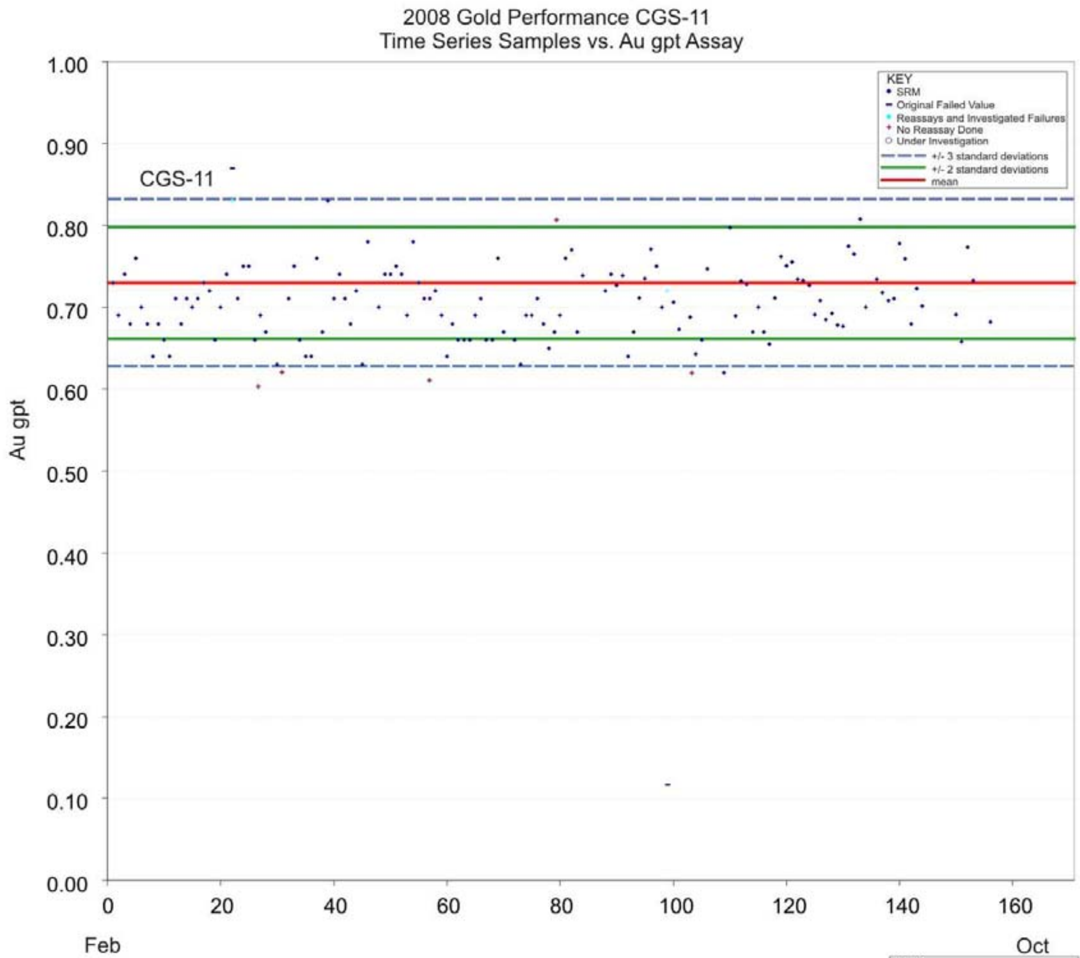
2008 Copper Performance SRM
Time Series Samples vs. Cu% Assay
KEY
SRM
Original Failed Value
Reassays and Investigated Failures
No Reassay Done
Under Investigation
+/- 3 standard deviations
+/- 2 standard deviations
mean
CM-3
Cu%
Not Rerun 2nd Time - Batch contained additional, passing SRM better suited to grade of interval
Feb
May

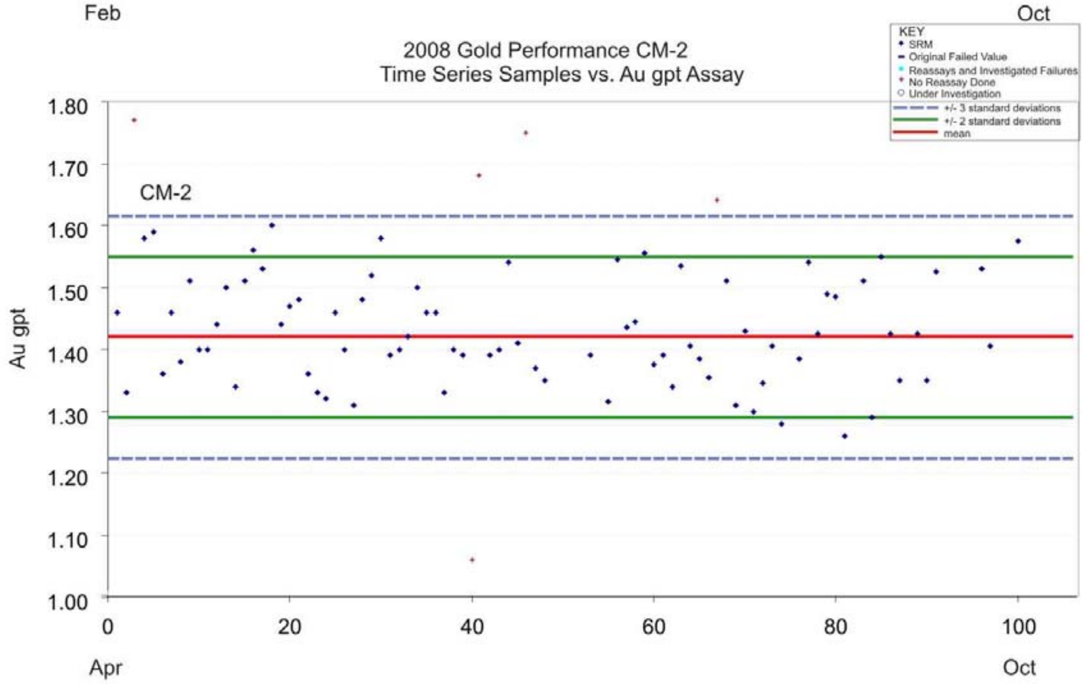


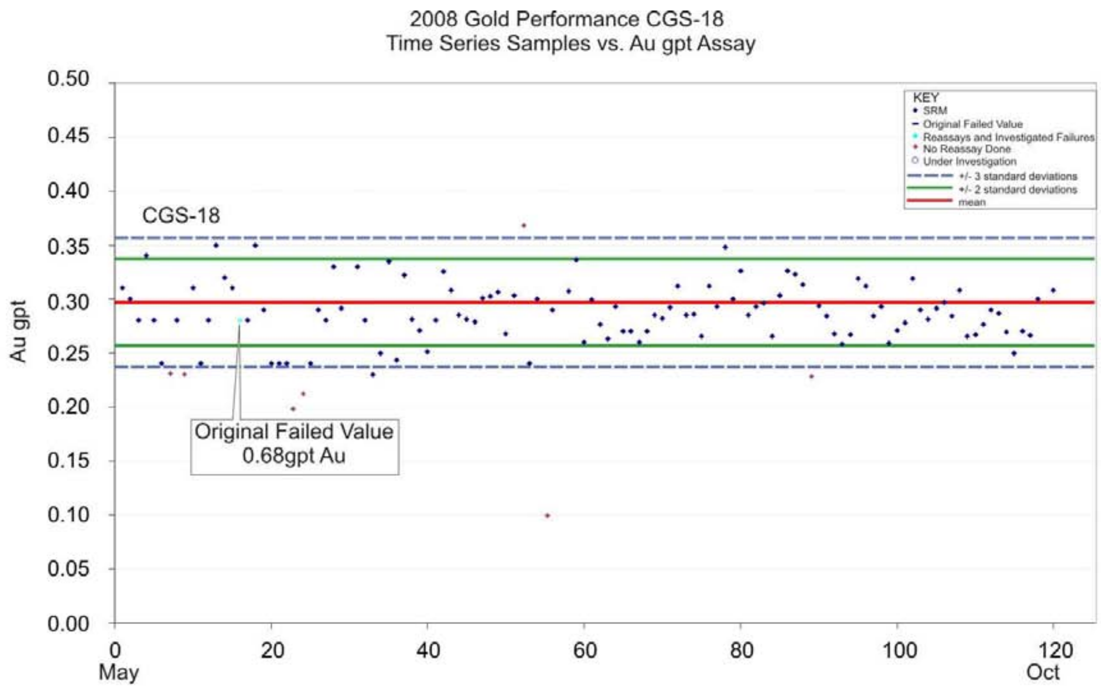

Copper Assay Results for Purchased SRM inserted with 2008 and 2009 Samples

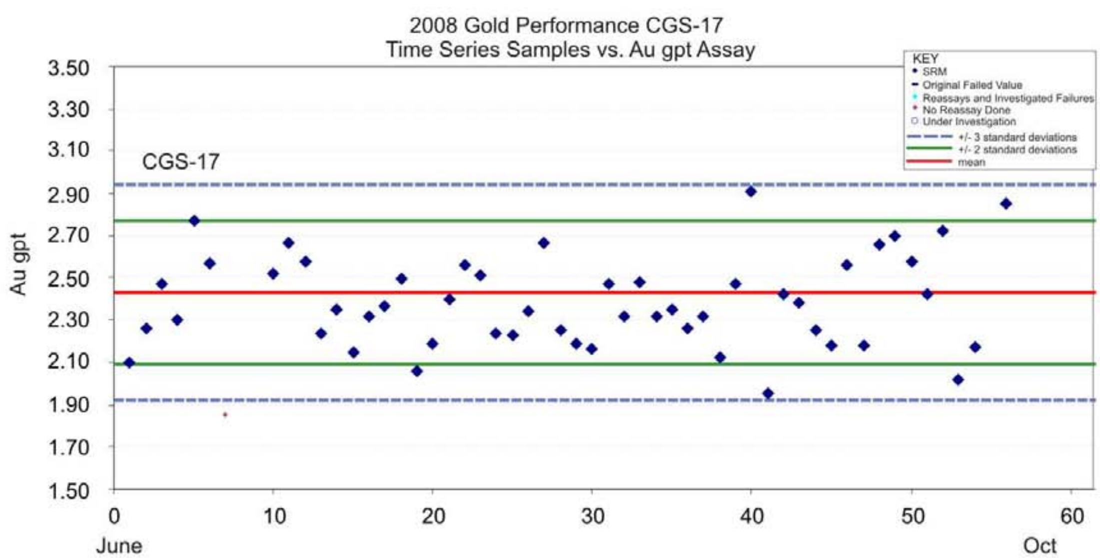


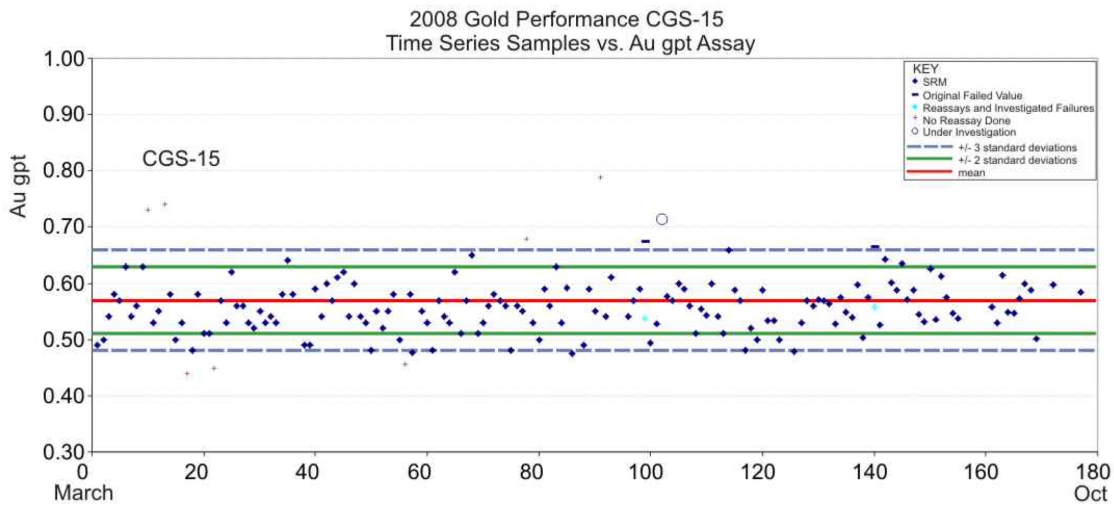
Gold Assay Results for Purchased SRM Inserted with 2008 Samples

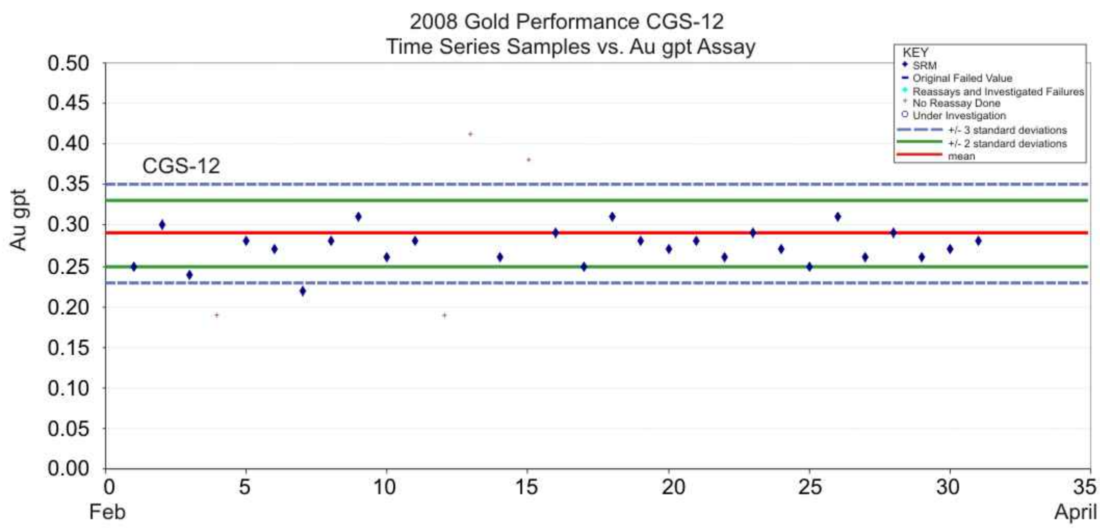


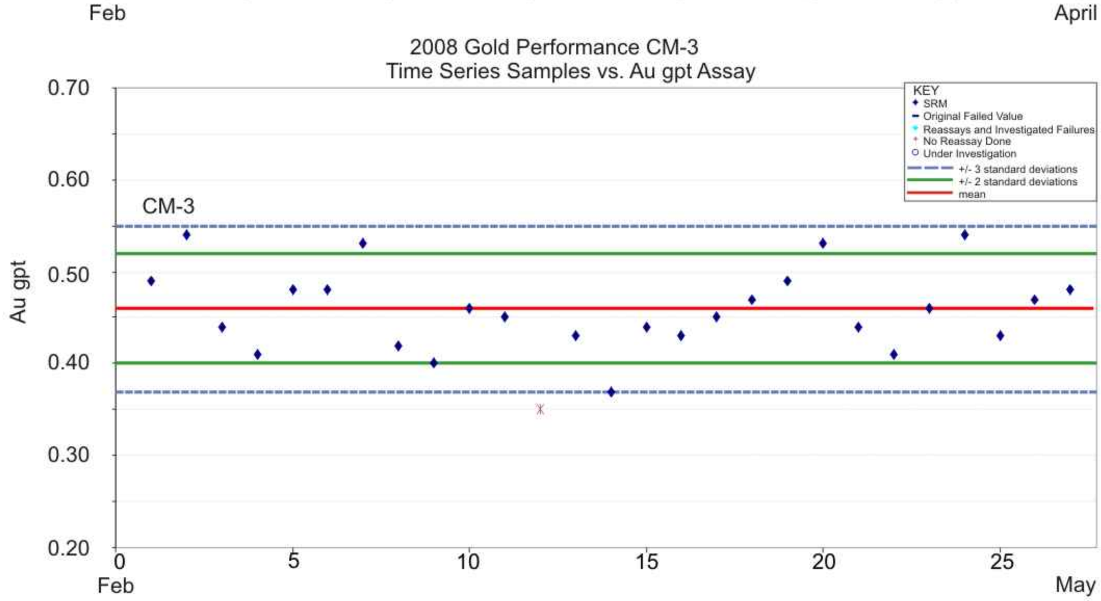



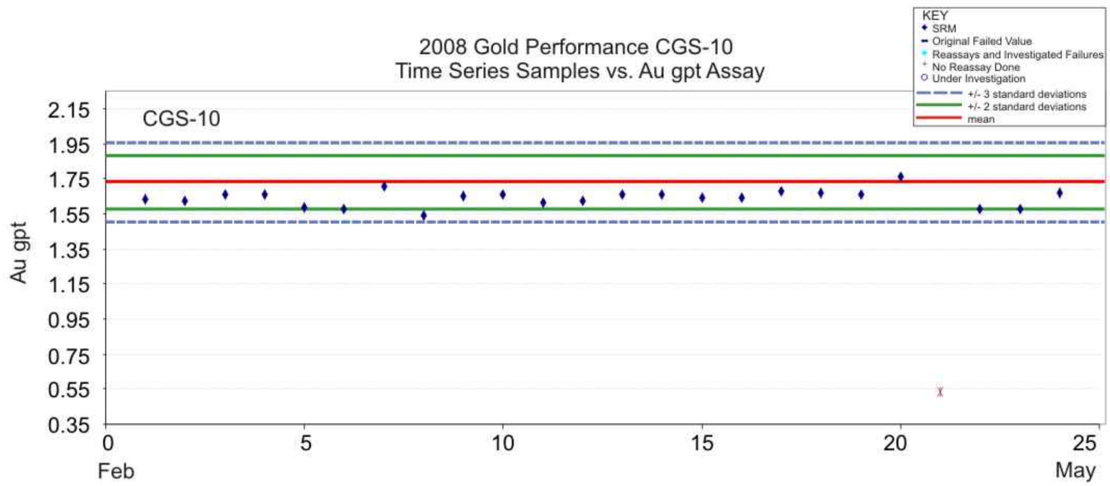

2008 Gold Performance CGS-18
Time Series Samples vs. Au gpt Assay
KEY
SRM
Original Failed Value
Reassays and Investigated Failures
No Reassay Done
Under Investigation
+/- 3 standard deviations
+/- 2 standard deviations
mean
CGS-18
Original Failed Value
0.68gpt Au
Au gpt
May
Oct


2008 Gold Performance CGS-17
Time Series Samples vs. Au gpt Assay
KEY
SRM
Original Failed Value
Reassays and Investigated Failures
No Reassay Done
Under Investigation
+/- 3 standard deviations
+/- 2 standard deviations
mean
CGS-17
Au gpt
June
Oct


2008 Gold Performance CGS-15
Time Series Samples vs. Au gpt Assay
CGS-15
Au gpt
1.00
0.90
0.80
0.70
0.60
0.50
0.40
0.30
0
20
40
60
80
100
120
140
160
180
March
Oct
KEY
SRM
Original Failed Value
Reassays and Investigated Failures
No Reassay Done
Under Investigation
+/- 3 standard deviations
+/- 2 standard deviations
mean

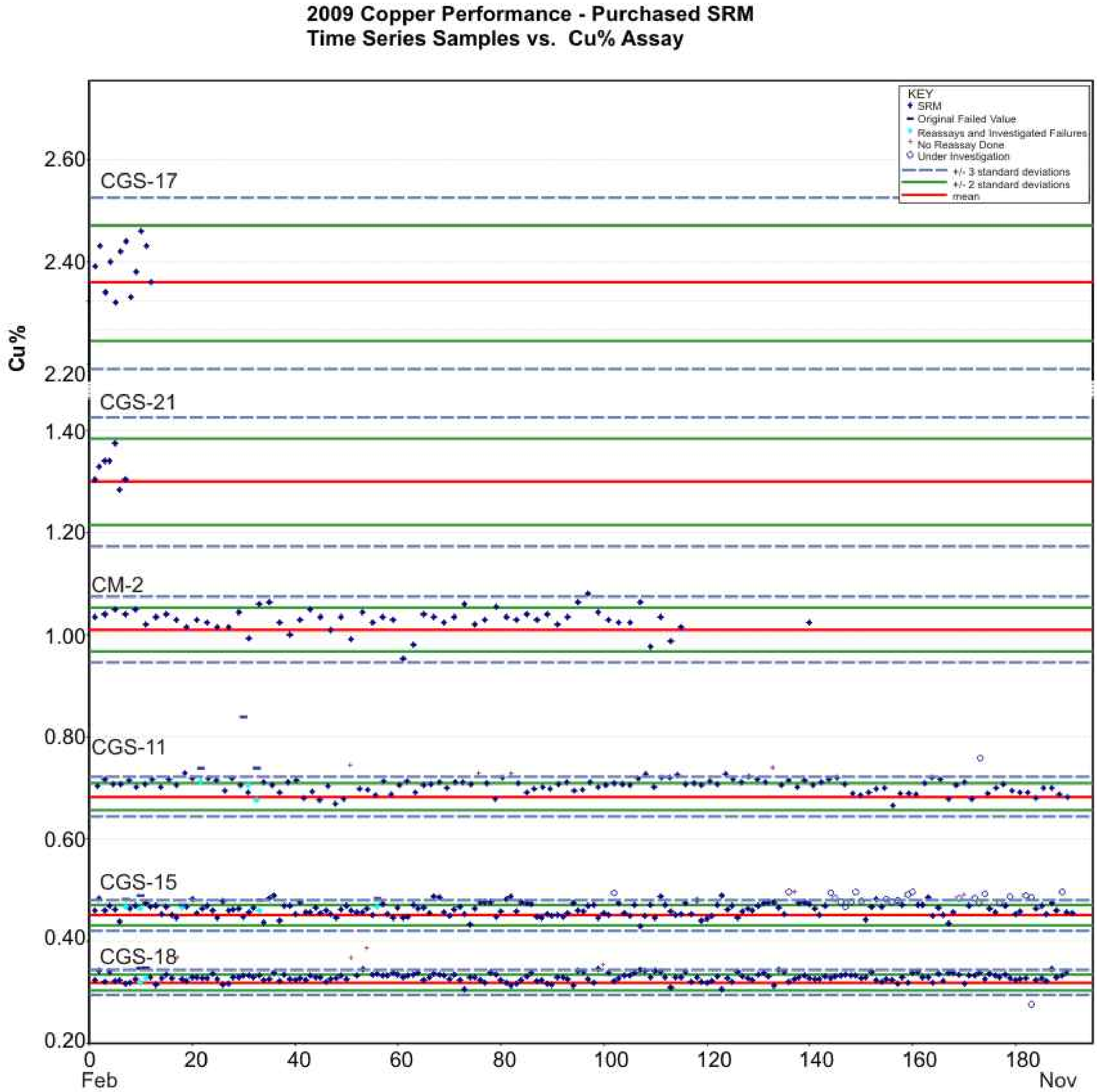

2008 Gold Performance CGS-12
Time Series Samples vs. Au gpt Assay
CGS-12
Au gpt
0.50
0.45
0.40
0.35
0.30
0.25
0.20
0.15
0.10
0.05
0.00
0
5
10
15
20
25
30
35
Feb
April
KEY
SRM
Original Failed Value
Reassays and Investigated Failures
No Reassay Done
Under Investigation
+/- 3 standard deviations
+/- 2 standard deviations
mean


2008 Gold Performance CM-3
Time Series Samples vs. Au gpt Assay
CM-3
Au gpt
0.70
0.60
0.50
0.40
0.30
0.20
0
5
10
15
20
25
Feb
May
KEY
SRM
Original Failed Value
Reassays and Investigated Failures
No Reassay Done
Under Investigation
+/- 3 standard deviations
+/- 2 standard deviations
mean

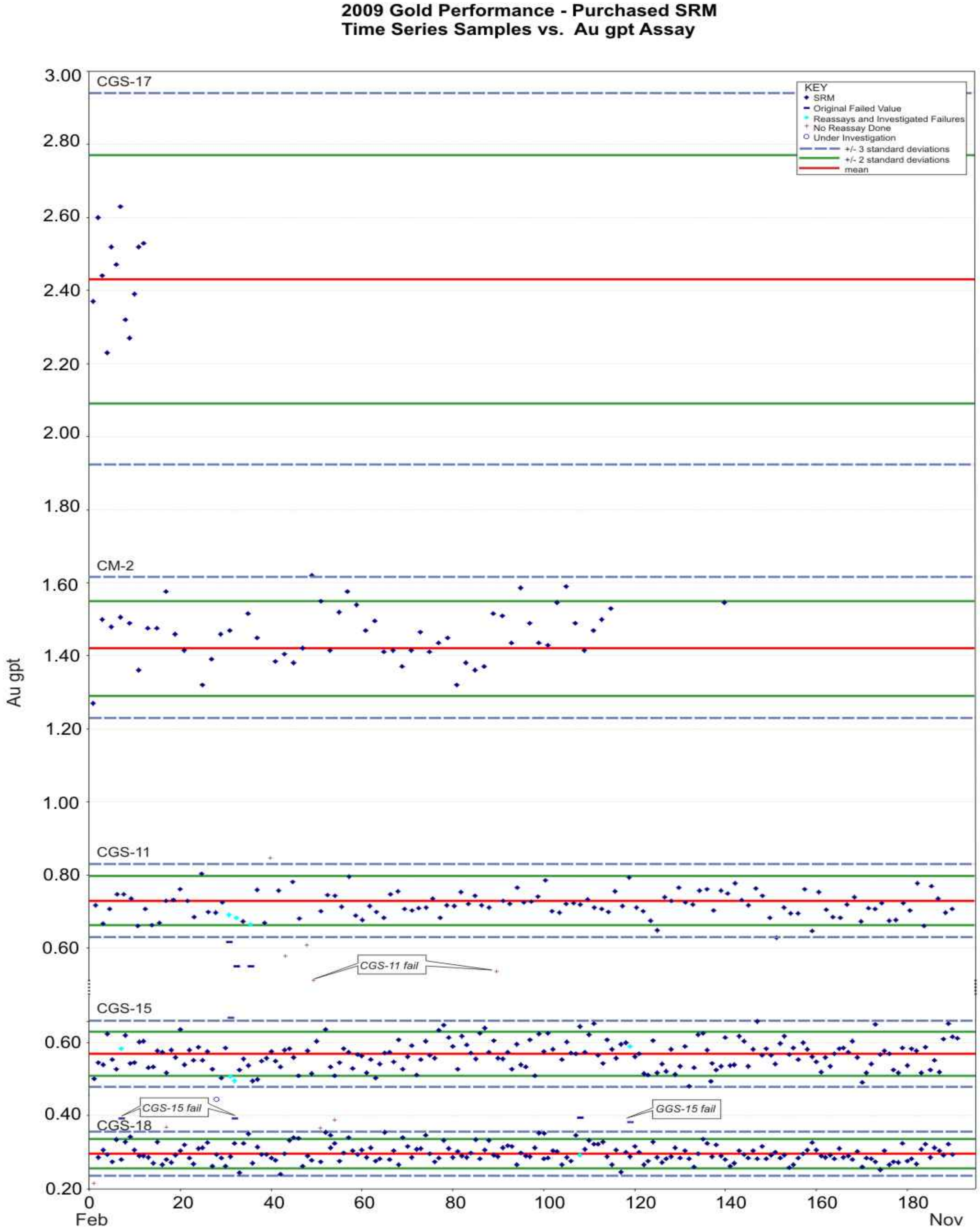


2008 Gold Performance CGS-10
Time Series Samples vs. Au gpt Assay
KEY
SRM
Original Failed Value
Reassays and Investigated Failures
No Reassay Done
Under Investigation
+/- 3 standard deviations
+/- 2 standard deviations
mean
CGS-10
Au gpt
2.15
1.95
1.75
1.55
1.35
1.15
0.95
0.75
0.55
0.35
0
5
10
15
20
25
Feb
May

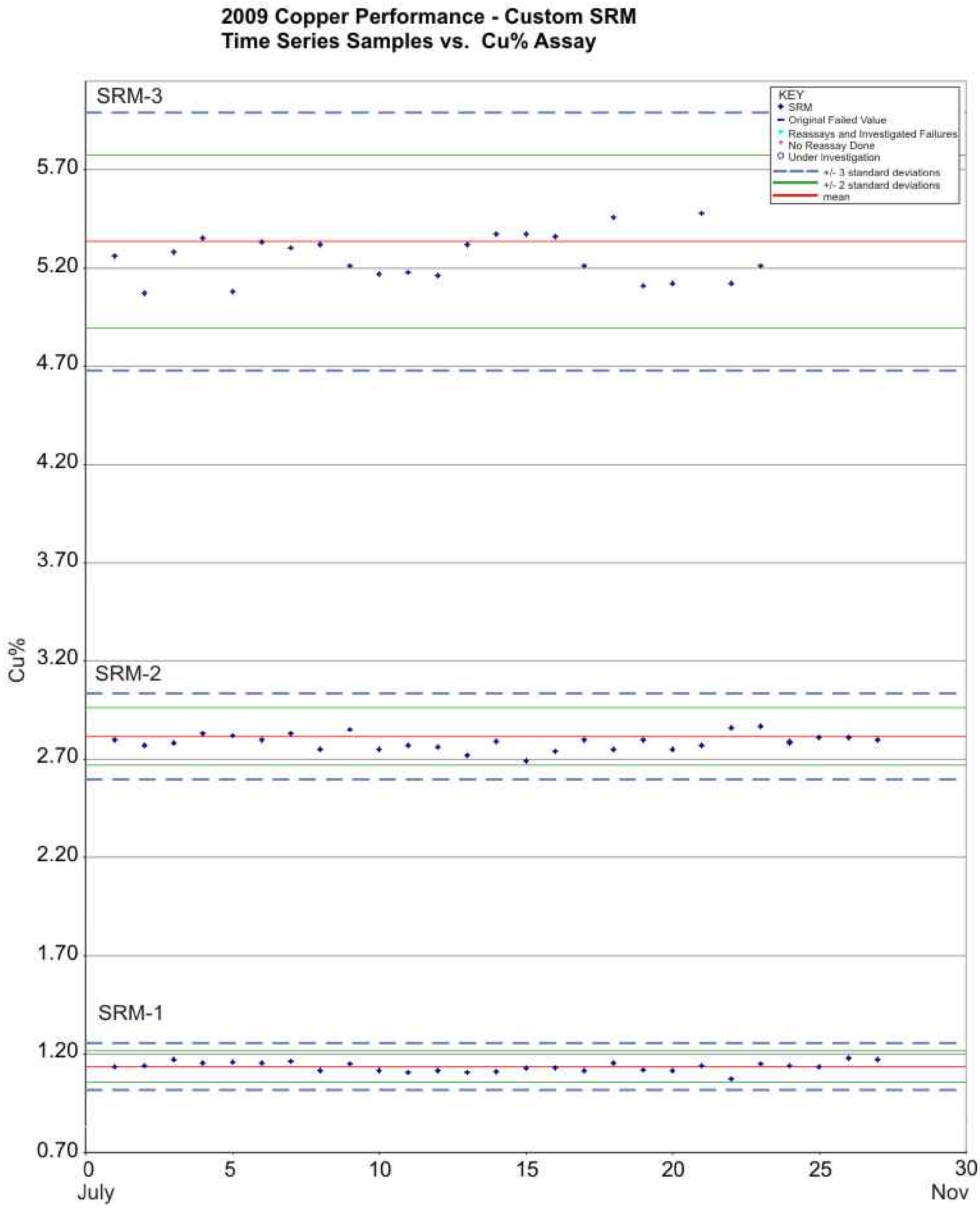
Copper Assay Results for Purchased SRM Inserted with 2009 Samples



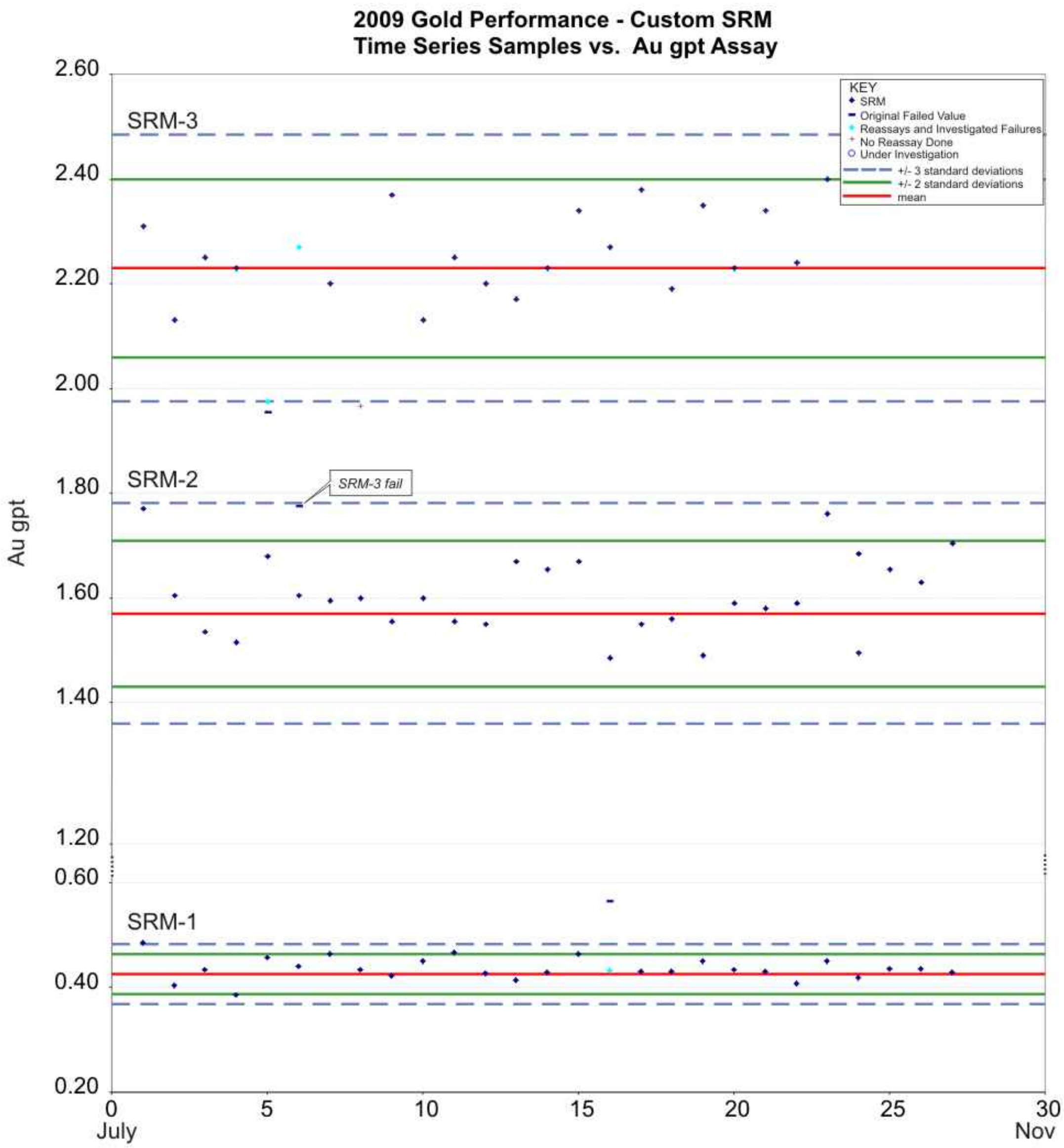
Gold Assay Results for Purchased SRM Inserted with 2009 Samples



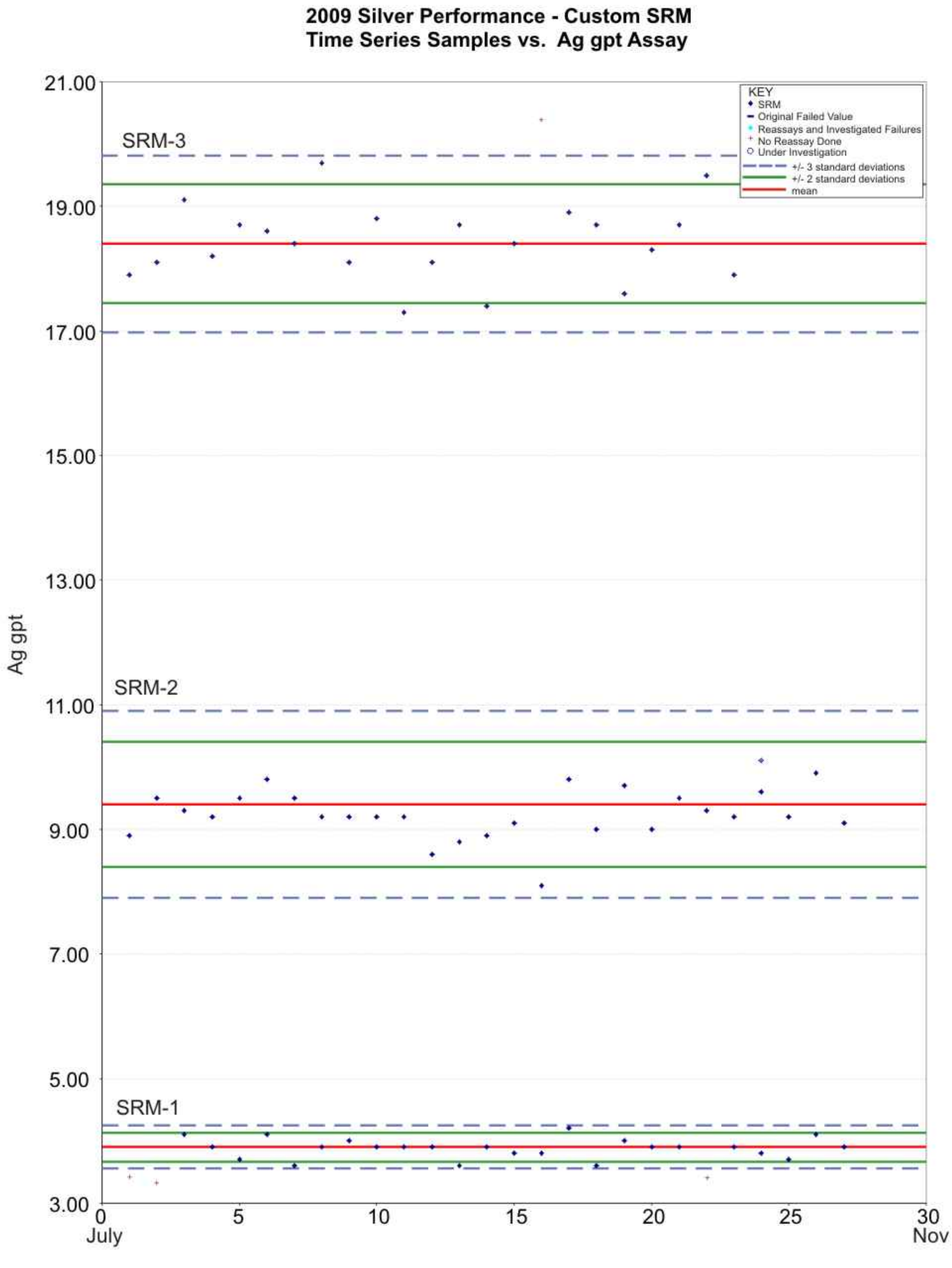
Copper Assay Results for Custom SRM Inserted with 2009 Samples



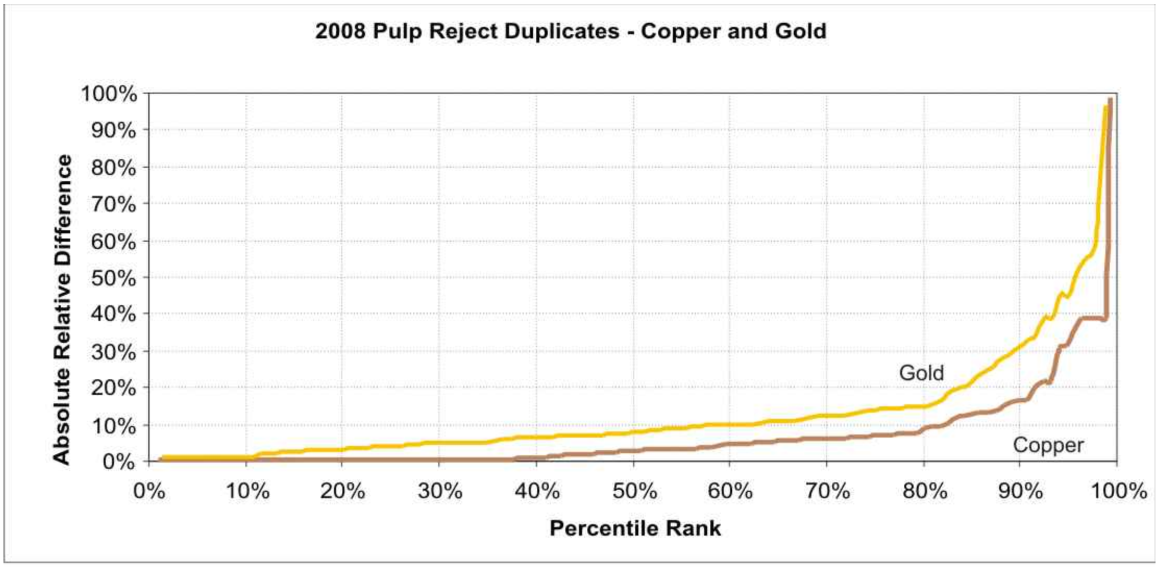
Gold Assay Results for Custom SRM Inserted with 2009 Samples

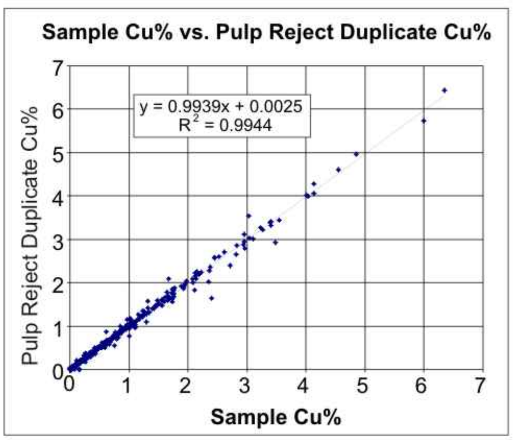


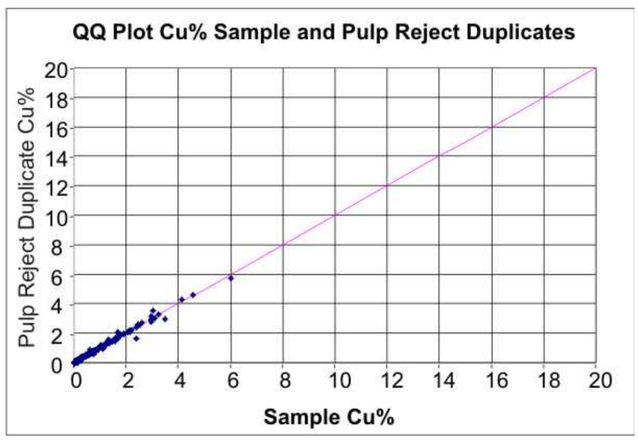
Silver Assay Results for Custom SRM Inserted with 2009 Samples

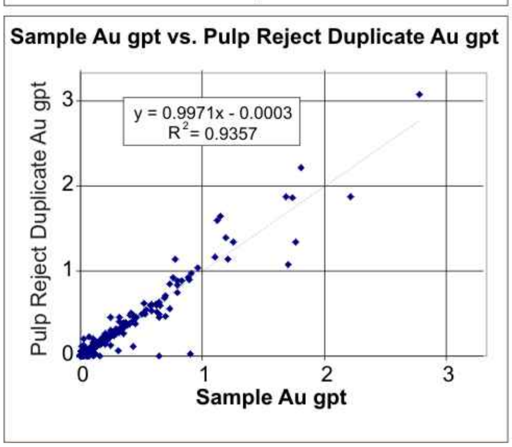


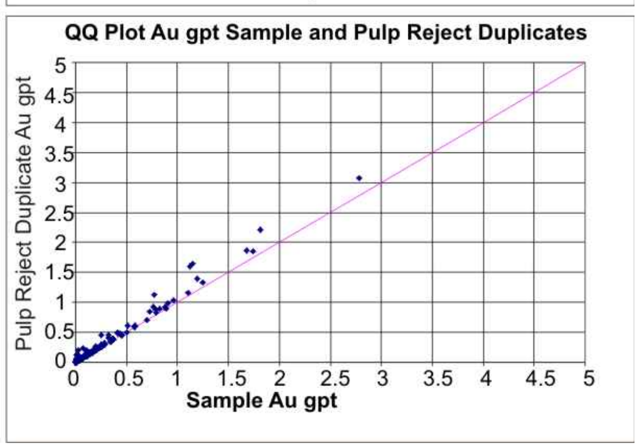
Assay Results for 2008 Pulp Reject Duplicate Sample Pairs



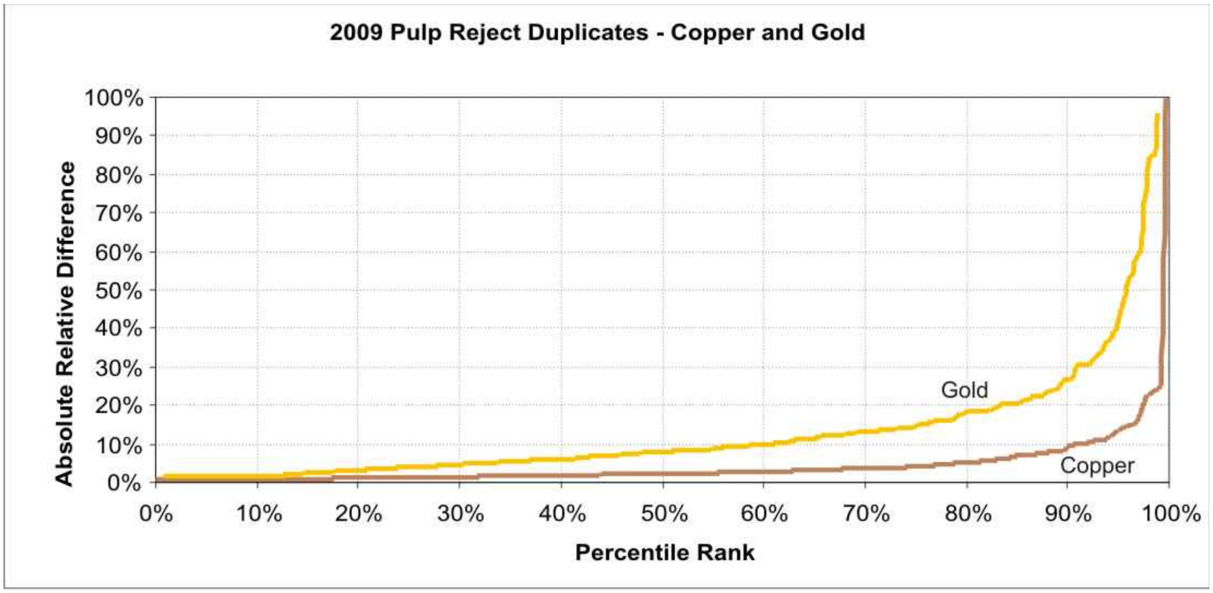



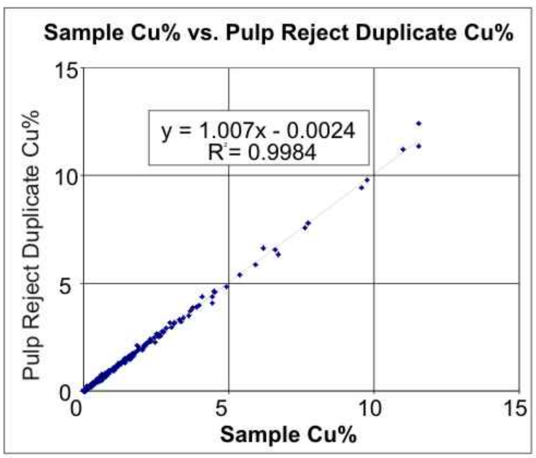



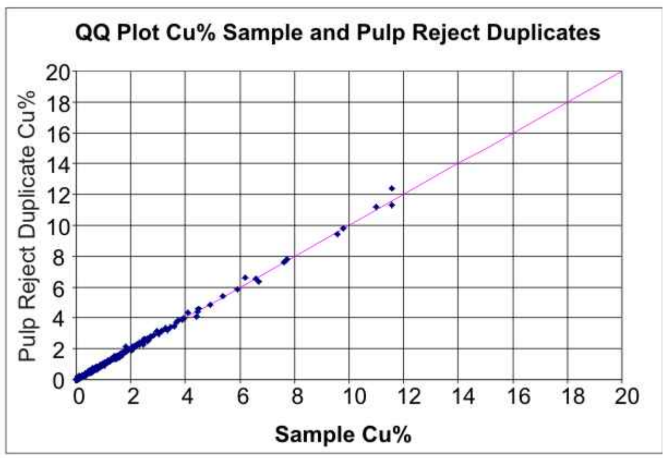



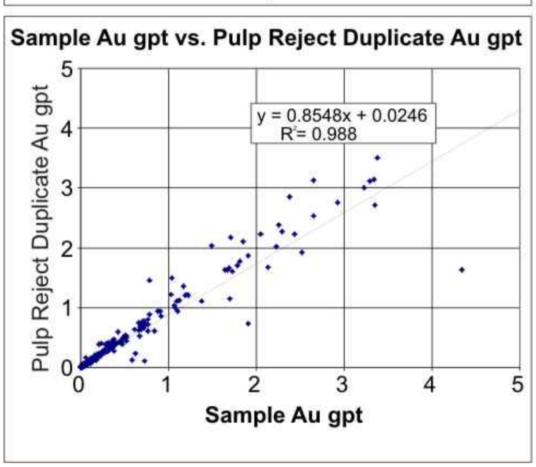



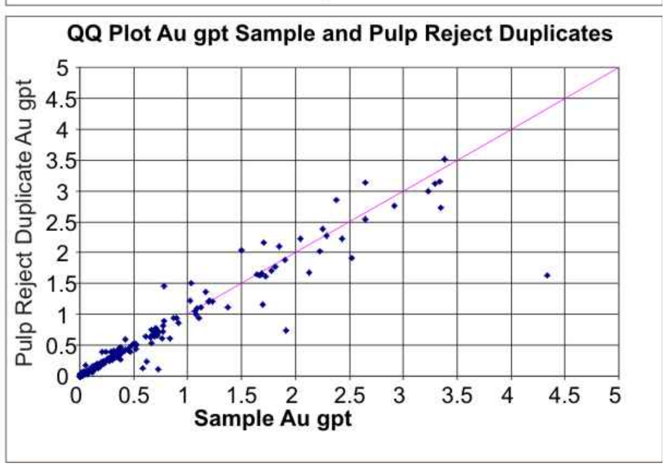
Assay Results for 2009 Pulp Reject Duplicate Sample Pairs



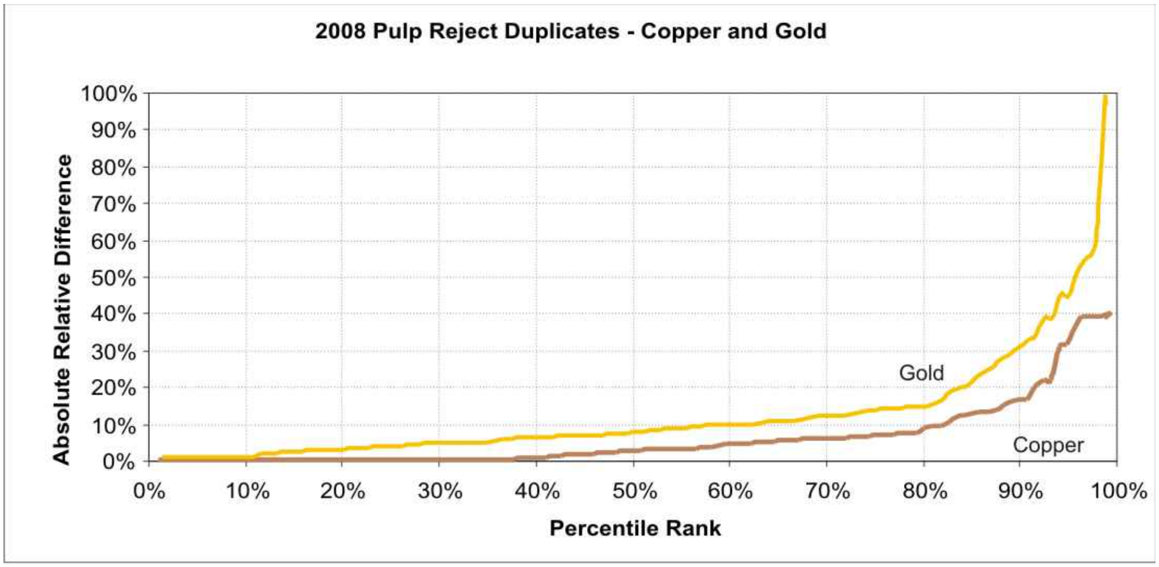



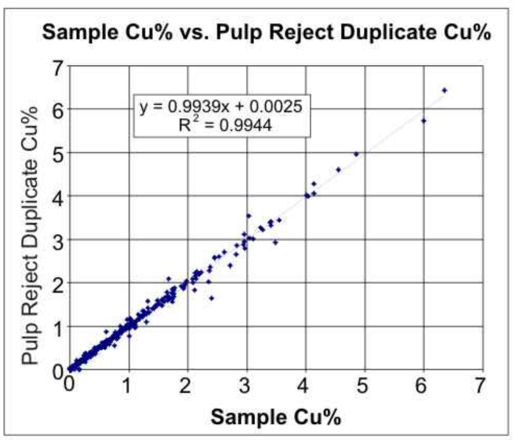



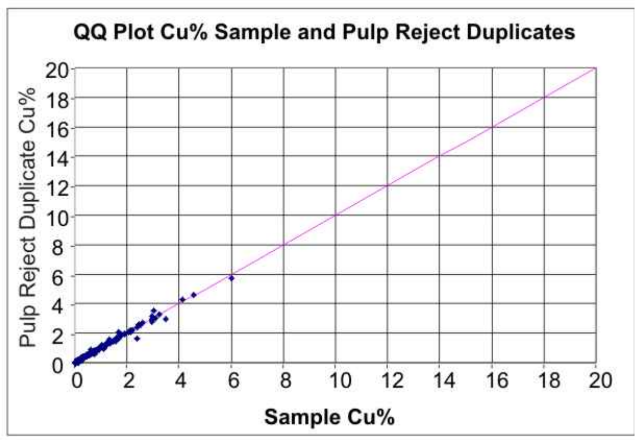



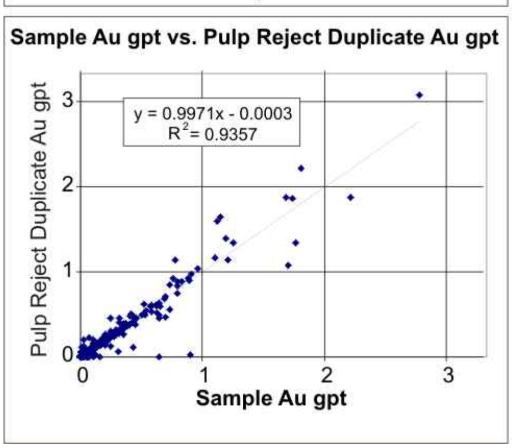



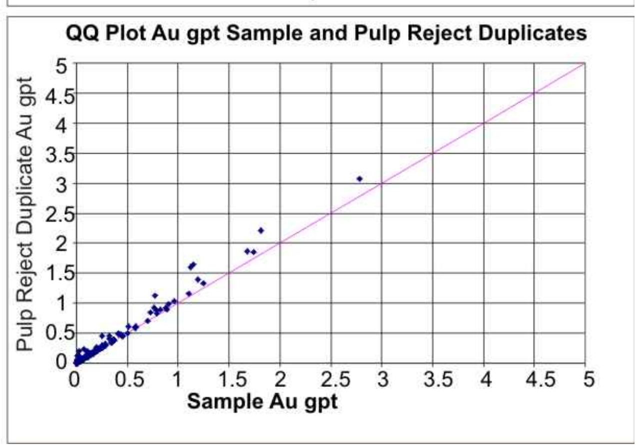
Assay Results for 2008 Coarse Reject Duplicate Sample Pairs



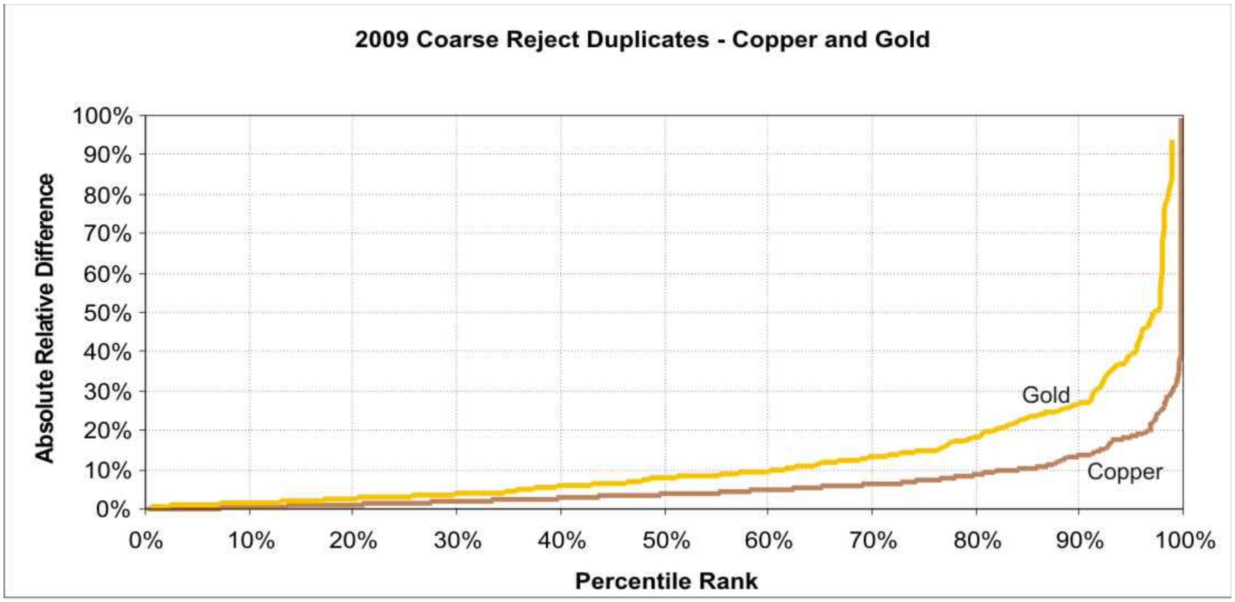



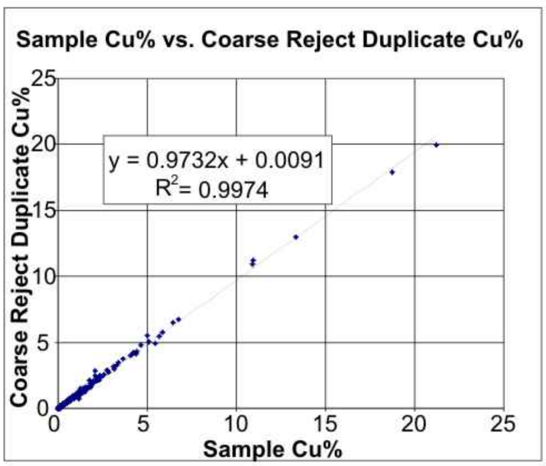



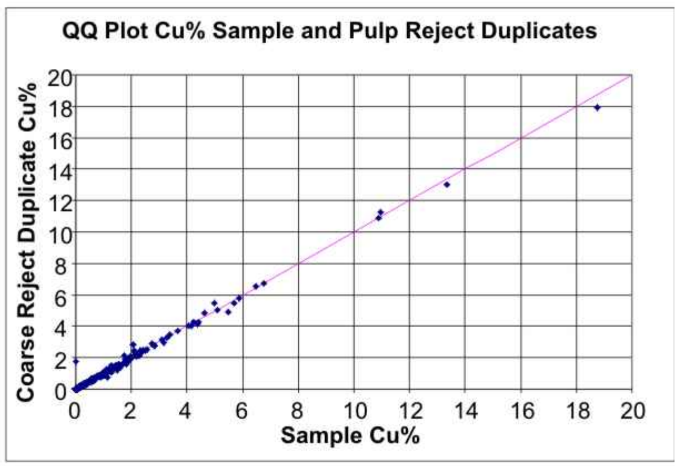



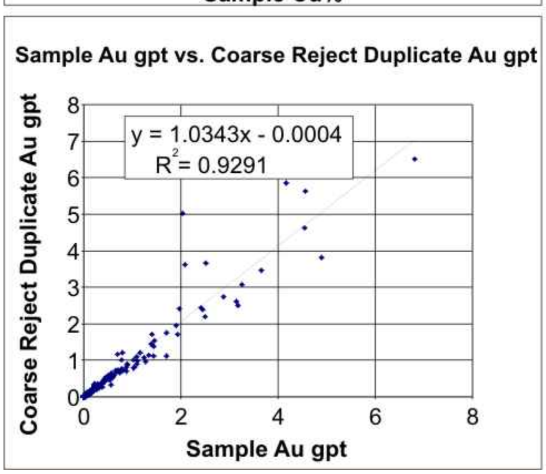



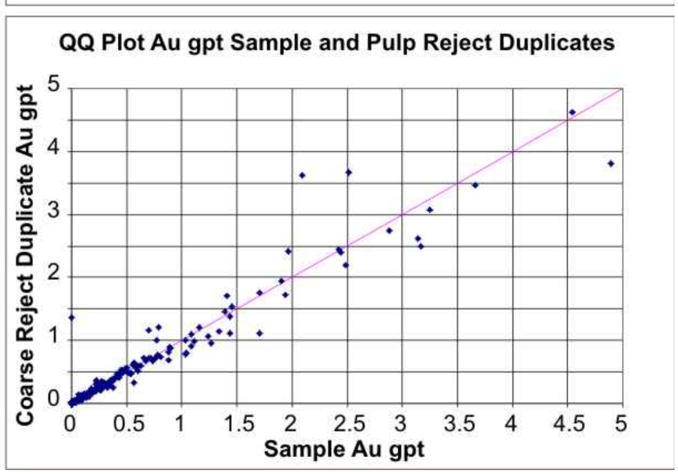
Assay Results for 2009 Coarse Reject Duplicate Sample Pairs



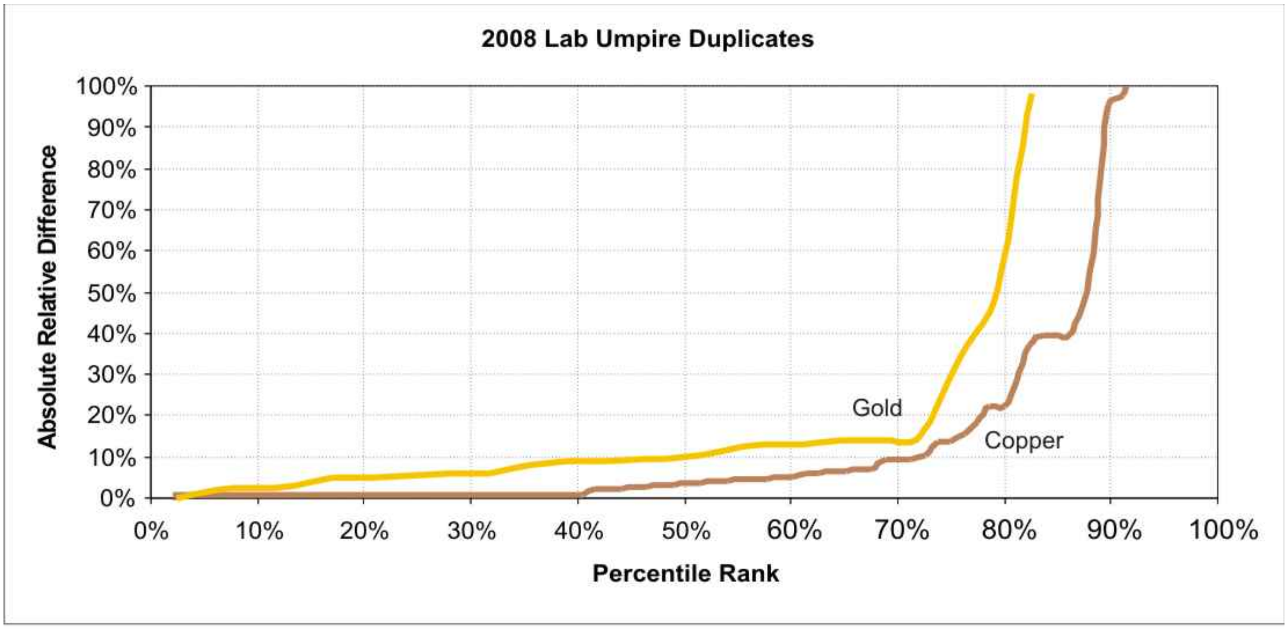



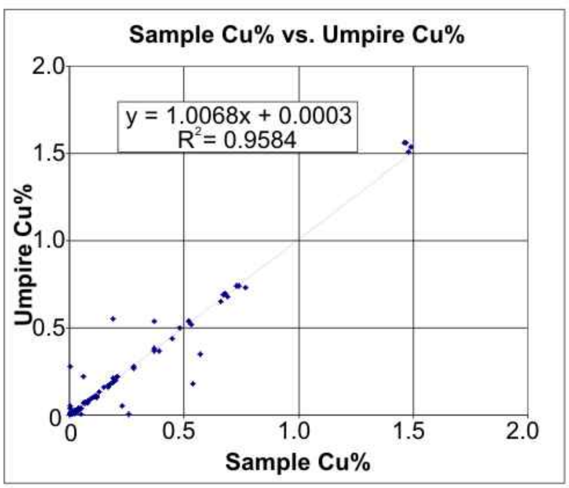



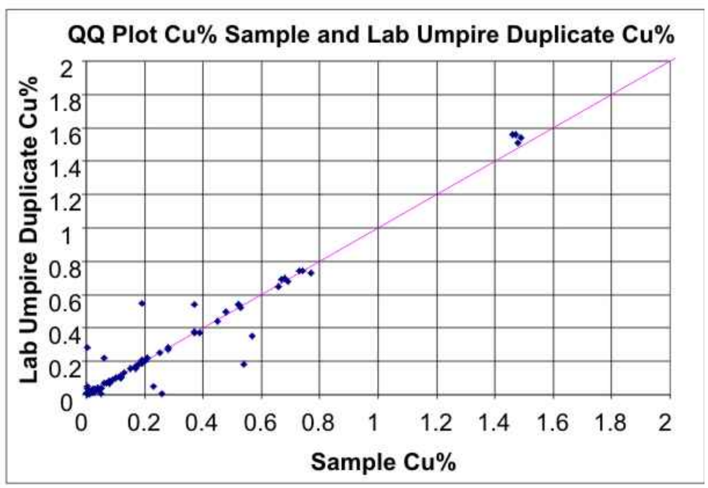



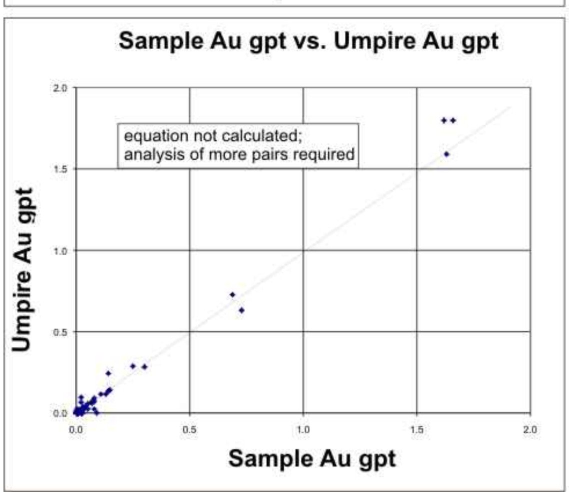



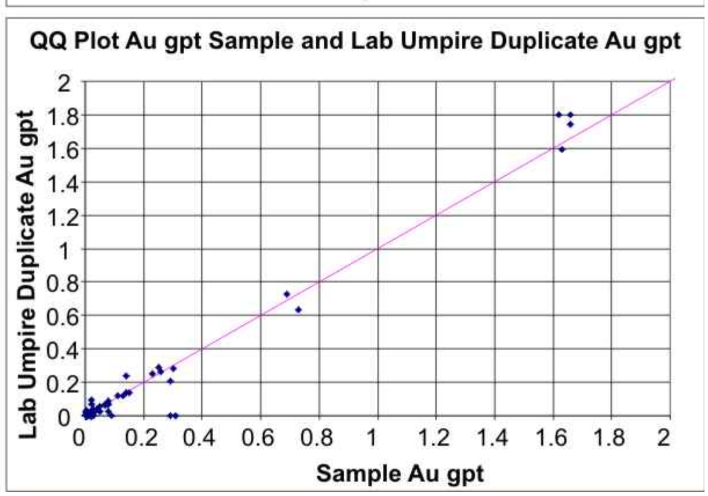
Assay Results for 2008 Umpire Duplicate Sample Pairs



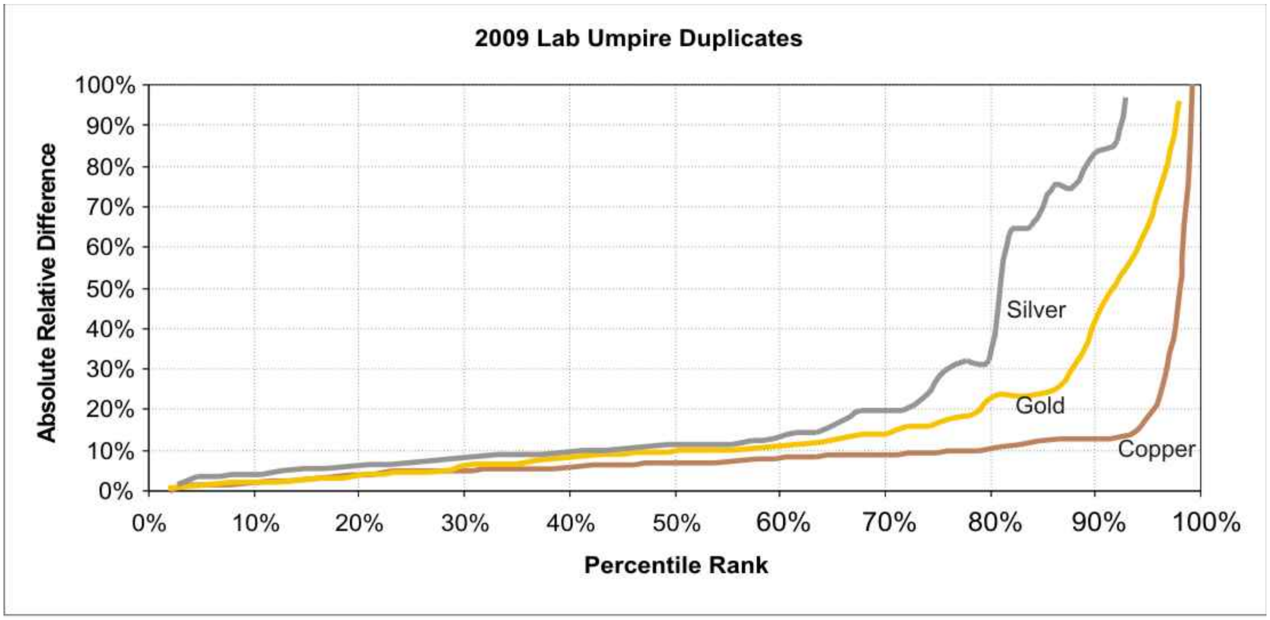



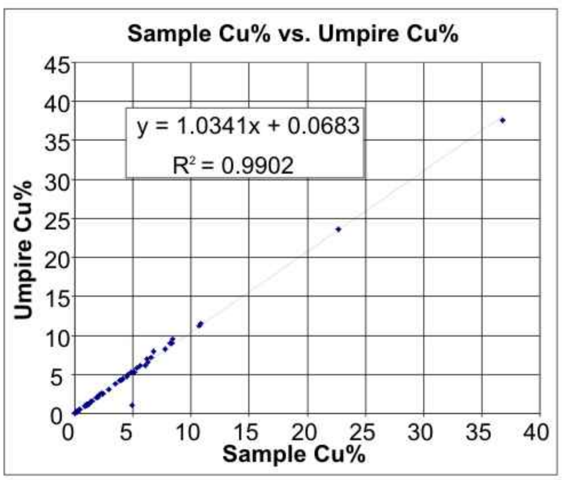



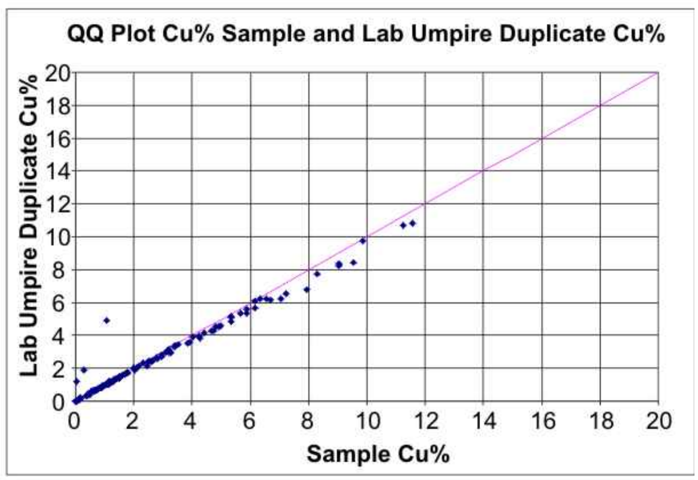



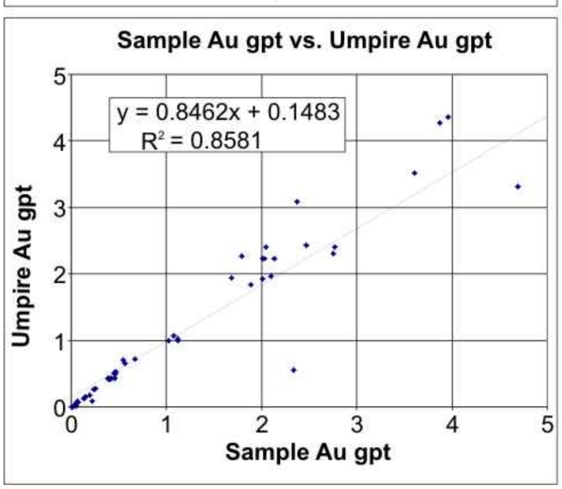



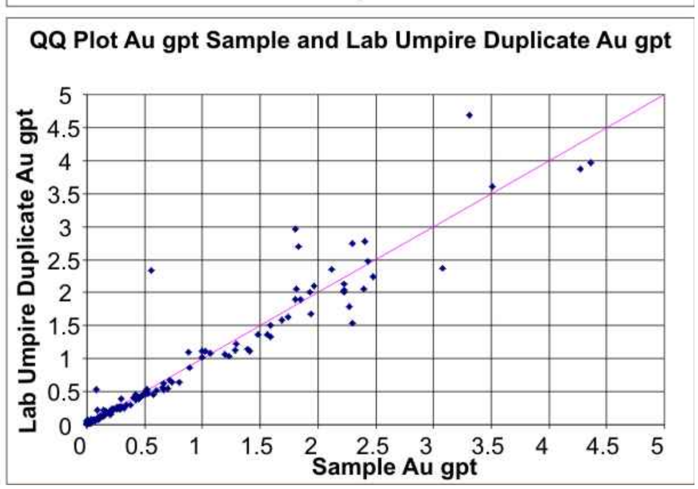
Assay Results for 2009 Umpire Duplicate Sample Pairs

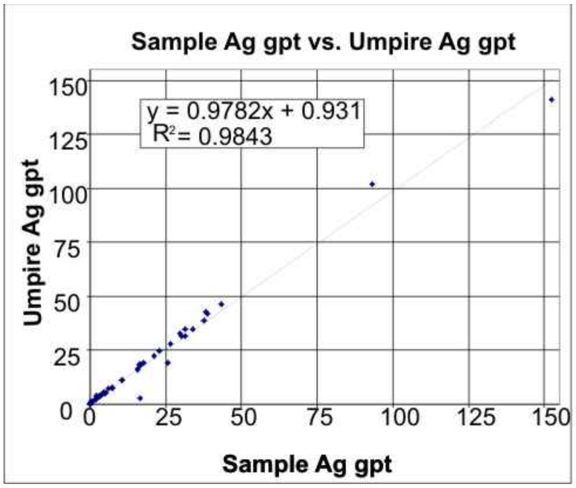


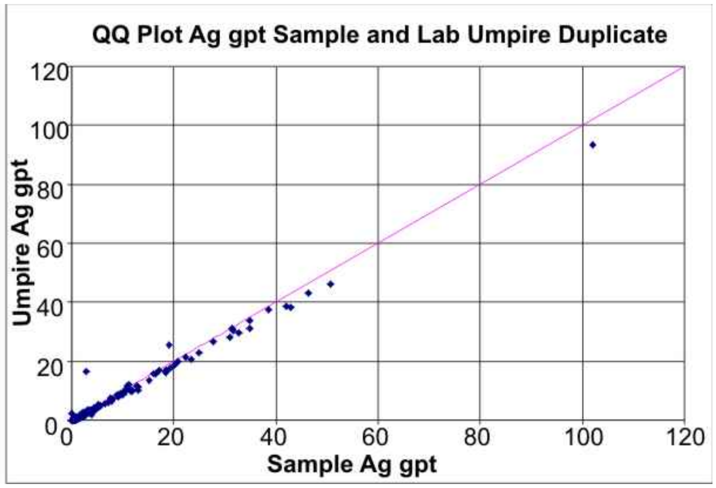




Sample Ag gpt vs. Umpire Ag gpt
y = 0.9782x + 0.931
R²= 0.9843
Umpire Ag gpt
Sample Ag gpt
0
25
50
75
100
125
150


QQ Plot Ag gpt Sample and Lab Umpire Duplicate
Umpire Ag gpt
Sample Ag gpt
0
20
40
60
80
100
120

# Appendix A2

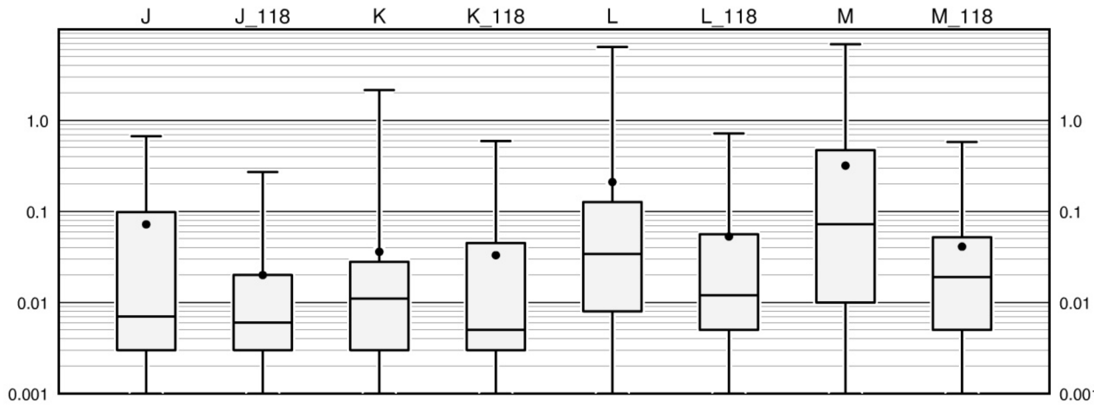
**Statistics of Gold and Silver Assays**

**and**

**Variogram Models of Gold Grades**

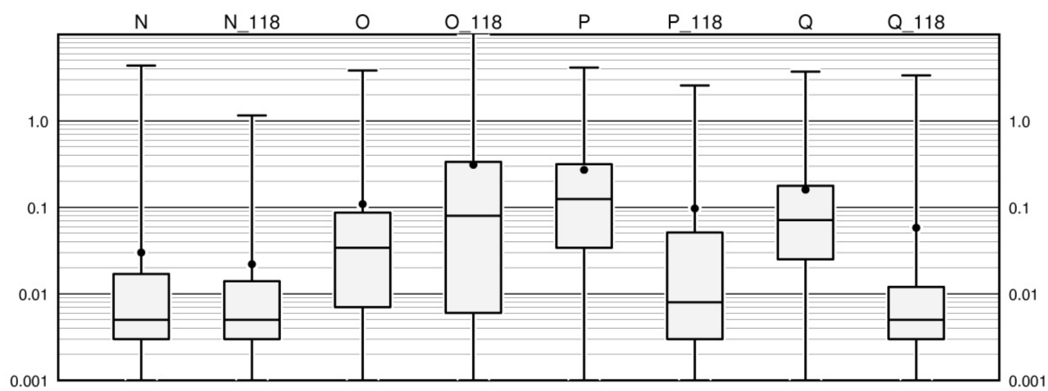
**Area 2/118 Deposit**

Au Declustered Composites




| | J | J_118 | K | K_118 | L | L_118 | M | M_118 | |
|---|---|---|---|---|---|---|---|---|---|
| Number of data | 248 | 113 | 675 | 332 | 1218 | 326 | 1915 | 373 | Number of data |
| Mean | 0.072 | 0.02 | 0.036 | 0.033 | 0.21 | 0.053 | 0.318 | 0.041 | Mean |
| Std. Dev. | 0.128 | 0.036 | 0.114 | 0.062 | 0.55 | 0.099 | 0.522 | 0.067 | Std. Dev. |
| Coef. of Var. | 1.776 | 1.837 | 3.183 | 1.866 | 2.625 | 1.874 | 1.642 | 1.611 | Coef. of Var. |
| Maximum | 0.668 | 0.27 | 2.15 | 0.592 | 6.37 | 0.715 | 6.85 | 0.58 | Maximum |
| Upper quartile | 0.098 | 0.02 | 0.028 | 0.045 | 0.127 | 0.056 | 0.47 | 0.052 | Upper quartile |
| Median | 0.007 | 0.006 | 0.011 | 0.005 | 0.034 | 0.012 | 0.072 | 0.019 | Median |
| Lower quartile | 0.003 | 0.003 | 0.003 | 0.003 | 0.008 | 0.005 | 0.01 | 0.005 | Lower quartile |
| Minimum | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | Minimum |

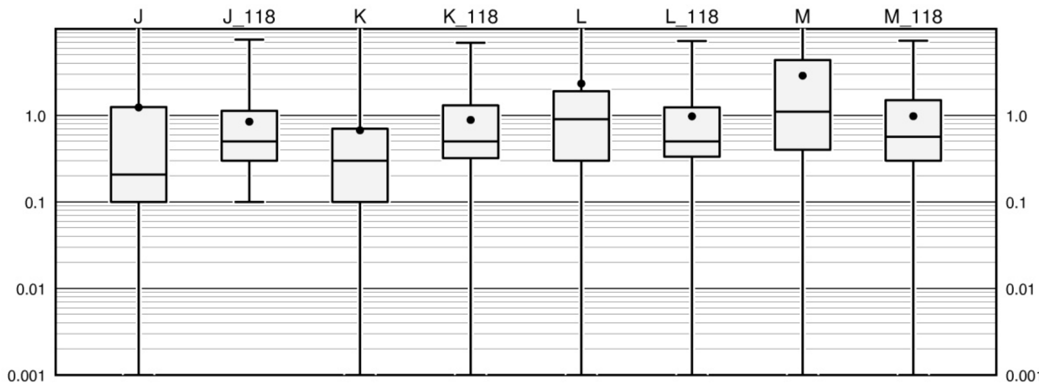
Au Declustered Composites




| | N | N_118 | O | O_118 | P | P_118 | Q | Q_118 | |
|---|---|---|---|---|---|---|---|---|---|
| Number of data | 3492 | 468 | 751 | 528 | 562 | 234 | 612 | 220 | Number of data |
| Mean | 0.03 | 0.022 | 0.109 | 0.31 | 0.271 | 0.097 | 0.16 | 0.058 | Mean |
| Std. Dev. | 0.172 | 0.071 | 0.271 | 0.721 | 0.424 | 0.307 | 0.301 | 0.278 | Std. Dev. |
| Coef. of Var. | 5.803 | 3.32 | 2.493 | 2.329 | 1.564 | 3.151 | 1.875 | 4.836 | Coef. of Var. |
| Maximum | 4.36 | 1.154 | 3.83 | 10.271 | 4.114 | 2.56 | 3.712 | 3.36 | Maximum |
| Upper quartile | 0.017 | 0.014 | 0.087 | 0.336 | 0.315 | 0.051 | 0.178 | 0.012 | Upper quartile |
| Median | 0.005 | 0.005 | 0.034 | 0.08 | 0.125 | 0.008 | 0.071 | 0.005 | Median |
| Lower quartile | 0.003 | 0.003 | 0.007 | 0.006 | 0.034 | 0.003 | 0.025 | 0.003 | Lower quartile |
| Minimum | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | Minimum |

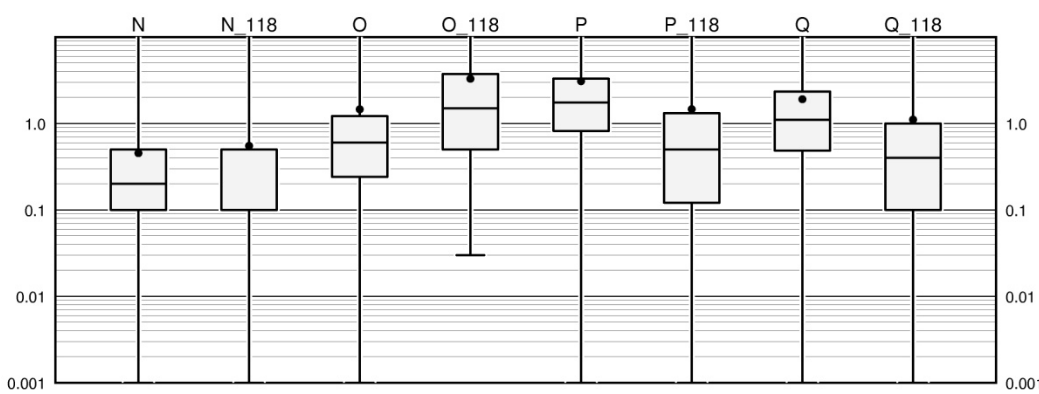
## Area 2/118 Deposit

### *Ag Declustered Composites*




| | J | J_118 | K | K_118 | L | L_118 | M | M_118 | |
|---|---|---|---|---|---|---|---|---|---|
| Number of data | 248 | 113 | 675 | 332 | 1218 | 326 | 1915 | 373 | Number of data |
| Mean | 1.235 | 0.846 | 0.672 | 0.885 | 2.331 | 0.975 | 2.874 | 0.979 | Mean |
| Std. Dev. | 2.703 | 0.993 | 1.524 | 0.907 | 5.187 | 1.175 | 3.744 | 0.989 | Std. Dev. |
| Coef. of Var. | 2.189 | 1.174 | 2.266 | 1.024 | 2.225 | 1.204 | 1.303 | 1.01 | Coef. of Var. |
| Maximum | 19.89 | 7.5 | 20.57 | 6.878 | 65.413 | 7.2 | 30.3 | 7.3 | Maximum |
| Upper quartile | 1.25 | 1.132 | 0.7 | 1.302 | 1.9 | 1.24 | 4.355 | 1.5 | Upper quartile |
| Median | 0.207 | 0.5 | 0.3 | 0.5 | 0.9 | 0.5 | 1.1 | 0.567 | Median |
| Lower quartile | 0.1 | 0.3 | 0.1 | 0.32 | 0.3 | 0.332 | 0.4 | 0.3 | Lower quartile |
| Minimum | 0.001 | 0.1 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | Minimum |

### *Ag Declustered Composites*



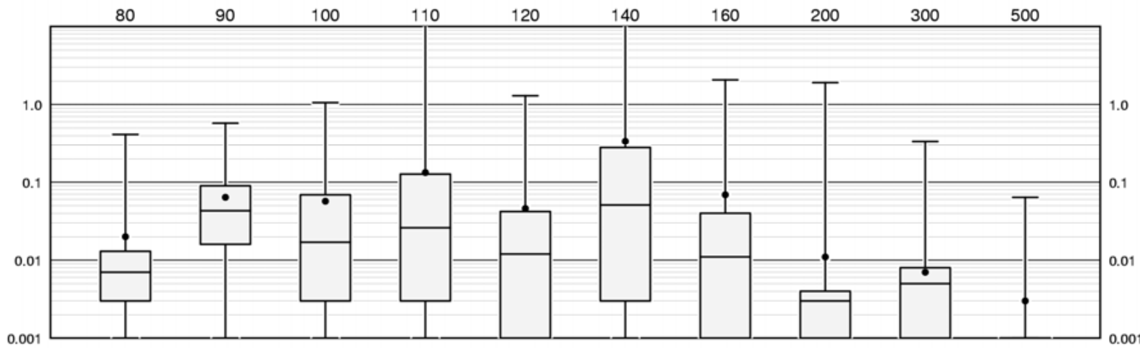

| | N | N_118 | O | O_118 | P | P_118 | Q | Q_118 | |
|---|---|---|---|---|---|---|---|---|---|
| Number of data | 3492 | 468 | 751 | 528 | 562 | 234 | 612 | 220 | Number of data |
| Mean | 0.453 | 0.551 | 1.458 | 3.287 | 3.082 | 1.462 | 1.906 | 1.108 | Mean |
| Std. Dev. | 1.248 | 1.113 | 3.452 | 5.017 | 4.739 | 3.465 | 2.909 | 3.909 | Std. Dev. |
| Coef. of Var. | 2.753 | 2.021 | 2.367 | 1.526 | 1.537 | 2.371 | 1.526 | 3.527 | Coef. of Var. |
| Maximum | 29.533 | 19.22 | 45.6 | 35.84 | 51.09 | 40.0 | 38.0 | 49.9 | Maximum |
| Upper quartile | 0.5 | 0.5 | 1.215 | 3.734 | 3.3 | 1.313 | 2.341 | 1.0 | Upper quartile |
| Median | 0.2 | 0.5 | 0.6 | 1.5 | 1.744 | 0.5 | 1.103 | 0.399 | Median |
| Lower quartile | 0.1 | 0.1 | 0.241 | 0.5 | 0.82 | 0.121 | 0.486 | 0.1 | Lower quartile |
| Minimum | 0.001 | 0.001 | 0.001 | 0.03 | 0.001 | 0.001 | 0.001 | 0.001 | 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 | 0.35 | -60 | 0 | 0 | 90 | 30 | 10 |
| | | 0.60 | | | | 180 | 50 | 20 |
| K | 0.25 | 0.60 | 45 | 0 | 0 | 20 | 50 | 15 |
| | | 0.15 | | | | 40 | 100 | 17 |
| L | 0.20 | 0.60 | 45 | 0 | 0 | 75 | 160 | 30 |
| | | 0.20 | | | | 600 | 200 | 60 |
| M | 0.20 | 0.55 | 100 | 18 | -37 | 50 | 180 | 35 |
| | | 0.25 | | | | 450 | 200 | 45 |
| N | 0.10 | 0.50 | 45 | 15 | 0 | 15 | 60 | 10 |
| | | 0.40 | | | | 40 | 70 | 30 |
| O | 0.30 | 0.60 | 45 | 0 | 0 | 30 | 150 | 18 |
| | | 0.10 | | | | 80 | 180 | 22 |
| P | 0.15 | 0.30 | 45 | 15 | 0 | 20 | 20 | 20 |
| | | 0.55 | | | | 120 | 120 | 25 |
| Q | 0.10 | 0.50 | 75 | 15 | 0 | 50 | 80 | 15 |
| | | 0.40 | | | | 90 | 170 | 80 |
| All 118 | 0.10 | 0.60 | 60 | 15 | 0 | 90 | 70 | 20 |
| | | 0.30 | | | | 110 | 90 | 75 |

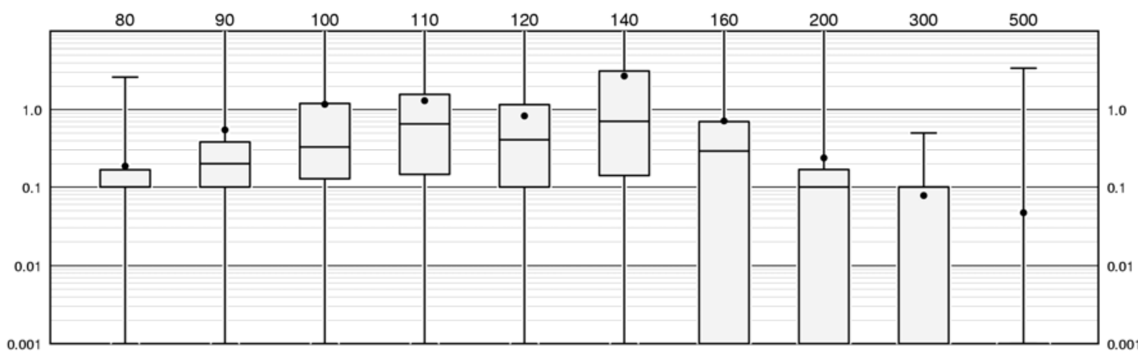
## Rigetop Deposit

**Au Declustered 1.5m Composite Data**




| | 80 | 90 | 100 | 110 | 120 | 140 | 160 | 200 | 300 | 500 | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Number of data | 169 | 259 | 778 | 1041 | 750 | 723 | 902 | 5928 | 100 | 484 | Number of data |
| Mean | 0.02 | 0.064 | 0.057 | 0.133 | 0.046 | 0.337 | 0.069 | 0.011 | 0.007 | 0.003 | Mean |
| Std. Dev. | 0.047 | 0.071 | 0.097 | 0.445 | 0.1 | 1.632 | 0.218 | 0.056 | 0.018 | 0.01 | Std. Dev. |
| Coef. of Var. | 2.419 | 1.119 | 1.695 | 3.348 | 2.158 | 4.84 | 3.157 | 5.283 | 2.539 | 2.937 | Coef. of Var. |
| Maximum | 0.412 | 0.577 | 1.058 | 16.132 | 1.297 | 34.855 | 2.071 | 1.909 | 0.335 | 0.064 | Maximum |
| Upper quartile | 0.013 | 0.09 | 0.069 | 0.127 | 0.042 | 0.279 | 0.04 | 0.004 | 0.008 | 0.001 | Upper quartile |
| Median | 0.007 | 0.043 | 0.017 | 0.026 | 0.012 | 0.051 | 0.011 | 0.003 | 0.005 | 0.001 | Median |
| Lower quartile | 0.003 | 0.016 | 0.003 | 0.003 | 0.001 | 0.003 | 0.001 | 0.001 | 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 |

**Ag Declustered 1.5m Composite Data**



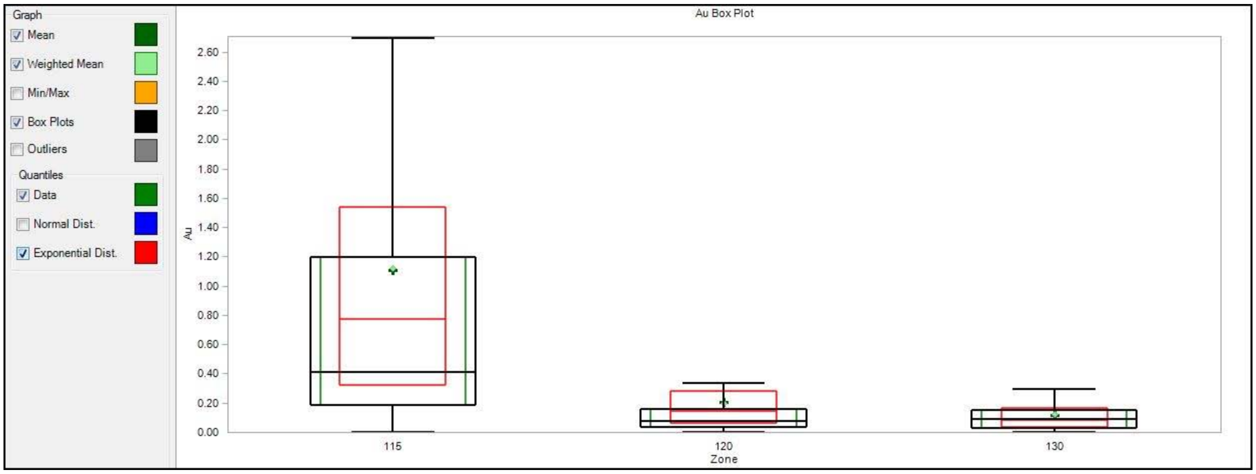

| | 80 | 90 | 100 | 110 | 120 | 140 | 160 | 200 | 300 | 500 | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Number of data | 169 | 259 | 778 | 1041 | 750 | 723 | 902 | 5928 | 100 | 484 | Number of data |
| Mean | 0.186 | 0.548 | 1.169 | 1.301 | 0.83 | 2.703 | 0.715 | 0.237 | 0.078 | 0.047 | Mean |
| Std. Dev. | 0.256 | 2.005 | 2.849 | 2.089 | 1.308 | 5.026 | 1.745 | 0.655 | 0.074 | 0.199 | Std. Dev. |
| Coef. of Var. | 1.376 | 3.66 | 2.437 | 1.605 | 1.577 | 1.86 | 2.44 | 2.769 | 0.953 | 4.268 | Coef. of Var. |
| Maximum | 2.616 | 30.3 | 77.8 | 23.772 | 12.91 | 38.9 | 20.142 | 15.464 | 0.5 | 3.4 | Maximum |
| Upper quartile | 0.167 | 0.384 | 1.2 | 1.568 | 1.157 | 3.128 | 0.7 | 0.168 | 0.1 | 0.001 | Upper quartile |
| Median | 0.1 | 0.2 | 0.33 | 0.653 | 0.409 | 0.705 | 0.29 | 0.1 | 0.1 | 0.001 | Median |
| Lower quartile | 0.1 | 0.1 | 0.128 | 0.146 | 0.1 | 0.141 | 0.001 | 0.001 | 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**

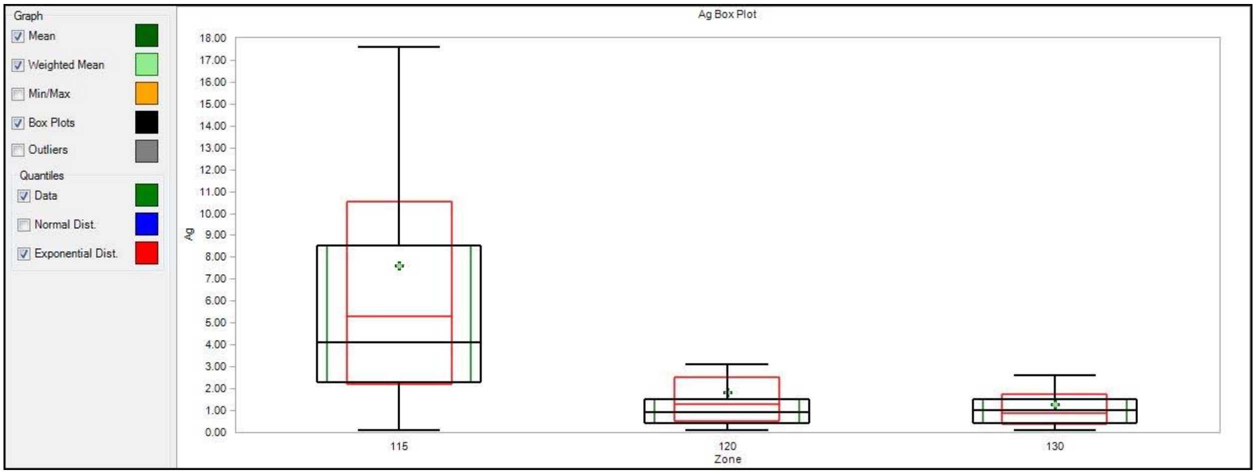
| Zone | | Nugget $C_0$ | Sill $C_1$ and $C_2$ | Gemcom Rotations (RRR rule) | | | Ranges $a_1$, $a_2$ | | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | around Z | around Y | around Z | X-Rot | Y-Rot | Z-Rot |
| 80* | | 0.05 | 0.75 | 50 | 24 | -48 | 100 | 40 | 20 |
| | | | 0.20 | | | | 200 | 140 | 25 |
| 90* | | 0.05 | 0.75 | 50 | 24 | -48 | 100 | 40 | 20 |
| | | | 0.20 | | | | 200 | 140 | 25 |
| 100 | | 0.05 | 0.75 | 50 | 24 | -48 | 100 | 40 | 20 |
| | | | 0.20 | | | | 200 | 140 | 25 |
| 110 | | 0.10 | 0.75 | 50 | 24 | -22 | 25 | 45 | 20 |
| | | | 0.15 | | | | 100 | 600 | 25 |
| 120 | | 0.20 | 0.30 | 50 | 24 | -48 | 90 | 25 | 8 |
| | | | 0.50 | | | | 110 | 60 | 15 |
| 140 | | 0.10 | 0.50 | 50 | 24 | -48 | 45 | 30 | 10 |
| | | | 0.40 | | | | 120 | 60 | 45 |
| 160 | | 0.10 | 0.30 | 50 | 24 | -48 | 60 | 30 | 18 |
| | | | 0.60 | | | | 120 | 70 | 25 |

* Variogram models assigned from Domain 100

## 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 |

# Appendix B
# Geotechnical

# Pre-Feasibility Geotechnical Evaluation
# Phase IV
# Minto Mine
# Yukon Territory, Canada

**Report Prepared for**

## Minto Explorations Ltd.

**Report Prepared by**

SRK Consulting
Engineers and Scientists

**December 2009**

# Pre-Feasibility Geotechnical Evaluation for Phase IV
# Minto Mine
# Yukon Territory, Canada

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**SRK Project Number 2CM022.006**

**December 2009**

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# Executive Summary

SRK Consulting (US), Inc. (SRK) was requested by Minto Explorations Ltd. (Minto) to carry out a pre-feasibility level geotechnical evaluation for the Area 2, Area 118, Ridgetop and Minto North deposit areas at the Minto Mine in the Yukon Territory, Canada. 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;
- Recommendation of optimal pit slope angles and pit architecture for mine design purposes; and,
- Recommendation of room and pillar dimensions as well as ground support requirements for the alternative underground development of Area 118.

As commissioned, the work reported herein 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 coreholes drilled for this investigation, data from three additional geotechnical coreholes drilled in 2007 as part of the previous SRK (2007) Area 2 Pre-feasibility Pit Slope Evaluation were also considered in the analyses.

## Laboratory 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.

Triaxial compressive strength (TCS) testing was conducted on six samples of core, yielding compressive strengths ( $_1$) ranging between 213.8 and 294 .1MPa with a mean value of 262.1 MPa under confining pressures ( $_3$) ranging between 6.9 and 20.7 MPa, with a mean value of 13.8 MPa.

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 disk tension 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/m3 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.

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 do 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.

## 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 90m 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 1m 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 15m in depth in the southwest portion, with up 20 to 45m along much of the north and east walls, and reaching a maximum depth of 70m 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 5m to 15m of completely weathered, in-situ bedrock (granular soil) or residuum.

Based on geotechnical drillhole data, the Area 2 weathered domain is adjudged to extend to depths of approximately 50 to 100m below the current ground surface.

## Area 118

The majority of the proposed Area 118 open pit footprint is covered with up to approximately 5m of overburden, except in its southwestern portion, where the soil locally deepens to approximately16m. The depth of bedrock weathering at Area 118 is generally to about 30 to 60m below the current ground surface.

## Ridgetop

The western regions of the proposed Ridgetop pits are anticipated to contain 1 to 5m of soil overburden, deepening to the east to from 5 to 15m on the east side and with a maximum depth of 21m 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 70m below current ground surface.

## 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.

## 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 ($m_i$) 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 (Slide), 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.

### Results of Interramp/Overall Slope Stability Modeling

| Deposit | Sector | Height (m) | Mean FoS | PoF (%) |
|---|---|---|---|---|
| Area 2 | Northeast | 130m | 2.5 | 0.7 |
| Area 2 | Southwest | 214m | 2.1 | 2.9 |
| Ridgetop | - | 130m | 2.3 | 2.4 |
| Minto North | - | 130m | 2.3 | 0.0 |

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 modeling 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 of the 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 southwestern 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 279m and 243m, 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 drillholes 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.

Major faults at similar orientations are also anticipated through the Area 118 underground mining areas and development.

## 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 modeled 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 porewater 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 sufficient persistence, 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, backbreak 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.

In analyses, the interramp angle is typically incrementally increased until a suitable probability of failure equal to or greater than 30% is achieved. The probabilities of instability are plotted against their respective interramp slope angles for each model and the slope angle expected to yield a suitable probability of instability (5% or 30%, depending on failure consequence) is determined.

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, pit slope design recommendations for each of the Minto areas are summarized below.

### Summary of Pit Slope Design Recommendations

| Deposit Area | Sector(s) | Max. Slope Height (m) | Interramp Angle (°) | Bench Face Angle (°) | Bench Height (m) | Berm Width (m) | Stepout 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 7m, a 15m offset or stepout should be incorporated at, or vertically near, the contact between the overburden and the bedrock.

## Area 118 Underground Pillar Assessment

In addition to the small open pit at Area 118 previously discussed, underground mining is also planned for Area 118. Based on the geotechnical data previously described, pillar strengths were evaluated in order to recommend suitable pillar dimensions for room and pillar mining. Based on estimates of ore deposit depth and thickness variability, pillar heights of 5m, 10m and 15m were assessed and ore depths, and respective overburden stresses, of 150m, 200m and 250m were considered.

In-situ Rock Mass Rating (IRMR) and Rock Mass Strength (RMS) values were evaluated for the ore zone as well as materials above and below the ore zone in geotechnical drillholes C09-01 and C09-02. An average IRMR and RMS of 55 and 60MPa, respectively, were conservatively estimated for pillar, roof and floor materials. Using Laubscher's (1990) method, the IRMR of 55 was reduced to a Mining Rock Mass Rating (MRMR) of 47 and the 60 MPa RMS to a Design Rock Mass Strength (DRMS) of 51 MPa by applying appropriate reductions for joint orientation, blasting and water.

Based on empirical data presented by Ouchi (2004), assuming a RMR value of 55, the maximum unsupported span distance was estimated to be 6m for all pillar height/deposit depth combinations considered. Subsequently, the tributary area method was used to estimate minimum pillar dimensions required to support 6m x 6m or, if required, lesser, roof spans based on pillar height and overburden stresses. The resultant recommended room and pillar dimensions and extraction ratios are summarized below.

### Room and Pillar Size Recommendations

| Depth (m) | Pillar Height (m) | Pillar Dimensions (m) | Room Dimensions (m) | Extraction Ratio |
|---|---|---|---|---|
| 150 | 5 | 4x4 | 6x6 | 84% |
| 150 | 10 | 5x5 | 6x6 | 79% |
| 150 | 15 | 6x6 | 6x6 | 75% |
| 200 | 5 | 4.5x4.5 | 6x6 | 82% |
| 200 | 10 | 6x6 | 6x6 | 75% |
| 200 | 15 | 7.5x7.5 | 6x6 | 69% |
| 250 | 5 | 5x5 | 6x6 | 79% |
| 250 | 10 | 7x7 | 6x6 | 71% |
| 250 | 15 | 8x8 | 5x5 | 62% |

Based on geotechnical conditions previously described, ground support requirements for development such as declines were estimated as follows:

- Pattern bolting with 2.4m long bolts at a 2m spacing within and between rings; and,
- Welded wire mesh in back and top of walls.

## Recommendations for Additional 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 drillholes 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 portion of Area 118 will also require additional geotechnical drilling for rock mass characterization at the feasibility and design levels. The Area 118 and Ridgetop open pits most likely will not require additional geotechnical drilling unless major changes are made to the current plans.

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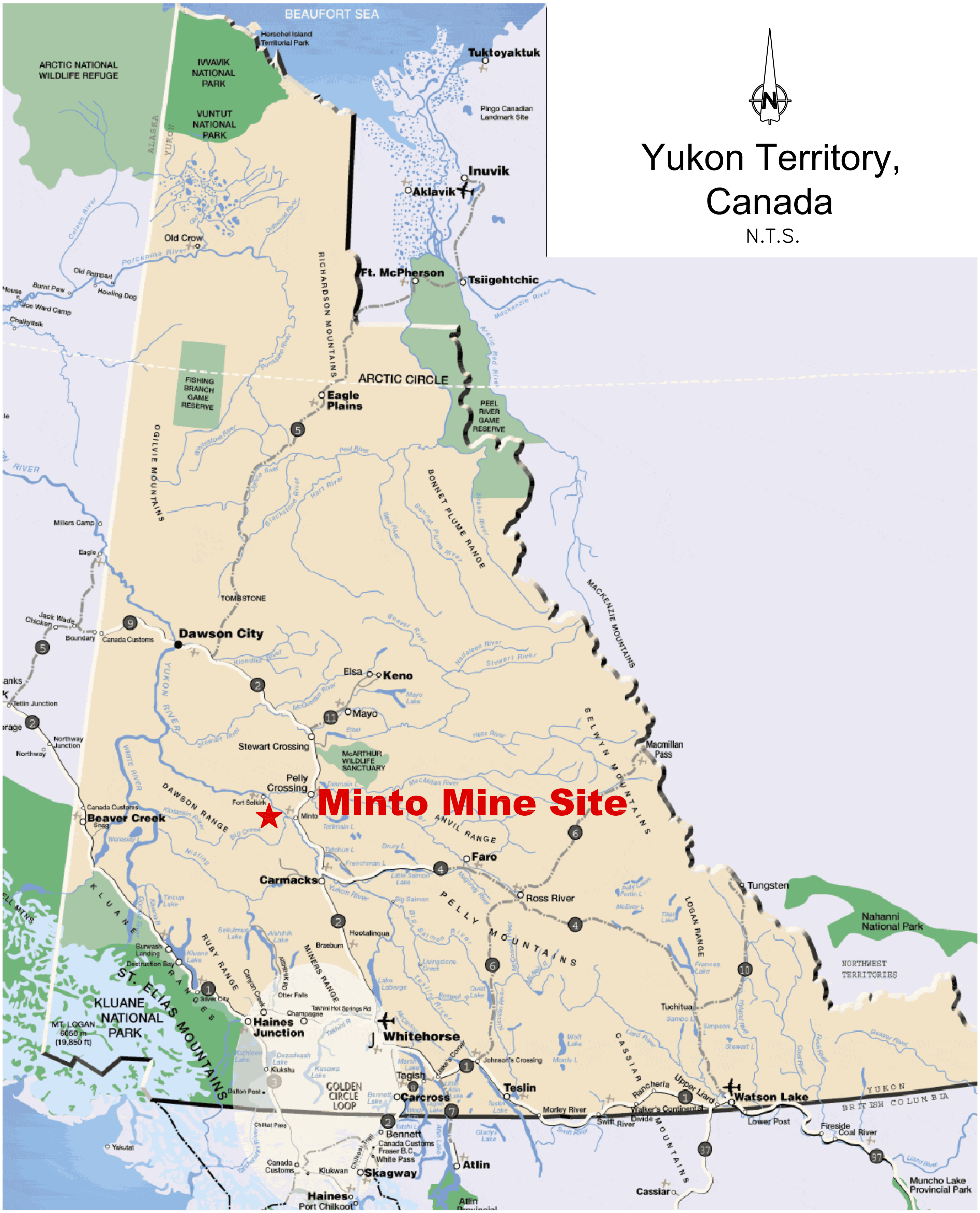
# 1 Introduction and Background

SRK Consulting (US), Inc. (SRK) was requested by Minto Explorations Ltd. (Minto) to carry out a pre-feasibility level geotechnical evaluation for the Area 2, Area 118, Ridgetop and Minto North deposit areas at the Minto Mine in the Yukon Territory, Canada (Figure 1).

This report presents a complete description of the methods used to collect pertinent information, the information so gathered, the analytical tools employed to produce assessments of the anticipated behavior of the geologic environments to the development of the open pits and, in the case of Area 118, the underground, and the recommendations based upon those assessments.




# Yukon Territory, Canada

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MINTO EXPLORATIONS LTD.

PREFEASIBILITY GEOTECHNICAL EVALUATION

SITE LOCATION MAP

MINTO MINE

SRK JOB NO.: 2CM022.006

FILE NAME: 02CM022.06.Fig.1-1.Rev.A.Site.Location.Map.2009-11-23.dwg

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# 2 Program Objectives and Work Program

## 2.1 Program Objectives

The primary objectives of the pre-feasibility geotechnical evaluation for each Minto area were:

- To collect and to assimilate geotechnical information pertaining to the in-situ materials;
- To geotechnically characterize the in-situ materials;
- To undertake laboratory testing of geomechanical properties of samples of the in-situ materials;
- To develop a geotechnical model to serve as the basis for geomechanical analyses;
- To conduct geomechanical analyses;
- To make recommendations pertaining to optimal slope angles and pit architecture for mine design purposes; and,
- To make recommendations pertaining to pillar and room dimensions for the potential Area 118 underground development.

## 2.2 Work Program

The principle stages of the geotechnical evaluation work program were comprised of the following:

- Recommendation of the number, location and orientation of core holes necessary to characterize the in-situ materials in each of the areas;
- Geotechnical core logging and discontinuity orientation of core recovered from the holes;
- Selection of representative drill core samples from the respective lithological units encountered in the geotechnical drill holes;
- Submission of the representative samples to the University of Arizona Rock Mechanics Laboratory in Tucson, Arizona, for geomechanical testing;
- Analyses and interpretation of the geotechnical data and laboratory test results to produce a comprehensive analytical model of in-situ conditions for each of the study areas;
- Examination of the behavior of each geotechnical model to expected mining-induced stresses, using various analytical methods; and,
- The compilation of a pre-feasibility geotechnical evaluation report incorporating recommendations pertaining to optimal pit slope angles and pit architecture for mine design purposes as well as room and pillar dimensions for the Area 118 underground.

As commissioned, the work reported herein was performed at a pre-feasibility design level.

# 3 Geologic Setting

The Minto region is located within the central portion of the accretionary complex known as the Yukon-Tanana (YT) terrane which lies between continental margin rocks of ancestral North America to the east and arc and oceanic terranes accreted in Mesozoic time to the west. The pericratonic YT terrane is comprised of Proterozoic and Paleozoic metamorphic rock intruded by Mesozoic plutons and covered by extrusive volcanics of Upper Cretaceous and Tertiary age (Colpron 2006).

The YT terrane is located within the western portion of the Omineca Belt of the Cordillera which is composed of variably metamorphosed sedimentary and igneous rocks that have undergone similar geomorphologic processes over the past billion years of geological history, climate and glaciation. Much of the north-western portion of the Omineca Belt including the Minto region was not glaciated during the most recent event resulting in a thicker cover of soil and weathered rock in some areas of the region (Hart 2002).

The Minto Mine site is located within the Klotassin batholith, an intrusive granitic pluton which intruded the YT terrane in early Jurassic time. The Klotassin batholith consists primarily of granodiorite but varies in composition from quartz diorite to quartz monzonite. The area to the south of the Minto mine site is covered with basalt and andesite flows of the Upper Cretaceous, Carmacks Group. The batholith is intruded by basalt and andesite dikes believed to have been feeders of the Carmacks Group volcanics. Quartz-feldspar pegmatite veins and dikes are also common in the Klatossin batholith (Hatch 2006).

Four separate deposits of mineralization were considered for this evaluation. They are the Area 2, Area 118, Ridgetop and Minto North deposits. Each of these deposits has similar shallow dipping copper sulphide mineralized zones. Area 2 and Area 118 area located immediately south of Main Minto deposit which is already exposed in open pit mining. The Ridgetop deposit is located just over 300m south of the Area 2 and Area 118 deposits. The Minto North deposit is located about 700m north of the Main Minto deposit. These deposits define a general north-northwest trend.

Seismically, the Minto deposits lie within an area of moderate to low seismic activity. According to information available from the Canadian Geological Survey (CGS), the Minto area can expect to experience a maximum seismically-induced acceleration of approximately 0.1g (percent of gravity) with a recurrence interval of 50 years. Since each of the Minto deposits are scheduled to be relatively short lived, i.e., on the order of 8 years, the CGS guideline equates to a maximum anticipated acceleration of approximately 0.01g during mine life. This maximum design acceleration is so inconsequential that no seismic loadings were considered in the analyses conducted for this study.

# 4 Field Data Collection

## 4.1 Geotechnical Core Logging

Geotechnical logging, field point load testing and discontinuity orientation of core recovered from a total of eight drillholes were conducted for this investigation. Based on the current understanding of the deposits, drillhole locations and orientations were selected to provide the best coverage possible of rock likely to form pit walls in the Ridgetop, Area 118, Area 2 and Minto North areas. The geotechnical drillhole locations were chosen based on preliminary and historic pit shells and, in some instances, drillhole intersections with the final pre-feasibility pit slopes presented herein were not optimal. It is believed, however, that this factor does not adversely impact the analyses conducted to a significant degree.

The geotechnical core drilling program was also designed to collect data for rock mass characterization for potential underground mining at Area 118. In addition to the eight geotechnical coreholes drilled in 2009 for this investigation, data from three additional geotechnical coreholes drilled in 2007 for the previous SRK (2007) Area 2 Pre-feasibility Pit Slope Evaluation were also considered in the analyses.

Drillhole inclinations of approximately 60 degrees below the horizontal were selected since they were judged to be more likely, than would vertical holes, to intersect geologic structures such as joints and fracture systems which, if present, will influence slope stability.

Collar locations and the drillhole azimuths of the eight geotechnical holes drilled for this investigation as well as the three holes considered from the previous (SRK, 2007) investigation are summarized in Table 1 and presented on Figure 2.

**Table 1: Drillholes Oriented and Logged for Geotechnical Data**

| SRK Hole ID | Minto Hole ID | Collar Coordinates | | | Azimuth (deg) | Inclination (deg) | Length (m) |
|---|---|---|---|---|---|---|---|
| | | Northing | Easting | Elevation | | | |
| 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 | 058 | -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 | 047 | -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 |

### 4.1.1 Geotechnical Logging Procedures

Core retrieved from the eight geotechnical coreholes were logged on a 24 hour per day basis, at the rig, in the liners, or splits, prior to boxing and transporting. The 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 characterization and frequency of discontinuities. Specific parameters that were logged included:

- General lithology and structures;
- Total core recovery;
- Rock Quality Designation (RQD);
- Rock weathering and intact strength indices;
- Frequency of discontinuities;
- Discontinuity characteristics (type, roughness, infillings and wall condition); and,
- Discontinuity orientation.

Geotechnical corehole logs are presented in Appendix A.

During core logging, samples of the core were collected to provide redundant specimens for laboratory strength testing. Samples were collected at approximately 30 meter intervals, or when significant rock type or strength changes were apparent. Each sample was sealed and safely stored at the time of collection. Upon completion of the drilling, samples were shipped to SRK's office in Denver, Colorado, for test sample selection. Select samples were then repackaged and shipped to the University of Arizona Rock Mechanics Laboratory in Tucson, Arizona, for testing.

### 4.1.2 Core Drilling Method

The coreholes were drilled by Driftwood Diamond Drilling, Ltd., from Smithers, British Columbia, using a skid mounted drill rig with a 45.1mm I.D.(NQ3), 1.5m long triple-tube sampling barrel. The coreholes were advanced with a face discharge bit system using a polymer mixture to facilitate core recovery. This coring method allowed for the recovery of continuous core samples as the holes advanced.

Downhole surveys were conducted by Driftwood upon completion of drilling; subsequently, the surface casing was pulled and the hole allowed to collapse. Depth to groundwater could not be determined at the time of hole advancement due to the 24 hour per day drilling schedule with continuous fluid injection and circulation.

## 4.2 Discontinuity Orientation

Orientation of discontinuities in each run was accomplished using an A.C.T. core orientation system manufactured by Reflex Instruments. The depth, alpha angle and beta angle were measured for each discontinuity on all core runs that were successfully oriented. The beta angle, i.e., the angle from the lowest part of the ellipse formed by the intersection of each discontinuity with the core, was measured from the bottom of the core in a clockwise direction when looking down hole. The alpha angle was measured as the maximum angle made by the discontinuity with respect to the core axis.

It was possible to orient a total of 4,328 discontinuities out of the total 5,161 discontinuities logged (84%) in the eight geotechnical coreholes drilled for this evaluation. A summary of oriented core information by hole is presented in Table 2.

**Table 2: Summary of Discontinuity Orientation**

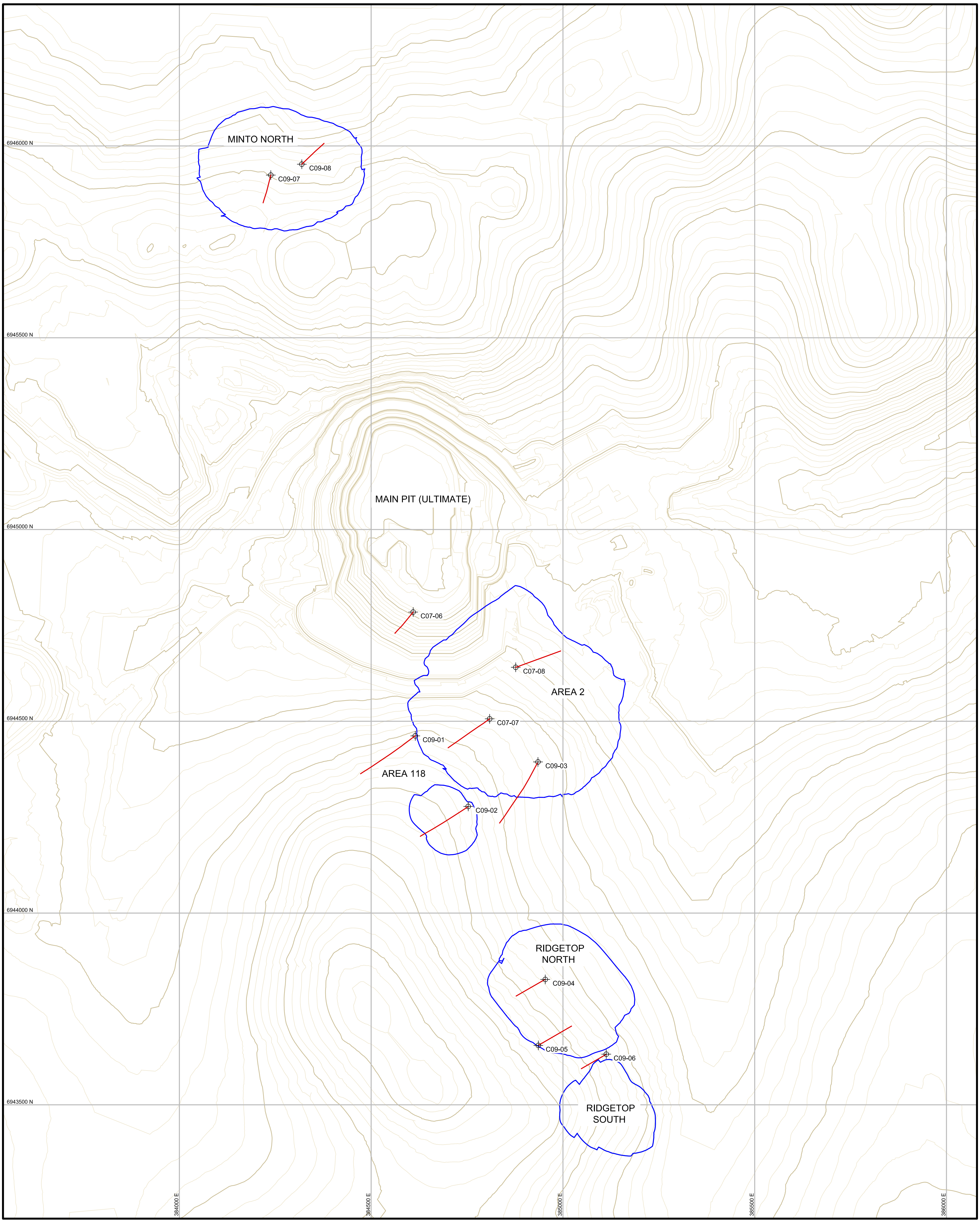
| SRK Hole ID | Drillhole Length (m) | Core Length Oriented (m) | Total Discontinuities Logged | Percentage of Discontinuities Oriented |
|---|---|---|---|---|
| C09-01 | 325.0 | 316.5 | 841 | 82% |
| C09-02 | 280.5 | 270.5 | 821 | 90% |
| C09-03 | 376.5 | 268.5 | 815 | 87% |
| C09-04 | 175.5 | 370.4 | 515 | 76% |
| C09-05 | 199.5 | 154.5 | 573 | 80% |
| C09-06 | 150.0 | 193.5 | 472 | 75% |
| C09-07 | 153.0 | 132.0 | 602 | 93% |
| C09-08 | 141.0 | 135.0 | 522 | 83% |
| C07-06 | 155.1 | 82.1 | 315 | 47% |
| C07-07 | 243.5 | 229.9 | 560 | 44% |
| C07-08 | 249.6 | 120.6 | 1194 | 60% |

## 4.3 Point Load Testing

A Point Load Test (PLT) was performed during core logging at a frequency of approximately one test per every 2 to 3m using a Roctest Pil-7 test machine to provide detailed and nearly continuous profiles of relative rock strength. PLTs were conducted according to International Society for Rock Mechanics (ISRM, 1985) procedures. Both axial (parallel to the long axis of the core) and diametral (perpendicular to the long axis of the core) loading tests were conducted. Axial point load testing was performed as samples suitable for testing in an axial orientation were obtained from coring or were produced by breaking especially long sticks of core in diametral tests.

A combined total of 640 point load tests were conducted on core from the eight geotechnical coreholes; of those, 496 met test criteria for passing test results. Point load indices ($Is_{(50)}$) were calculated from the field PLT data using the ISRM (1985) suggested method. Calculated point load index strengths ($Is_{(50)}$) ranged between 0.1 and 11.1 MPa, with an average of 4.6 MPa.

In addition to the tests routinely conducted at 2 to 3 meter intervals, at least one PLT was also performed adjacent to each UCS sample obtained for laboratory testing. The reason for the paired PLT and UCS samples was for estimation of a correlation factor for conversion of the field PLT tests to laboratory UCS values.

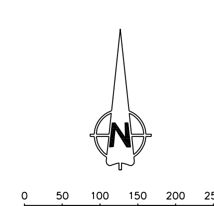



LEGEND

EXISTING GROUND CONTOURS (MAJOR/MINOR) 5 METER INTERVAL

C09-02 GEOTECHNICAL DRILLHOLE COLLAR LOCATION AND HORIZONTAL BOREHOLE PROJECTION

CURRENT PFS ULTIMATE PIT BOUNDARY

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| | | | | | |
|---|---|---|---|---|---|
| SRK Consulting Engineers and Scientists 7175 West Jefferson Ave. Suite 3000 Denver, Colorado 80235 303-985-1333 | MINTO EXPLORATIONS LTD. | PREFEASIBILITY GEOTECHNICAL EVALUATION | | | |
| | | LOCATIONS OF GEOTECHNICAL DRILLHOLES | | | |
| SRK JOB NO.: 02CM022.06 | MINTO MINE | DATE: DEC. 2009 | APPROVED: ML | FIGURE: 2 | REVISION NO. A |
| FILE NAME: 02CM022.06.Fig.4-1.Rev.A.Site.Map.and.Collar.Locations.2009-11-18.dwg | | | | | |

# 5 Laboratory Testing

Geomechanical testing was conducted at The University of Arizona Rock Mechanics Laboratory in Tucson, Arizona, to determine strength characteristics for the in-situ materials. The overall laboratory program consisted of direct shear, uniaxial and triaxial compressive strength, and direct tensile strength testing and measurements 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. After completion of the laboratory testing program, the tested samples were returned to SRK for further evaluation. Raw laboratory test data is included in Appendix B.

## 5.1 Unconfined Compressive Strength and Elastic Properties

Uniaxial compressive strength (UCS) testing was conducted on 30 samples according to ASTM Method D7012. Elastic properties (Young's Modulus and Poisson's Ratio) were measured for seven of the 30 UCS samples. Test results indicated UCS values 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. Results of the UCS and elastic properties testing are summarized in Table 3.

Three samples had an L/D ratio of less than 2.0 and, as a result, a correction factor was applied to more properly estimate UCS.

Valid tests produced UCS values ranging from 48.9 to 172.3 MPa, with a mean 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.

**Table 3: Uniaxial Compressive Strength Testing**

| SRK Hole ID | Sample Depth (m) | UCS (MPa) | Young's Modulus (GPa) | Poisson's Ratio | Unit Wt. (kN/m$^3$) |
|---|---|---|---|---|---|
| C09-01 | 32.10 | 88.21 | 50.5 | 0.217 | 26.12 |
| C09-01 | 89.50 | 119.56 | | | 26.25 |
| C09-01 | 187.00 | 150.39 | | | 26.34 |
| C09-01 | 220.30 | 164.68 | 66.5 | 0.302 | 26.61 |
| C09-01 | 293.16 | 156.10 | | | 26.31 |
| C09-02 | 122.67 | 71.69 | 49.2 | 0.214 | 26.17 |
| C09-02 | 179.54 | 128.30 | | | 26.59 |
| C09-02 | 271.90 | 149.87 | | | 26.20 |
| C09-03 | 38.00 | 48.94 | 14.9 | 0.084 | 25.79 |
| C09-03 | 77.33 | 72.30 | | | 24.90 |
| C09-03 | 130.84 | 66.03 | | | 25.90 |
| C09-03 | 161.03 | 104.39 | 47.3 | 0.228 | 26.48 |
| C09-03 | 282.10 | 102.63 | | | 26.34 |
| C09-03 | 361.70 | 149.58* | | | 26.56 |
| C09-04 | 30.40 | 63.15 | | | 25.32 |
| C09-04 | 91.10 | 140.72 | | | 26.34 |
| C09-04 | 150.25 | 153.42 | | | 26.52 |
| C09-05 | 33.00 | 70.92 | | | 26.01 |
| C09-05 | 92.70 | 74.34** | | | 25.67 |
| C09-05 | 150.11 | 86.71 | 53.9 | 0.262 | 26.26 |
| C09-06 | 37.20 | 121.20 | | | 26.08 |
| C09-06 | 71.22 | 131.32 | 52.5 | 0.294 | 26.01 |
| C09-06 | 108.35 | 122.78* | | | 26.04 |
| C09-06 | 138.00 | 100.70* | | | 26.30 |
| C09-07 | 29.32 | 172.29 | | | 26.70 |
| C09-07 | 86.34 | 139.69 | | | 26.56 |
| C09-07 | 124.57 | 124.68 | | | 26.33 |
| C09-08 | 47.53 | 157.71 | | | 26.53 |
| C09-08 | 89.15 | 94.31 | | | 26.47 |
| C09-08 | 129.40 | 153.60 | | | 26.37 |

* Correction factor applied to account sample L/D ratio of less than 2.0.
** UCS test results considered invalid and excluded from further analysis.

The intact Young's Moduli determined from laboratory testing were used for empirical calculations of a rock mass deformation modulus for each domain by methods presented by Hoek and Diederichs (2006).

## 5.2 Direct Shear Testing

Direct shear testing is commonly used for estimating the expected shear strength along natural rock discontinuities such as joints, fractures and faults. Since the stress levels developed within open pits are usually much lower than the rock substance or intact strength, displacement frequently occurs along pre-existing geologic discontinuities, making the determination of discontinuity shear strength a necessity. For open pit design, direct shear testing is preferred over other methods of estimating discontinuity shear strength, such as triaxial compression testing, because direct shear testing permits a higher degree of control over the selection of the actual surface tested.

For this project, ten core samples were selected for four-point, small-scale direct shear (SSDS) tests (ASTM Method D5607) to obtain discontinuity shear strength data. Natural core discontinuities preserved in the field were used for direct shear testing.

The range of normal stresses applied during testing was selected to span estimated ranges of in-situ stresses that are expected to develop within the slopes and to reasonably define the characteristics of the shear strength envelopes. The selected normal loads ranged from approximately 170 to 1,700 kPa.

In order to fit a shear strength envelope to the laboratory data points, a linear or curvilinear regression analysis is typically conducted. For a linear fit, the envelope is presented according to the Mohr-Coulomb criterion, i.e., in the form of a friction angle (Φ), which corresponds to the inverse tangent of the slope of the least-squares regression line, and cohesion (c), which corresponds to the shear strength intercept at zero normal stress. When conducting a linear regression with discontinuity shear strength data, the line is commonly forced through the origin simulating zero cohesion.

A curvilinear strength envelope can be presented in terms of a power curve with k and m values as described by Jeager (1971) or other nonlinear relationships such as the Hoek-Brown (Hoek et al, 2002) criterion. For sufficiently strong rock, the curvilinear fit is considered a more realistic representation of the shear strength/normal stress relationship, particularly at relatively low normal stresses, which typify conditions in a majority of open pit mine slopes.

Based on the direct shear testing results, shear strengths were typified using the Mohr-Coulomb and power curve shear strength/normal stress relationships. The results are summarized in Table 4.

**Table 4: Summary of Residual Shear Strengths**

| SRK Hole ID | Sample Depth (m) | Linear Regression | | | Power Regression | | Discontinuity Type |
|---|---|---|---|---|---|---|---|
| | | Φ* (°) | C (kPa) | Φ**(°) | k | m | |
| C09-01 | 49.87 | 40.7 | 21.6 | 49.2 | 4.9745 | 0.6630 | Natural Joint |
| C09-01 | 103.00 | 35.0 | 20.5 | 38.7 | 2.9505 | 0.7589 | Natural Joint |
| C09-01 | 212.15 | 33.4 | 1.3 | 33.8 | 0.7014 | 0.9911 | Natural Joint |
| C09-02 | 211.14 | 32.9 | 5.7 | 34.0 | 0.8961 | 0.9474 | Natural Joint |
| C09-03 | 162.55 | 33.7 | 10.0 | 35.7 | 1.4628 | 0.8671 | Natural Joint |
| C09-04 | 52.02 | 45.8 | 6.8 | 48.7 | 2.0405 | 0.8603 | Natural Joint |
| C09-05 | 61.07 | 37.6 | 12.7 | 40.0 | 1.9037 | 0.8465 | Natural Joint |
| C09-06 | 51.94 | 37.6 | 6.0 | 40.2 | 1.4533 | 0.8775 | Natural Joint |
| C09-07 | 137.2 | 33.7 | 13.1 | 36.3 | 1.6814 | 0.8462 | Natural Joint |
| C09-08 | 54.9 | 34.2 | 5.0 | 36.4 | 1.1906 | 0.8935 | Natural Joint |

* Best linear fit friction angle given the apparent cohesion calculated and noted
** Best linear fit friction angle assuming a zero apparent cohesion.

## 5.3 Triaxial Compressive Strength Testing

For this project, triaxial compressive strength (TCS) tests were conducted on six samples using ASTM Method D7012. The samples were tested at confining pressures selected to range from zero to approximately one-half of the UCS values as suggested by Hoek and Brown (1997).

TCS testing was conducted on six samples of core, yielding compressive strengths ($\sigma_1$) ranging between 213.8 and 294.1 MPa with a mean value of 262.1 MPa under confining pressures ($\sigma_3$) ranging between 6.9 and 20.7 MPa, with a mean of 13.8 MPa. The results of the TCS testing are summarized in Table 5.

**Table 5: Triaxial Compressive Strength Testing**

| SRK Hole ID | Sample Depth (m) | $\sigma_3$ (MPa) | $\sigma_1$ (MPa) | Unit Wt. (kN/m3) |
|---|---|---|---|---|
| C09-01 | 59.88 | 6.9 | 222.1 | 26.4 |
| C09-01 | 153.30 | 17.2 | 276.8 | 26.2 |
| C09-02 | 150.10 | 10.3 | 213.8 | 26.4 |
| C09-02 | 209.69 | 13.8 | 294.1 | 26.4 |
| C09-03 | 250.17 | 13.8 | 288.2 | 26.5 |
| C09-04 | 123.25 | 20.7 | 277.5 | 26.3 |

## 5.4 Direct Tensile Strength Testing

Brazilian disk tension testing according to ASTM method D3967 was conducted on five samples indicating intact tensile strengths ranging from 7.2 to 10.8 MPa, with a mean value of 8.8 MPa. Results of the direct tensile strength testing are summarized in Table 6.

**Table 6: Direct Tensile Strength Testing**

| SRK Hole ID | Sample Depth (m) | Tensile Strength (Mpa) |
|---|---|---|
| C09-02 | 150.10 | 10.8 |
| C09-02 | 271.90 | 9.4 |
| C09-03 | 161.03 | 7.6 |
| C09-05 | 150.11 | 7.2 |
| C09-06 | 37.20 | 8.9 |

## 5.5 Unit Weight Measurements

Prior to actual testing of UCS and TCS core sample, 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 mean value of 26.2 kN/m$^3$. Unit weights are summarized along with the various strength measurements in the preceding Tables 3 and 5.

# 6 Rock Mass Assessment

Rock mass models were developed for each of the deposit areas at Minto to provide a framework for interramp/overall slope stability modeling by mathematically simulating site geotechnical conditions. The term “rock mass” refers to the entire body of rock, including discontinuities; in contrast, “intact rock” or “substance strength” refers to the rock between discontinuities in a rock mass. Primary inputs to the rock mass models included intact rock strength, degree of fracturing and strength of fractures.

## 6.1 Data Analysis

Evaluation of the field and laboratory data collection programs indicates a high degree of variability in rock strength and geologic structure at Minto. This natural variation in rock strength and structure suggests that a probability-based method of analysis is most appropriate, yielding less conservative slope angles than would the selection of a unique, potentially over-conservative value as is typical in strictly deterministic analyses.

Probabilistic methods differ from deterministic methods in that each model parameter is characterized by a statistical distribution of values having a central tendency and some variation around that central tendency, rather than by a single, unique value. Further details of the probabilistic method used in this evaluation follow. Details of the data analysis methods are discussed in subsequent sections.

### 6.1.1 Intact Rock Strength

Intact rock strengths were assessed in the field qualitatively using ISRM (1978) methods and by conducting point load tests (PLT) as discussed in Section 4.3. Several samples of core were also selected for laboratory uniaxial compressive strength (UCS) and triaxial compressive strength testing as described in Sections 5.1 and 5.3, respectively. UCS and $Is_{(50)}$ values, as well as the field estimates of intact rock strength, are plotted with depth on the geotechnical logs presented in Appendix A.

Each laboratory UCS test was paired with an adjacent field PLT $Is_{(50)}$ value for estimation of a correlation factor for conversion of the field PLT tests to laboratory UCS values. Overall, a relatively linear relationship was apparent between the two variables, yielding a correlation factor of 23 (UCS:$Is_{(50)}$). The correlation between the laboratory UCS tests and the PLTs is demonstrated on Figure 3.

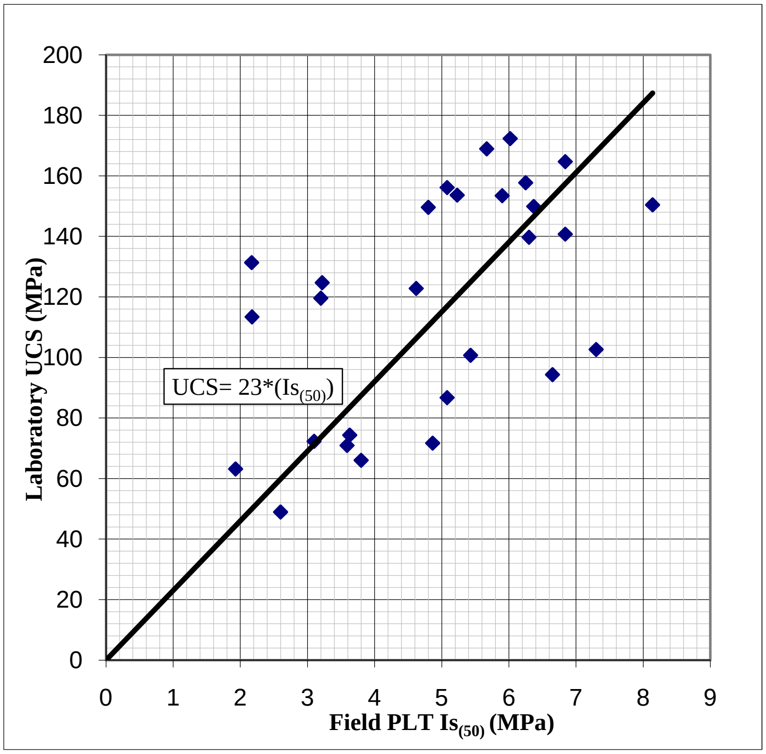



**Figure 3: Point Load Index – UCS Correlation Factor**

The conversion of the field PLTs to laboratory UCS values allowed nearly continuous profiles of rock strength for each corehole and provided a large population for defining UCS statistical distributions for the probabilistic analyses.

As demonstrated in the plots contained on Figures 4 through 7, the weathered domains have distinctively lower distributions of UCS than do the fresh units. The weathered domains have UCS strengths generally ranging up to about 120 MPa, with the mode (peak concentration) around 20 MPa, while the fresh domains typically have UCS values ranging up to about 240 MPa with the mode around 110 to 140 MPa.

TCS test results, as described in Section 5.3, were used for direct determination of the Hoek-Brown (Hoek, et al, 2002) material coefficient $m_i$. As described by Hoek (1983), the Hoek-Brown constant $m_i$ is very approximately analogous to the angle of friction of the conventional Mohr-Coulomb failure criterion. Higher $m_i$ values are characteristic of brittle igneous and metamorphic rocks producing relatively steeply inclined strength envelopes and high instantaneous friction angles at lower normal stress levels.

### 6.1.2 Discontinuity Frequency

The fracture (discontinuity) frequency or its inverse, fracture spacing, is a critical parameter influencing rock mass behavior. Fracture frequency is expressed as the number of fractures per unit length and fracture spacing is defined as the distance between fractures. Fracture frequency per meter was recorded during drilling for each run, thereby enabling calculation of mean fracture spacings for use in rock mass characterization and bench scale analyses, both of which are discussed in more detail in the following sections. For expedience, it was assumed that each measurement began and ended with a fracture, thereby resulting in a maximum possible spacing of about 1.5 meters, the length of the core barrel.

### 6.1.3 Discontinuity Shear Strength

Discontinuity shear strengths are a function of geologic history as well as rock mass weathering, alteration and/or infilling. Direct shear testing was conducted on a number of rock samples as previously discussed in Section 5.2 to provide information on the distribution of discontinuity shear strengths. Although results of direct shear testing of discontinuities on some of the samples tested demonstrated curvilinear shear strength/normal stress envelopes, most analytical stability models, including those used by SRK for backbreak analyses, utilize linear, Mohr-Coulomb parameters.

Tests results indicate similar shear strengths between the different domains and areas; consequently, discontinuity shear strengths were grouped together into one distribution. For samples tested from the recent 2009 geotechnical coreholes, calculated friction angles (assuming zero apparent cohesion as discussed in Section 5.2) ranged from 33° to 46° with apparent cohesion values ranging from 1 to 22 kPa. The mean friction angle was 36° with an apparent cohesion of 10 kPa. The distribution of friction angles obtained from testing the recent natural fractures as well as six saw cut direct shears from the previous Area 2 (SRK 2007) investigation is shown on Figure 4.

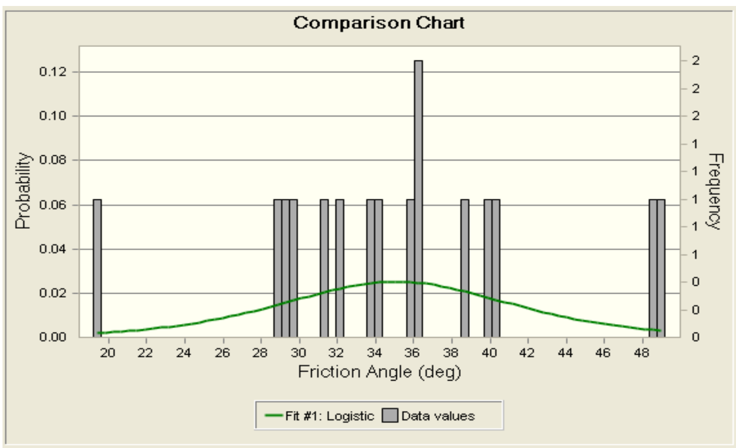



**Figure 4: Distribution of measured discontinuity shear strengths**

## 6.2 Rock Mass Classification

Rock mass characterization is a largely empirical process of classification based on information obtained primarily from field data and enhanced with further data analysis and laboratory testing. For typical slope stability applications, materials from ground surface to a depth of approximately 30% of the ultimate slope height below final pit bottom and for a distance approximately two times the ultimate pit height behind the slope crest are characterized and represented within the geotechnical model.

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. The Laubscher IRMR system consists of three primary parameters; intact rock strength (IRS), fracture frequency per meter (FF/m) and joint conditions (Jc). The individual parameters as well as the IRMR value out of a total of 100 for each run are displayed on the geotechnical core logs presented in Appendix A. A large scale joint expression of slight undulation and dry conditions were assumed.

The in-situ RMR is typically adjusted to account for the expected mining environment, namely the influence of weathering, structural orientations, induced or changes to stresses and blasting.

The adjustments to the in-situ RMR are introduced in recognition of the type of excavation proposed and the time dependant behavior of the rock mass. These adjustments were not incorporated for the pit slope analyses as they are accounted for in other ways. They were, however, considered for the Area 118 underground, as discussed in Section 9.

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, and, in the case of Area 118, the underground excavation 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.

Due to the relatively shallow depth of the Minto North pit and the presence of multiple sub-horizontal 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.

A summary of IRMR values per domain is presented in Table 7.

**Table 7: In-situ RMR Distributions per Domain**

| Deposit | Domain | Distribution | Sample No. | Mean | Std. Dev. | Min | Max |
|---|---|---|---|---|---|---|---|
| Area 2 | Weathered | Weibull | 162 | 46.4 | 8.6 | 18 | 68 |
| Area 2 | Fresh | Min. Extreme | 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 |

## 6.3 Geotechnical Domains

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, based on engineering properties that are determined during field data collection and laboratory testing programs.

Based on the results of data analysis and rock mass classification previously described as well as available site geology information, geotechnical domains were delineated for each area. 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. The weathered and fresh rock domains are very similar in terms of discontinuity orientations; however, they possess distinctly different rock mass properties.

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 do 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.

### 6.3.1 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 90m 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 1m is “active”, i.e., seasonally freezing and thawing.

Based on available information from resource and geotechnical drilling, Area 2 is covered with overburden ranging from about 5 to 15m in depth in the southwest portion, with up 20 to 45m along much of the north and east walls, and reaching a maximum depth of 70m at the far north.

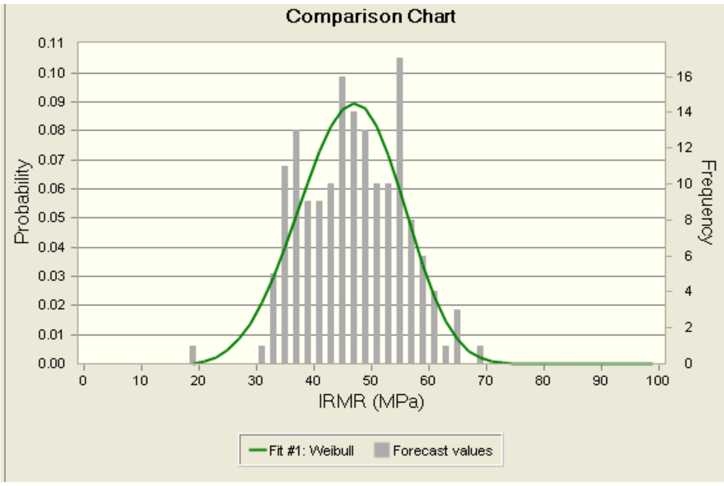
While it is possible that the frozen overburden may extend farther south, available information suggests that the overburden at the south and west ends of the proposed Area 2 pit consists of a thin veneer of organic soil underlain by approximately 5m to 15m of completely weathered, in-situ bedrock (granular soil) or residuum.

Based on geotechnical drillhole data, the Area 2 weathered domain is adjudged to extend to depths of approximately 50 to 100m below the current ground surface.

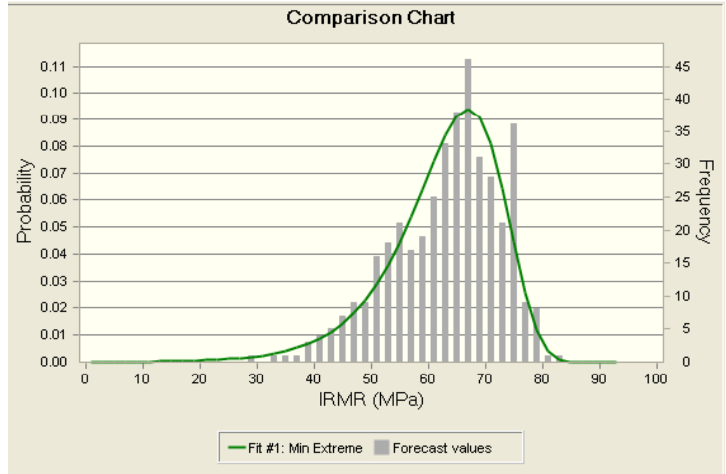
Distributions of UCS, fracture frequency and IRMR for the Area 2 weathered and fresh rock domains are presented on Figure 5. Cross sections showing the geotechnical domains of the Area 2 west and east walls are presented in Figures 6 and 7 respectively.

## Area 2 Weathered Rock Domain

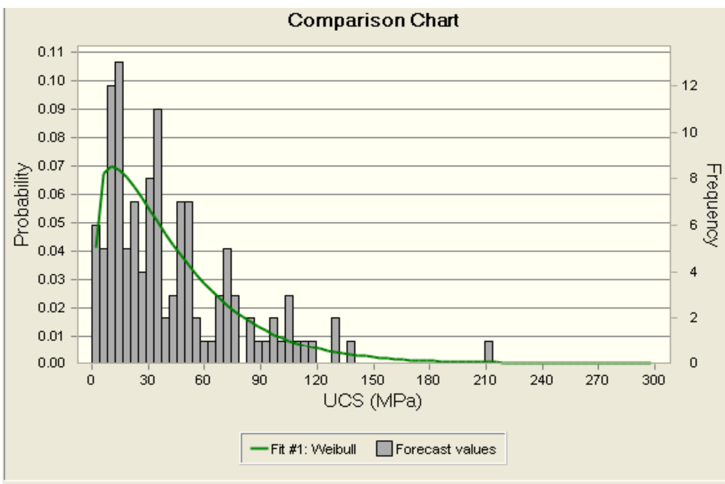
## Area 2 Fresh Rock Domain



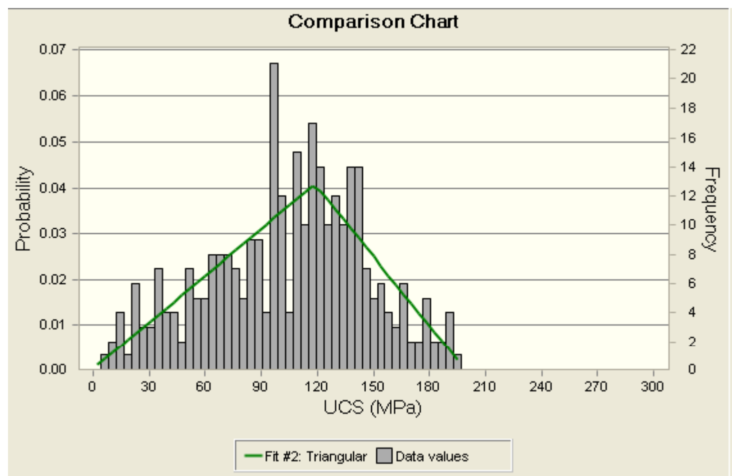

Mean IRMR = 46.4 (162)



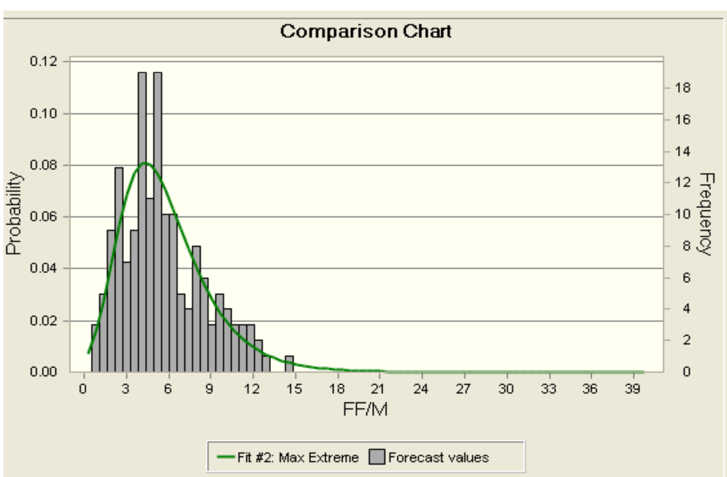

Mean IRMR = 59.8 (409)



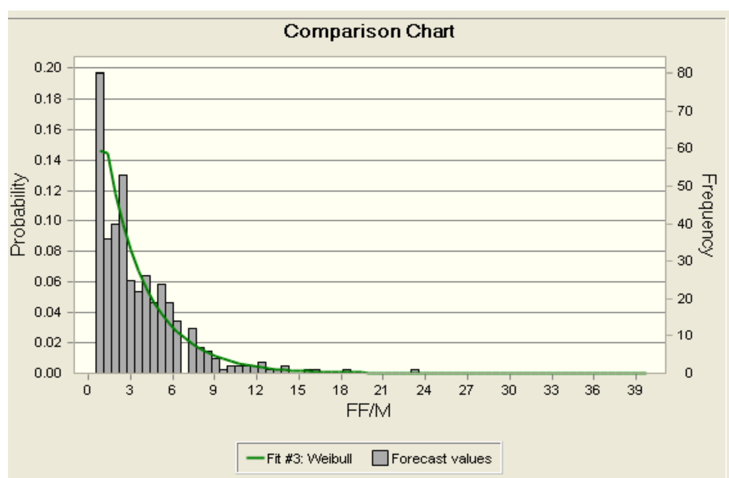

Mean UCS = 42.5 MPa (122)




Mean UCS = 104.2 MPa (314)




Mean ff/m = 5.81 (164)




Mean ff/m = 3.67 (407)

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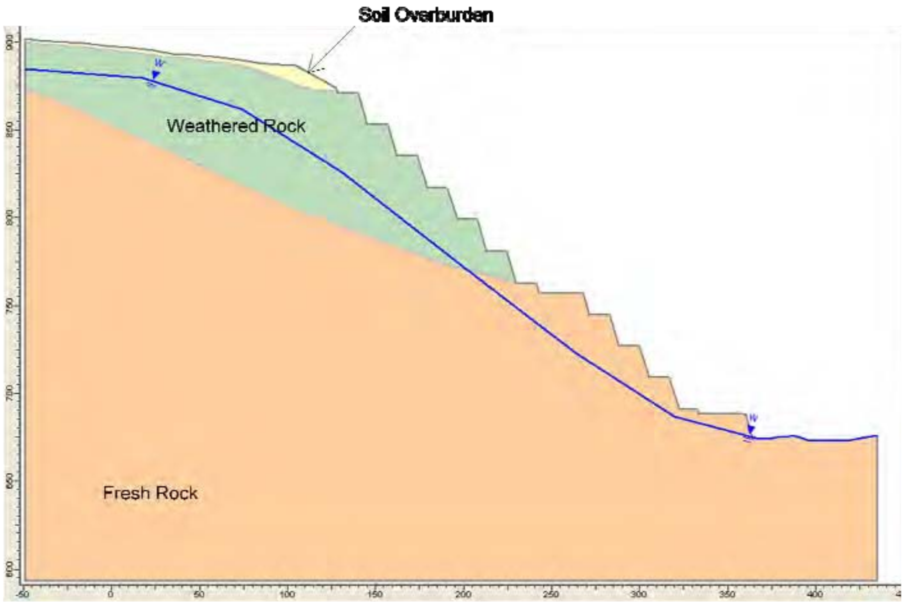



**Figure 6: Critical section through Area 2 geotechnical model: west wall**

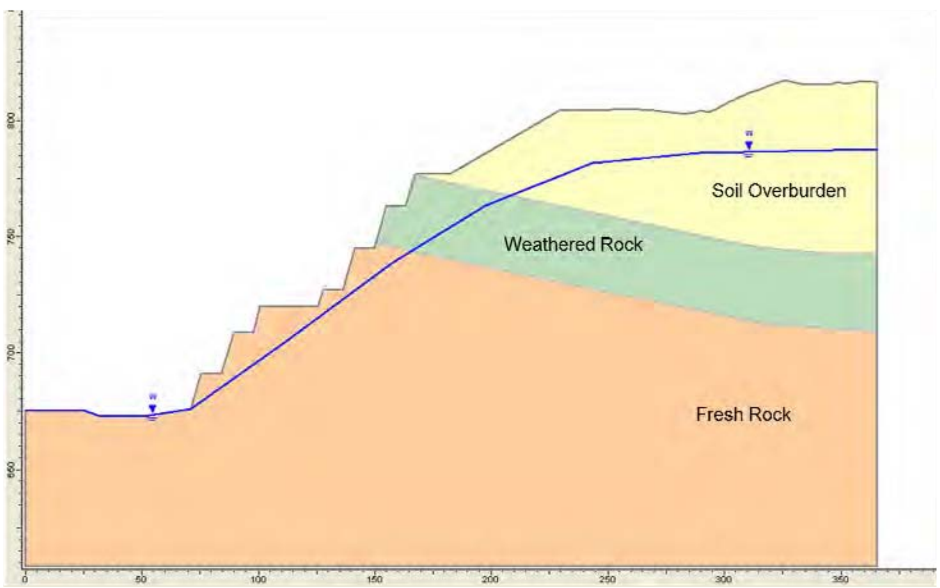



**Figure 7: Critical section through Area 2 geotechnical model: east wall**

### 6.3.2 Area 118

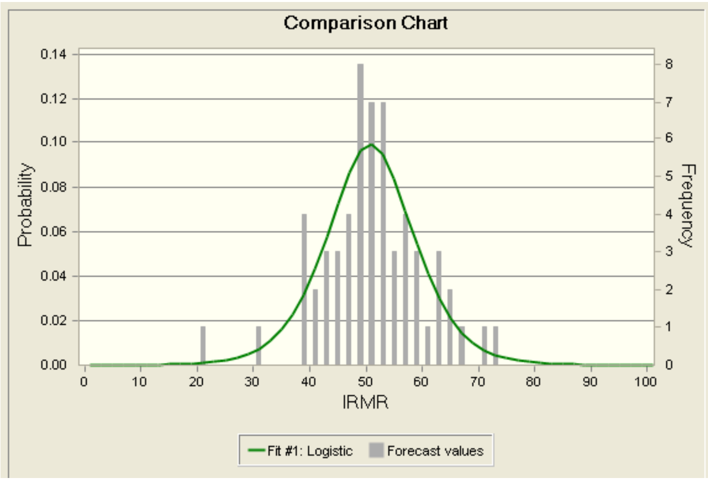
The majority of the proposed Area 118 open pit footprint is covered with up to approximately 5m of soil overburden except the southwest portion where the overburden locally deepens to approximately 16m. The depth of bedrock weathering at Area 118 is generally to about 30 to 60m below ground surface.

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 modeling was not deemed necessary for Area 118 at the current, Pre-feasibility level. Consequentially, a detailed geotechnical model cross section was not created for Area 118.

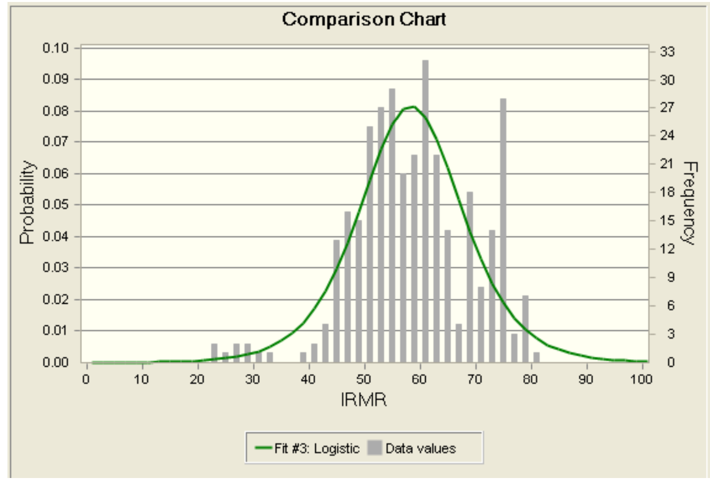
Distributions of UCS, fracture frequency and IRMR for the Area 118 weathered and fresh rock domains are presented on Figure 8.

## Area 118 Weathered Rock Domain

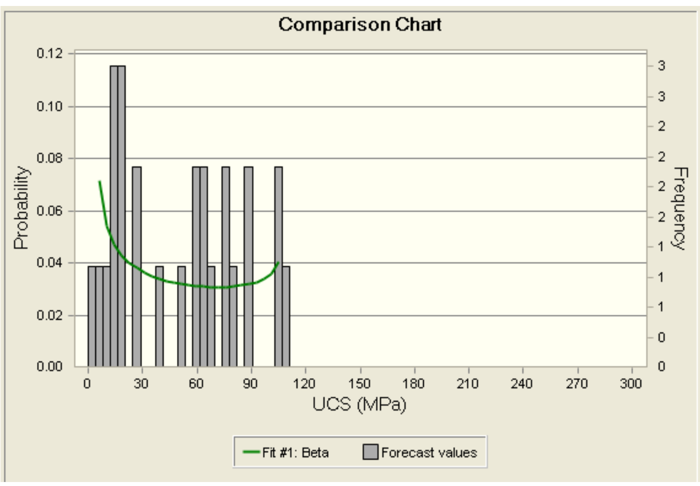
## Area 118 Fresh Rock Domain



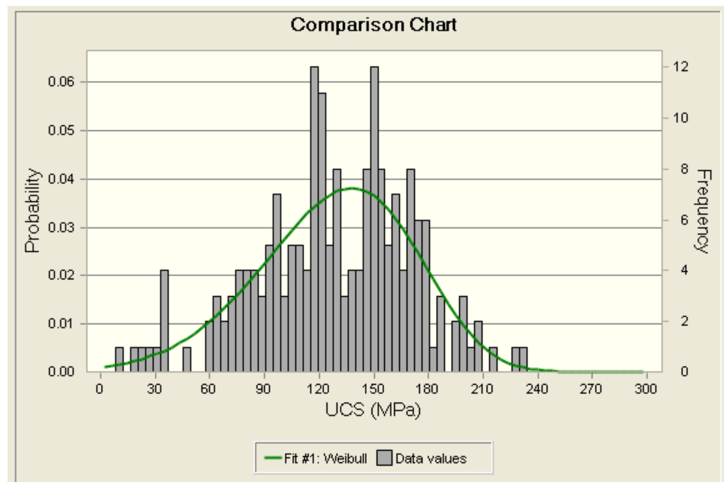

Mean IRMR = 50.8 (59)



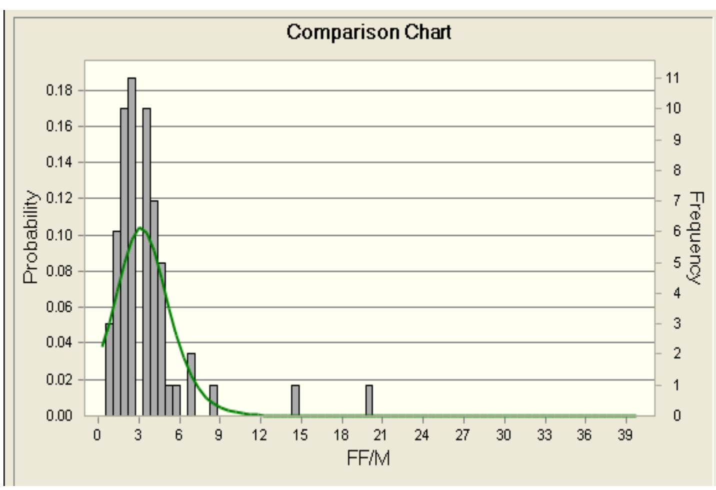

Mean IRMR = 59.3 (334)



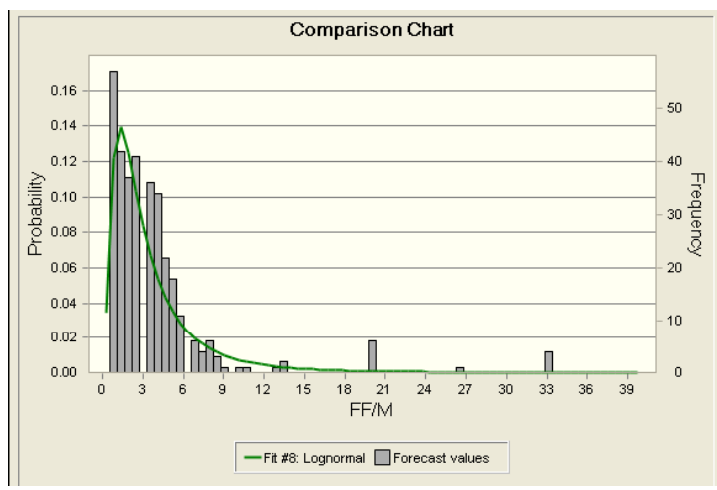

Mean UCS = 50.3 MPa (26)




Mean UCS = 128.3 MPa (190)




Mean ff/m = 3.60 (59)




Mean ff/m = 3.87 (334)

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ROCK MASS PARAMETERS:
AREA 118

SRK PROJECT NO.: 2CM022.006

FILE NAME:

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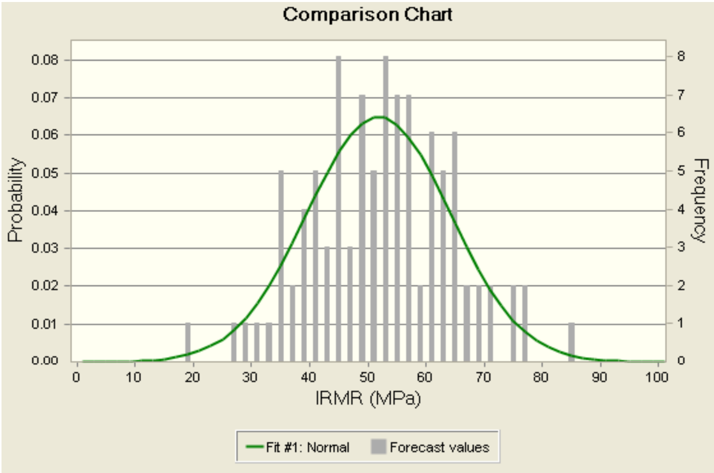
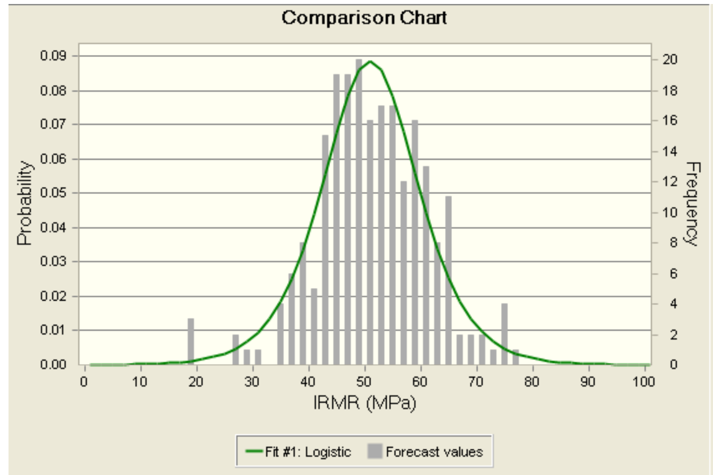
## Ridgetop Weathered Rock Domain

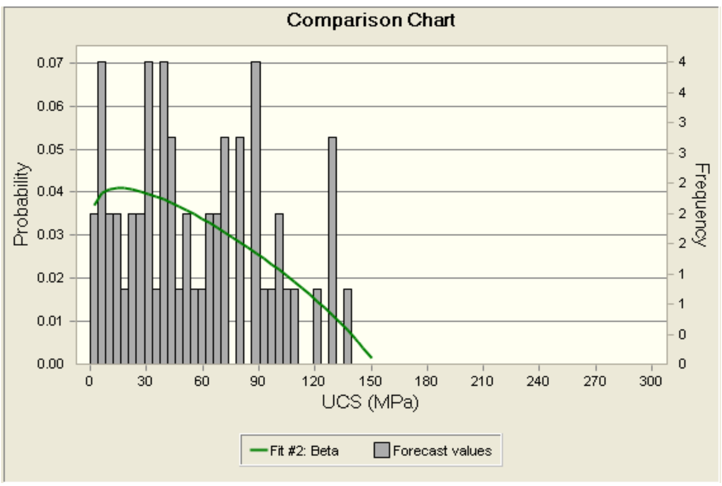
## Ridgetop Fresh Rock Domain



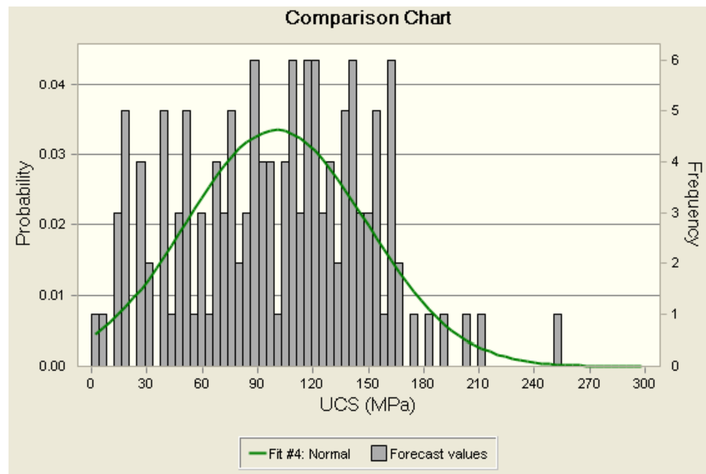

Mean IRMR = 51.8 (99)



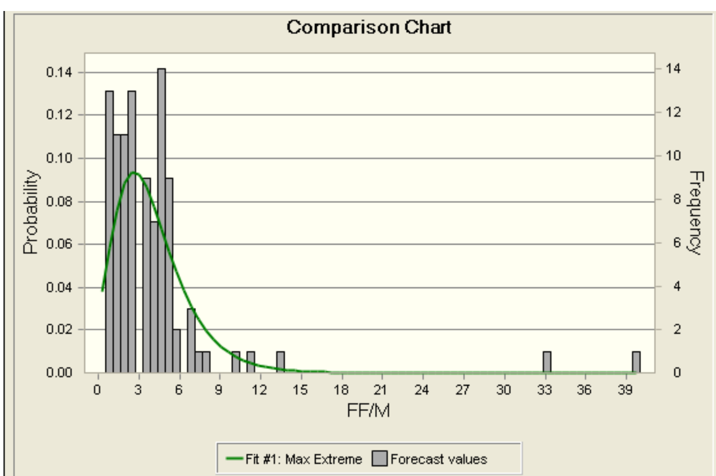

Mean IRMR = 51.0 (225)



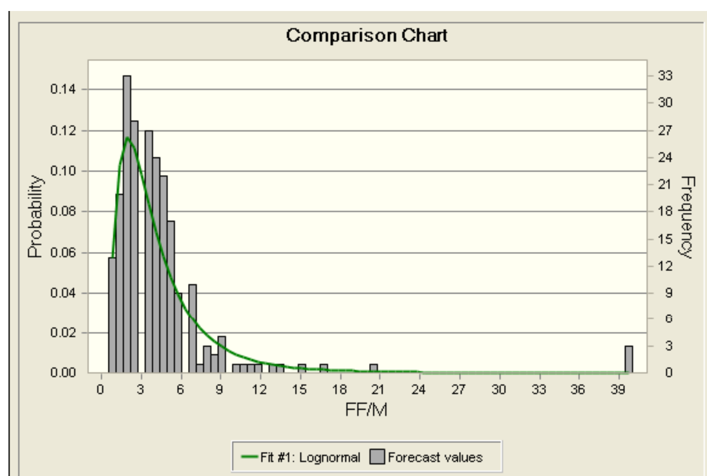

Mean UCS = 56.6 MPa (57)




Mean UCS = 100.0 MPa (138)




Mean ff/m = 4.11 (99)




Mean ff/m = 4.47 (225)

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### 6.3.3 Ridgetop

The western portion of the proposed Ridgetop pits are anticipated to contain 1 to 5m of soil overburden deepening to the east to generally about 5 to 15m at the eastern edge, with a maximum depth of 21m at the far northeast portion of Ridgetop North and at the far east portion of Ridgetop South. The bedrock at Ridgetop is generally weathered to a depth of approximately 45 to 70m below ground surface. Distributions of UCS, fracture frequency and IRMR for the weathered and fresh rock domains are presented on Figure 9. A generalized cross section showing the geotechnical domains at Ridgetop is presented in Figure 10.

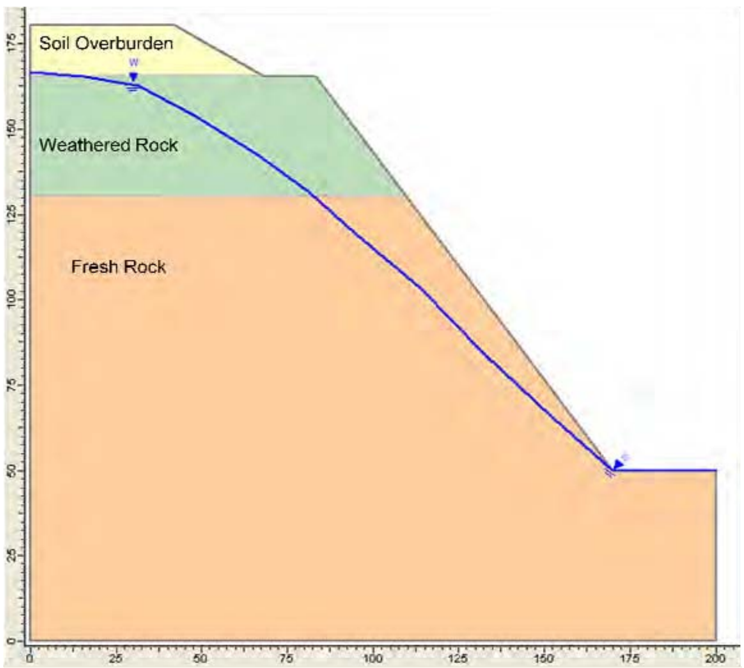


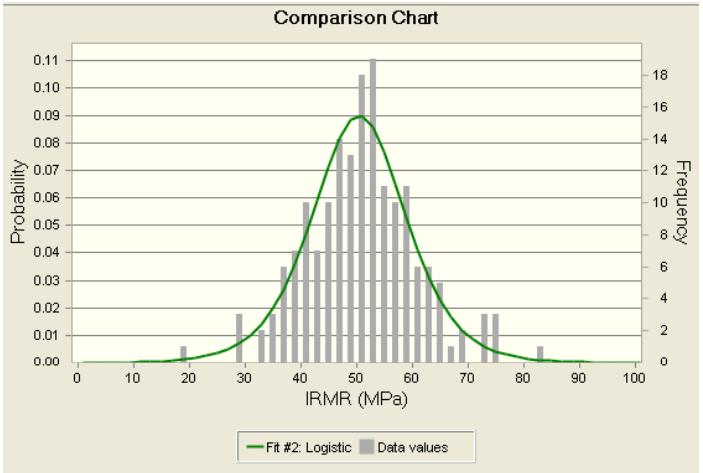

**Figure 10: Critical section through generalized Ridgetop geotechnical model**

### 6.3.4 Minto North

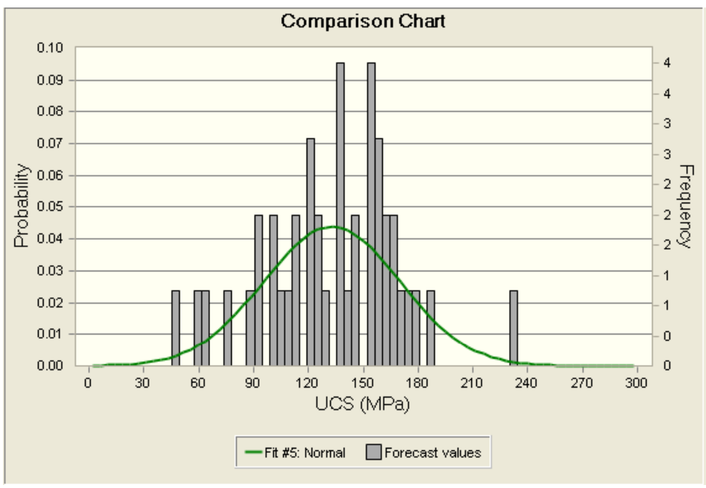
Based on geotechnical drillhole C09-07, bedrock weathering is very shallow at Minto North and fairly competent fresh rock lies beneath the soil overburden. Geotechnical drillhole C09-08 also does not indicate extensive weathering at the bedrock surface but did encounter a relatively thick fault zone beneath the overburden.

Due to the relatively shallow depth of the Minto North pit and the presence of multiple structures, there is a less significant distinction, if any, between the weathered and fresh rock materials;, consequentially, materials at Minto North were combined together into a single domain for modeling. As such, a detailed cross section through the Minto North geotechnical model is not presented. A distribution of UCS, fracture frequency and IRMR for the Minto North domain is presented on Figure 11.

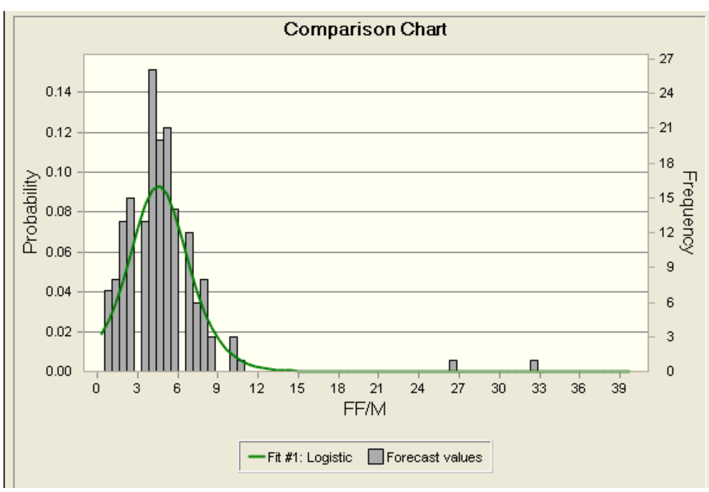
# Minto North Domain




Mean IRMR = 50.6 (172)




Mean UCS = 132.8 MPa (42)




Mean ff/m = 4.84 (172)

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| SRK PROJECT NO.: 2CM022.006 | MINTO MINE | ROCK MASS PARAMETERS: MINTO NORTH |
| FILE NAME: | | DATE: DEC. 2009 / APPROVED: MEL / FIGURE NO.: 11 / REVISION NO. A |

## 6.4 Rock Mass Shear Strength

The shear strength/normal stress relationship describes the ultimate shear strength available at a given point within a slope as a function of the effective normal stress acting on that point. Rock mass shear strength/normal stress relationships were developed for weathered and fresh rock domains at each area using the Generalized Hoek-Brown criterion (Hoek et al, 2002).

The Generalized Hoek-Brown criterion defines curvilinear shear strength envelopes that are considered effective representations of intact rock and heavily jointed rock mass behavior. Primary input parameters for the Generalized Hoek-Brown jointed rock mass criterion include the Geological Strength Index (GSI), a material constant ($m_i$) and a disturbance factor (D), as defined by Hoek et al, (2002). Probability density functions (PDF) were selected to represent stochastic (statistical) distributions of each of the primary parameters for each domain. The distributions selected were based upon the results of field and laboratory testing as well as upon SRK's experience.

After the PDFs were selected to represent the three primary Generalized Hoek-Brown parameters ($m_i$, GSI and D), Crystal Ball 7.3.2 (Crystal Ball), commercial software available from Oracle, was utilized to perform a large number of stochastic simulations, sampling each of the three parameter distributions during each simulation. From each set of primary parameters sampled, respective Hoek-Brown secondary parameters ($m_b$, s and a) were calculated producing PDFs for each of the secondary parameters.

PDFs representing the UCS for each domain were also defined using a mathematical, "best-fit" technique available in Crystal Ball. The distribution types and defining parameters for the Hoek-Brown secondary parameters and for UCS selected for the analyses are summarized in Table 8.

### Table 8: Secondary Hoek-Brown Parameters Stochastic Input

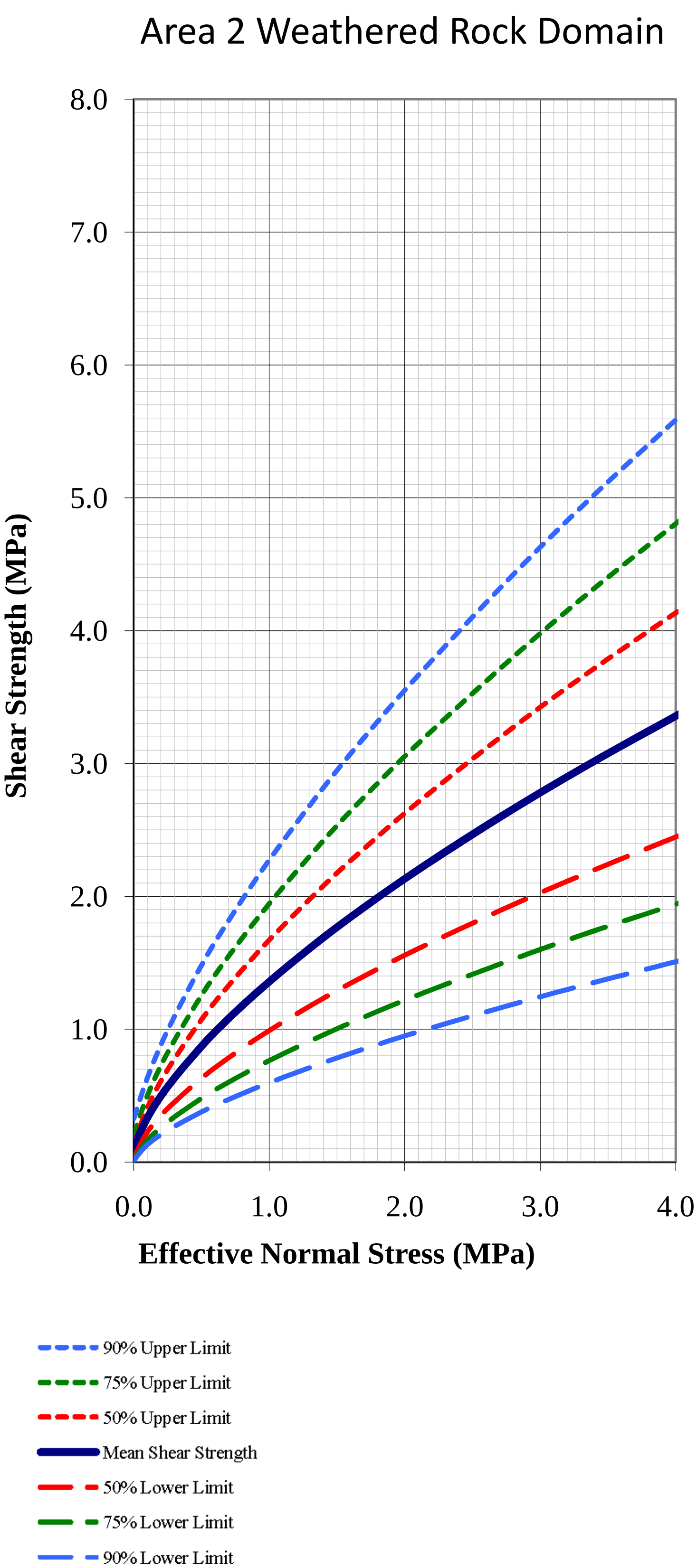
| Deposit | Domain | Parameter | Distribution | Mean | Std. Dev. | Min | Max |
|---|---|---|---|---|---|---|---|
| Area 2 | Weathered | Hoek-Brown a parameter | Gamma | 0.5087 | 0.0102 | 0.5007 | 0.524 |
| Area 2 | Weathered | Hoek-Brown m parameter | Lognormal | 1.11 | 0.64 | 0.135 | 3.03 |
| Area 2 | Weathered | Hoek-Brown s parameter | Gamma | 5.85E-04 | 1.38E-03 | 0.00E+00 | 4.73E-03 |
| Area 2 | Weathered | UCS (intact) MPa | Beta | 42.51 | 35.54 | 0.00 | 878.22 |
| Area 2 | Fresh | Hoek-Brown a parameter | Gamma | 0.5036 | 0.0101 | 0.5001 | 0.5108 |
| Area 2 | Fresh | Hoek-Brown m parameter | Lognormal | 2.69 | 1.84 | 0.00 | 8.21 |
| Area 2 | Fresh | Hoek-Brown s parameter | Lognormal | 5.86E-03 | 1.73E-02 | 0.00E+00 | 5.78E-02 |
| Area 2 | Fresh | UCS (intact) MPa | Triangular | 105.68 | 42.19 | 0.00 | 199.77 |
| North | - | Hoek-Brown a parameter | Gamma | 0.5072 | 0.01015 | 0.5000 | 0.5228 |
| North | - | Hoek-Brown m parameter | Lognormal | 1.41 | 1.08 | 0.00 | 4.65 |
| North | - | Hoek-Brown s parameter | Lognormal | 1.65E-03 | 6.01E-03 | 0.00E+00 | 1.97E-02 |
| North | - | UCS (intact) MPa | Normal | 132.76 | 37.34 | 0.00 | 282.12 |
| Ridgetop | Weathered | Hoek-Brown a parameter | Lognormal | 0.5072 | 0.0058 | 0.5000 | 0.5246 |
| Ridgetop | Weathered | Hoek-Brown m parameter | Lognormal | 1.66 | 1.56 | 0.00 | 6.34 |
| Ridgetop | Weathered | Hoek-Brown s parameter | Lognormal | 3.37E-03 | 2.16E-02 | 0.00E+00 | 6.82E-02 |
| Ridgetop | Weathered | UCS (intact) MPa | Beta | 56.64 | 37.33 | 0.00 | 151.59 |
| Ridgetop | Fresh | Hoek-Brown a parameter | Gamma | 0.5068 | 0.0102 | 0.5000 | 0.5209 |
| Ridgetop | Fresh | Hoek-Brown m parameter | Lognormal | 1.45 | 1.1 | 0.00 | 4.75 |
| Ridgetop | Fresh | Hoek-Brown s parameter | Lognormal | 1.73E-03 | 6.14E-03 | 0.00E+00 | 2.02E-02 |
| Ridgetop | Fresh | UCS (intact) MPa | Normal | 100.01 | 48.94 | 0.00 | 246.83 |

From the repeated, randomized samplings of the secondary Hoek-Brown parameters and UCS, distributions of the shear strength/normal stress relationships were calculated. Graphical representations of the range of shear strength/normal stress envelopes used by the model for each domain are presented on Figures 12 through 15, respectively. In Figures 12 and 15, the 50%, 75% and 90% Upper and Lower Limits represent the ranges within which the shear strength lies, with 50%, 75% and 90% reliability, respectively.

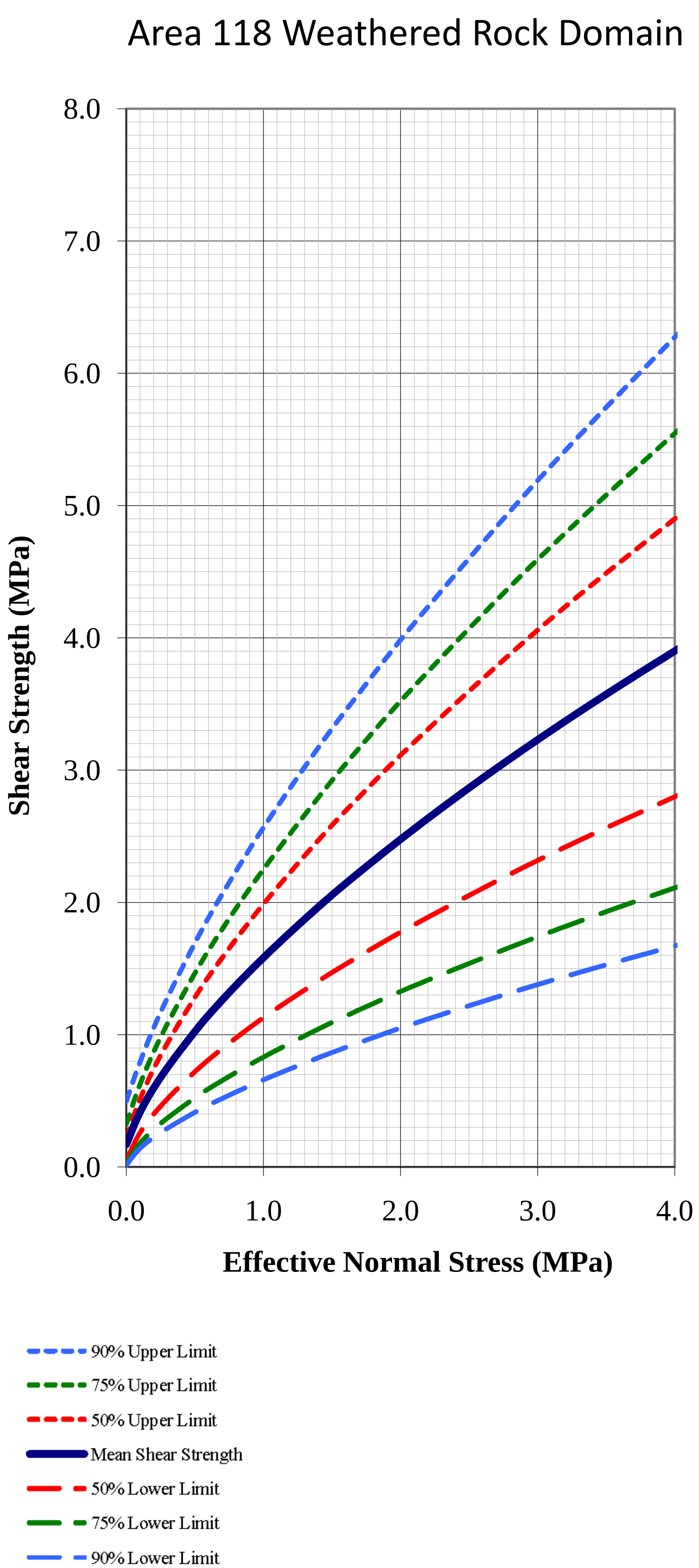
## 6.5 Groundwater

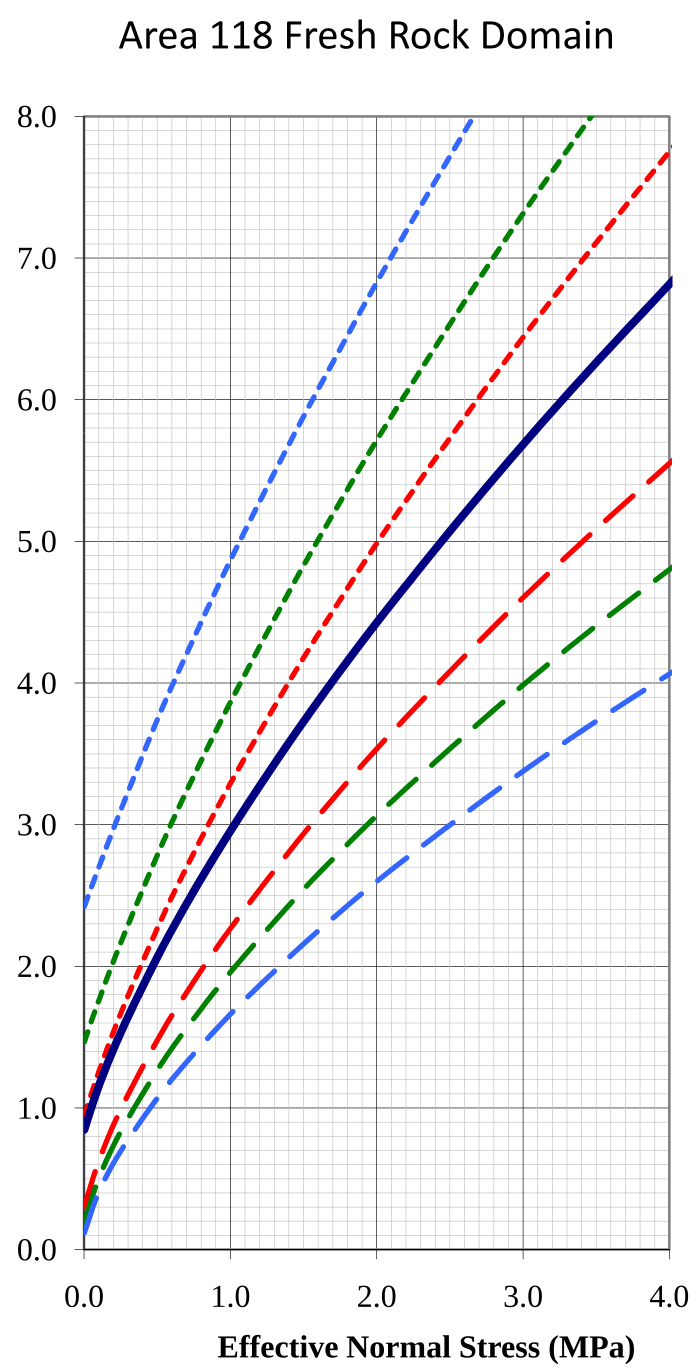
Groundwater (porewater) pressure is an important component of slope stability. Porewater pressures act in direct opposition (as buoyant forces) to stabilizing forces, and as such, must be considered for the results of stability modeling to be realistic. A relatively free-draining slope will typically allow drawdown of the groundwater surface sufficiently deep within the slope so that porewater pressures are of minimal impact to slope stability. Since the rock mass comprising open pit benches has usually been at least moderately disrupted by production blasting, such rock masses are usually free-draining and, in recognition, porewater pressures are seldom considered in bench scale stability analyses. However, deeper within rock masses that have been intensively weathered, altered and/or sheared, clay-filled discontinuities and/or faults are common, compartmentalizing groundwater and resulting in a greatly reduced rock mass permeability. A lower permeability rock mass frequently inhibits free drainage, leading to a much steeper groundwater drawdown surface closer to the pit face. As a result, significant porewater pressures may be present on potential slip surfaces, thereby reducing effective normal stresses which, in turn, reduce resisting forces within the slope, and, consequentially, adversely impact the stability of the slope.

No recent groundwater data is available in the immediate area of the subject deposits. As a result of the lack of available groundwater information and the very difficult nature of groundwater prediction, SRK approximated a relatively high groundwater drawdown surface for use in slope stability modeling. The purpose of this approach is to determine the sensitivity of groundwater levels on the stability of pit slopes in order to provide guidance regarding the extent of groundwater drawdown which may be necessary for global pit slope stability.




| SRK Consulting Engineers and Scientists | MINTO EXPLORATIONS LTD. | PREFEASIBILITY GEOTECHNICAL EVALUATION | | | |
|---|---|---|---|---|---|
| SRK PROJECT NO.: 2CM022.006 | MINTO MINE | ROCK MASS SHEAR STRENGTH: AREA 2 | | | |
| FILE NAME: | | DATE: NOV. 2009 | APPROVED: MEL | FIGURE NO.: 12 | REVISION NO. A |

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PREFEASIBILITY GEOTECHNICAL EVALUATION

ROCK MASS SHEAR STRENGTH:
AREA 118

SRK PROJECT NO.: 2CM022.006

FILE NAME:

MINTO MINE

| DATE: | APPROVED: | FIGURE NO.: | REVISION NO. |
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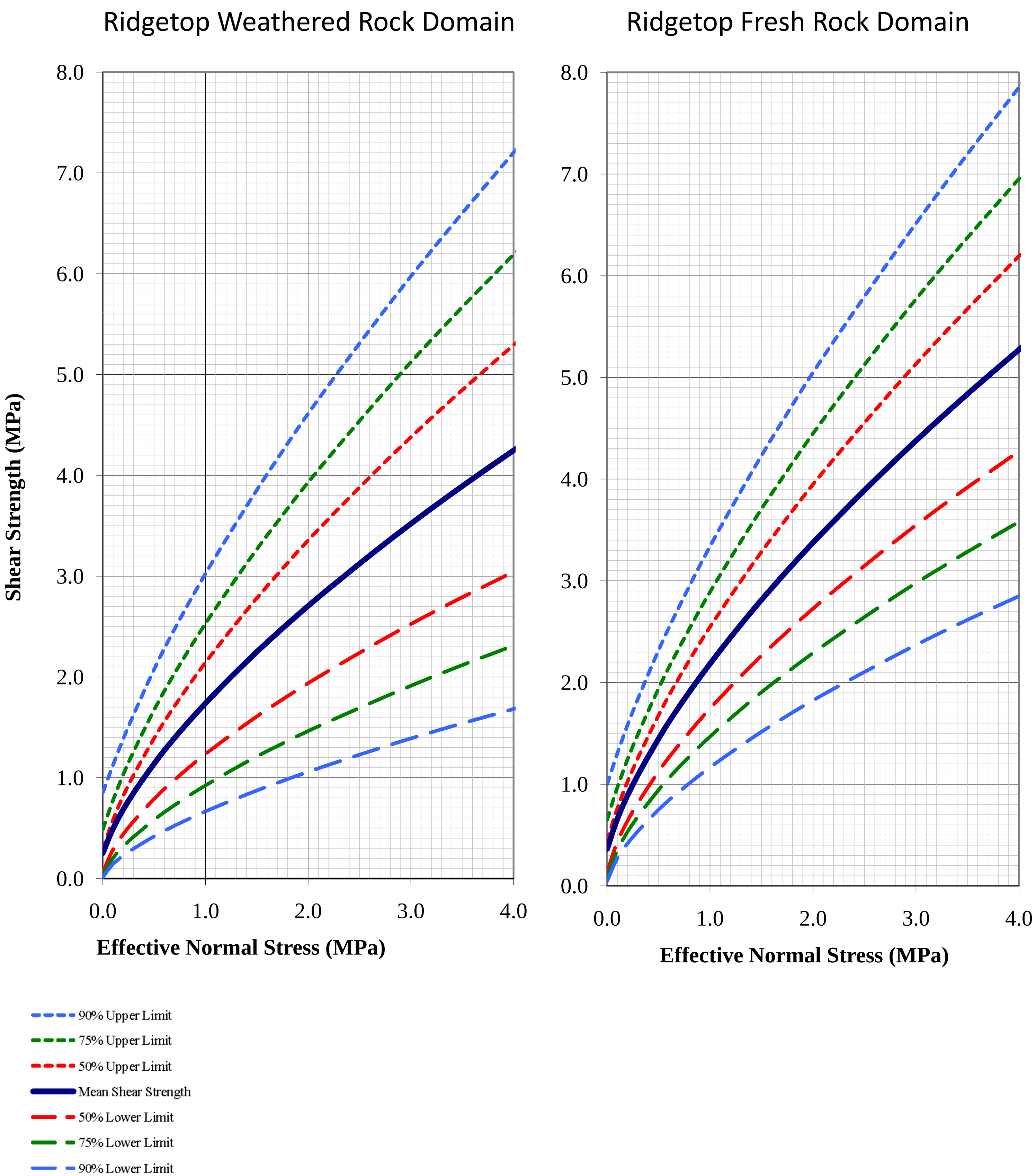


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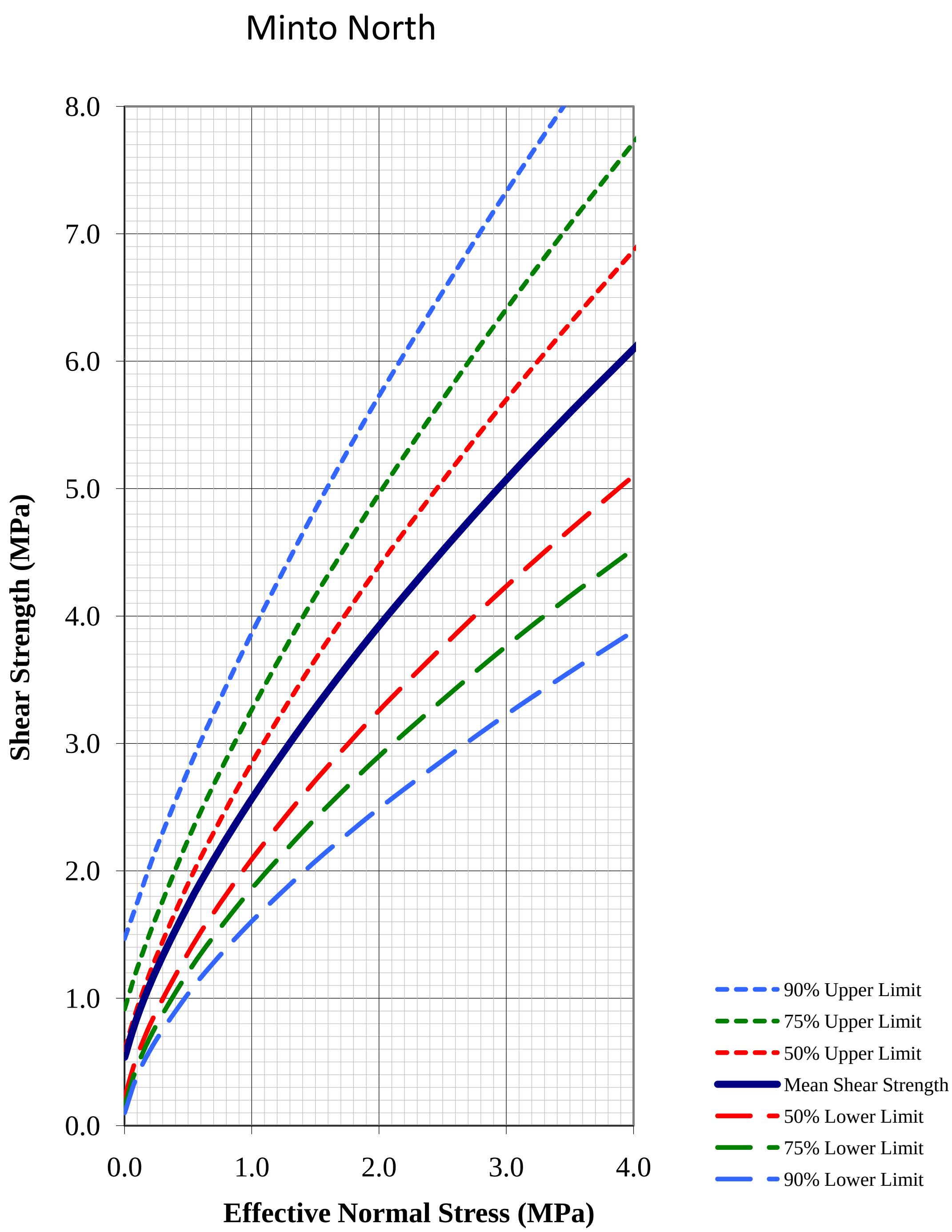
| SRK Consulting Engineers and Scientists | MINTO EXPLORATIONS LTD. | PREFEASIBILITY GEOTECHNICAL EVALUATION<br>ROCK MASS SHEAR STRENGTH: RIDGETOP |
| SRK PROJECT NO.: 2CM022.006<br>FILE NAME: | MINTO MINE | DATE: NOV. 2009 · APPROVED: MEL · FIGURE NO.: 14 · REVISION NO. A |

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| SRK Consulting Engineers and Scientists | MINTO EXPLORATIONS LTD. | PREFEASIBILITY GEOTECHNICAL EVALUATION |
| SRK PROJECT NO.: 2CM022.006 | MINTO MINE | ROCK MASS SHEAR STRENGTH: MINTO NORTH |
| FILE NAME: | | DATE: NOV. 2009 / APPROVED: MEL / FIGURE NO.: 15 / REVISION NO. A |

# 7 Interramp/Overall Slope Stability Modeling

Slope design involves analysis of the three major components of a pit slope, i.e., bench configuration, interramp angle and overall slope angle, all as defined on Figure 16. The bench configuration, which is controlled by the bench face angle, bench height, and berm width, defines the interramp angle. The overall slope angle consists of interramp sections separated by wide step-outs for haulage roads or mine infrastructure. The overall slope angle at Minto will be approximately equal to the interramp angle except in areas where a wide step-out may be planned, e.g., at the contact between overburden and the underlying rock. In order to refine the recommendations of this study, a range of slope angles was analyzed.

As discussed in Section 3, the maximum anticipated seismic acceleration which any of the Minto pits may be subject to during their relatively short lives is sufficiently low, that no analyses were conducted for seismic conditions.

SRK evaluated both global and bench scale stability for the proposed Minto open pits, where global failure is defined as one that occurs relatively deep through the rock mass, is pseudo-rotational, and is of sufficient scale to impact interramp and/or overall slopes. Bench scale failures typically involve only one or two bench levels and can be described as a block type failure involving the translation of a block delineated by one or more structural features, such as discontinuities, within the rock mass. Techniques used by SRK for the global analyses are presented in the remainder of this section. Details regarding bench scale stability analyses are presented in Section 8.

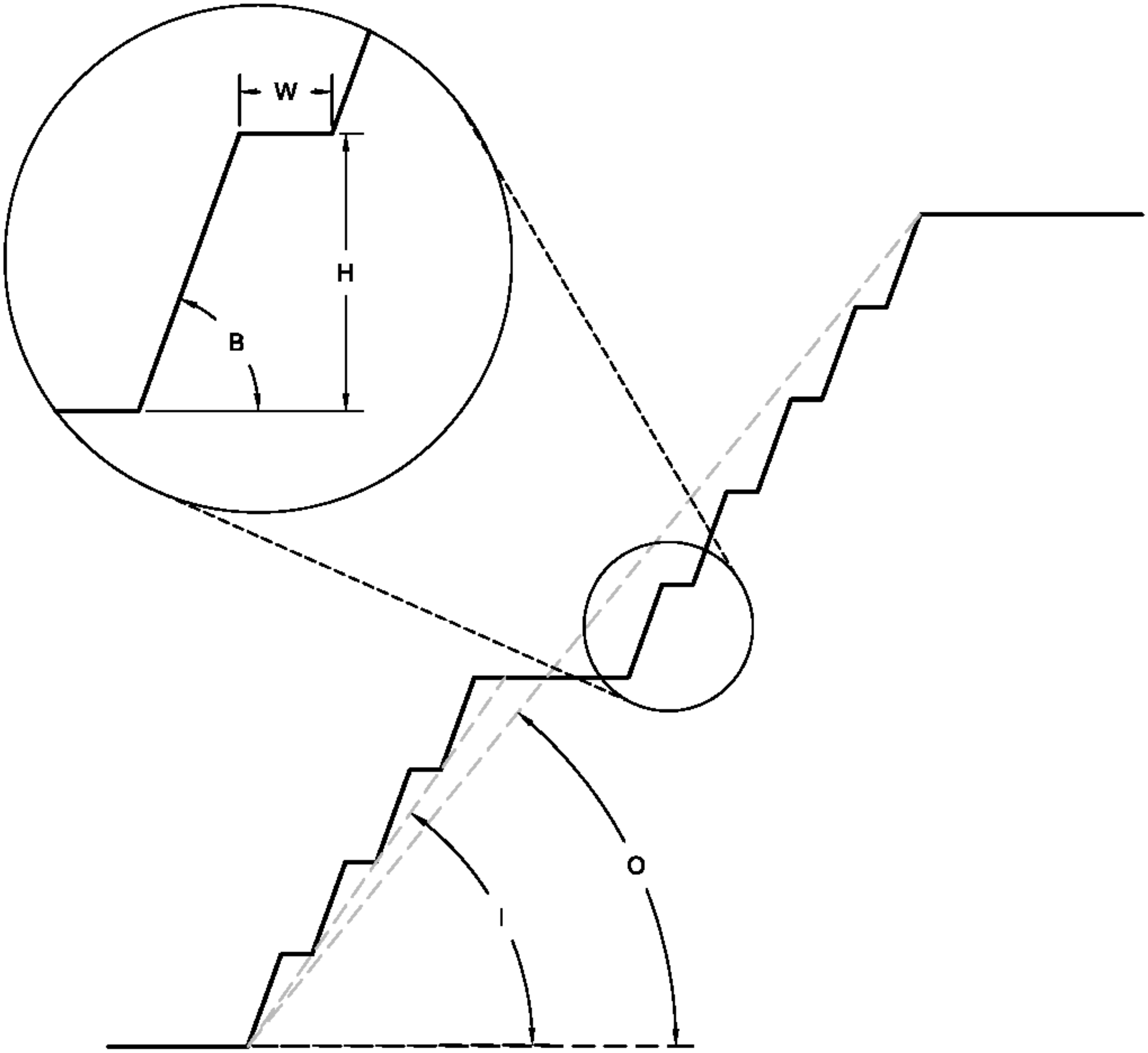
The mathematical geotechnical model was input into the commercially available geotechnical modeling software package Slide 5.039 (Slide), developed by Rocscience, Inc. (2003). Slide is a two-dimensional, limit equilibrium slope stability analysis program that analyzes slope stability by various methods of slices. Spencer's method was selected for the limit equilibrium analyses of this evaluation due to its consideration of both force and moment equilibrium.

Vertical profiles considered most critical and representative of conditions were selected for analysis based on the ultimate pit configurations and the geotechnical model at each deposit location. For Area 2, profiles of the highest sections of the west and east walls were selected for the interramp and overall stability analyses. Given the relatively shallow depths and low interramp slope heights at Ridgetop and Minto North, generalized sections were constructed containing ultimate values for each component. This method represents a conservative or worst case scenario.

The slope angles were optimized in terms of risk, i.e. Probability of Failure (PoF), to ensure that the design slope angles were the optimum based on a quantitative evaluation of alternative designs. The PoF value incorporates the variations associated with the input parameters and the concept of risk into the design.

## 7.1 Results of Interramp/Overall Stability Analysis

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% 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.




O = OVERALL ANGLE

I = INTERRAMP ANGLE

B = BENCH FACE ANGLE

H = BENCH HEIGHT

W = CATCH BENCH WIDTH = $H\left(\frac{1}{TAN\ I} - \frac{1}{TAN\ B}\right)$

| | | |
|---|---|---|
| SRK Consulting<br>Engineers and Scientists | MINTO EXPLORATIONS LTD. | PREFEASIBILITY GEOTECHNICAL EVALUATION<br>EXPLANATION OF PIT SLOPE TERMINOLOGY |
| SRK PROJECT NO.: 2CM022.006<br>FILE NAME: | MINTO MINE | DATE: NOV. 2009 \| APPROVED: MEL \| FIGURE NO.: 16 \| REVISION NO. A |

In analyses, the interramp angle is typically incrementally increased until a suitable probability of failure equal to or greater than 30% is achieved. The probabilities of instability are plotted against their respective interramp slope angles for each model and the slope angle expected to yield a suitable probability of instability (5% or 30% depending on failure consequence) is determined.

Results of slope stability modeling are summarized in Table 9 and 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.

**Table 9: Results of Interramp/Overall Slope Stability Modeling**

| Deposit | Sector | Height (m) | Mean FoS | PoF (%) |
|---|---|---|---|---|
| Area 2 | Northeast | 130m | 2.5 | 0.7 |
| Area 2 | Southwest | 214m | 2.1 | 2.9 |
| Ridgetop | - | 130m | 2.3 | 2.4 |
| Minto North | - | 130m | 2.3 | 0.0 |

### 7.1.1 Area 2 and Area 118

Results of the interramp/overall slope stability analysis of the Area 2 east wall are shown graphically in Figure 17. The hatched area is the Critical Deterministic Surface which is defined as the slip surface with the lowest safety factor when all the input parameters are equal to their mean values. The remaining surfaces shown are all of the Global Minimum Surfaces that were located by the analyses when the properties were sampled randomly.

The critical slip surface for the east wall is a circular surface initiating at the base of the weathered bedrock. Surfaces initiating at the toe of the slope were also evaluated.

Although the critical failure surface shown in Figure 17 represent a relatively low interramp slope failure, its location directly above the main haul road and suggests that a failure through weathered bedrock materials could have a significant impact on mine operations. Consequentially, critical surfaces were evaluated both at the toe of the slope and at the interface between weathered and fresh bedrock for this model.

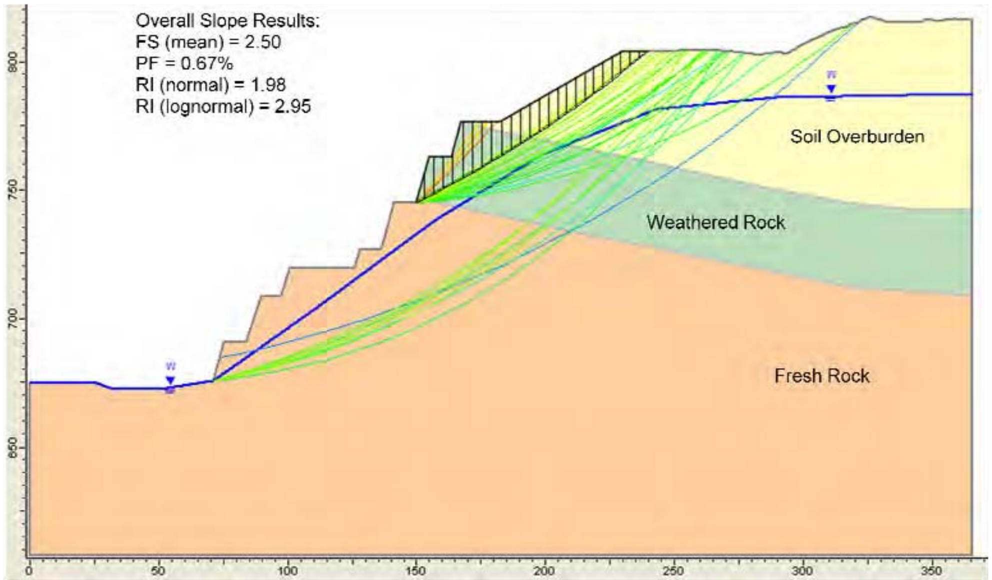



**Figure 17: Interramp and overall stability modeling results: Area 2 east wall**

Results of interramp/overall slope stability modeling of the Area 2 west wall are shown in Figure 18. Surfaces initiating at the base of the weathered domain and the toe of the overall slope were again considered due to the proximity of the weathered rock to the main haul road. The critical slip surface initiates at the base of the weathered rock.

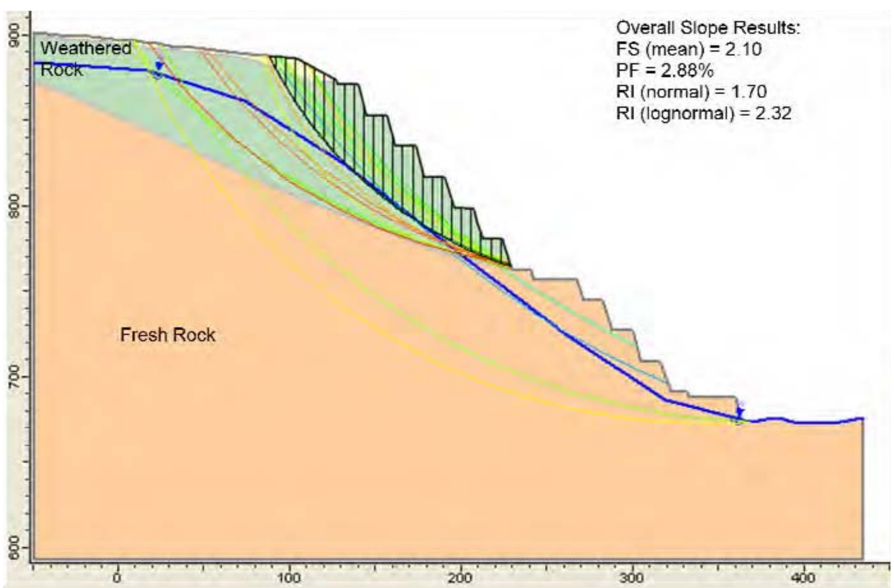



**Figure 18: Interramp and overall stability modeling results: Area 2 west wall**

Given the small size of the Area 118 pit as well as its close proximity and geotechnical similarities to Area 2, additional interramp slope stability modeling was not deemed necessary for Area 118 at the pre-feasibility level.

### 7.1.2 Ridgetop

Results of interramp stability modeling of the generalized Ridgetop section indicate a probability of failure of approximately 2.4% and critical slip surface initiating at toe of the slope (Figure 19). Surfaces initiating at the base of the weathered bedrock were also evaluated during the analysis.

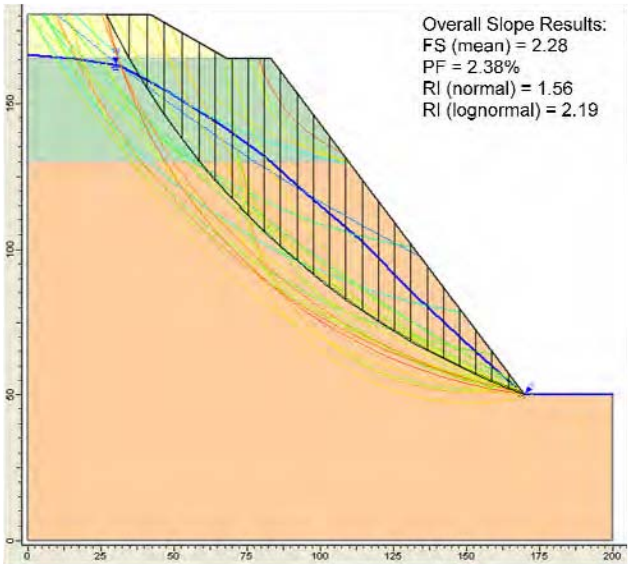



**Figure 19: Interramp stability modeling results: Area 2 east wall**

### 7.1.3 Minto North

An interramp slope angle of 52 degrees yields a probability of failure approaching zero percent. Results of the interramp/overall slope stability analysis of the Minto North section are shown graphically in Figure 20.

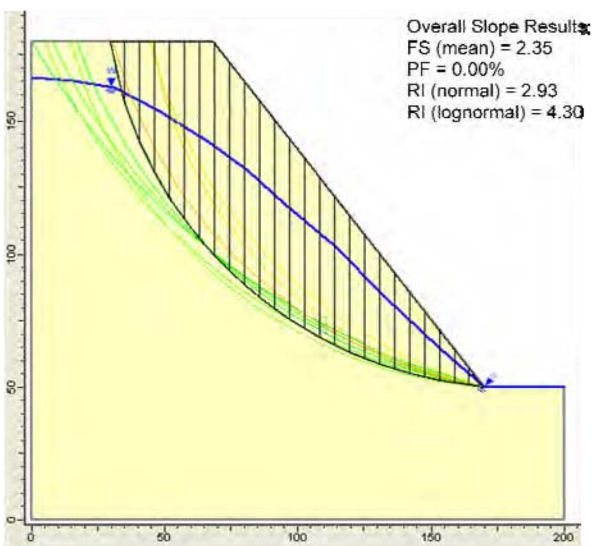



**Figure 20: Interramp stability modeling results: Minto North**

# 8 Geologic Discontinuity Analysis

Geologic discontinuity influenced failure mechanisms were analyzed at both the pit wall and bench scales. The term discontinuity refers to any significant mechanical break or fracture having negligible tensile strength in the rock. 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.

## 8.1 Major Geologic 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.

Several faults or shear zones have been identified in resource and geotechnical drilling at all of the subject sites. Most of these structures are not anticipated to significantly impact pit slope stability due to their apparent lack of persistence and associated limited degree of rock degradation. However, the potential for one or more major structures to adversely impact stability of the Area 2 west wall has been identified and should be investigated further as the project advances.

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. For a pre-feasibility level evaluation, geotechnical drilling is targeted to obtain data representative of overall rock mass conditions, and to a lesser extent, individual structures such as those previously mentioned.

### 8.1.1 Area 2 and Area 118

Both resource and geotechnical drilling in southwestern Area 2 suggest a major fault(s) potentially striking northwest, 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 disrupted zones at down hole depths of approximately 279m and 243m, respectively. However, the same indications were not observed in adjacent holes, thereby suggesting a high dip angle for the structure.

Geotechnical drillholes C09-03 and C07-07 also intersected major structures at shallower depths that would be consistent with the potential structure(s) and would coincide with the western Area 2 ultimate pit wall.

Major faults at similar orientations are also anticipated through the Area 118 underground mining areas and development.

During the recent geotechnical core logging program, three orientations were measured on different striations contained within two different fault zones in core from drillholes C09-02 and C09-03.

### 8.1.2 Ridgetop

During logging of geotechnical drillholes C09-04 and C09-05, orientation measurements were obtained on seven different zones believed to be related to faulting. Poles to the discontinuities bounding these zones are shown on Figure 21.

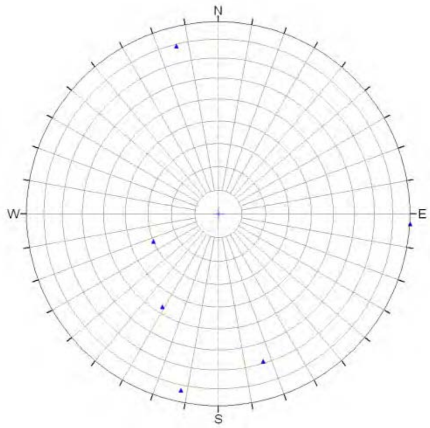



**Figure 21: Pole plot of oriented faults at Ridgetop**

### 8.1.3 Minto North

Geotechnical and resource drilling at Minto North suggests multiple sub-horizontal structures above the ore zone as well as a sub-vertical fault striking approximately north-south through the mid portion of the pit. Given the relatively shallow pit depth at Minto North, the fault zones associated with these structures could potentially form a significant portion of the pit walls.

Two orientations were obtained on potential fault zones in geotechnical drillhole C09-08; poles to the two faults logged are shown on Figure 22.

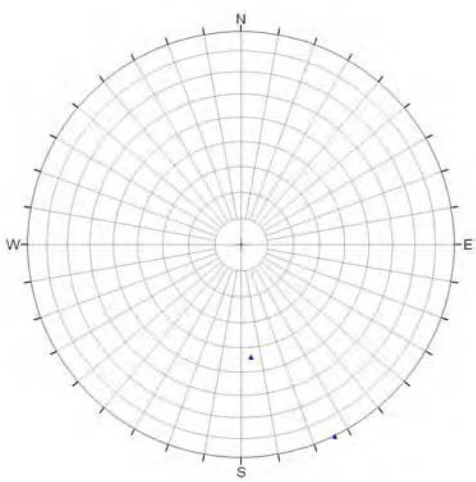



**Figure 22: Pole plot of oriented faults at Minto North**

## 8.2 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 modeled 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, i.e., dip direction and dip, are designated as a single discontinuity set.

### 8.2.1 Discontinuity Orientation

The depth of intercept and the angles of the discontinuities relative to the core axis and perpendicular to the core axis, (alpha and beta angles, respectively) were measured during logging to enable the calculation of the true dip direction and dip.

Accounting for the plunge and azimuth of each drillhole, discontinuity alpha and beta angles were converted to dip and dip direction using the commercially available software package, Dips developed by Rocscience, Inc. (2003). Discontinuity data from each of the geotechnical coreholes was contoured on an equal area percent plot for analysis of structural stability. In most cases, visual inspection of these plots revealed preferred discontinuity orientations. The contour plots are presented on Figure 23 through 25.

After the discontinuity measurements were converted into in situ orientations, the combined data set of discontinuities was divided into categories of which, given sufficient persistence, 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.

A summary of discontinuity sets delineated and incorporated in the analysis of bench stability is presented in Table 10.

**Table 10: Design Discontinuity Sets**

| Discontinuity Set Information | | | | Dip | | DDR | |
|---|---|---|---|---|---|---|---|
| **Deposit** | **Sector** | **Set ID** | **No.** | **Mean** | **Stdev.** | **Mean** | **Stdev.** |
| Ridgetop | - | J1 | 275 | 37.1 | 12.3 | 104.3 | 21.8 |
| Ridgetop | - | J2 | 174 | 47.4 | 9.6 | 35.8 | 18.9 |
| Area 2 | South | J1 | 135 | 51.1 | 8.5 | 47.9 | 12.6 |
| Area 2 | South | J2 | 150 | 46.0 | 12.9 | 1.4 | 13.8 |
| Area 2 | West | J3 | 142 | 17.7 | 7.8 | 25.7 | 36.0 |
| Area 2 | West | J4 | 107 | 69.7 | 11.4 | 16.4 | 13.9 |
| Area 2 | West | J5 | 86 | 48.5 | 7.8 | 92.6 | 17.3 |
| Area 2 | North | J6 | 206 | 62.5 | 13.4 | 13.6 | 23.8 |
| Area 2 | North | J7 | 123 | 19.1 | 9.4 | 21.3 | 40.3 |
| Area 2 | North | J8 | 73 | 50.9 | 8.1 | 92.4 | 16.4 |

### 8.2.2 Design Sectors

Slope angles within an open pit mine are influenced not only by geologic structure, rock mass strength and porewater 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. Design sectors for Area 2 and Ridgetop are shown on Figures 26 and 27, respectively.

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.

### 8.2.3 Backbreak Analysis

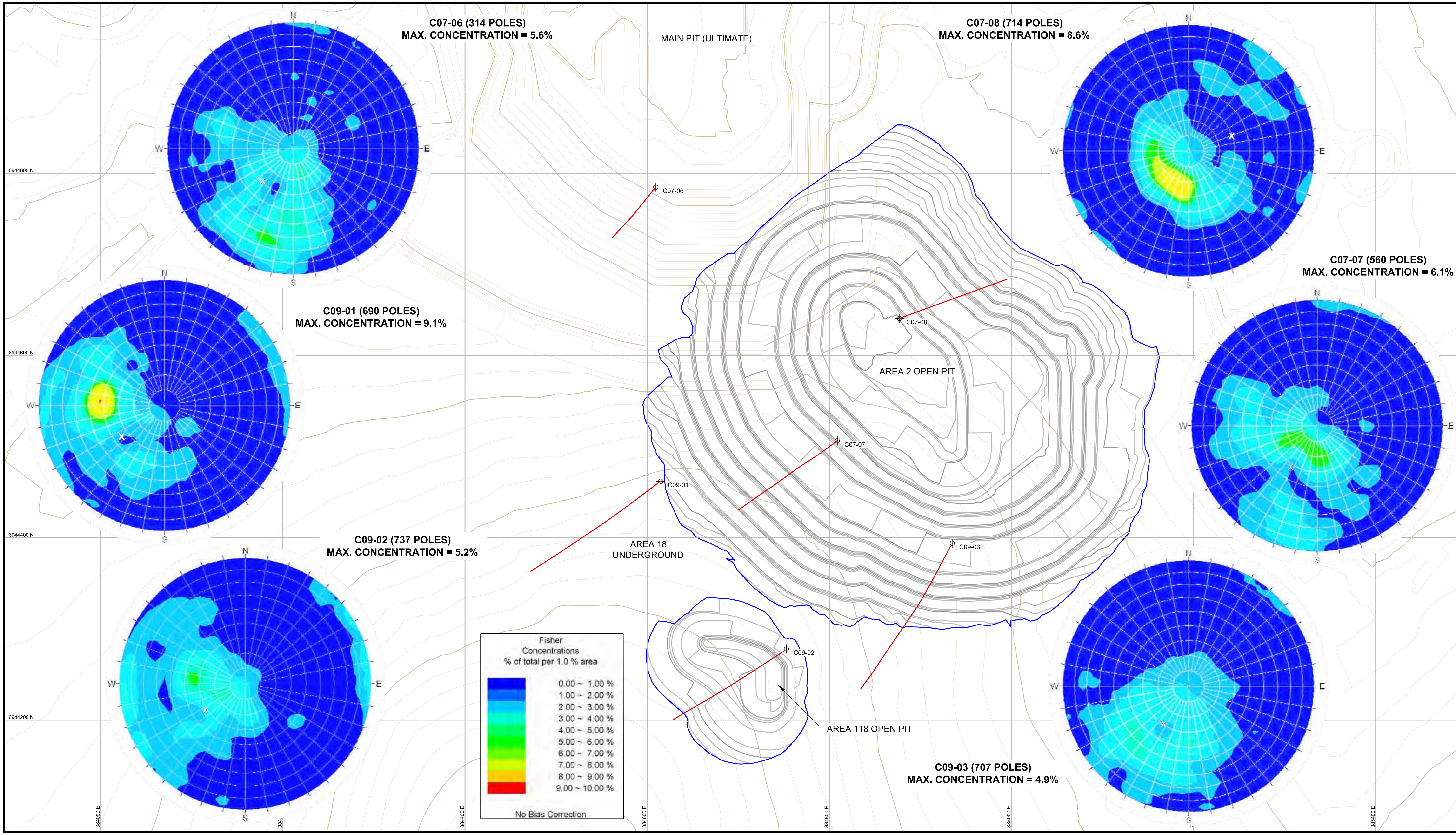
Preliminary kinematic analyses indicated that the south and west sectors of Area 2, Area 118 and Ridgetop had potential for bench scale instabilities; consequentially, additional, backbreak analyses were carried out for those sectors. SRK's backbreak analyses use stochastic simulations of discontinuity 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. Pit sectors selected for backbreak analyses and their respective discontinuity sets are summarized in Table 11.

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 shallow discontinuity dip angles relative to the anticipated shear strength of the discontinuities at Ridgetop, 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.

**Table 11: Summary of backbreak analyses per sector**

| Area | Sector | Sub-sector | Plane Shear | Wedge |
|---|---|---|---|---|
| Area 2 | Northwest | - | J8 | J6/J8 |
| Area 2 | West | W1 | - | J4/J5 |
| Area 2 | West | W2 | - | - |
| Area 2 | South | S1 | - | J1/J2 |
| Area 2 | South | S2 | J2 | J1/J2 |
| Area 2 | Northeast | - | - | - |
| Ridgetop | West | - | J1 | J1/J2 |
| Ridgetop | Southwest | - | J2 | - |
| Ridgetop | Northeast | - | - | - |




LEGEND

EXISTING GROUND CONTOURS (MAJOR/MINOR) 5 METER INTERVAL

CURRENT PFS ULTIMATE PIT CONTOURS (MAJOR/MINOR) 5 METER INTERVAL

C09-02 GEOTECHNICAL DRILLHOLE COLLAR LOCATION AND HORIZONTAL BOREHOLE PROJECTION

N

0 20 40 60 80 100
Scale in Metres

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| | | |
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| SRK Consulting | MINTO EXPLORATIONS LTD. | PREFEASIBILITY GEOTECHNICAL EVALUATION<br>DISCONTINUITY POLE PLOTS: AREA 2 AND AREA 118 |
| SRK JOB NO.: 2CM022.006<br>FILE NAME: 02CM022.06.Fig.8-3.Rev.A.Discontinuity.Pole.Plots.A2.and.A118.2009-11-18.dwg | MINTO MINE | DATE: DEC. 2009 / APPROVED: ML / FIGURE: 23 / REVISION NO. A |

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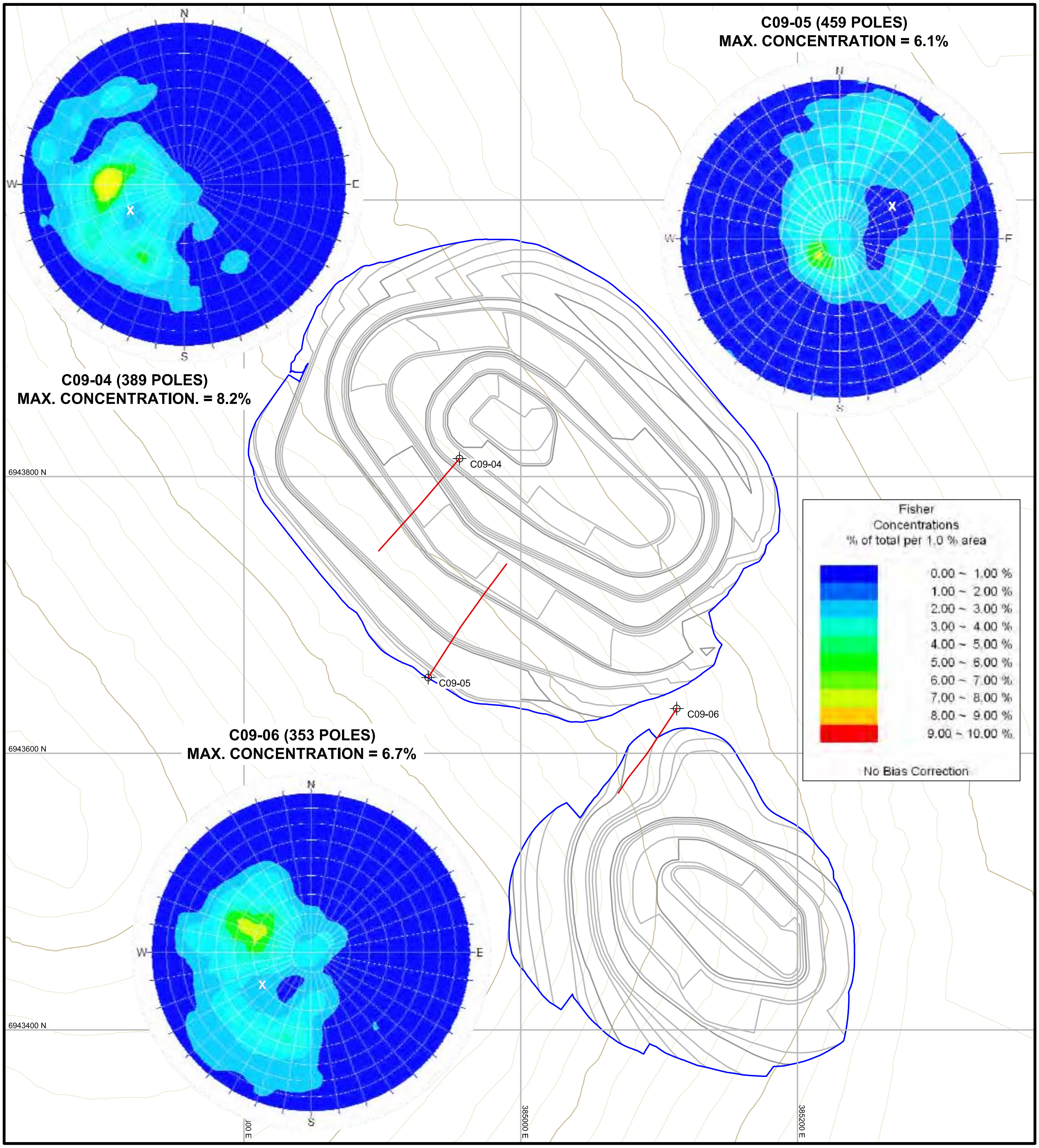


LEGEND

- EXISTING GROUND CONTOURS (MAJOR/MINOR) 5 METER INTERVAL
- CURRENT PFS ULTIMATE PIT CONTOURS (MAJOR/MINOR) 5 METER INTERVAL
- C09-02 GEOTECHNICAL DRILLHOLE COLLAR LOCATION AND HORIZONTAL BOREHOLE PROJECTION

0 20 40 60 80 100
Scale in Metres

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| FILE NAME: 02CM022.06.Fig.8-4.Rev.A.Discontinuity.Pole.Plots.Ridge.Top.2009-11-18.dwg | MINTO MINE | DISCONTINUITY POLE PLOTS: RIDGETOP |

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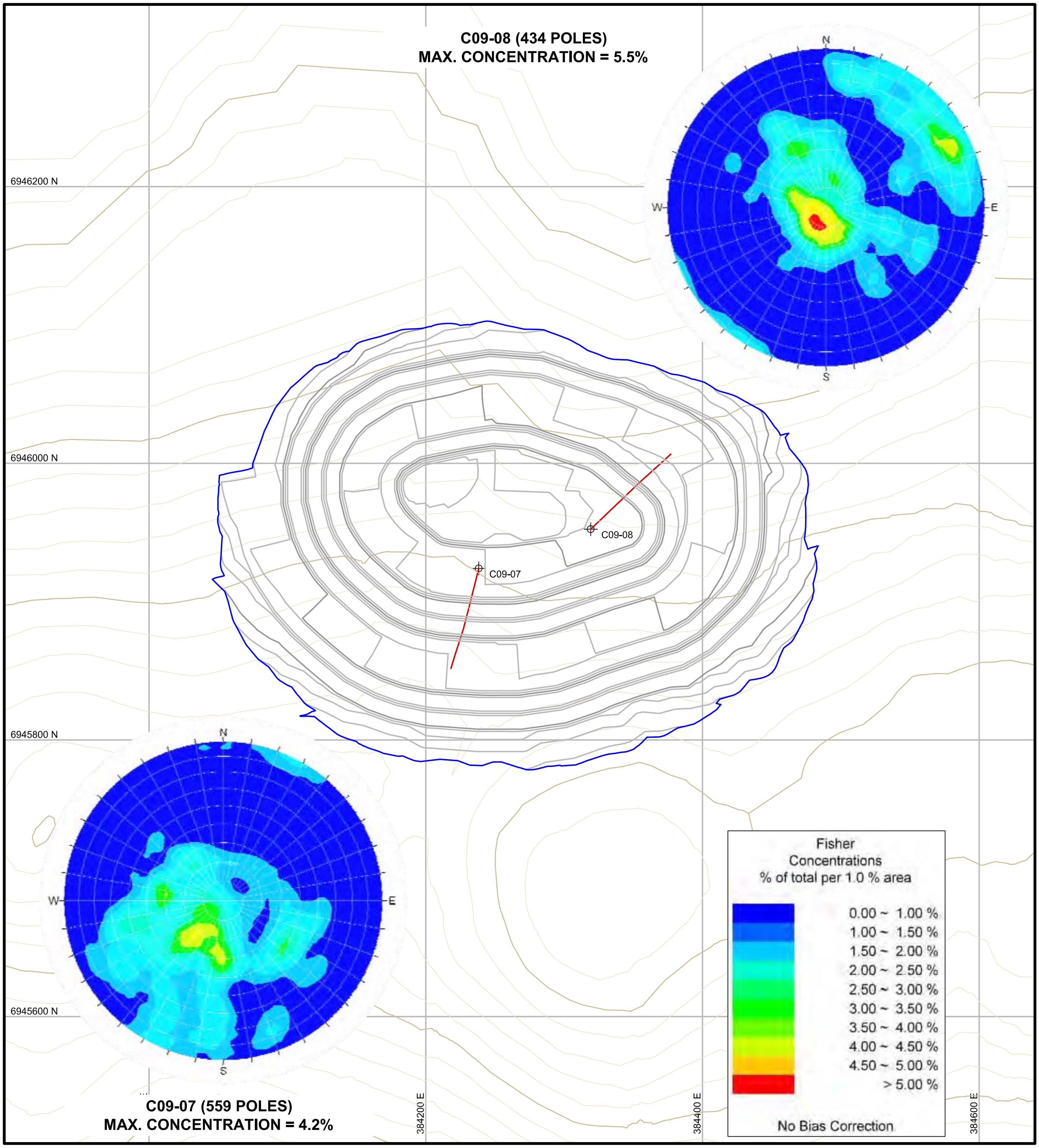


LEGEND

- 5280 — EXISTING GROUND CONTOURS (MAJOR/MINOR) 5 METER INTERVAL
- 5280 — CURRENT PFS ULTIMATE PIT CONTOURS (MAJOR/MINOR) 5 METER INTERVAL
- C09-02 — GEOTECHNICAL DRILLHOLE COLLAR LOCATION AND HORIZONTAL BOREHOLE PROJECTION

N

0 20 40 60 80 100
Scale in Metres

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PREFEASIBILITY GEOTECHNICAL EVALUATION

DISCONTINUITY POLE PLOTS:
MINTO NORTH

MINTO MINE

| SRK JOB NO.: | 2CM022.006 |
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| FILE NAME: | 02CM022.06.Fig.8-5.Rev.A.Discontinuity.Pole.Plots.North.2009-11-18.dwg |

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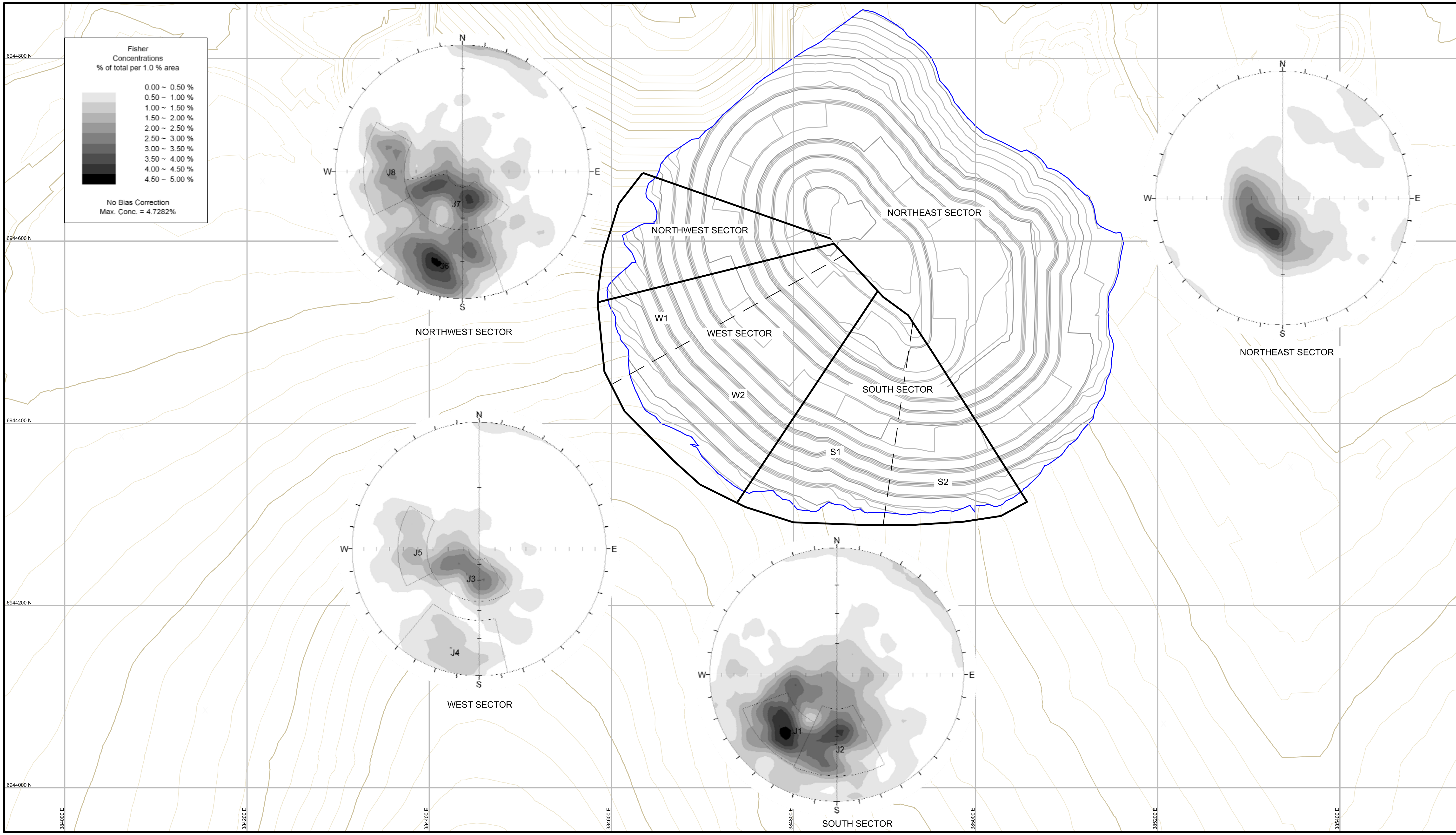


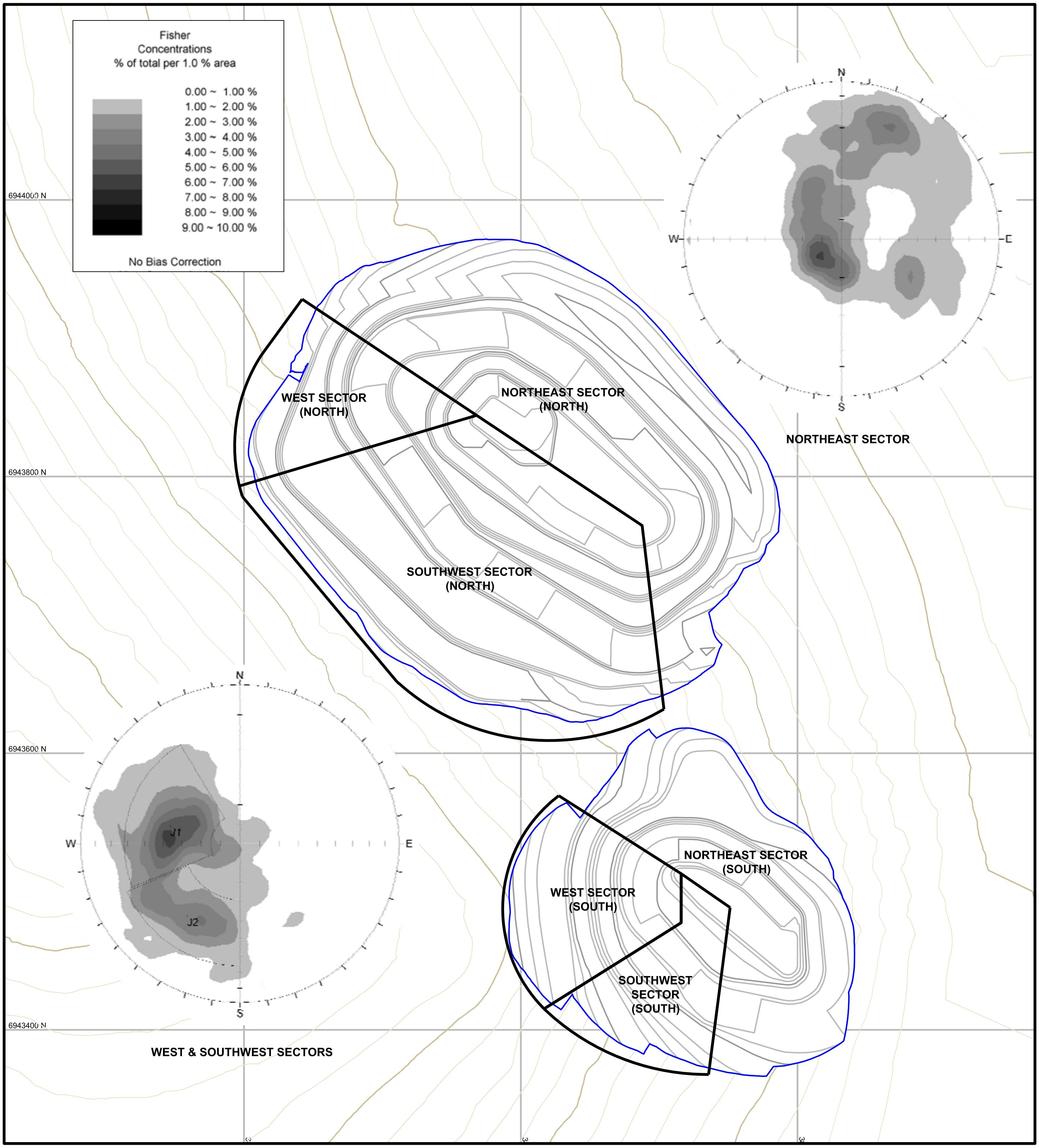

LEGEND

- EXISTING GROUND CONTOURS (MAJOR/MINOR) 5 METER INTERVAL
- CURRENT PFS ULTIMATE PIT CONTOURS (MAJOR/MINOR) 5 METER INTERVAL
- C09-02 GEOTECHNICAL DRILLHOLE COLLAR LOCATION AND HORIZONTAL BOREHOLE PROJECTION

0 20 40 60 80 100
Scale in Metres

| | | |
|---|---|---|
| SRK Consulting<br>Engineers and Scientists<br>7175 West Jefferson Ave. Suite 3000<br>Denver, Colorado 80235<br>303-985-1333 | MINTO EXPLORATIONS LTD. | PREFEASIBILITY GEOTECHNICAL EVALUATION<br>DESIGN SECTORS:<br>AREA 2 |
| SRK JOB NO.: 2CM022.006<br>FILE NAME: 02CM022.06.Fig.8-6.Rev.A.Design.Sectors.A2.and.A118.2009-11-23.dwg | MINTO MINE | DATE: DEC. 2009 · APPROVED: ML · FIGURE: 26 · REVISION NO. A |

T:\Minto Mine Yukon Territory\!040_AutoCAD\A118,A2, Ridge Top & N PFS Figures\Figures\02CM022.06.Fig.8-6.Rev.A.Design.Sectors.A2.and.A118.2009-11-23.dwg




LEGEND

- EXISTING GROUND CONTOURS (MAJOR/MINOR) 5 METER INTERVAL
- CURRENT PFS ULTIMATE PIT CONTOURS (MAJOR/MINOR) 5 METER INTERVAL
- C09-02 GEOTECHNICAL DRILLHOLE COLLAR LOCATION AND HORIZONTAL BOREHOLE PROJECTION

0 20 40 60 80 100
Scale in Metres

SRK Consulting
Engineers and Scientists
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Denver, Colorado 80235
303-985-1333

MINTO EXPLORATIONS LTD.

PREFEASIBILITY GEOTECHNICAL EVALUATION

DESIGN SECTORS:
RIDGETOP

SRK JOB NO.: 2CM022.006

FILE NAME: 02CM022.06.Fig.8-7.Rev.A.Design.Sectors.Ridge.Top.2009-11-21.dwg

MINTO MINE

| DATE: | APPROVED: | FIGURE: | REVISION NO. |
|---|---|---|---|
| DEC. 2009 | ML | 27 | A |

T:\Minto Mine Yukon Territory\040_AutoCAD\A118.A2, Ridge Top & N PFS Figures\Figures\02CM022.06.Fig.8-7.Rev.A.Design.Sectors.Ridge.Top.2009-11-21.dwg

# 9 Pit Slope Design Recommendations

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 effectively control rockfall and/or overbank slough accumulation. In such instances, recommended interramp angles are flattened sufficiently to provide adequately wide average catch benches.

Pit slope design recommendations for each area are summarized in Table 12.

**Table 12: Summary of Pit Slope Design Recommendations**

| Deposit Area | Sector(s) | Max. Slope Height (m) | Interramp Angle (°) | Bench Face Angle (°) | Bench Height (m) | Berm Width (m) | Stepout 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 7m, a 15m offset or stepout should be incorporated at, or vertically near, the contact between the overburden and the bedrock.

The Area 2 pit sectors are depicted in Figure 26. A similar delineation of the Area 118 pit, i.e., one based on relative position, is recommended for the Area 118 pit.

# 10 Area 118 Underground Pillar Assessment

In addition to the small open pit at Area 118 previously discussed, underground mining is also planned for Area 118. Based on the geotechnical data previously described, pillar strengths were evaluated in order to recommend suitable pillar dimensions for room and pillar mining. Based on estimates of ore deposit depth and thickness variability, pillar heights of 5m, 10m and 15m were assessed and ore depths, and respective overburden stresses, of 150m, 200m and 250m were considered.

In-situ Rock Mass Rating (IRMR) and Rock Mass Strength (RMS) values were evaluated for the ore zone as well as materials above and below the ore zone in geotechnical drillholes C09-01 and C09-02. A design IRMR and RMS of 55 and 60 MPa, respectively, were conservatively estimated for pillar, roof and floor materials. Using Laubscher's (1990) method, the IRMR of 55 was reduced to a Mining Rock Mass Rating (MRMR) of 47 and the 60 MPa RMS to a Design Rock Mass Strength (DRMS) of 51 MPa by applying appropriate reductions for joint orientation, blasting and water.

Based on empirical data presented by Ouchi (2004), assuming a RMR value of 55, the maximum unsupported span distance was estimated to be 6m for all pillar height/deposit depth combinations considered, as shown in Figure 28.

**Figure 28: Critical span curve (Ouchi 2004)**

Subsequently, the tributary area method was used to estimate minimum pillar dimensions required to support 6m x 6m or, if required, lesser, roof spans based on pillar height and overburden stresses.  The resultant recommended room and pillar dimensions and extraction ratios are summarized in Table 13.

### Table 13: Summary of Room and Pillar Size Recommendations

| Depth (m) | Pillar Height (m) | Pillar Dimensions (m) | Room Dimensions (m) | Extraction Ratio |
|---|---|---|---|---|
| 150 | 5 | 4x4 | 6x6 | 84% |
| 150 | 10 | 5x5 | 6x6 | 79% |
| 150 | 15 | 6x6 | 6x6 | 75% |
| 200 | 5 | 4.5x4.5 | 6x6 | 82% |
| 200 | 10 | 6x6 | 6x6 | 75% |
| 200 | 15 | 7.5x7.5 | 6x6 | 69% |
| 250 | 5 | 5x5 | 6x6 | 79% |
| 250 | 10 | 7x7 | 6x6 | 71% |
| 250 | 15 | 8x8 | 5x5 | 62% |

Based on geotechnical conditions previously described, ground support requirements for development such as the 5mx5m decline were estimated as follows:

Recommendations for ground support for development include:

- Pattern bolting with 2.4m long bolts at a 2m spacing within and between rings; and,
- Welded wire mesh in back and top of walls.

# 11 Assessment of Future 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 drillholes 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 portion of Area 118 will also require additional geotechnical drilling for rock mass characterization at the feasibility and design levels. The Area 118 and Ridgetop open pits most likely will not require additional geotechnical drilling unless major changes are made to the current plans.

# 12 References

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# Appendices

# Appendix A: Geotechnical Core Logs

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-04 **TO** 2009-03-08
**DIP:** -57.00 **AZIMUTH:** 236.00
**COORDINATES:** 6944462.53N 384615.22E **DATUM:** Nad83

**BOREHOLE:** C09-01
**PAGE:** 1 **OF** 7
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC424
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa): 50, 100, 150, 200 | FF/m | FF/m, FF/m CJ+J: 10, 20, 30; TCR, RQD (%): 25, 50, 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa): 20, 40, 60, 80

Depth scale: 25, 50, 75, 100, 125, 150 ft; 25 m

Stratigraphy (top to bottom): OVB, pG, fG, pG, fG, pG, Peg, pG, Peg, pG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 1.3 | 6 | 37 | 2 |
| 3.3 | 19 | 50 | 32 |
| 4.7 | 21 | 46 | 28 |
| 6.0 | 11 | 30 | 10 |
| 6.7 | 22 | 43 | 26 |
| 8.7 | 21 | 40 | 24 |
| 6.7 | 18 | 41 | 25 |
| 3.3 | 22 | 50 | 32 |
| 2.7 | 26 | 55 | 35 |
| 2.0 | 16 | 43 | 12 |
| 20.0 | 7 | 21 | 5 |
| 4.7 | 24 | 49 | 31 |
| 3.3 | 25 | 53 | 34 |
| 2.7 | 22 | 51 | 32 |
| 2.7 | 20 | 49 | 31 |
| 1.3 | 18 | 52 | 33 |
| 2.0 | 17 | 48 | 30 |
| 3.3 | 21 | 49 | 31 |
| 4.0 | 19 | 46 | 28 |
| 3.3 | 21 | 57 | 68 |
| 2.0 | 11 | 52 | 60 |
| 1.3 | 9 | 56 | 66 |
| 3.3 | 21 | 57 | 68 |
| 1.3 | 21 | 68 | 88 |
| 2.0 | 17 | 58 | 70 |
| 2.7 | 9 | 48 | 52 |
| 7.3 | 21 | 51 | 58 |
| 4.7 | 18 | 51 | 58 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-04 **TO** 2009-03-08
**DIP:** -57.00 **AZIMUTH:** 236.00
**COORDINATES:** 6944462.53N 384615.22E **DATUM:** Nad83

**BOREHOLE:** C09-01
**PAGE:** 2 **OF** 7
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC424
**PLAN No:**

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RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

- Gouge
- Sheared
- Jointed
- Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

Column headings: DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth - ft: 175, 200, 225, 250, 275, 300, 325
Depth - m: 75

Stratigraphy (top to bottom): fG, pG, fG, pG, Peg, pG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 4.7 | 21 | 56 | 66 |
| 2.7 | 14 | 51 | 58 |
| 2.0 | 20 | 61 | 75 |
| 3.3 | 14 | 52 | 60 |
| 2.7 | 21 | 60 | 74 |
| 2.7 | 9 | 51 | 58 |
| 1.3 | 21 | 68 | 88 |
| 4.7 | 21 | 54 | 63 |
| 3.3 | 20 | 58 | 70 |
| 2.7 | 21 | 58 | 70 |
| 2.0 | 21 | 65 | 82 |
| 4.7 | 21 | 59 | 72 |
| 2.7 | 9 | 51 | 58 |
| 2.0 | 30 | 71 | 93 |
| 6.0 | 17 | 49 | 54 |
| 3.3 | 21 | 59 | 72 |
| 2.7 | 21 | 58 | 70 |
| 3.3 | 16 | 54 | 63 |
| 2.0 | 21 | 62 | 77 |
| 0.7 | 22 | 74 | 98 |
| 4.0 | 19 | 53 | 61 |
| 2.0 | 17 | 56 | 66 |
| 2.7 | 18 | 57 | 68 |
| 2.7 | 12 | 51 | 58 |
| 1.3 | 8 | 55 | 65 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 8 | 55 | 65 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 3.3 | 19 | 55 | 65 |
| 1.3 | 21 | 68 | 88 |
| 0.7 | 21 | 73 | 96 |

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# GEOTECHNICAL CORE LOG

| | | | |
|---|---|---|---|
| **PROJECT:** Minto PFS Geotechnical Evaluation | | **BOREHOLE:** C09-01 | |
| **LOCATION:** Yukon Territory, Canada | | **PAGE:** 3 | **OF** 7 |
| **SITE & PROJECT No:** MINTO (2CM022.006) | | **DRILL TYPE:** | |
| **BORING DATE:** 2009-03-04 | **TO** 2009-03-08 | **DRILL:** | |
| **DIP:** -57.00 | **AZIMUTH:** 236.00 | **CLIENT HOLE ID:** 09SWC424 | |
| **COORDINATES:** 6944462.53N 384615.22E | **DATUM:** Nad83 | **PLAN No:** | |

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RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

Column headings: DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth markers: 350, 375, 400, 425, 450, 475 ft; 125 m

Stratigraphy (top to bottom): fG, pG, fG, pG, fG, eG, fG, eG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 0.7 | 9 | 61 | 75 |
| 0.7 | 8 | 60 | 74 |
| 1.3 | 8 | 52 | 60 |
| 0.7 | 26 | 78 | 105 |
| 2.0 | 19 | 63 | 79 |
| 0.7 | 10 | 62 | 77 |
| 0.7 | 22 | 74 | 98 |
| 5.3 | 21 | 58 | 70 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 22 | 69 | 89 |
| 2.0 | 19 | 60 | 74 |
| 0.7 | 19 | 71 | 93 |
| 1.3 | 19 | 63 | 79 |
| 0.7 | 24 | 76 | 102 |
| 2.0 | 26 | 70 | 91 |
| 0.7 | 19 | 71 | 93 |
| 3.3 | 21 | 57 | 68 |
| 0.7 | 26 | 78 | 105 |
| 3.3 | 23 | 61 | 75 |
| 4.0 | 25 | 62 | 77 |
| 1.3 | 19 | 66 | 84 |
| 2.7 | 19 | 61 | 75 |
| 3.3 | 22 | 60 | 74 |
| 5.3 | 24 | 58 | 70 |
| 4.7 | 23 | 58 | 70 |
| 2.7 | 16 | 55 | 65 |
| 0.7 | 22 | 68 | 59 |
| 0.7 | 22 | 64 | 39 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 19 | 66 | 84 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 29 | 81 | 110 |
| 0.7 | 22 | 74 | 98 |

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**COORDINATES:** 6944462.53N 384615.22E **DATUM:** Nad83

**BOREHOLE:** C09-01
**PAGE:** 4 **OF** 7
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC424
**PLAN No:**

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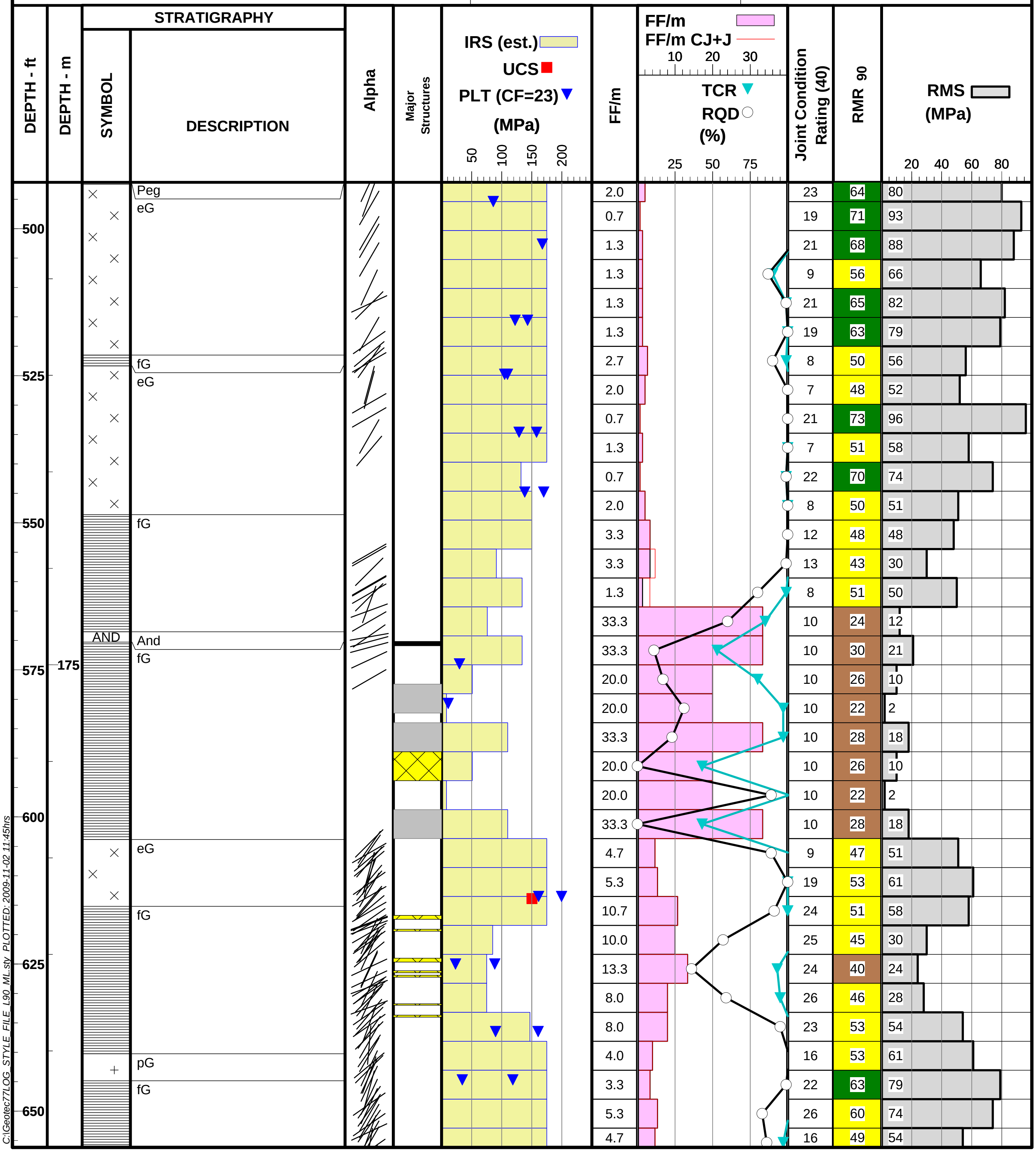
**LEGEND OF MAJOR STRUCTURES**

- Gouge
- Sheared
- Jointed
- Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100




| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.0 | 23 | 64 | 80 |
| 0.7 | 19 | 71 | 93 |
| 1.3 | 21 | 68 | 88 |
| 1.3 | 9 | 56 | 66 |
| 1.3 | 21 | 65 | 82 |
| 1.3 | 19 | 63 | 79 |
| 2.7 | 8 | 50 | 56 |
| 2.0 | 7 | 48 | 52 |
| 0.7 | 21 | 73 | 96 |
| 1.3 | 7 | 51 | 58 |
| 0.7 | 22 | 70 | 74 |
| 2.0 | 8 | 50 | 51 |
| 3.3 | 12 | 48 | 48 |
| 3.3 | 13 | 43 | 30 |
| 1.3 | 8 | 51 | 50 |
| 33.3 | 10 | 24 | 12 |
| 33.3 | 10 | 30 | 21 |
| 20.0 | 10 | 26 | 10 |
| 20.0 | 10 | 22 | 2 |
| 33.3 | 10 | 28 | 18 |
| 20.0 | 10 | 26 | 10 |
| 20.0 | 10 | 22 | 2 |
| 33.3 | 10 | 28 | 18 |
| 4.7 | 9 | 47 | 51 |
| 5.3 | 19 | 53 | 61 |
| 10.7 | 24 | 51 | 58 |
| 10.0 | 25 | 45 | 30 |
| 13.3 | 24 | 40 | 24 |
| 8.0 | 26 | 46 | 28 |
| 8.0 | 23 | 53 | 54 |
| 4.0 | 16 | 53 | 61 |
| 3.3 | 22 | 63 | 79 |
| 5.3 | 26 | 60 | 74 |
| 4.7 | 16 | 49 | 54 |

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**DIP:** -57.00 **AZIMUTH:** 236.00
**COORDINATES:** 6944462.53N 384615.22E **DATUM:** Nad83

**BOREHOLE:** C09-01
**PAGE:** 5 **OF** 7
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**DRILL:**
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**PLAN No:**

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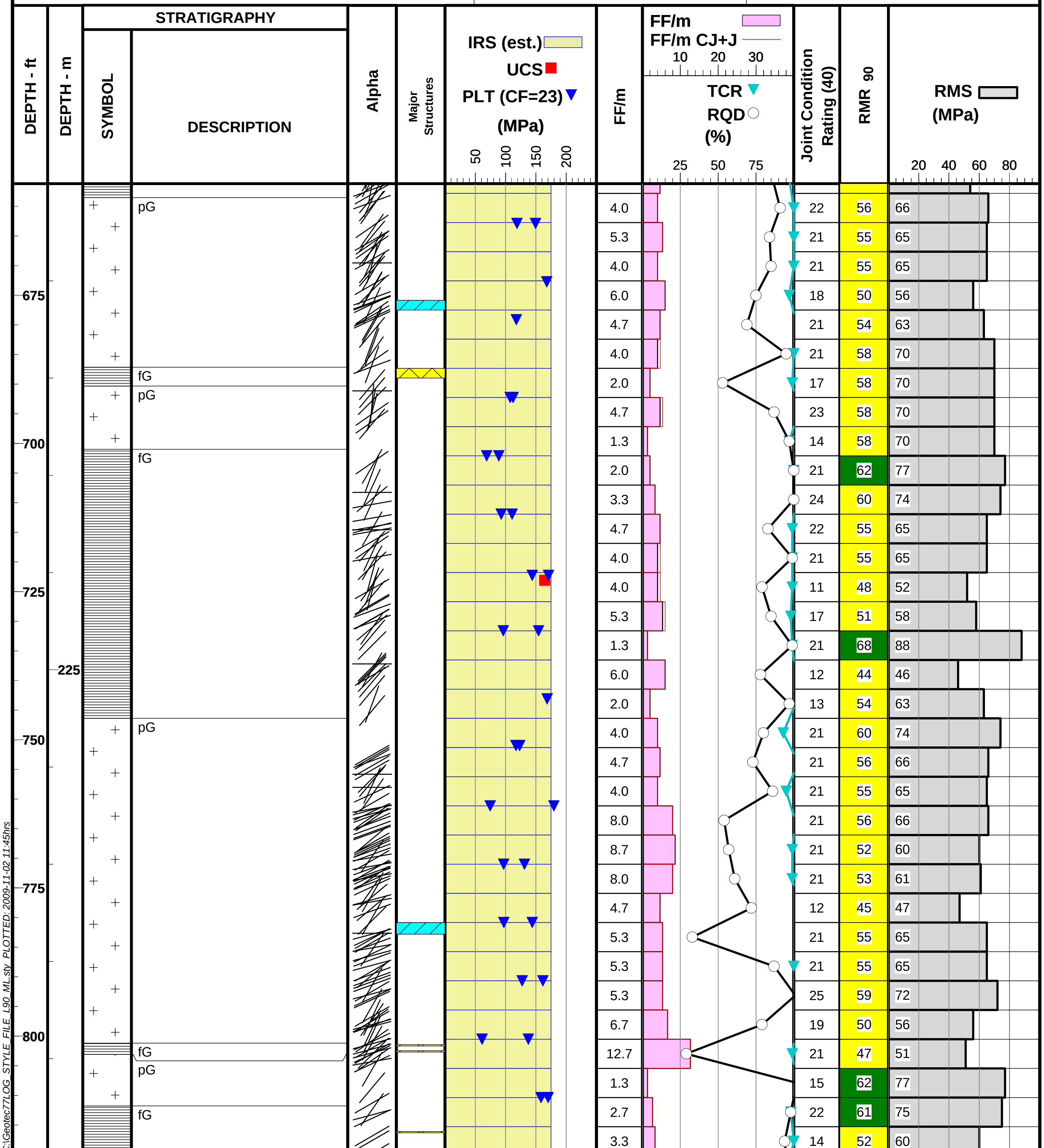
**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100




| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 4.0 | 22 | 56 | 66 |
| 5.3 | 21 | 55 | 65 |
| 4.0 | 21 | 55 | 65 |
| 6.0 | 18 | 50 | 56 |
| 4.7 | 21 | 54 | 63 |
| 4.0 | 21 | 58 | 70 |
| 2.0 | 17 | 58 | 70 |
| 4.7 | 23 | 58 | 70 |
| 1.3 | 14 | 58 | 70 |
| 2.0 | 21 | 62 | 77 |
| 3.3 | 24 | 60 | 74 |
| 4.7 | 22 | 55 | 65 |
| 4.0 | 21 | 55 | 65 |
| 4.0 | 11 | 48 | 52 |
| 5.3 | 17 | 51 | 58 |
| 1.3 | 21 | 68 | 88 |
| 6.0 | 12 | 44 | 46 |
| 2.0 | 13 | 54 | 63 |
| 4.0 | 21 | 60 | 74 |
| 4.7 | 21 | 56 | 66 |
| 4.0 | 21 | 55 | 65 |
| 8.0 | 21 | 56 | 66 |
| 8.7 | 21 | 52 | 60 |
| 8.0 | 21 | 53 | 61 |
| 4.7 | 12 | 45 | 47 |
| 5.3 | 21 | 55 | 65 |
| 5.3 | 21 | 55 | 65 |
| 5.3 | 25 | 59 | 72 |
| 6.7 | 19 | 50 | 56 |
| 12.7 | 21 | 47 | 51 |
| 1.3 | 15 | 62 | 77 |
| 2.7 | 22 | 61 | 75 |
| 3.3 | 14 | 52 | 60 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-04 **TO** 2009-03-08
**DIP:** -57.00 **AZIMUTH:** 236.00
**COORDINATES:** 6944462.53N 384615.22E **DATUM:** Nad83

**BOREHOLE:** C09-01

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC424
**PLAN No:**

**DEFINITIONS**

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**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

Columns: DEPTH - ft | DEPTH - m | STRATIGRAPHY (SYMBOL, DESCRIPTION) | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth (ft): 850, 875, 900, 925, 950, 975
Depth (m): 275

Stratigraphy: pG, fG, pG, fG, pG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.0 | 16 | 60 | 74 |
| 1.3 | 20 | 67 | 86 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 21 | 68 | 88 |
| 2.7 | 21 | 60 | 74 |
| 1.3 | 20 | 67 | 86 |
| 2.0 | 22 | 66 | 84 |
| 0.7 | 20 | 72 | 94 |
| 1.3 | 24 | 71 | 93 |
| 3.3 | 22 | 60 | 74 |
| 2.0 | 21 | 62 | 77 |
| 0.7 | 24 | 76 | 102 |
| 1.3 | 20 | 67 | 86 |
| 1.3 | 21 | 68 | 88 |
| 1.3 | 26 | 73 | 96 |
| 0.7 | 26 | 78 | 105 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 21 | 73 | 96 |
| 1.3 | 21 | 68 | 88 |
| 0.7 | 26 | 78 | 105 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 26 | 78 | 105 |
| 1.3 | 21 | 68 | 88 |
| 1.3 | 17 | 64 | 80 |
| 2.0 | 12 | 56 | 66 |
| 2.0 | 17 | 61 | 75 |
| 0.7 | 21 | 73 | 96 |
| 0.7 | 22 | 74 | 98 |
| 2.0 | 21 | 65 | 82 |
| 2.0 | 17 | 58 | 70 |
| 1.3 | 6 | 53 | 61 |
| 2.7 | 12 | 54 | 63 |
| 1.3 | 18 | 62 | 77 |

# GEOTECHNICAL CORE LOG

SRK Consulting — Engineers and Scientists

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-04 **TO** 2009-03-08
**DIP:** -57.00 **AZIMUTH:** 236.00
**COORDINATES:** 6944462.53N 384615.22E **DATUM:** Nad83

**BOREHOLE:** C09-01

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC424
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

- Gouge
- Sheared
- Jointed
- Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

Column headings: DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) (scale 50, 100, 150, 200) | FF/m | FF/m, FF/m CJ+J (scale 10, 20, 30); TCR, RQD (%) (scale 25, 50, 75) | Joint Condition Rating (40) | RMR 90 | RMS (MPa) (scale 20, 40, 60, 80)

Depth markers (ft): 1000, 1025, 1050, 1075, 1100, 1125. Depth marker (m): 325 (end of logged interval).

Stratigraphy (top to bottom): fG, pG (above 1000 ft), fG (from ~1025 ft), pG (from ~1043 ft).

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 1.3 | 8 | 55 | 65 |
| 1.3 | 13 | 57 | 68 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 12 | 56 | 66 |
| 2.0 | 7 | 48 | 52 |
| 0.7 | 26 | 78 | 105 |
| 0.7 | 7 | 59 | 72 |
| 0.7 | 21 | 73 | 96 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 26 | 78 | 105 |
| 0.7 | 29 | 81 | 110 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 21 | 68 | 88 |
| 1.3 | 16 | 63 | 79 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 24 | 71 | 93 |

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SRK Consulting
Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-01 **TO** 2009-03-04
**DIP:** -58.00 **AZIMUTH:** 239.00
**COORDINATES:** 6944276.36N 384751.29E **DATUM:** Nad83

**BOREHOLE:** C09-02
**PAGE:** 1 **OF** 6
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC422
**PLAN No:**

**DEFINITIONS**

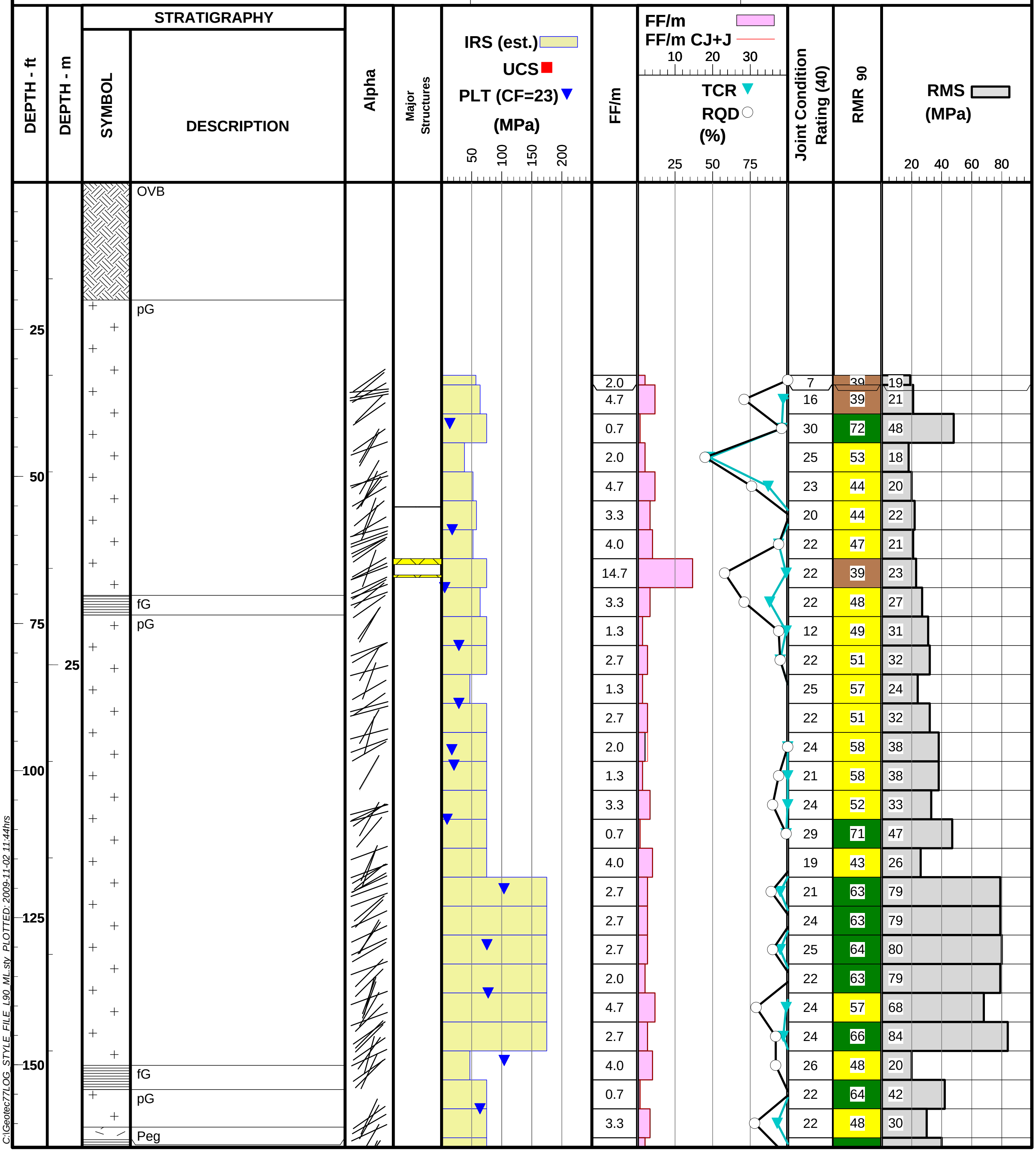
IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100




| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.0 | 7 | 39 | 19 |
| 4.7 | 16 | 39 | 21 |
| 0.7 | 30 | 72 | 48 |
| 2.0 | 25 | 53 | 18 |
| 4.7 | 23 | 44 | 20 |
| 3.3 | 20 | 44 | 22 |
| 4.0 | 22 | 47 | 21 |
| 14.7 | 22 | 39 | 23 |
| 3.3 | 22 | 48 | 27 |
| 1.3 | 12 | 49 | 31 |
| 2.7 | 22 | 51 | 32 |
| 1.3 | 25 | 57 | 24 |
| 2.7 | 22 | 51 | 32 |
| 2.0 | 24 | 58 | 38 |
| 1.3 | 21 | 58 | 38 |
| 3.3 | 24 | 52 | 33 |
| 0.7 | 29 | 71 | 47 |
| 4.0 | 19 | 43 | 26 |
| 2.7 | 21 | 63 | 79 |
| 2.7 | 24 | 63 | 79 |
| 2.7 | 25 | 64 | 80 |
| 2.0 | 22 | 63 | 79 |
| 4.7 | 24 | 57 | 68 |
| 2.7 | 24 | 66 | 84 |
| 4.0 | 26 | 48 | 20 |
| 0.7 | 22 | 64 | 42 |
| 3.3 | 22 | 48 | 30 |

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SRK Consulting
Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-01 **TO** 2009-03-04
**DIP:** -58.00 **AZIMUTH:** 239.00
**COORDINATES:** 6944276.36N 384751.29E **DATUM:** Nad83

**BOREHOLE:** C09-02

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC422
**PLAN No:**

**DEFINITIONS**

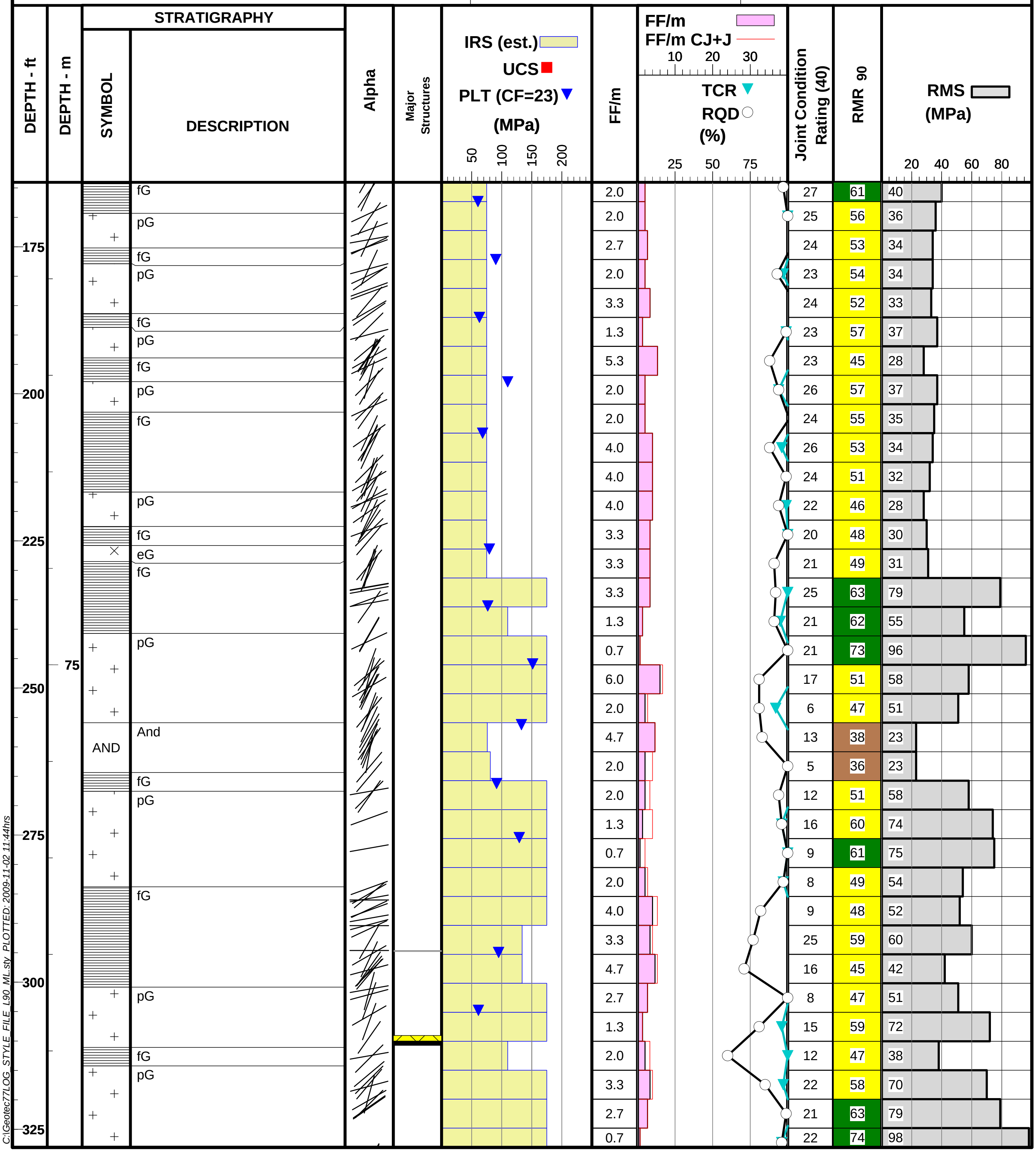
IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100




| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.0 | 27 | 61 | 40 |
| 2.0 | 25 | 56 | 36 |
| 2.7 | 24 | 53 | 34 |
| 2.0 | 23 | 54 | 34 |
| 3.3 | 24 | 52 | 33 |
| 1.3 | 23 | 57 | 37 |
| 5.3 | 23 | 45 | 28 |
| 2.0 | 26 | 57 | 37 |
| 2.0 | 24 | 55 | 35 |
| 4.0 | 26 | 53 | 34 |
| 4.0 | 24 | 51 | 32 |
| 4.0 | 22 | 46 | 28 |
| 3.3 | 20 | 48 | 30 |
| 3.3 | 21 | 49 | 31 |
| 3.3 | 25 | 63 | 79 |
| 1.3 | 21 | 62 | 55 |
| 0.7 | 21 | 73 | 96 |
| 6.0 | 17 | 51 | 58 |
| 2.0 | 6 | 47 | 51 |
| 4.7 | 13 | 38 | 23 |
| 2.0 | 5 | 36 | 23 |
| 2.0 | 12 | 51 | 58 |
| 1.3 | 16 | 60 | 74 |
| 0.7 | 9 | 61 | 75 |
| 2.0 | 8 | 49 | 54 |
| 4.0 | 9 | 48 | 52 |
| 3.3 | 25 | 59 | 60 |
| 4.7 | 16 | 45 | 42 |
| 2.7 | 8 | 47 | 51 |
| 1.3 | 15 | 59 | 72 |
| 2.0 | 12 | 47 | 38 |
| 3.3 | 22 | 58 | 70 |
| 2.7 | 21 | 63 | 79 |
| 0.7 | 22 | 74 | 98 |

# GEOTECHNICAL CORE LOG

SRK Consulting
Engineers and Scientists

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-01 **TO** 2009-03-04
**DIP:** -58.00 **AZIMUTH:** 239.00
**COORDINATES:** 6944276.36N 384751.29E **DATUM:** Nad83

**BOREHOLE:** C09-02
**PAGE:** 3 **OF** 6
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC422
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft: 350, 375, 400, 425, 450, 475
DEPTH - m: 125

STRATIGRAPHY — SYMBOL / DESCRIPTION: fG, pG, fG, Peg, pG, fG

Alpha | Major Structures | IRS (est.) | UCS | PLT (CF=23) (MPa): 50, 100, 150, 200 | FF/m | FF/m, FF/m CJ+J: 10, 20, 30 | TCR, RQD (%): 25, 50, 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa): 20, 40, 60, 80

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.0 | 8 | 49 | 54 |
| 2.0 | 7 | 48 | 52 |
| 2.7 | 8 | 47 | 51 |
| 2.7 | 10 | 49 | 54 |
| 1.3 | 4 | 51 | 58 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 3.3 | 18 | 54 | 63 |
| 1.3 | 21 | 68 | 88 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 26 | 70 | 91 |
| 4.0 | 22 | 59 | 72 |
| 4.0 | 24 | 58 | 70 |
| 3.3 | 14 | 52 | 60 |
| 6.0 | 22 | 56 | 66 |
| 5.3 | 17 | 49 | 54 |
| 3.3 | 23 | 61 | 75 |
| 4.0 | 15 | 52 | 60 |
| 2.7 | 26 | 65 | 82 |
| 5.3 | 24 | 56 | 66 |
| 4.0 | 21 | 58 | 70 |
| 0.7 | 16 | 68 | 88 |
| 6.0 | 20 | 52 | 60 |
| 2.7 | 21 | 58 | 70 |
| 3.3 | 22 | 60 | 74 |
| 3.3 | 24 | 62 | 77 |
| 2.7 | 24 | 63 | 79 |
| 2.0 | 23 | 64 | 80 |
| 2.0 | 18 | 59 | 72 |
| 3.3 | 22 | 60 | 74 |
| 2.0 | 19 | 63 | 79 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |

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**SRK Consulting** Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-01 **TO** 2009-03-04
**DIP:** -58.00 **AZIMUTH:** 239.00
**COORDINATES:** 6944276.36N 384751.29E **DATUM:** Nad83

**BOREHOLE:** C09-02

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC422
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

- Gouge
- Sheared
- Jointed
- Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY (SYMBOL, DESCRIPTION) | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa): 50, 100, 150, 200 | FF/m | FF/m, FF/m CJ+J: 10, 20, 30; TCR, RQD (%): 25, 50, 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa): 20, 40, 60, 80

Depth (ft): 500, 525, 550, 575, 600, 625, 650. Depth (m): 175.

Stratigraphy (top to bottom): fG, pG, fG, pG, fG, pG, fG, pG, fG, pG, fG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.0 | 19 | 60 | 74 |
| 0.7 | 24 | 76 | 102 |
| 1.3 | 24 | 71 | 93 |
| 0.7 | 26 | 72 | 67 |
| 4.0 | 22 | 51 | 31 |
| 1.3 | 6 | 47 | 39 |
| 3.3 | 22 | 54 | 47 |
| 3.3 | 26 | 64 | 80 |
| 1.3 | 20 | 67 | 86 |
| 2.7 | 24 | 63 | 79 |
| 2.0 | 20 | 64 | 80 |
| 1.3 | 22 | 63 | 57 |
| 1.3 | 16 | 60 | 74 |
| 2.0 | 19 | 60 | 74 |
| 1.3 | 15 | 59 | 72 |
| 0.7 | 6 | 58 | 70 |
| 2.0 | 6 | 47 | 51 |
| 1.3 | 10 | 54 | 63 |
| 2.0 | 13 | 54 | 63 |
| 2.0 | 10 | 49 | 54 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 21 | 73 | 96 |
| 0.7 | 9 | 61 | 75 |
| 2.0 | 7 | 48 | 52 |
| 2.7 | 18 | 57 | 68 |
| 3.3 | 9 | 47 | 51 |
| 0.7 | 8 | 60 | 74 |
| 3.3 | 9 | 47 | 51 |
| 1.3 | 8 | 52 | 60 |
| 2.0 | 10 | 54 | 63 |
| 2.0 | 11 | 50 | 56 |
| 2.7 | 8 | 47 | 51 |
| 3.3 | 21 | 59 | 72 |

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SRK Consulting
Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-01 **TO** 2009-03-04
**DIP:** -58.00 **AZIMUTH:** 239.00
**COORDINATES:** 6944276.36N 384751.29E **DATUM:** Nad83

**BOREHOLE:** C09-02

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC422
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

Columns: DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth markers: 675, 700, 725, 750, 775, 800 ft; 225 m

Stratigraphy (top to bottom): pG, Peg, pG, Peg, pG, fG, pG, fG, pG, Peg, pG, fG, pG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 6.0 | 21 | 55 | 65 |
| 3.3 | 21 | 57 | 68 |
| 3.3 | 21 | 59 | 72 |
| 0.7 | 9 | 61 | 75 |
| 4.0 | 21 | 58 | 70 |
| 2.0 | 21 | 62 | 77 |
| 2.7 | 12 | 51 | 58 |
| 2.7 | 22 | 61 | 75 |
| 3.3 | 7 | 45 | 47 |
| 1.3 | 26 | 73 | 96 |
| 2.7 | 26 | 68 | 88 |
| 2.0 | 26 | 70 | 91 |
| 0.7 | 20 | 72 | 94 |
| 2.0 | 19 | 63 | 79 |
| 2.0 | 19 | 63 | 79 |
| 6.7 | 20 | 51 | 58 |
| 4.7 | 24 | 59 | 72 |
| 4.0 | 22 | 61 | 75 |
| 4.7 | 19 | 54 | 63 |
| 4.0 | 25 | 62 | 77 |
| 3.3 | 20 | 58 | 70 |
| 8.0 | 21 | 51 | 58 |
| 6.0 | 20 | 54 | 63 |
| 2.7 | 23 | 62 | 77 |
| 8.7 | 21 | 55 | 65 |
| 4.0 | 24 | 58 | 70 |
| 2.0 | 24 | 68 | 88 |
| 5.3 | 24 | 61 | 75 |
| 2.7 | 24 | 66 | 84 |
| 4.7 | 22 | 60 | 74 |
| 8.0 | 26 | 61 | 75 |
| 0.7 | 21 | 73 | 96 |
| 3.3 | 22 | 60 | 74 |
| 4.0 | 25 | 62 | 77 |

![SRK Consulting — Engineers and Scientists]

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-01 **TO** 2009-03-04
**DIP:** -58.00 **AZIMUTH:** 239.00
**COORDINATES:** 6944276.36N 384751.29E **DATUM:** Nad83

**BOREHOLE:** C09-02
**PAGE:** 6 **OF** 6
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC422
**PLAN No:**

**DEFINITIONS**

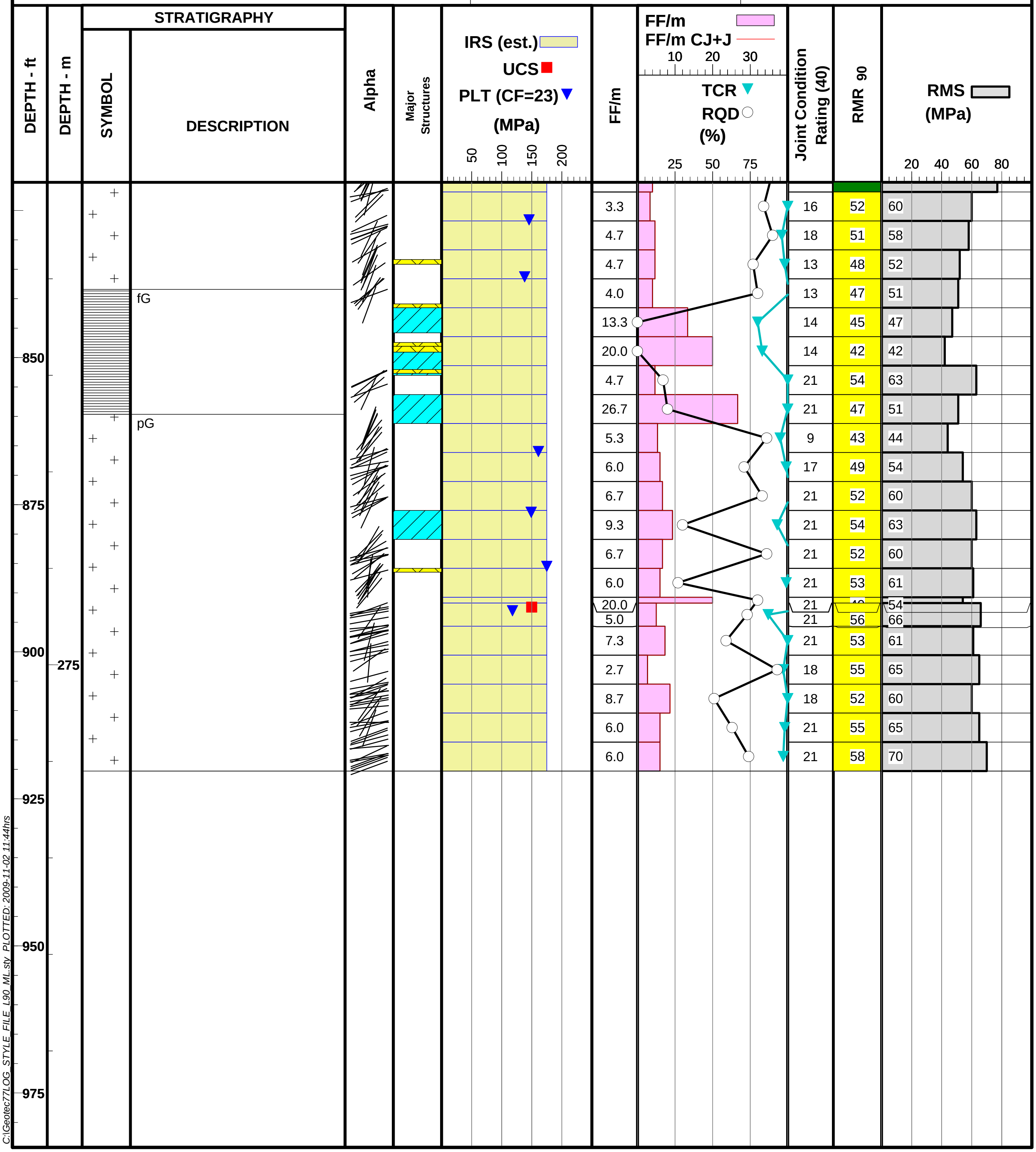
IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

- Gouge
- Sheared
- Jointed
- Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100




| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 3.3 | 16 | 52 | 60 |
| 4.7 | 18 | 51 | 58 |
| 4.7 | 13 | 48 | 52 |
| 4.0 | 13 | 47 | 51 |
| 13.3 | 14 | 45 | 47 |
| 20.0 | 14 | 42 | 42 |
| 4.7 | 21 | 54 | 63 |
| 26.7 | 21 | 47 | 51 |
| 5.3 | 9 | 43 | 44 |
| 6.0 | 17 | 49 | 54 |
| 6.7 | 21 | 52 | 60 |
| 9.3 | 21 | 54 | 63 |
| 6.7 | 21 | 52 | 60 |
| 6.0 | 21 | 53 | 61 |
| 20.0 | 21 | [illegible] | 54 |
| 5.0 | 21 | 56 | 66 |
| 7.3 | 21 | 53 | 61 |
| 2.7 | 18 | 55 | 65 |
| 8.7 | 18 | 52 | 60 |
| 6.0 | 21 | 55 | 65 |
| 6.0 | 21 | 58 | 70 |

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SRK Consulting
Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-02-24 **TO** 2009-03-01
**DIP:** -61.00 **AZIMUTH:** 213.00
**COORDINATES:** 6944390.77N 384933.12E **DATUM:** Nad83

**BOREHOLE:** C09-03
**PAGE:** 1 **OF** 8
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC420
**PLAN No:**

**DEFINITIONS**

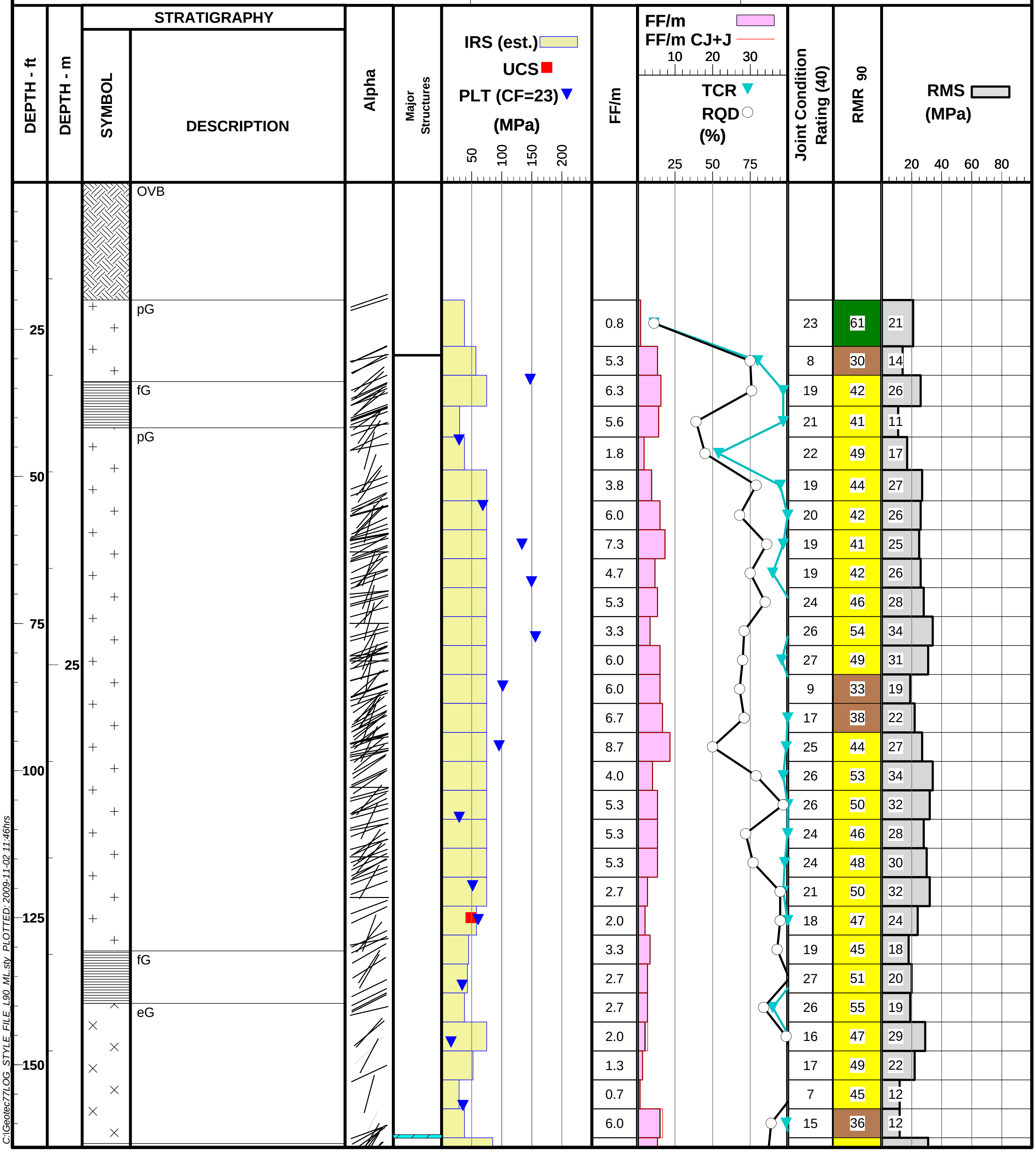
IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100




| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 0.8 | 23 | 61 | 21 |
| 5.3 | 8 | 30 | 14 |
| 6.3 | 19 | 42 | 26 |
| 5.6 | 21 | 41 | 11 |
| 1.8 | 22 | 49 | 17 |
| 3.8 | 19 | 44 | 27 |
| 6.0 | 20 | 42 | 26 |
| 7.3 | 19 | 41 | 25 |
| 4.7 | 19 | 42 | 26 |
| 5.3 | 24 | 46 | 28 |
| 3.3 | 26 | 54 | 34 |
| 6.0 | 27 | 49 | 31 |
| 6.0 | 9 | 33 | 19 |
| 6.7 | 17 | 38 | 22 |
| 8.7 | 25 | 44 | 27 |
| 4.0 | 26 | 53 | 34 |
| 5.3 | 26 | 50 | 32 |
| 5.3 | 24 | 46 | 28 |
| 5.3 | 24 | 48 | 30 |
| 2.7 | 21 | 50 | 32 |
| 2.0 | 18 | 47 | 24 |
| 3.3 | 19 | 45 | 18 |
| 2.7 | 27 | 51 | 20 |
| 2.7 | 26 | 55 | 19 |
| 2.0 | 16 | 47 | 29 |
| 1.3 | 17 | 49 | 22 |
| 0.7 | 7 | 45 | 12 |
| 6.0 | 15 | 36 | 12 |

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# GEOTECHNICAL CORE LOG

SRK Consulting
Engineers and Scientists

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-02-24 **TO** 2009-03-01
**DIP:** -61.00 **AZIMUTH:** 213.00
**COORDINATES:** 6944390.77N 384933.12E **DATUM:** Nad83

**BOREHOLE:** C09-03
**PAGE:** 2 **OF** 8
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC420
**PLAN No:**

**DEFINITIONS**

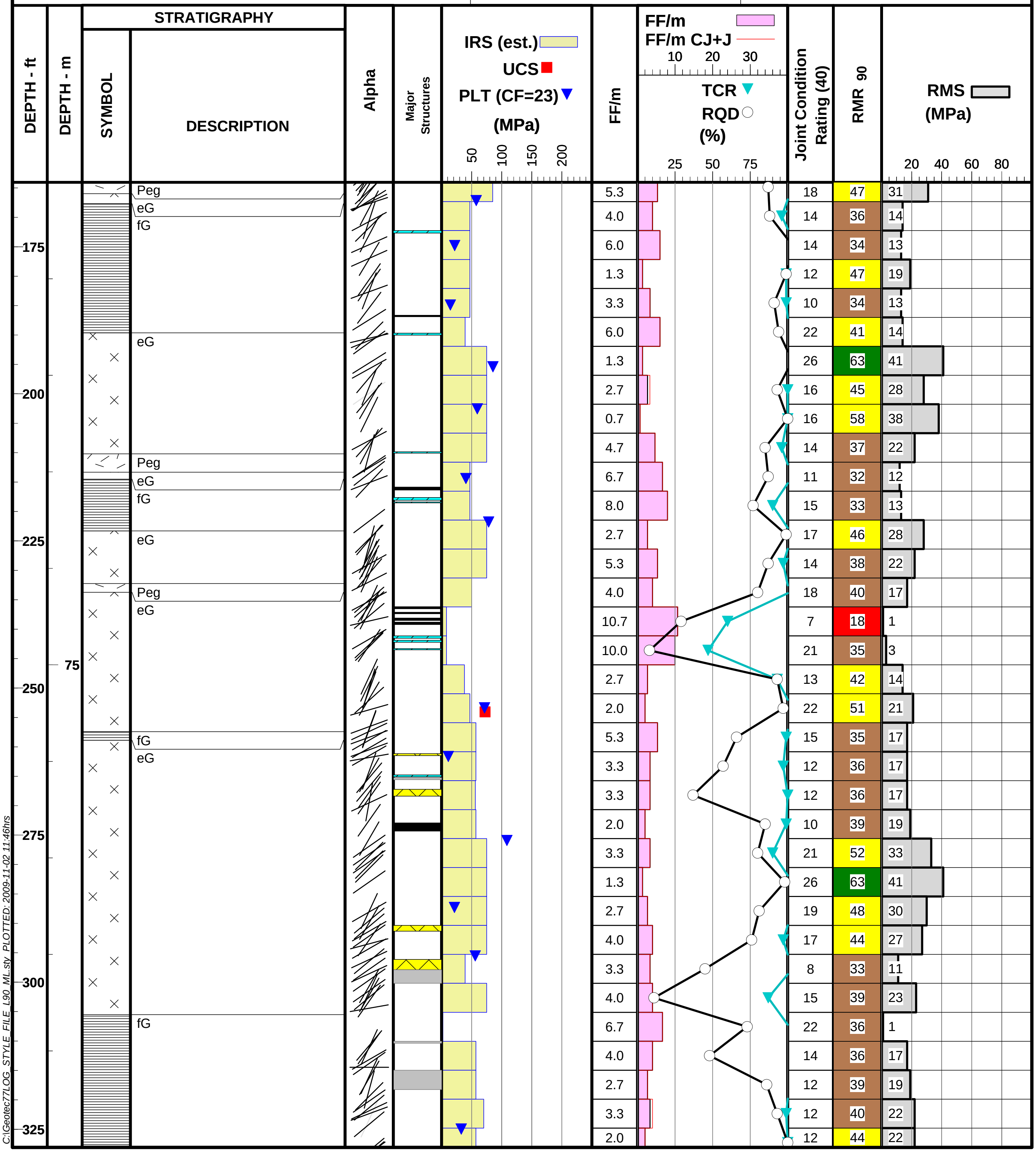
IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100




| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 5.3 | 18 | 47 | 31 |
| 4.0 | 14 | 36 | 14 |
| 6.0 | 14 | 34 | 13 |
| 1.3 | 12 | 47 | 19 |
| 3.3 | 10 | 34 | 13 |
| 6.0 | 22 | 41 | 14 |
| 1.3 | 26 | 63 | 41 |
| 2.7 | 16 | 45 | 28 |
| 0.7 | 16 | 58 | 38 |
| 4.7 | 14 | 37 | 22 |
| 6.7 | 11 | 32 | 12 |
| 8.0 | 15 | 33 | 13 |
| 2.7 | 17 | 46 | 28 |
| 5.3 | 14 | 38 | 22 |
| 4.0 | 18 | 40 | 17 |
| 10.7 | 7 | 18 | 1 |
| 10.0 | 21 | 35 | 3 |
| 2.7 | 13 | 42 | 14 |
| 2.0 | 22 | 51 | 21 |
| 5.3 | 15 | 35 | 17 |
| 3.3 | 12 | 36 | 17 |
| 3.3 | 12 | 36 | 17 |
| 2.0 | 10 | 39 | 19 |
| 3.3 | 21 | 52 | 33 |
| 1.3 | 26 | 63 | 41 |
| 2.7 | 19 | 48 | 30 |
| 4.0 | 17 | 44 | 27 |
| 3.3 | 8 | 33 | 11 |
| 4.0 | 15 | 39 | 23 |
| 6.7 | 22 | 36 | 1 |
| 4.0 | 14 | 36 | 17 |
| 2.7 | 12 | 39 | 19 |
| 3.3 | 12 | 40 | 22 |
| 2.0 | 12 | 44 | 22 |

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**SRK Consulting**
Engineers and Scientists

# GEOTECHNICAL CORE LOG

| | | | |
|---|---|---|---|
| **PROJECT:** Minto PFS Geotechnical Evaluation | | **BOREHOLE:** C09-03 | |
| **LOCATION:** Yukon Territory, Canada | | **PAGE:** 3 | **OF** 8 |
| **SITE & PROJECT No:** MINTO (2CM022.006) | | **DRILL TYPE:** | |
| **BORING DATE:** 2009-02-24 | **TO** 2009-03-01 | **DRILL:** | |
| **DIP:** -61.00 | **AZIMUTH:** 213.00 | **CLIENT HOLE ID:** 09SWC420 | |
| **COORDINATES:** 6944390.77N 384933.12E | **DATUM:** Nad83 | **PLAN No:** | |

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

Columns: DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth - ft: 350, 375, 400, 425, 450, 475
Depth - m: 125

Stratigraphy descriptions: Peg fG; Peg fG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.7 | 12 | 41 | 25 |
| 2.7 | 22 | 51 | 32 |
| 3.3 | 23 | 49 | 31 |
| 0.7 | 15 | 55 | 28 |
| 1.3 | 16 | 50 | 32 |
| 2.7 | 19 | 48 | 30 |
| 2.0 | 12 | 46 | 28 |
| 0.7 | 10 | 52 | 33 |
| 1.3 | 9 | 43 | 26 |
| 2.0 | 8 | 39 | 23 |
| 1.3 | 8 | 45 | 28 |
| 2.7 | 15 | 44 | 25 |
| 4.0 | 18 | 45 | 28 |
| 1.3 | 13 | 50 | 32 |
| 3.3 | 9 | 35 | 19 |
| 2.7 | 24 | 51 | 26 |
| 1.3 | 25 | 59 | 38 |
| 2.0 | 24 | 55 | 35 |
| 10.0 | 15 | 33 | 19 |
| 12.0 | 15 | 29 | 9 |
| 2.7 | 22 | 51 | 32 |
| 0.7 | 24 | 66 | 44 |
| 0.7 | 26 | 68 | 45 |
| 1.0 | 18 | 54 | 34 |
| 1.3 | 15 | 49 | 31 |
| 2.7 | 18 | 47 | 29 |
| 1.3 | 22 | 66 | 84 |
| 0.7 | 29 | 81 | 110 |
| 2.0 | 24 | 65 | 82 |
| 0.7 | 21 | 73 | 96 |
| 3.3 | 25 | 55 | 38 |
| 2.7 | 21 | 50 | 34 |

**SRK Consulting** Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-02-24 **TO** 2009-03-01
**DIP:** -61.00 **AZIMUTH:** 213.00
**COORDINATES:** 6944390.77N 384933.12E **DATUM:** Nad83

**BOREHOLE:** C09-03
**PAGE:** 4 **OF** 8
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC420
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

Columns: DEPTH - ft | DEPTH - m | STRATIGRAPHY (SYMBOL, DESCRIPTION) | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth (ft) marks: 500, 525, 550, 575, 600, 625, 650. Depth (m) mark: 175 (at about 575 ft).

Stratigraphy (top to bottom): Peg (about 575 ft); fG; eG; fG (about 625 ft); pG; fG; pG (about 652 ft)

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 3.3 | 19 | 57 | 68 |
| 0.7 | 15 | 67 | 86 |
| 2.0 | 24 | 65 | 82 |
| 1.3 | 18 | 62 | 77 |
| 1.3 | 22 | 69 | 89 |
| 0.7 | 20 | 72 | 94 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 2.0 | 8 | 49 | 54 |
| 1.3 | 15 | 59 | 72 |
| 0.7 | 15 | 67 | 86 |
| 2.0 | 18 | 59 | 72 |
| 2.7 | 21 | 60 | 74 |
| 2.7 | 17 | 54 | 63 |
| 2.7 | 18 | 55 | 65 |
| 3.3 | 16 | 52 | 60 |
| 0.7 | 21 | 73 | 96 |
| 2.7 | 14 | 51 | 58 |
| 1.3 | 13 | 57 | 68 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 13 | 57 | 68 |
| 1.3 | 19 | 66 | 84 |
| 0.7 | 7 | 59 | 72 |
| 2.7 | 12 | 54 | 63 |
| 3.3 | 7 | 45 | 47 |
| 2.0 | 12 | 53 | 61 |
| 0.7 | 22 | 70 | 75 |
| 1.3 | 12 | 59 | 72 |
| 1.3 | 10 | 54 | 63 |
| 0.7 | 9 | 61 | 75 |
| 2.0 | 12 | 56 | 66 |
| 1.3 | 4 | 51 | 58 |
| 2.7 | 15 | 54 | 63 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-02-24 **TO** 2009-03-01
**DIP:** -61.00 **AZIMUTH:** 213.00
**COORDINATES:** 6944390.77N 384933.12E **DATUM:** Nad83

**BOREHOLE:** C09-03
**PAGE:** 5 **OF** 8
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC420
**PLAN No:**

**DEFINITIONS**

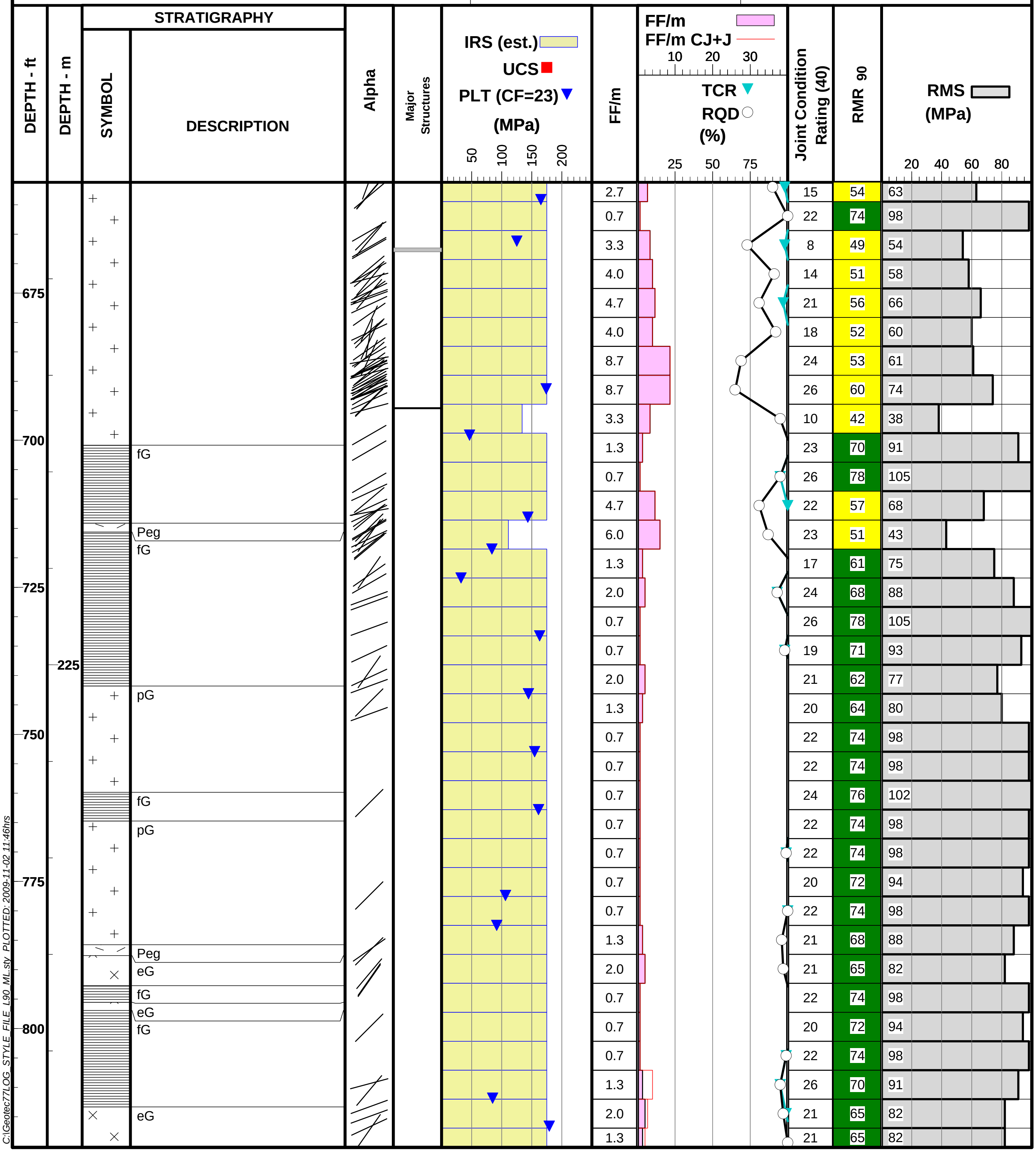
IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100




| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.7 | 15 | 54 | 63 |
| 0.7 | 22 | 74 | 98 |
| 3.3 | 8 | 49 | 54 |
| 4.0 | 14 | 51 | 58 |
| 4.7 | 21 | 56 | 66 |
| 4.0 | 18 | 52 | 60 |
| 8.7 | 24 | 53 | 61 |
| 8.7 | 26 | 60 | 74 |
| 3.3 | 10 | 42 | 38 |
| 1.3 | 23 | 70 | 91 |
| 0.7 | 26 | 78 | 105 |
| 4.7 | 22 | 57 | 68 |
| 6.0 | 23 | 51 | 43 |
| 1.3 | 17 | 61 | 75 |
| 2.0 | 24 | 68 | 88 |
| 0.7 | 26 | 78 | 105 |
| 0.7 | 19 | 71 | 93 |
| 2.0 | 21 | 62 | 77 |
| 1.3 | 20 | 64 | 80 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 24 | 76 | 102 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 20 | 72 | 94 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 21 | 68 | 88 |
| 2.0 | 21 | 65 | 82 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 20 | 72 | 94 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 26 | 70 | 91 |
| 2.0 | 21 | 65 | 82 |
| 1.3 | 21 | 65 | 82 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-02-24 **TO** 2009-03-01
**DIP:** -61.00 **AZIMUTH:** 213.00
**COORDINATES:** 6944390.77N 384933.12E **DATUM:** Nad83

**BOREHOLE:** C09-03
**PAGE:** 6 **OF** 8
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC420
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth (ft): 850, 875, 900, 925, 950, 975
Depth (m): 275

Stratigraphy (top to bottom): fG, eG, Peg, eG, fG, eG, AND, eG, Peg, eG, fG, eG, fG, eG, fG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 0.7 | 22 | 74 | 98 |
| 2.0 | 21 | 62 | 77 |
| 2.0 | 22 | 63 | 79 |
| 3.3 | 15 | 53 | 61 |
| 0.8 | 22 | 73 | 96 |
| 2.7 | 10 | 49 | 54 |
| 3.3 | 12 | [illegible] | 61 |
| 1.3 | 9 | 53 | 61 |
| 1.3 | 8 | 52 | 60 |
| 2.0 | 7 | 48 | 52 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 21 | 73 | 96 |
| 1.3 | 5 | 52 | 60 |
| 0.7 | 21 | 73 | 96 |
| 0.7 | 15 | 67 | 86 |
| 0.7 | 22 | 74 | 98 |
| 2.0 | 9 | 50 | 56 |
| 1.3 | 21 | 68 | 88 |
| 2.7 | 19 | 61 | 75 |
| 2.7 | 21 | 60 | 74 |
| 2.7 | 20 | 57 | 68 |
| 5.3 | 22 | 54 | 63 |
| 2.0 | 19 | 58 | 70 |
| 2.7 | 24 | 61 | 75 |
| 0.7 | 24 | 76 | 102 |
| 2.0 | 23 | 64 | 80 |
| 1.3 | 19 | 63 | 79 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 18 | 70 | 91 |
| 3.3 | 6 | 42 | 42 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 24 | 76 | 102 |
| 0.7 | 24 | 76 | 102 |
| 0.7 | 26 | 78 | 105 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-02-24 **TO** 2009-03-01
**DIP:** -61.00 **AZIMUTH:** 213.00
**COORDINATES:** 6944390.77N 384933.12E **DATUM:** Nad83

**BOREHOLE:** C09-03

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC420
**PLAN No:**

**DEFINITIONS**
IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**
Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**
0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa): 50, 100, 150, 200 | FF/m | FF/m, FF/m CJ+J: 10, 20, 30; TCR, RQD (%): 25, 50, 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa): 20, 40, 60, 80

Depth (ft) marks: 1000, 1025, 1050, 1075, 1100, 1125
Depth (m) mark: 325

Stratigraphy (top to bottom): pG, fG, pG, fG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 0.7 | 21 | 73 | 96 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 20 | 72 | 94 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 9 | 56 | 66 |
| 1.3 | 9 | 56 | 66 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 3.3 | 8 | 44 | 46 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 12 | 56 | 66 |
| 0.7 | 21 | 73 | 96 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 21 | 73 | 96 |
| 0.7 | 26 | 78 | 105 |
| 0.7 | 22 | 74 | 98 |
| 2.7 | 9 | 51 | 58 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 9 | 56 | 66 |
| 0.7 | 30 | 82 | 112 |
| 1.3 | 14 | 58 | 70 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 21 | 73 | 96 |
| 0.7 | 6 | 58 | 70 |
| 1.3 | 13 | 57 | 68 |
| 0.7 | 26 | 78 | 105 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-02-24 **TO** 2009-03-01
**DIP:** -61.00 **AZIMUTH:** 213.00
**COORDINATES:** 6944390.77N 384933.12E **DATUM:** Nad83

**BOREHOLE:** C09-03

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC420
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20 | 21 - 40 | 41 - 60 | 61 - 80 | 81 - 100

| DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL | STRATIGRAPHY: DESCRIPTION | Alpha | Major Structures | IRS (est.) / UCS / PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m / FF/m CJ+J 10 20 30; TCR / RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | pG | | | | 0.7 | | 22 | 74 | 98 |
| | | | | | | | 0.7 | | 21 | 73 | 96 |
| | | | | | | | 0.7 | | 22 | 74 | 98 |
| | | | | | | | 0.7 | | 22 | 74 | 98 |
| | | | | | | | 0.7 | | 24 | 76 | 102 |
| 1175 | | | | | | | 0.7 | | 22 | 74 | 98 |
| | | | | | | | 0.7 | | 22 | 74 | 98 |
| | | | | | | | 0.7 | | 24 | 76 | 102 |
| | | | | | | | 0.7 | | 26 | 78 | 105 |
| | | | | | | | 0.7 | | 22 | 74 | 98 |
| 1200 | | | | | | | 2.7 | | 22 | 64 | 80 |
| | | | | | | | 0.7 | | 22 | 74 | 98 |
| | | | fG | | | | 0.7 | | 21 | 73 | 96 |
| | | | pG | | | | 0.7 | | 7 | 59 | 72 |
| | | | fG | | | | 0.7 | | 22 | 74 | 98 |
| 1225 | | | | | | | 0.7 | | 22 | 74 | 98 |
| | 375 | | | | | | 0.7 | | 21 | 73 | 96 |
| | | | | | | | 2.0 | | 7 | 48 | 52 |
| 1250 | | | | | | | | | | | |
| 1275 | | | | | | | | | | | |
| 1300 | | | | | | | | | | | |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-10 **TO** 2009-10-11
**DIP:** -60.00 **AZIMUTH:** 245.00
**COORDINATES:** 6943813.04N 384955.72E **DATUM:** Nad83

**BOREHOLE:** C09-04
**PAGE:** 1 **OF** 4
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC427
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

Columns: DEPTH - ft; DEPTH - m; STRATIGRAPHY (SYMBOL, DESCRIPTION); Alpha; Major Structures; IRS (est.), UCS, PLT (CF=23) (MPa) [50, 100, 150, 200]; FF/m; FF/m, FF/m CJ+J [10, 20, 30]; TCR, RQD (%) [25, 50, 75]; Joint Condition Rating (40); RMR 90; RMS (MPa) [20, 40, 60, 80]

Stratigraphy (top down): OVB; eG; fG; eG; fG; eG; fG

Depth markers: 25, 50, 75, 100, 125, 150 ft; 25 m

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 5.3 | 14 | 36 | 14 |
| 6.7 | 24 | 47 | 29 |
| 2.7 | 20 | 49 | 17 |
| 11.3 | 21 | 38 | 22 |
| 5.3 | 24 | 48 | 30 |
| 5.3 | 14 | 35 | 11 |
| 6.0 | 21 | 45 | 28 |
| 3.3 | 20 | 46 | 19 |
| 5.3 | 9 | 26 | 6 |
| 0.7 | 22 | 61 | 21 |
| 0.7 | 22 | 60 | 15 |
| 4.7 | 15 | 38 | 15 |
| 4.7 | 13 | 40 | 29 |
| 10.0 | 10 | 37 | 26 |
| 4.0 | 16 | 42 | 29 |
| 4.7 | 21 | 55 | 55 |
| 4.7 | 19 | 50 | 48 |
| 2.7 | 20 | 51 | 43 |
| 2.0 | 16 | 53 | 52 |
| 2.0 | 17 | 58 | 70 |
| 4.0 | 21 | 60 | 74 |
| 2.7 | 16 | 58 | 70 |
| 13.3 | 8 | 33 | 23 |
| 3.3 | 16 | 54 | 63 |
| 2.0 | 21 | 65 | 82 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-10 **TO** 2009-10-11
**DIP:** -60.00 **AZIMUTH:** 245.00
**COORDINATES:** 6943813.04N 384955.72E **DATUM:** Nad83

**BOREHOLE:** C09-04
**PAGE:** 2 **OF** 4
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC427
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.) / UCS / PLT (CF=23) (MPa): 50, 100, 150, 200 | FF/m | FF/m, FF/m CJ+J: 10, 20, 30 / TCR, RQD (%): 25, 50, 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa): 20, 40, 60, 80

Depth (ft): 175, 200, 225, 250, 275, 300, 325
Depth (m): 75

Stratigraphy (top to bottom): Peg, fG, pG, fG, pG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 4.7 | 16 | 51 | 58 |
| 4.0 | 13 | 50 | 56 |
| 2.7 | 19 | 58 | 70 |
| 4.7 | 26 | 64 | 80 |
| 2.7 | 15 | 54 | 63 |
| 3.3 | 18 | 56 | 66 |
| 3.3 | 16 | 52 | 60 |
| 4.7 | 16 | 51 | 58 |
| 2.0 | 11 | 52 | 60 |
| 2.7 | 9 | 48 | 52 |
| 0.7 | 8 | 60 | 74 |
| 4.0 | 11 | 45 | 47 |
| 1.3 | 11 | 55 | 65 |
| 5.3 | 21 | 53 | 61 |
| 3.3 | 15 | 51 | 58 |
| 1.3 | 13 | 57 | 68 |
| 4.0 | 16 | 50 | 56 |
| 2.7 | 20 | 59 | 72 |
| 2.7 | 22 | 61 | 75 |
| 6.0 | 20 | 52 | 60 |
| 2.7 | 19 | 56 | 66 |
| 2.7 | 21 | 63 | 79 |
| 2.7 | 22 | 59 | 72 |
| 3.3 | 11 | 49 | 54 |
| 4.0 | 7 | 44 | 46 |
| 0.7 | 22 | 74 | 98 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 17 | 61 | 75 |
| 2.0 | 9 | 50 | 56 |
| 2.7 | 22 | 64 | 80 |
| 2.0 | 17 | 58 | 70 |
| 3.3 | 13 | 47 | 44 |
| 1.3 | 21 | 68 | 88 |
| 2.0 | 16 | 60 | 74 |

# GEOTECHNICAL CORE LOG

SRK Consulting — Engineers and Scientists

| | |
|---|---|
| **PROJECT:** Minto PFS Geotechnical Evaluation | **BOREHOLE:** C09-04 |
| **LOCATION:** Yukon Territory, Canada | **PAGE:** 3 **OF** 4 |
| **SITE & PROJECT No:** MINTO (2CM022.006) | **DRILL TYPE:** |
| **BORING DATE:** 2009-03-10 **TO** 2009-10-11 | **DRILL:** |
| **DIP:** -60.00 **AZIMUTH:** 245.00 | **CLIENT HOLE ID:** 09SWC427 |
| **COORDINATES:** 6943813.04N 384955.72E **DATUM:** Nad83 | **PLAN No:** |

**DEFINITIONS**

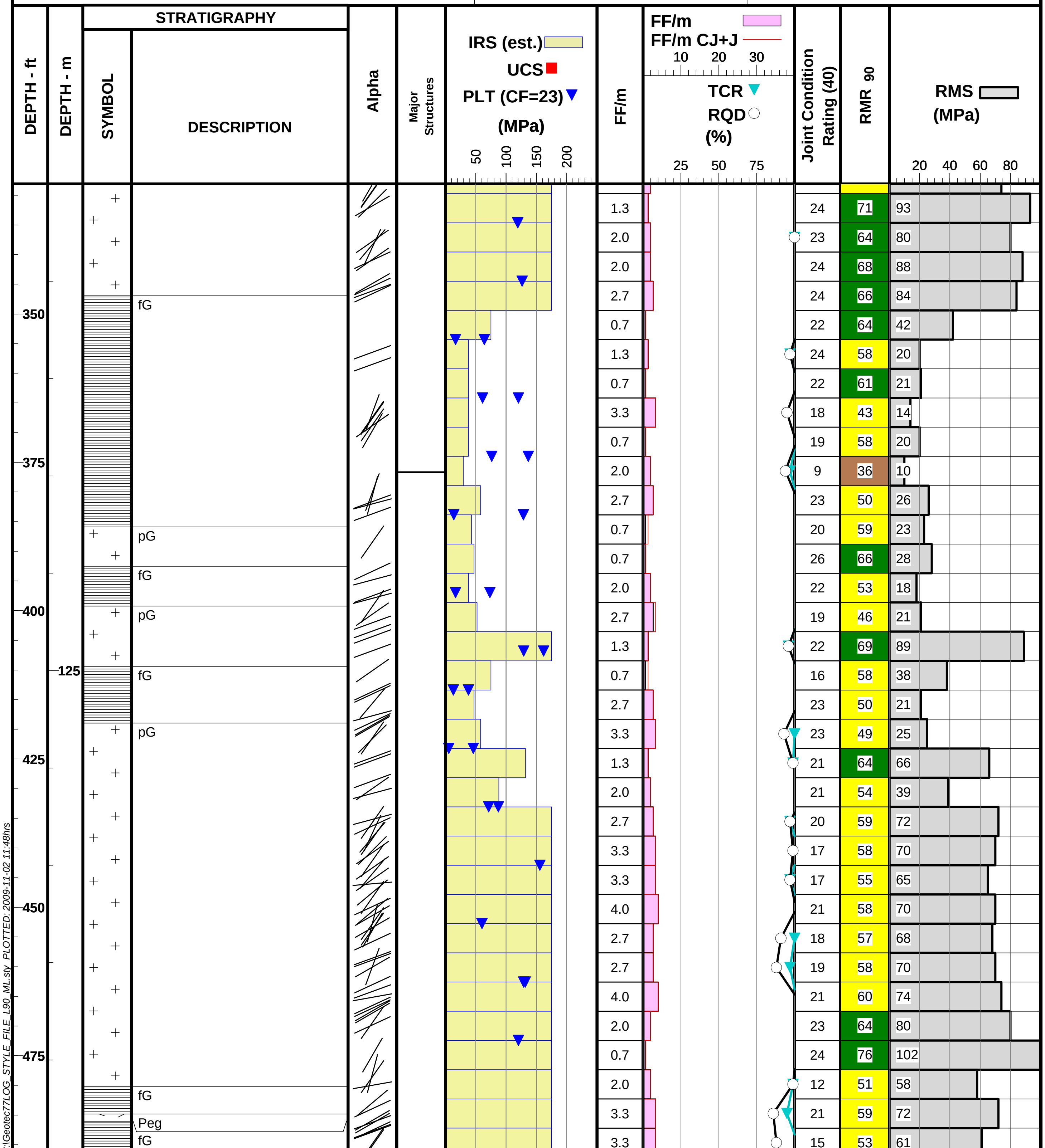
IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100




| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 1.3 | 24 | 71 | 93 |
| 2.0 | 23 | 64 | 80 |
| 2.0 | 24 | 68 | 88 |
| 2.7 | 24 | 66 | 84 |
| 0.7 | 22 | 64 | 42 |
| 1.3 | 24 | 58 | 20 |
| 0.7 | 22 | 61 | 21 |
| 3.3 | 18 | 43 | 14 |
| 0.7 | 19 | 58 | 20 |
| 2.0 | 9 | 36 | 10 |
| 2.7 | 23 | 50 | 26 |
| 0.7 | 20 | 59 | 23 |
| 0.7 | 26 | 66 | 28 |
| 2.0 | 22 | 53 | 18 |
| 2.7 | 19 | 46 | 21 |
| 1.3 | 22 | 69 | 89 |
| 0.7 | 16 | 58 | 38 |
| 2.7 | 23 | 50 | 21 |
| 3.3 | 23 | 49 | 25 |
| 1.3 | 21 | 64 | 66 |
| 2.0 | 21 | 54 | 39 |
| 2.7 | 20 | 59 | 72 |
| 3.3 | 17 | 58 | 70 |
| 3.3 | 17 | 55 | 65 |
| 4.0 | 21 | 58 | 70 |
| 2.7 | 18 | 57 | 68 |
| 2.7 | 19 | 58 | 70 |
| 4.0 | 21 | 60 | 74 |
| 2.0 | 23 | 64 | 80 |
| 0.7 | 24 | 76 | 102 |
| 2.0 | 12 | 51 | 58 |
| 3.3 | 21 | 59 | 72 |
| 3.3 | 15 | 53 | 61 |

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SRK Consulting
Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-10 **TO** 2009-10-11
**DIP:** -60.00 **AZIMUTH:** 245.00
**COORDINATES:** 6943813.04N 384955.72E **DATUM:** Nad83

**BOREHOLE:** C09-04
**PAGE:** 4 **OF** 4
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC427
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

- Gouge
- Sheared
- Jointed
- Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

| DEPTH - ft | DEPTH - m | SYMBOL | DESCRIPTION | Alpha | Major Structures | IRS (est.) / UCS / PLT (CF=23) (MPa) | FF/m | FF/m, FF/m CJ+J / TCR, RQD (%) | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | pG | | | | 0.7 | | 9 | 61 | 75 |
| 500 | | | fG | | | | 2.7 | | 15 | 52 | 60 |
| | | | | | | | 0.7 | | 22 | 74 | 98 |
| | | | pG | | | | 2.0 | | 11 | 52 | 60 |
| | | | | | | | 2.7 | | 8 | 47 | 51 |
| | | | | | | | 6.0 | | 14 | 46 | 49 |
| 525 | | | | | | | 4.0 | | 9 | 46 | 49 |
| | | | | | | | 3.3 | | 9 | 50 | 56 |
| | | | fG | | | | 2.0 | | 16 | 57 | 68 |
| | | | pG | | | | 2.7 | | 8 | 47 | 51 |
| | | | | | | | 6.0 | | 17 | 49 | 54 |
| 550 | | | | | | | 12.0 | | 22 | 49 | 54 |
| | | | | | | | 13.3 | | 26 | 57 | 68 |
| | | | fG | | | | 10.0 | | 6 | 35 | 28 |
| | | | pG | | | | 4.7 | | 14 | 43 | 39 |
| | | | | | | | 9.3 | | 15 | 43 | 44 |
| 575 | 175 | | | | | | 7.3 | | 8 | 38 | 35 |
| 600 | | | | | | | | | | | |
| 625 | | | | | | | | | | | |
| 650 | | | | | | | | | | | |

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SRK Consulting
Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-12 **TO** 2009-03-13
**DIP:** -59.00 **AZIMUTH:** 58.00
**COORDINATES:** 6943654.77N 384933.13E **DATUM:** Nad83

**BOREHOLE:** C09-05

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC429
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

| DEPTH - ft | DEPTH - m | SYMBOL | DESCRIPTION | Alpha | Major Structures | IRS (est.) / UCS / PLT (CF=23) (MPa) | FF/m | FF/m, FF/m CJ+J / TCR, RQD (%) | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|---|---|---|---|---|---|---|---|

Stratigraphy: Ob (0 to ~70 ft); eG (~70 to ~155 ft); fG (from ~155 ft)

Depth scale (ft): 25, 50, 75, 100, 125, 150; (m): 25

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 1.3 | 21 | 58 | 38 |
| 8.0 | 21 | 46 | 28 |
| 3.3 | 24 | 55 | 35 |
| 4.0 | 21 | 50 | 32 |
| 2.7 | 21 | 50 | 17 |
| 5.3 | 20 | 41 | 14 |
| 6.0 | 20 | 41 | 14 |
| 6.7 | 20 | 38 | 13 |
| 4.7 | 15 | 40 | 13 |
| 4.7 | 16 | 38 | 13 |
| 5.3 | 20 | 44 | 15 |
| 4.7 | 19 | 44 | 27 |
| 4.0 | 19 | 48 | 30 |
| 3.3 | 17 | 45 | 28 |
| 2.7 | 18 | 50 | 32 |
| 1.3 | 26 | 63 | 41 |
| 2.0 | 19 | 50 | 32 |
| 5.3 | 9 | 33 | 19 |
| 33.3 | 10 | 18 | 2 |

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# SRK Consulting
Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-12 **TO** 2009-03-13
**DIP:** -59.00 **AZIMUTH:** 58.00
**COORDINATES:** 6943654.77N 384933.13E **DATUM:** Nad83

**BOREHOLE:** C09-05
**PAGE:** 2 **OF** 4
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC429
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

Columns: DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth - ft: 175, 200, 225, 250, 275, 300, 325
Depth - m: 75
Description: eG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 1.3 | 10 | 42 | 16 |
| 2.7 | 8 | 41 | 32 |
| 4.7 | 8 | 31 | 15 |
| 3.3 | 16 | 44 | 27 |
| 1.3 | 16 | 53 | 34 |
| 4.7 | 11 | 34 | 20 |
| 4.0 | 8 | 47 | 51 |
| 3.3 | 11 | 47 | 51 |
| 8.0 | 16 | 45 | 39 |
| 2.0 | 21 | 65 | 82 |
| 4.0 | 15 | 49 | 54 |
| 5.3 | 10 | 38 | 32 |
| 0.7 | 8 | 54 | 44 |
| 1.3 | 16 | 60 | 74 |
| 2.7 | 14 | 53 | 61 |
| 2.0 | 7 | 38 | 24 |
| 4.0 | 9 | 48 | 52 |
| 1.3 | 9 | 53 | 61 |
| 2.0 | 9 | 50 | 56 |
| 2.0 | 8 | 49 | 54 |
| 5.3 | 14 | 48 | 52 |
| 4.7 | 13 | 48 | 52 |
| 2.7 | 10 | 47 | 51 |
| 1.3 | 10 | 57 | 68 |
| 10.0 | 10 | 43 | 44 |
| 2.0 | 7 | 39 | 19 |
| 13.3 | 12 | 43 | 44 |
| 3.3 | 13 | 49 | 54 |
| 3.3 | 7 | 45 | 47 |
| 3.3 | 7 | 45 | 47 |
| 1.3 | 14 | 50 | 34 |
| 2.0 | 8 | 52 | 60 |
| 3.3 | 9 | 47 | 51 |
| 3.3 | 10 | 51 | 58 |

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# GEOTECHNICAL CORE LOG

SRK Consulting — Engineers and Scientists

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-12 **TO** 2009-03-13
**DIP:** -59.00 **AZIMUTH:** 58.00
**COORDINATES:** 6943654.77N 384933.13E **DATUM:** Nad83

**BOREHOLE:** C09-05

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC429
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

Column headings: DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth markers: 350, 375, 400, 425, 450, 475 ft; 125 m

Stratigraphy (top to bottom): fG; pG; fG; pG; Peg; pG; fG; Peg; pG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 5.3 | 17 | 54 | 63 |
| 3.3 | 17 | 58 | 70 |
| 4.0 | 22 | 59 | 72 |
| 4.7 | 6 | 44 | 46 |
| 6.7 | 16 | 49 | 54 |
| 4.7 | 9 | 42 | 42 |
| 5.3 | 20 | 54 | 63 |
| 4.7 | 15 | 48 | 52 |
| 0.7 | 7 | 59 | 72 |
| 3.3 | 10 | 48 | 52 |
| 4.7 | 17 | 50 | 56 |
| 3.3 | 10 | 46 | 49 |
| 4.7 | 14 | 49 | 54 |
| 4.0 | 20 | 57 | 68 |
| 6.7 | 14 | 41 | 32 |
| 8.7 | 12 | 29 | 13 |
| 6.7 | 12 | 38 | 22 |
| 6.0 | 12 | 36 | 21 |
| 9.3 | 11 | 31 | 17 |
| 5.3 | 19 | 53 | 61 |
| 2.0 | 19 | 60 | 74 |
| 5.3 | 13 | 37 | 22 |
| 2.0 | 14 | 45 | 28 |
| 4.0 | 16 | 53 | 61 |
| 5.3 | 18 | 42 | 26 |
| 1.3 | 12 | 46 | 28 |
| 2.0 | 11 | 45 | 28 |
| 4.0 | 18 | 47 | 29 |
| 3.3 | 10 | 51 | 58 |
| 2.7 | 10 | 49 | 54 |
| 2.0 | 17 | 61 | 75 |
| 2.7 | 12 | 54 | 63 |
| 2.0 | 16 | 57 | 68 |

# GEOTECHNICAL CORE LOG

SRK Consulting — Engineers and Scientists

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-12 **TO** 2009-03-13
**DIP:** -59.00 **AZIMUTH:** 58.00
**COORDINATES:** 6943654.77N 384933.13E **DATUM:** Nad83

**BOREHOLE:** C09-05

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC429
**PLAN No:**

**DEFINITIONS**

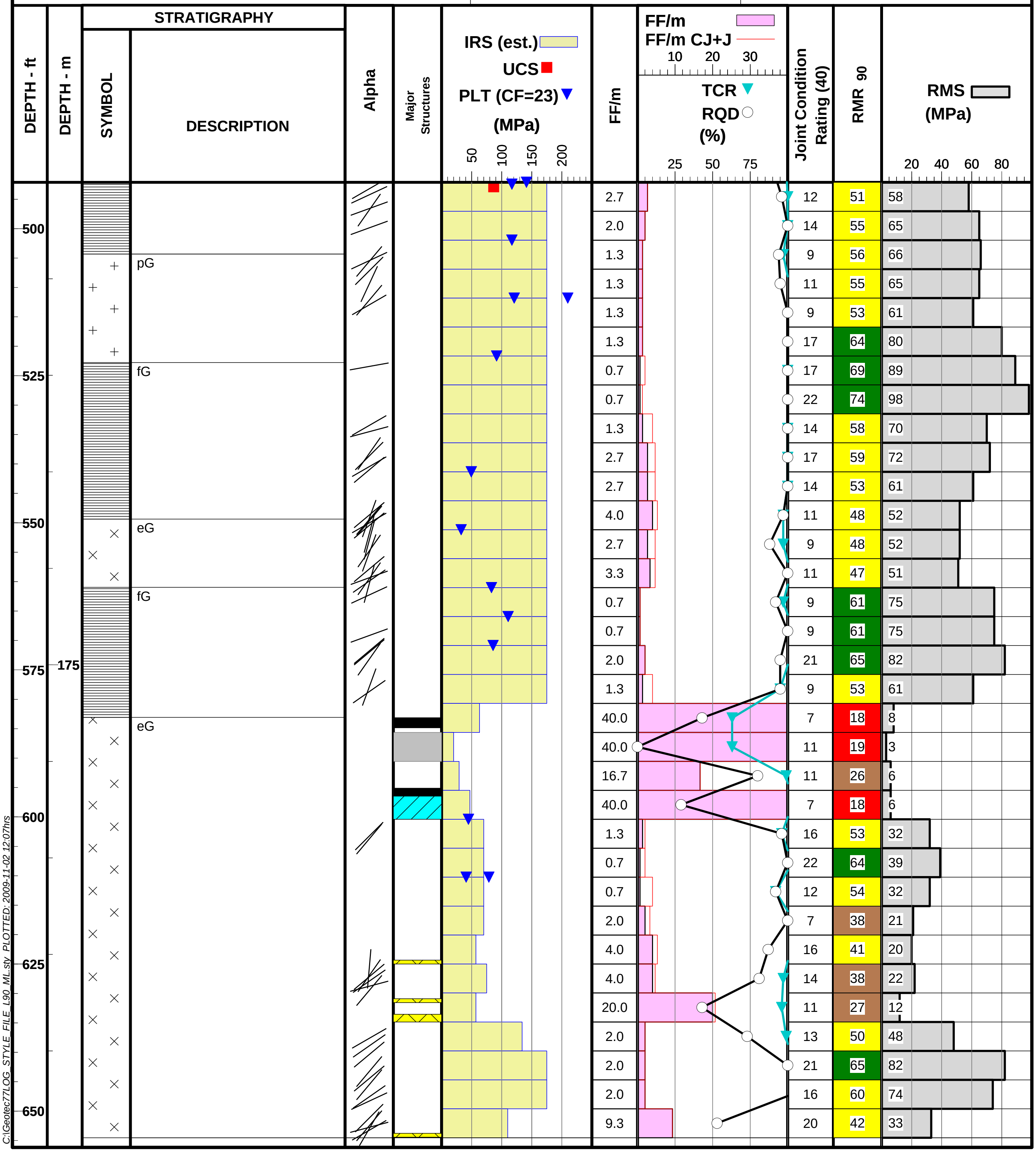
IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

- Gouge
- Sheared
- Jointed
- Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100




| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.7 | 12 | 51 | 58 |
| 2.0 | 14 | 55 | 65 |
| 1.3 | 9 | 56 | 66 |
| 1.3 | 11 | 55 | 65 |
| 1.3 | 9 | 53 | 61 |
| 1.3 | 17 | 64 | 80 |
| 0.7 | 17 | 69 | 89 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 14 | 58 | 70 |
| 2.7 | 17 | 59 | 72 |
| 2.7 | 14 | 53 | 61 |
| 4.0 | 11 | 48 | 52 |
| 2.7 | 9 | 48 | 52 |
| 3.3 | 11 | 47 | 51 |
| 0.7 | 9 | 61 | 75 |
| 0.7 | 9 | 61 | 75 |
| 2.0 | 21 | 65 | 82 |
| 1.3 | 9 | 53 | 61 |
| 40.0 | 7 | 18 | 8 |
| 40.0 | 11 | 19 | 3 |
| 16.7 | 11 | 26 | 6 |
| 40.0 | 7 | 18 | 6 |
| 1.3 | 16 | 53 | 32 |
| 0.7 | 22 | 64 | 39 |
| 0.7 | 12 | 54 | 32 |
| 2.0 | 7 | 38 | 21 |
| 4.0 | 16 | 41 | 20 |
| 4.0 | 14 | 38 | 22 |
| 20.0 | 11 | 27 | 12 |
| 2.0 | 13 | 50 | 48 |
| 2.0 | 21 | 65 | 82 |
| 2.0 | 16 | 60 | 74 |
| 9.3 | 20 | 42 | 33 |

# GEOTECHNICAL CORE LOG

SRK Consulting
Engineers and Scientists

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-14 **TO** 2009-03-14
**DIP:** -60.00 **AZIMUTH:** 238.00
**COORDINATES:** 6943632.33N 385112.67E **DATUM:** Nad83

**BOREHOLE:** C09-06
**PAGE:** 1 **OF** 3
**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC431
**PLAN No:**

**DEFINITIONS**

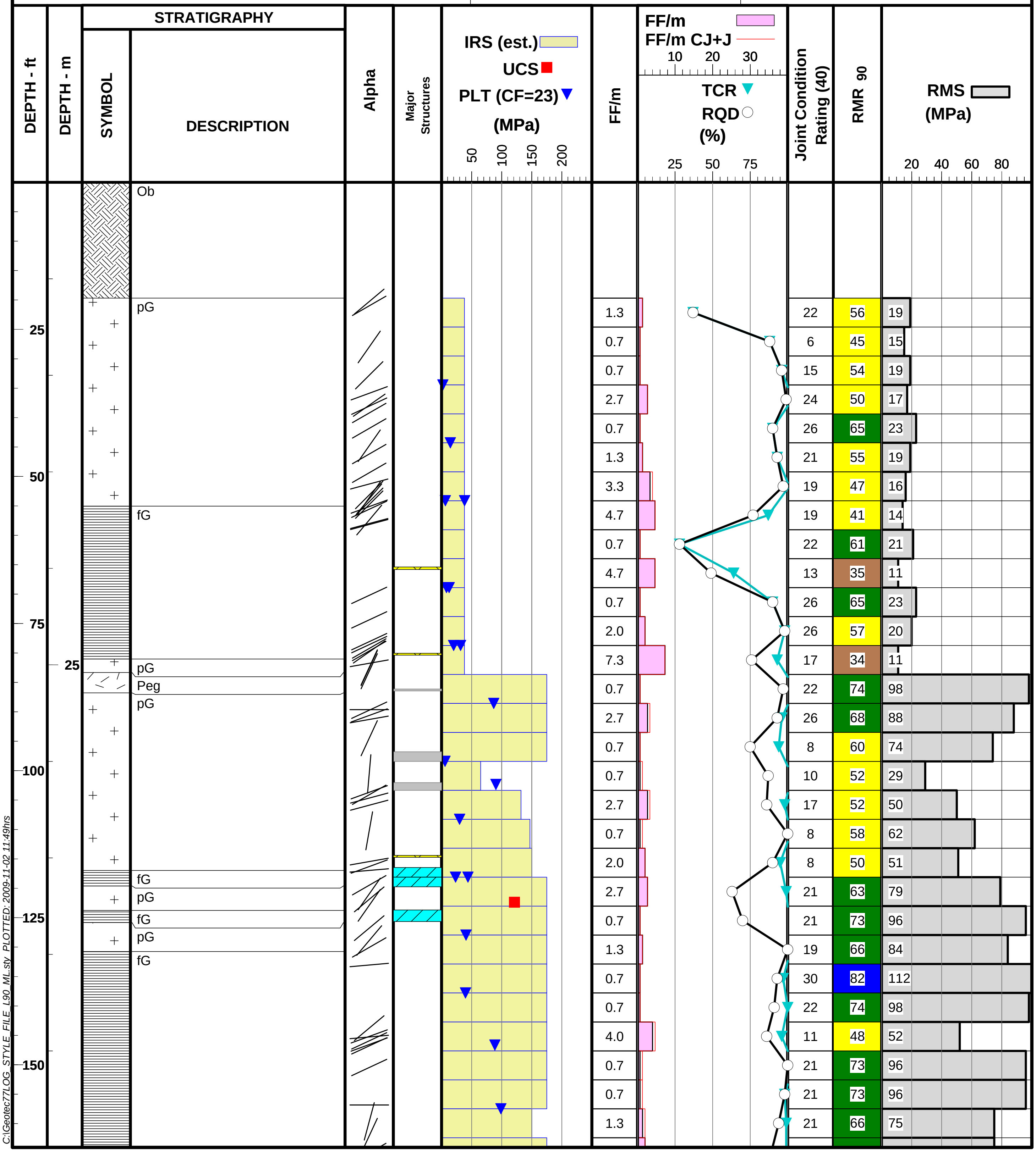
IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

- Gouge
- Sheared
- Jointed
- Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100




| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 1.3 | 22 | 56 | 19 |
| 0.7 | 6 | 45 | 15 |
| 0.7 | 15 | 54 | 19 |
| 2.7 | 24 | 50 | 17 |
| 0.7 | 26 | 65 | 23 |
| 1.3 | 21 | 55 | 19 |
| 3.3 | 19 | 47 | 16 |
| 4.7 | 19 | 41 | 14 |
| 0.7 | 22 | 61 | 21 |
| 4.7 | 13 | 35 | 11 |
| 0.7 | 26 | 65 | 23 |
| 2.0 | 26 | 57 | 20 |
| 7.3 | 17 | 34 | 11 |
| 0.7 | 22 | 74 | 98 |
| 2.7 | 26 | 68 | 88 |
| 0.7 | 8 | 60 | 74 |
| 0.7 | 10 | 52 | 29 |
| 2.7 | 17 | 52 | 50 |
| 0.7 | 8 | 58 | 62 |
| 2.0 | 8 | 50 | 51 |
| 2.7 | 21 | 63 | 79 |
| 0.7 | 21 | 73 | 96 |
| 1.3 | 19 | 66 | 84 |
| 0.7 | 30 | 82 | 112 |
| 0.7 | 22 | 74 | 98 |
| 4.0 | 11 | 48 | 52 |
| 0.7 | 21 | 73 | 96 |
| 0.7 | 21 | 73 | 96 |
| 1.3 | 21 | 66 | 75 |

SRK Consulting
Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-14 **TO** 2009-03-14
**DIP:** -60.00 **AZIMUTH:** 238.00
**COORDINATES:** 6943632.33N 385112.67E **DATUM:** Nad83

**BOREHOLE:** C09-06

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC431
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa): 50, 100, 150, 200 | FF/m | FF/m, FF/m CJ+J: 10, 20, 30; TCR, RQD (%): 25, 50, 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa): 20, 40, 60, 80

Depth - ft: 175, 200, 225, 250, 275, 300, 325
Depth - m: 75

Stratigraphy: pG (to about 300 ft); fG; pG; fG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.0 | 20 | 61 | 75 |
| 2.0 | 21 | 65 | 82 |
| 2.7 | 22 | 59 | 69 |
| 2.7 | 21 | 60 | 74 |
| 2.0 | 24 | 65 | 82 |
| 4.0 | 20 | 57 | 68 |
| 0.7 | 21 | 73 | 96 |
| 1.3 | 9 | 56 | 66 |
| 2.0 | 15 | 52 | 51 |
| 5.3 | 21 | 55 | 65 |
| 3.3 | 21 | 57 | 68 |
| 1.3 | 10 | 54 | 63 |
| 0.7 | 16 | 62 | 55 |
| 0.7 | 17 | 69 | 89 |
| 2.7 | 8 | 47 | 51 |
| 2.7 | 8 | 47 | 51 |
| 1.3 | 16 | 60 | 74 |
| 40.0 | 7 | 26 | 16 |
| 2.0 | 17 | 58 | 70 |
| 0.7 | 12 | 64 | 80 |
| 3.3 | 15 | 53 | 61 |
| 4.0 | 12 | 49 | 54 |
| 6.0 | 9 | 43 | 44 |
| 4.7 | 21 | 59 | 72 |
| 3.3 | 26 | 67 | 86 |
| 7.3 | 21 | 53 | 61 |
| 3.3 | 17 | 53 | 61 |
| 4.0 | 22 | 59 | 72 |
| 4.0 | 18 | 57 | 68 |
| 3.3 | 18 | 56 | 66 |
| 2.0 | 21 | 62 | 77 |
| 3.3 | 14 | 39 | 13 |
| 6.0 | 20 | 44 | 15 |
| 6.7 | 24 | 44 | 17 |

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Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-03-14 **TO** 2009-03-14
**DIP:** -60.00 **AZIMUTH:** 238.00
**COORDINATES:** 6943632.33N 385112.67E **DATUM:** Nad83

**BOREHOLE:** C09-06

**DRILL TYPE:**
**DRILL:**
**CLIENT HOLE ID:** 09SWC431
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

- Gouge
- Sheared
- Jointed
- Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa): 50, 100, 150, 200 | FF/m | FF/m, FF/m CJ+J: 10, 20, 30; TCR, RQD (%): 25, 50, 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa): 20, 40, 60, 80

Depth marks: 350, 375, 400, 425, 450, 475 (ft); 125 (m)

**STRATIGRAPHY (DESCRIPTION), top to bottom:** Peg, fG, Peg, fG, eG, fG, eG, fG, pG, fG, pG, fG, pG, fG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 5.3 | 20 | 39 | 13 |
| 6.7 | 20 | 41 | 18 |
| 9.3 | 22 | 40 | 24 |
| 8.7 | 21 | 42 | 26 |
| 4.7 | 18 | 44 | 22 |
| 5.3 | 15 | 37 | 22 |
| 4.7 | 21 | 43 | 14 |
| 8.0 | 20 | 39 | 13 |
| 4.0 | 21 | 45 | 15 |
| 3.3 | 21 | 49 | 18 |
| 4.0 | 19 | 46 | 28 |
| 6.7 | 20 | 43 | 26 |
| 5.3 | 21 | 48 | 30 |
| 5.3 | 22 | 44 | 27 |
| 12.7 | 19 | 35 | 20 |
| 10.7 | 21 | 38 | 22 |
| 4.7 | 22 | 45 | 28 |
| 6.0 | 20 | 44 | 27 |
| 4.0 | 19 | 46 | 28 |
| 6.0 | 19 | 46 | 28 |
| 2.0 | 20 | 54 | 34 |
| 5.3 | 20 | 42 | 26 |
| 4.7 | 21 | 55 | 52 |
| 3.3 | 21 | 62 | 77 |
| 2.7 | 21 | 63 | 79 |
| 2.7 | 21 | 63 | 79 |
| 5.3 | 17 | 51 | 58 |
| 2.7 | 21 | 63 | 79 |
| 3.3 | 21 | 59 | 72 |
| 1.3 | 6 | 53 | 61 |
| 2.0 | 21 | 62 | 77 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 7 | 54 | 63 |

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Engineers and Scientists

# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-06-26 **TO** 2009-09-27
**DIP:** -60.00 **AZIMUTH:** 196.00
**COORDINATES:** 6945925.00N 384238.00E **DATUM:** Nad83

**BOREHOLE:** C09-07
**PAGE:** 1 **OF** 4
**DRILL TYPE:**
**DRILL:** DD
**CLIENT HOLE ID:** 09SWC495
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY (SYMBOL, DESCRIPTION) | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth - ft: 25, 50, 75, 100, 125, 150
Depth - m: 25

Stratigraphy: Ob, pG, fG, pG, fG, pG, fG, pG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 4.7 | 23 | 56 | 66 |
| 4.7 | 24 | 59 | 72 |
| 8.0 | 21 | 53 | 61 |
| 5.3 | 21 | 55 | 65 |
| 4.0 | 25 | 62 | 77 |
| 4.0 | 22 | 56 | 66 |
| 4.0 | 26 | 63 | 79 |
| 4.0 | 12 | 44 | 41 |
| 2.0 | 17 | 58 | 70 |
| 6.0 | 19 | 51 | 58 |
| 4.7 | 24 | 59 | 72 |
| 6.7 | 22 | 51 | 51 |
| 5.3 | 22 | 54 | 63 |
| 8.0 | 21 | 53 | 61 |
| 6.7 | 21 | 52 | 60 |
| 5.3 | 24 | 56 | 66 |
| 8.0 | 18 | 48 | 52 |
| 4.7 | 26 | 61 | 75 |
| 4.7 | 26 | 51 | 32 |
| 4.7 | 23 | 58 | 70 |
| 8.7 | 22 | 53 | 61 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-06-26 **TO** 2009-09-27
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**COORDINATES:** 6945925.00N 384238.00E **DATUM:** Nad83

**BOREHOLE:** C09-07
**PAGE:** 2 **OF** 4
**DRILL TYPE:**
**DRILL:** DD
**CLIENT HOLE ID:** 09SWC495
**PLAN No:**

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RMR: Rock Mass Rating
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**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.) UCS PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth (ft): 175, 200, 225, 250, 275, 300, 325
Depth (m): 75

Stratigraphy (from top): fG, Ap, pG, fG, Ap, fG, pG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 7.3 | 23 | 53 | 61 |
| 5.3 | 17 | 49 | 54 |
| 4.7 | 19 | 54 | 63 |
| 5.3 | 11 | 43 | 44 |
| 5.3 | 17 | 49 | 54 |
| 6.7 | 10 | 41 | 40 |
| 4.0 | 21 | 52 | 44 |
| 5.3 | 23 | 55 | 65 |
| 6.0 | 18 | 52 | 60 |
| 2.7 | 24 | 63 | 79 |
| 6.0 | 16 | 48 | 52 |
| 6.0 | 11 | 45 | 47 |
| 2.7 | 21 | 60 | 74 |
| 26.7 | 7 | 33 | 26 |
| 4.0 | 18 | 55 | 65 |
| 4.0 | 14 | 51 | 58 |
| 5.3 | 13 | 47 | 51 |
| 2.0 | 17 | 56 | 66 |
| 6.7 | 16 | 49 | 54 |
| 4.7 | 15 | 50 | 56 |
| 3.3 | 15 | 51 | 58 |
| 4.0 | 10 | 47 | 51 |
| 3.3 | 20 | 58 | 70 |
| 0.7 | 21 | 73 | 96 |
| 4.0 | 17 | 54 | 63 |
| 2.0 | 21 | 65 | 82 |
| 4.0 | 15 | 52 | 60 |
| 0.7 | 26 | 78 | 105 |
| 4.0 | 12 | 49 | 54 |
| 4.0 | 25 | 62 | 77 |
| 2.7 | 13 | 52 | 60 |
| 3.3 | 25 | 63 | 79 |
| 4.0 | 24 | 58 | 70 |
| 2.7 | 21 | 60 | 74 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-06-26 **TO** 2009-09-27
**DIP:** -60.00 **AZIMUTH:** 196.00
**COORDINATES:** 6945925.00N 384238.00E **DATUM:** Nad83

**BOREHOLE:** C09-07
**PAGE:** 3 **OF** 4
**DRILL TYPE:**
**DRILL:** DD
**CLIENT HOLE ID:** 09SWC495
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa) 50 100 150 200 | FF/m | FF/m, FF/m CJ+J 10 20 30; TCR, RQD (%) 25 50 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa) 20 40 60 80

Depth - ft: 350, 375, 400, 425, 450, 475
Depth - m: 125

Stratigraphy: And, pG, fG, eG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.0 | 21 | 65 | 82 |
| 2.7 | 19 | 58 | 70 |
| 4.7 | 12 | 47 | 51 |
| 4.7 | 16 | 41 | 24 |
| 8.7 | 10 | 39 | 37 |
| 3.3 | 22 | 53 | 34 |
| 5.3 | 9 | 28 | 4 |
| 5.3 | 10 | 29 | 6 |
| 4.7 | 11 | 29 | 6 |
| 5.3 | 10 | 32 | 12 |
| 32.0 | 7 | 20 | 6 |
| 5.3 | 14 | 46 | 49 |
| 5.3 | 17 | 43 | 34 |
| 5.3 | 15 | 41 | 32 |
| 5.3 | 15 | 46 | 37 |
| 7.3 | 8 | 38 | 35 |
| 4.7 | 15 | 53 | 61 |
| 6.0 | 9 | 37 | 31 |
| 3.3 | 28 | 66 | 84 |
| 4.7 | 16 | 51 | 58 |
| 6.7 | 10 | 39 | 33 |
| 4.0 | 17 | 49 | 53 |
| 4.0 | 13 | 50 | 54 |
| 4.0 | 7 | 44 | 46 |
| 2.0 | 9 | 53 | 61 |
| 4.0 | 21 | 58 | 70 |
| 4.7 | 20 | 51 | 51 |
| 3.3 | 15 | 49 | 49 |
| 5.3 | 10 | 37 | 22 |
| 4.0 | 13 | 50 | 56 |
| 1.3 | 13 | 60 | 74 |
| 4.0 | 13 | 50 | 56 |
| 1.3 | 9 | 56 | 66 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-06-26 **TO** 2009-09-27
**DIP:** -60.00 **AZIMUTH:** 196.00
**COORDINATES:** 6945925.00N 384238.00E **DATUM:** Nad83

**BOREHOLE:** C09-07

**DRILL TYPE:**
**DRILL:** DD
**CLIENT HOLE ID:** 09SWC495
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

| DEPTH - ft | DEPTH - m | SYMBOL | DESCRIPTION | Alpha | Major Structures | IRS (est.) / UCS / PLT (CF=23) (MPa) | FF/m | FF/m, FF/m CJ+J / TCR, RQD (%) | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Ap | | | | 2.7 | | 12 | 54 | 63 |
| 500 | | | fG | | | | 5.3 | | 19 | 53 | 61 |
| 525 | | | | | | | | | | | |
| 550 | | | | | | | | | | | |
| 575 | 175 | | | | | | | | | | |
| 600 | | | | | | | | | | | |
| 625 | | | | | | | | | | | |
| 650 | | | | | | | | | | | |

IRS scale: 50, 100, 150, 200
FF/m scale: 10, 20, 30
TCR/RQD scale: 25, 50, 75
RMS scale: 20, 40, 60, 80

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-06-27 **TO** 2009-06-27
**DIP:** -55.00 **AZIMUTH:** 47.00
**COORDINATES:** 6945953.00N 384320.00E **DATUM:** Nad83

**BOREHOLE:** C09-08

**DRILL TYPE:**
**DRILL:** DD
**CLIENT HOLE ID:** 09SWC497
**PLAN No:**

**DEFINITIONS**
IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**
Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**
0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

**DEPTH - ft** | **DEPTH - m** | **STRATIGRAPHY: SYMBOL, DESCRIPTION** | **Alpha** | **Major Structures** | **IRS (est.)**, **UCS**, **PLT (CF=23)** **(MPa)** 50 100 150 200 | **FF/m** | **FF/m**, **FF/m CJ+J** 10 20 30; **TCR**, **RQD** **(%)** 25 50 75 | **Joint Condition Rating (40)** | **RMR 90** | **RMS (MPa)** 20 40 60 80

Depth - ft: 25, 50, 75, 100, 125, 150
Depth - m: 25

Stratigraphy: Ob; pG; And; pG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 0.7 | 23 | 65 | 43 |
| 4.7 | 14 | 41 | 32 |
| 2.0 | 11 | 40 | 16 |
| 4.7 | 12 | 39 | 30 |
| 7.3 | 11 | 39 | 34 |
| 6.0 | 14 | 46 | 49 |
| 10.7 | 19 | 36 | 21 |
| 6.0 | 20 | 42 | 26 |
| 5.3 | 10 | 34 | 20 |
| 8.0 | 21 | 51 | 58 |
| 7.3 | 24 | 56 | 66 |
| 8.0 | 13 | 43 | 44 |
| 6.0 | 21 | 53 | 61 |
| 6.7 | 25 | 48 | 37 |
| 6.0 | 26 | 48 | 19 |
| 6.0 | 17 | 43 | 32 |
| 9.3 | 21 | 45 | 39 |
| 10.0 | 22 | 48 | 47 |
| 6.0 | 23 | 57 | 68 |
| 8.0 | 21 | 53 | 61 |
| 4.7 | 21 | 59 | 72 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-06-27 **TO** 2009-06-27
**DIP:** -55.00 **AZIMUTH:** 47.00
**COORDINATES:** 6945953.00N 384320.00E **DATUM:** Nad83

**BOREHOLE:** C09-08
**PAGE:** 2 **OF** 3
**DRILL TYPE:**
**DRILL:** DD
**CLIENT HOLE ID:** 09SWC497
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa): 50, 100, 150, 200 | FF/m | FF/m, FF/m CJ+J: 10, 20, 30; TCR, RQD (%): 25, 50, 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa): 20, 40, 60, 80

Depth (ft): 175, 200, 225, 250, 275, 300, 325
Depth (m): 75

Stratigraphy (top to bottom): Peg, pG, fG, Peg, fG, pG, Peg, pG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 6.7 | 17 | 50 | 56 |
| 5.3 | 19 | 53 | 61 |
| 4.0 | 7 | 44 | 46 |
| 3.3 | 12 | 50 | 56 |
| 5.3 | 10 | 42 | 42 |
| 3.3 | 12 | 46 | 43 |
| 3.3 | 13 | 51 | 58 |
| 4.0 | 14 | 51 | 58 |
| 6.0 | 23 | 57 | 68 |
| 3.3 | 9 | 50 | 56 |
| 5.3 | 12 | 44 | 41 |
| 3.3 | 9 | 41 | 32 |
| 6.0 | 12 | 46 | 49 |
| 6.0 | 7 | 39 | 37 |
| 6.7 | 9 | 38 | 35 |
| 7.3 | 10 | 34 | 26 |
| 3.3 | 8 | 42 | 38 |
| 4.0 | 11 | 37 | 25 |
| 6.7 | 11 | 36 | 25 |
| 2.7 | 10 | 47 | 51 |
| 2.0 | 12 | 56 | 66 |
| 4.7 | 6 | 41 | 40 |
| 4.7 | 9 | 47 | 51 |
| 2.0 | 12 | 53 | 61 |
| 1.3 | 15 | 59 | 72 |
| 2.0 | 21 | 65 | 82 |
| 2.0 | 22 | 63 | 79 |
| 2.7 | 18 | 57 | 68 |
| 0.7 | 22 | 74 | 98 |
| 2.7 | 9 | 48 | 52 |
| 2.7 | 6 | 45 | 47 |
| 4.7 | 7 | 42 | 42 |
| 2.7 | 6 | 45 | 47 |
| 0.7 | 30 | 82 | 112 |

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# GEOTECHNICAL CORE LOG

**PROJECT:** Minto PFS Geotechnical Evaluation
**LOCATION:** Yukon Territory, Canada
**SITE & PROJECT No:** MINTO (2CM022.006)
**BORING DATE:** 2009-06-27 **TO** 2009-06-27
**DIP:** -55.00 **AZIMUTH:** 47.00
**COORDINATES:** 6945953.00N 384320.00E **DATUM:** Nad83

**BOREHOLE:** C09-08
**PAGE:** 3 **OF** 3
**DRILL TYPE:**
**DRILL:** DD
**CLIENT HOLE ID:** 09SWC497
**PLAN No:**

**DEFINITIONS**

IRS: Intact Rock Strength (subjective)
UCS: Uniaxial Compressive Strength (MPa)
Pt LOAD: Point Load Test (MPa)
FF/m: Fracture Frequency per m
TCR: Total Core Recovery
RQD: Rock Quality Designation
RMR: Rock Mass Rating
RMS: Rock Mass Strength

**LEGEND OF MAJOR STRUCTURES**

Gouge
Sheared
Jointed
Broken

**LEGEND OF RMR (90)**

0 - 20, 21 - 40, 41 - 60, 61 - 80, 81 - 100

DEPTH - ft | DEPTH - m | STRATIGRAPHY: SYMBOL, DESCRIPTION | Alpha | Major Structures | IRS (est.), UCS, PLT (CF=23) (MPa): 50, 100, 150, 200 | FF/m | FF/m, FF/m CJ+J: 10, 20, 30; TCR, RQD (%): 25, 50, 75 | Joint Condition Rating (40) | RMR 90 | RMS (MPa): 20, 40, 60, 80

Depth - ft: 350, 375, 400, 425, 450, 475
Depth - m: 125

Stratigraphy: PEG, PG, PEG, PG

| FF/m | Joint Condition Rating (40) | RMR 90 | RMS (MPa) |
|---|---|---|---|
| 2.0 | 7 | 48 | 52 |
| 0.7 | 9 | 61 | 75 |
| 0.7 | 22 | 74 | 98 |
| 1.3 | 15 | 57 | 62 |
| 2.0 | 9 | 51 | 56 |
| 4.0 | 8 | 47 | 51 |
| 4.0 | 10 | 47 | 51 |
| 1.3 | 12 | 46 | 32 |
| 2.7 | 9 | 40 | 31 |
| 4.7 | 10 | 45 | 47 |
| 8.7 | 14 | 35 | 24 |
| 8.0 | 20 | 50 | 56 |
| 8.0 | 14 | 38 | 29 |
| 6.0 | 20 | 52 | 60 |
| 4.0 | 12 | 51 | 58 |
| 2.7 | 12 | 54 | 63 |
| 4.0 | 13 | 50 | 56 |
| 4.7 | 22 | 55 | 65 |
| 2.7 | 17 | 54 | 63 |
| 2.7 | 20 | 59 | 72 |
| 1.3 | 13 | 60 | 74 |
| 1.3 | 21 | 68 | 88 |
| 2.0 | 10 | 54 | 63 |
| 0.7 | 21 | 73 | 96 |
| 2.7 | 26 | 65 | 82 |
| 1.3 | 28 | 72 | 94 |
| 0.7 | 21 | 73 | 96 |

# Appendix B: Laboratory Testing

# Uniaxial Compressive Strength Testing

# *University of Arizona*

*GEOMECHANICAL LABORATORY*

*TUCSON, ARIZONA USA*

## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **01-003U** |
| Rock Type | GRANIT E |
| Density : | 167.1 (pcf) |
| | 2,677.0 (kg/m$^3$) |

Sample # **01-003U**

| | | |
|---|---|---|
| Fail Stress | **17,336** | psi |
| | 119.56 | Mpa |
| Modulus | | psi |
| Poisson's | | |

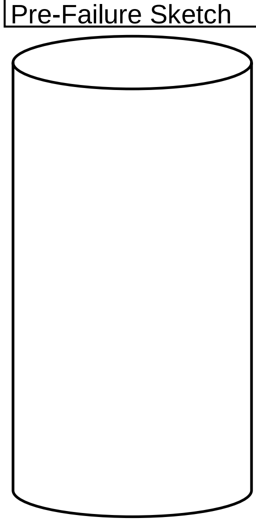
| **Sample Data :** | |
|---|---|
| Sample # : | **01-003U** |
| Rock Type: | GRANIT E |
| Hole # : | C09-01 |
| Depth : | 89.5 |
| Alterations: | |
| Diameter : | 1.775 (in) |
| Height : | 3.834 (in) |
| Weight : | 416.25 (gm) |
| Area : | 2.475 (in$^2$) |
| Volume : | 9.488 (in$^3$) |

| | |
|---|---|
| Fail Stress | 17,336 (psi) |
| | 119.6 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 42,900 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

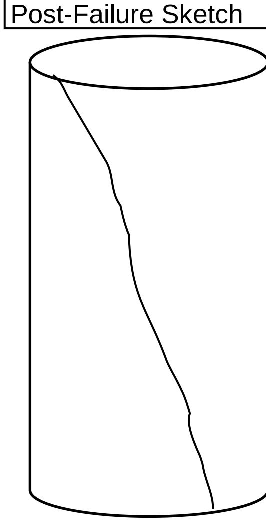
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.779 | Ht. 1 | 3.834 |
| Dia. 2 | 1.777 | Ht. 2 | 3.835 |
| Dia. 3 | 1.773 | Ht. 3 | 3.835 |
| Dia. 4 | 1.772 | Ht. 4 | 3.835 |
| Dia. 5 | 1.773 | Weight (gm) | 416.25 |
| Dia. 6 | 1.777 | Sample # | **01-003U** |

**Fail Load** 42900 lbs

# *University of Arizona*

*GEOMECHANICAL LABORATORY*

*TUCSON, ARIZONA USA*

| | | Uniaxial Compression Test Results | | |
|---|---|---|---|---|
| Project # | 2CM022 | | Client | SRK |
| Date | 5/4/2009 | | Location | |
| Technician | D.Streeter | | Sample # | **01-007U** |
| | | | Rock Type | GRANITE |
| | | | Density : | 167.7 (pcf) |
| | | | | 2,685.7 (kg/m$^3$) |

| | | |
|---|---|---|
| Sample # | **01-007U** | |
| Fail Stress | **21,807** | psi |
| | 150.39 | Mpa |
| Modulus | | psi |
| Poisson's | | |

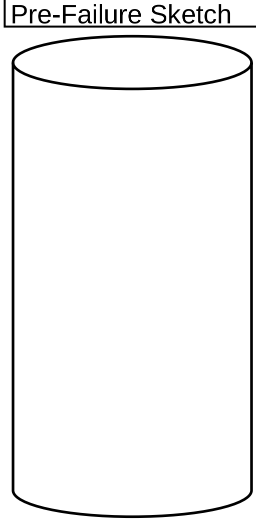
| **Sample Data :** | |
|---|---|
| Sample # : | **01-007U** |
| Rock Type: | GRANITE |
| Hole # : | C09-01 |
| Depth : | 187 |
| Alterations: | |
| Diameter : | 1.772 (in) |
| Height : | 3.878 (in) |
| Weight : | 421.13 (gm) |
| Area : | 2.467 (in$^2$) |
| Volume : | 9.569 (in$^3$) |

| | |
|---|---|
| Fail Stress | 21,807 (psi) |
| | 150.4 Mpa |

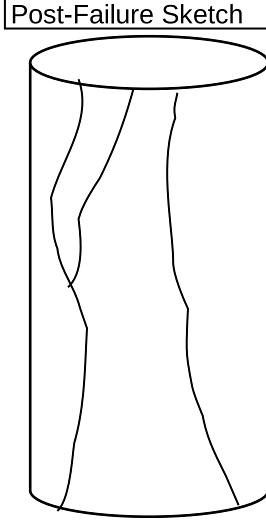
| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 53,800 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

Pre-Failure Sketch



Post-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

| | | | |
|---|---|---|---|
| Dia. 1 | 1.771 | Ht. 1 | 3.877 |
| Dia. 2 | 1.773 | Ht. 2 | 3.875 |
| Dia. 3 | 1.771 | Ht. 3 | 3.883 |
| Dia. 4 | 1.771 | Ht. 4 | 3.879 |
| Dia. 5 | 1.771 | Weight (gm) | 421.13 |
| Dia. 6 | 1.779 | Sample # | **01-007U** |

**Fail Load** 53800 lbs

# *University of Arizona*

*GEOMECHANICAL LABORATORY*

*TUCSON, ARIZONA USA*

## Uniaxial Compression Test Results

| | | | |
|---|---|---|---|
| Project # | 2CM022 | Client | SRK |
| Date | 5/4/2009 | Location | |
| Technician | D.Streeter | Sample # | **01-010U** |
| | | Rock Type | GRANITE |
| | | Density : | 167.5 (pcf) |
| | | | 2,683.3 (kg/m$^3$) |

Sample # **01-010U**

Fail Stress **22,634** psi

156.10 Mpa

Modulus psi

Poisson's

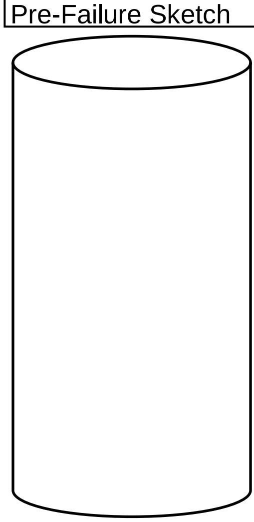
| Sample Data : | |
|---|---|
| Sample # : | **01-010U** |
| Rock Type: | GRANITE |
| Hole # : | C09-01 |
| Depth : | 293.16 |
| Alterations: | |
| Diameter : | 1.780 (in) |
| Height : | 3.876 (in) |
| Weight : | 423.95 (gm) |
| Area : | 2.487 (in$^2$) |
| Volume : | 9.641 (in$^3$) |

| | |
|---|---|
| Fail Stress | 22,634 (psi) |
| | 156.1 Mpa |

| Test Data: | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 56,300 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

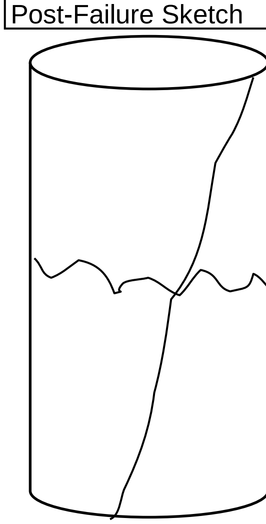
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.785 | Ht. 1 | 3.876 |
| Dia. 2 | 1.775 | Ht. 2 | 3.877 |
| Dia. 3 | 1.779 | Ht. 3 | 3.877 |
| Dia. 4 | 1.782 | Ht. 4 | 3.875 |
| Dia. 5 | 1.777 | Weight (gm) | 423.95 |
| Dia. 6 | 1.780 | Sample # | **01-010U** |

**Fail Load** 56300 lbs

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Uniaxial Compression Test Results | Client | SRK |
| Date | 5/4/2009 | | Location | |
| Technician | D.Streeter | | Sample # | **02-006U** |
| | | | Rock Type | GRANITE |
| | | | Density : | 169.3 (pcf) |
| | | | | 2,711.4 ($kg/m^3$) |

| | | |
|---|---|---|
| Sample # | **02-006U** | |
| Fail Stress | **18,603** | psi |
| | 128.30 | Mpa |
| Modulus | | psi |
| Poisson's | | |

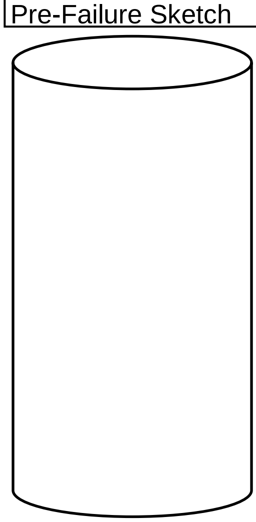
| **Sample Data :** | |
|---|---|
| Sample # : | **02-006U** |
| Rock Type: | GRANITE |
| Hole # : | C09-02 |
| Depth : | 179.54 |
| Alterations: | |
| Diameter : | 1.774 (in) |
| Height : | 3.855 (in) |
| Weight : | 423.52 (gm) |
| Area : | 2.473 ($in^2$) |
| Volume : | 9.532 ($in^3$) |

| | |
|---|---|
| Fail Stress | 18,603 (psi) |
| | 128.3 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 46,000 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

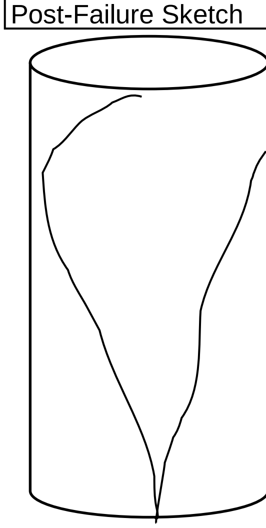
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.774 | Ht. 1 | 3.855 |
| Dia. 2 | 1.774 | Ht. 2 | 3.855 |
| Dia. 3 | 1.775 | Ht. 3 | 3.856 |
| Dia. 4 | 1.777 | Ht. 4 | 3.855 |
| Dia. 5 | 1.773 | Weight (gm) | 423.52 |
| Dia. 6 | 1.774 | Sample # | **02-006U** |

| | | |
|---|---|---|
| **Fail Load** | 46000 | lbs |

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## Uniaxial Compression Test Results

| | | | |
|---|---|---|---|
| Project # | 2CM022 | Client | SRK |
| Date | 5/4/2009 | Location | |
| Technician | D.Streeter | Sample # | **02-009U** |
| | | Rock Type | GRANITE |
| | | Density : | 166.8 (pcf) |
| | | | 2,671.7 (kg/m$^3$) |

| | | |
|---|---|---|
| Sample # | **02-009U** | |
| Fail Stress | **21,731** | psi |
| | 149.87 | Mpa |
| Modulus | | psi |
| Poisson's | | |

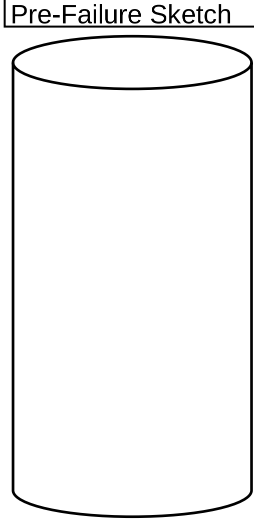
| **Sample Data :** | |
|---|---|
| Sample # : | **02-009U** |
| Rock Type: | GRANITE |
| Hole # : | C09-02 |
| Depth : | 271.9 |
| Alterations: | |
| Diameter : | 1.775 (in) |
| Height : | 3.768 (in) |
| Weight : | 408.47 (gm) |
| Area : | 2.476 (in$^2$) |
| Volume : | 9.330 (in$^3$) |

| | |
|---|---|
| Fail Stress | 21,731 (psi) |
| | 149.9 Mpa |

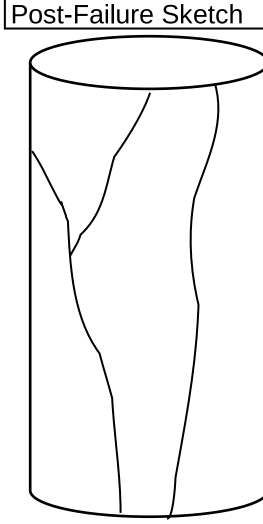
| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 53,800 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

### Worksheet

Pre-Failure Sketch



Post-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

| | | | |
|---|---|---|---|
| Dia. 1 | 1.774 | Ht. 1 | 3.768 |
| Dia. 2 | 1.772 | Ht. 2 | 3.771 |
| Dia. 3 | 1.777 | Ht. 3 | 3.768 |
| Dia. 4 | 1.783 | Ht. 4 | 3.768 |
| Dia. 5 | 1.776 | Weight (gm) | 408.47 |
| Dia. 6 | 1.771 | Sample # | **02-009U** |

**Fail Load** 53800 lbs

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Uniaxial Compression Test Results | Client | SRK |
| Date | 5/4/2009 | | Location | |
| Technician | D.Streeter | | Sample # | **03-003U** |
| | | Sample # **03-003U** | Rock Type | PK GRANITE |
| | | | Density : | 158.5 (pcf) |
| | | | | 2,539.1 (kg/$m^3$) |

| | | |
|---|---|---|
| Sample # | **03-003U** | |
| Fail Stress | **10,483** | psi |
| | 72.30 | Mpa |
| Modulus | | psi |
| Poisson's | | |

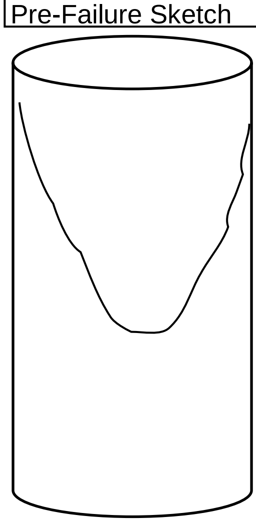
| | |
|---|---|
| Fail Stress | 10,483 (psi) |
| | 72.3 Mpa |

| **Sample Data :** | |
|---|---|
| Sample # : | **03-003U** |
| Rock Type: | PK GRANITE |
| Hole # : | C09-03 |
| Depth : | 77.33 |
| Alterations: | |
| Diameter : | 1.773 (in) |
| Height : | 3.876 (in) |
| Weight : | 398.03 (gm) |
| Area : | 2.468 ($in^2$) |
| Volume : | 9.566 ($in^3$) |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 25,870 (lbs) |
| Mode of Failure | Fracture |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

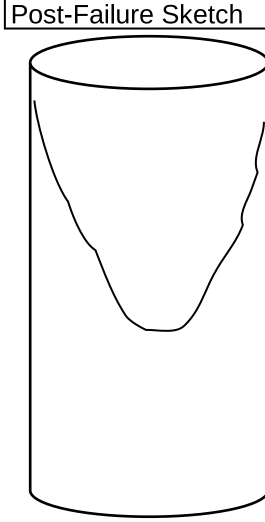
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | XX |
| Intact | |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.772 | Ht. 1 | 3.785 |
| Dia. 2 | 1.772 | Ht. 2 | 4.160 |
| Dia. 3 | 1.772 | Ht. 3 | 3.788 |
| Dia. 4 | 1.775 | Ht. 4 | 3.773 |
| Dia. 5 | 1.773 | Weight (gm) | 398.03 |
| Dia. 6 | 1.772 | Sample # | **03-003U** |

| | | |
|---|---|---|
| **Fail Load** | 25870 | lbs |

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## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **03-006U** |
| Rock Type | GRANITE |
| Density : | 164.9 (pcf) |
| | 2,641.0 (kg/$m^3$) |

Sample # **03-006U**

Fail Stress **9,574** psi

66.03 Mpa

Modulus psi

Poisson's

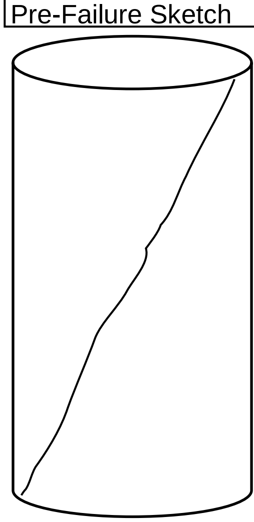
| **Sample Data :** | |
|---|---|
| Sample # : | **03-006U** |
| Rock Type: | GRANITE |
| Hole # : | C09-03 |
| Depth : | 130.84 |
| Alterations: | |
| Diameter : | 1.772 (in) |
| Height : | 3.833 (in) |
| Weight : | 408.90 (gm) |
| Area : | 2.465 ($in^2$) |
| Volume : | 9.448 ($in^3$) |

| | |
|---|---|
| Fail Stress | 9,574 (psi) |
| | 66.0 Mpa |

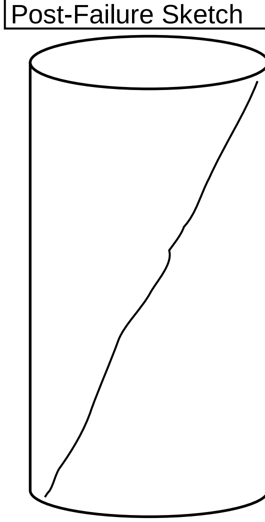
| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 23,600 (lbs) |
| Mode of Failure | Fracture |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

Pre-Failure Sketch



Post-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | XX |
| Intact | |
| Both | |

| | | | |
|---|---|---|---|
| Dia. 1 | 1.777 | Ht. 1 | 3.814 |
| Dia. 2 | 1.768 | Ht. 2 | 3.840 |
| Dia. 3 | 1.768 | Ht. 3 | 3.852 |
| Dia. 4 | 1.770 | Ht. 4 | 3.825 |
| Dia. 5 | 1.770 | Weight (gm) | 408.90 |
| Dia. 6 | 1.776 | Sample # | **03-006U** |

**Fail Load** 23600 lbs

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*TUCSON, ARIZONA USA*

| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Uniaxial Compression Test Results | Client | SRK |
| Date | 5/4/2009 | | Location | |
| Technician | D.Streeter | | Sample # | **03-011U** |
| | | | Rock Type | GRANITE |
| | | | Density : | 167.7 (pcf) |
| | | | | 2,686.1 (kg/m$^3$) |

Sample # **03-011U**

Fail Stress **14,881** psi

102.63 Mpa

Modulus psi

Poisson's

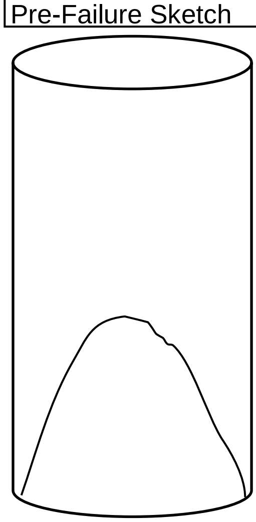
| Sample Data : | |
|---|---|
| Sample # : | **03-011U** |
| Rock Type: | GRANITE |
| Hole # : | C09-03 |
| Depth : | 282.1 |
| Alterations: | |
| Diameter : | 1.774 (in) |
| Height : | 3.953 (in) |
| Weight : | 430.35 (gm) |
| Area : | 2.473 (in$^2$) |
| Volume : | 9.777 (in$^3$) |

| | |
|---|---|
| Fail Stress | 14,881 (psi) |
| | 102.6 Mpa |

| Test Data: | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 36,800 (lbs) |
| Mode of Failure | Both |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

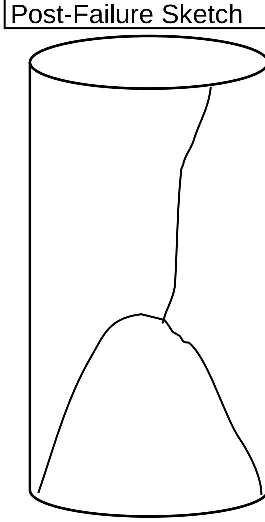
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | |
| Both | XX |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.772 | Ht. 1 | 3.957 |
| Dia. 2 | 1.775 | Ht. 2 | 3.953 |
| Dia. 3 | 1.773 | Ht. 3 | 3.951 |
| Dia. 4 | 1.775 | Ht. 4 | 3.953 |
| Dia. 5 | 1.777 | Weight (gm) | 430.35 |
| Dia. 6 | 1.775 | Sample # | **03-011U** |

**Fail Load** 36800 lbs

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*TUCSON, ARIZONA USA*

## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **03-014U** |
| Rock Type | GRANITE |
| Density : | 169.1 (pcf) |
| | 2,709.0 (kg/$m^3$) |

Sample # **03-014U**

Fail Stress **21,690** psi

149.58 Mpa

Modulus psi

Poisson's

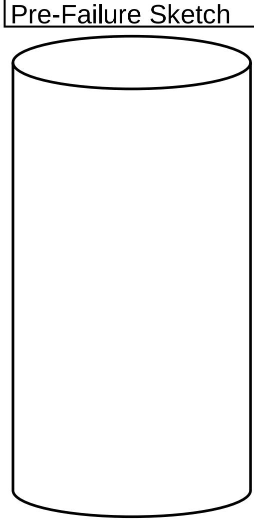
| **Sample Data :** | |
|---|---|
| Sample # : | **03-014U** |
| Rock Type: | GRANITE |
| Hole # : | C09-03 |
| Depth : | 361.7 |
| Alterations: | |
| Diameter : | 1.771 (in) |
| Height : | 1.991 (in) |
| Weight : | 217.79 (gm) |
| Area : | 2.464 ($in^2$) |
| Volume : | 4.906 ($in^3$) |

| | |
|---|---|
| Fail Stress | 21,690 (psi) |
| | 149.6 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 58,400 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 0.915 |

Worksheet

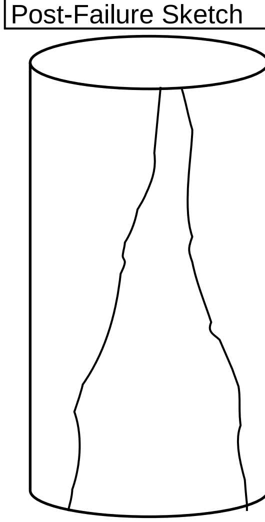
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.773 | Ht. 1 | 1.989 |
| Dia. 2 | 1.771 | Ht. 2 | 1.992 |
| Dia. 3 | 1.771 | Ht. 3 | 1.993 |
| Dia. 4 | 1.771 | Ht. 4 | 1.992 |
| Dia. 5 | 1.770 | Weight (gm) | 217.79 |
| Dia. 6 | 1.772 | Sample # | **03-014U** |

**Fail Load** 58400 lbs

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## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **04-001U** |
| Rock Type | LT GRANITE |
| Density : | 161.2 (pcf) |
| | 2,581.4 ($kg/m^3$) |

Sample # **04-001U**

Fail Stress **9,157** psi

63.15 Mpa

Modulus psi

Poisson's

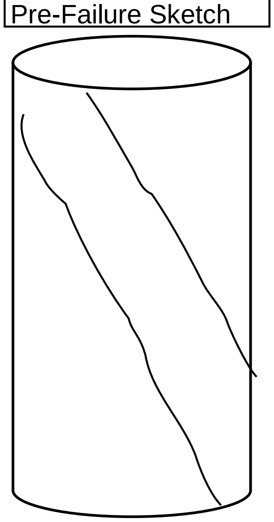
| **Sample Data :** | |
|---|---|
| Sample # : | **04-001U** |
| Rock Type: | LT GRANITE |
| Hole # : | C09-04 |
| Depth : | 30.4 |
| Alterations: | |
| Diameter : | 1.777 (in) |
| Height : | 3.653 (in) |
| Weight : | 383.03 (gm) |
| Area : | 2.479 ($in^2$) |
| Volume : | 9.055 ($in^3$) |

| | |
|---|---|
| Fail Stress | 9,157 (psi) |
| | 63.2 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 22,700 (lbs) |
| Mode of Failure | Fracture |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

Pre-Failure Sketch



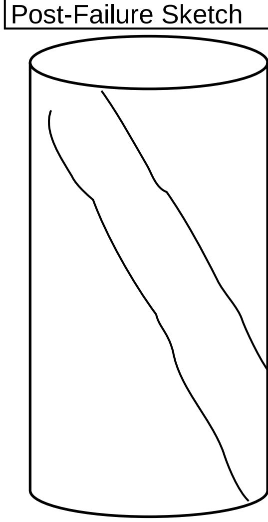
Mode of Failure :

Fracture XX

Intact

Both

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.772 | Ht. 1 | 3.653 |
| Dia. 2 | 1.772 | Ht. 2 | 3.653 |
| Dia. 3 | 1.771 | Ht. 3 | 3.653 |
| Dia. 4 | 1.795 | Ht. 4 | 3.652 |
| Dia. 5 | 1.773 | Weight (gm) | 383.03 |
| Dia. 6 | 1.776 | Sample # | **04-001U** |

**Fail Load** 22700 lbs

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## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **04-003U** |
| Rock Type | GRANITE |
| Density : | 167.7 (pcf) |
| | 2,686.2 (kg/m$^3$) |

Sample # **04-003U**

Fail Stress **20,404** psi

140.72 Mpa

Modulus psi

Poisson's

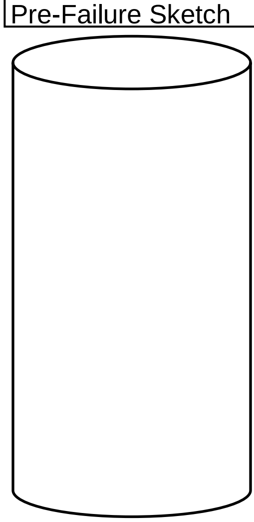
| Sample Data : | |
|---|---|
| Sample # : | **04-003U** |
| Rock Type: | GRANITE |
| Hole # : | C09-04 |
| Depth : | 91.1 |
| Alterations: | |
| Diameter : | 1.777 (in) |
| Height : | 3.911 (in) |
| Weight : | 426.91 (gm) |
| Area : | 2.480 (in$^2$) |
| Volume : | 9.698 (in$^3$) |

| | |
|---|---|
| Fail Stress | 20,404 (psi) |
| | 140.7 Mpa |

| Test Data: | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 50,600 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

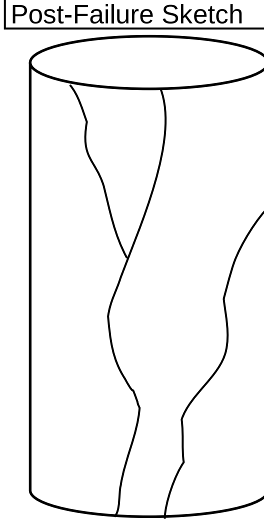
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.782 | Ht. 1 | 3.912 |
| Dia. 2 | 1.775 | Ht. 2 | 3.910 |
| Dia. 3 | 1.775 | Ht. 3 | 3.910 |
| Dia. 4 | 1.776 | Ht. 4 | 3.911 |
| Dia. 5 | 1.775 | Weight (gm) | 426.91 |
| Dia. 6 | 1.779 | Sample # | **04-003U** |

**Fail Load** 50600 lbs

# *University of Arizona*

*GEOMECHANICAL LABORATORY*

*TUCSON, ARIZONA USA*

## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **04-005U** |
| Rock Type | GRANITE |
| Density : | 168.8 (pcf) |
| | 2,703.6 (kg/m$^3$) |

| | | |
|---|---|---|
| Sample # | **04-005U** | |
| Fail Stress | **22,246** | psi |
| | 153.42 | Mpa |
| Modulus | | psi |
| Poisson's | | |

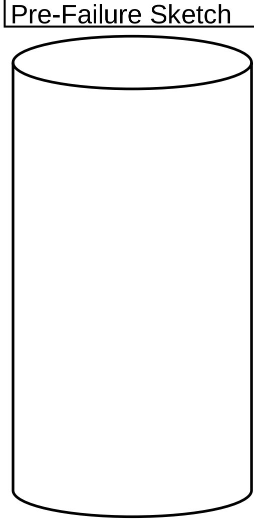
| | |
|---|---|
| Fail Stress | 22,246 (psi) |
| | 153.4 Mpa |

| **Sample Data :** | |
|---|---|
| Sample # : | **04-005U** |
| Rock Type: | GRANITE |
| Hole # : | C09-04 |
| Depth : | 150.25 |
| Alterations: | |
| Diameter : | 1.773 (in) |
| Height : | 3.921 (in) |
| Weight : | 428.69 (gm) |
| Area : | 2.468 (in$^2$) |
| Volume : | 9.676 (in$^3$) |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 54,900 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

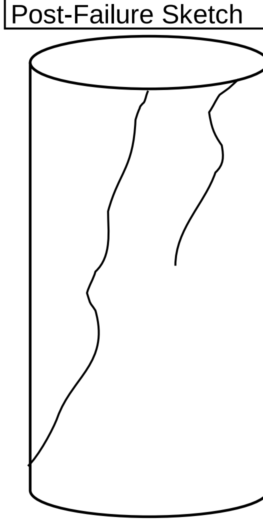
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.772 | Ht. 1 | 3.923 |
| Dia. 2 | 1.774 | Ht. 2 | 3.921 |
| Dia. 3 | 1.771 | Ht. 3 | 3.920 |
| Dia. 4 | 1.775 | Ht. 4 | 3.919 |
| Dia. 5 | 1.774 | Weight (gm) | 428.69 |
| Dia. 6 | 1.770 | Sample # | **04-005U** |

| | | |
|---|---|---|
| **Fail Load** | 54900 | lbs |

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## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **05-001U** |
| Rock Type | GRANITE |
| Density : | 165.6 (pcf) |
| | 2,653.4 (kg/m$^3$) |

Sample # **05-001U**

Fail Stress **10,284** psi

70.92 Mpa

Modulus psi

Poisson's

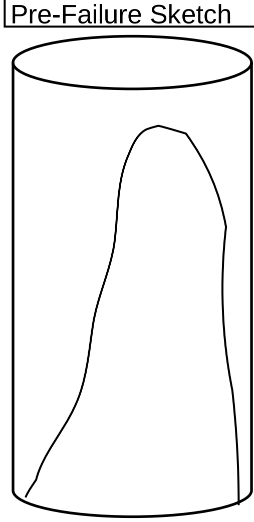
| Sample Data : | |
|---|---|
| Sample # : | **05-001U** |
| Rock Type: | GRANITE |
| Hole # : | C09-05 |
| Depth : | 33 |
| Alterations: | |
| Diameter : | 1.773 (in) |
| Height : | 3.892 (in) |
| Weight : | 417.99 (gm) |
| Area : | 2.470 (in$^2$) |
| Volume : | 9.613 (in$^3$) |

| | |
|---|---|
| Fail Stress | 10,284 (psi) |
| | 70.9 Mpa |

| Test Data: | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 25,400 (lbs) |
| Mode of Failure | Both |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

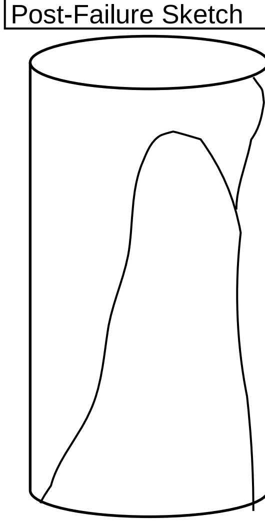
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | |
| Both | XX |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.772 | Ht. 1 | 3.893 |
| Dia. 2 | 1.767 | Ht. 2 | 3.893 |
| Dia. 3 | 1.789 | Ht. 3 | 3.892 |
| Dia. 4 | 1.775 | Ht. 4 | 3.891 |
| Dia. 5 | 1.768 | Weight (gm) | 417.99 |
| Dia. 6 | 1.770 | Sample # | **05-001U** |

**Fail Load** 25400 lbs

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Uniaxial Compression Test Results | Client | SRK |
| Date | 5/4/2009 | | Location | |
| Technician | D.Streeter | | Sample # | **05-003U** |
| | | Sample # **05-003U** | Rock Type | GRANITE |
| | | | Density : | 163.4 (pcf) |
| | | Fail Stress **10,780** psi | | 2,616.6 (kg/m$^3$) |
| | | 74.34 Mpa | | |

| **Sample Data :** | |
|---|---|
| Sample # : | **05-003U** |
| Rock Type: | GRANITE |
| Hole # : | C09-05 |
| Depth : | 92.7 |
| Alterations: | |
| Diameter : | 1.769 (in) |
| Height : | 3.818 (in) |
| Weight : | 402.49 (gm) |
| Area : | 2.458 (in$^2$) |
| Volume : | 9.387 (in$^3$) |

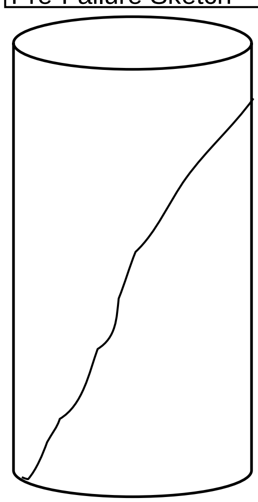
| | | |
|---|---|---|
| Modulus | | psi |
| Poisson's | | |

| | |
|---|---|
| Fail Stress | 10,780 (psi) |
| | 74.3 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 26,500 (lbs) |
| Mode of Failure | Fracture |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | XX |
| Intact | |
| Both | |

Post-Failure Sketch

| | | | |
|---|---|---|---|
| Dia. 1 | 1.771 | Ht. 1 | 3.819 |
| Dia. 2 | 1.767 | Ht. 2 | 3.819 |
| Dia. 3 | 1.767 | Ht. 3 | 3.818 |
| Dia. 4 | 1.773 | Ht. 4 | 3.818 |
| Dia. 5 | 1.771 | Weight (gm) | 402.49 |
| Dia. 6 | 1.767 | Sample # | **05-003U** |

| | | |
|---|---|---|
| **Fail Load** | 26500 | lbs |

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Uniaxial Compression Test Results | Client | SRK |
| Date | 5/4/2009 | | Location | |
| Technician | D.Streeter | | Sample # | **06-001U** |
| | | Sample # **06-001U** | Rock Type | GRANITE |
| | | | Density : | 166.0 (pcf) |
| | | Fail Stress **17,574** psi | | 2,659.8 (kg/m$^3$) |
| | | 121.20 Mpa | | |

| **Sample Data :** | |
|---|---|
| Sample # : | **06-001U** |
| Rock Type: | GRANITE |
| Hole # : | C09-06 |
| Depth : | 37.2 |
| Alterations: | |
| Diameter : | 1.771 (in) |
| Height : | 3.841 (in) |
| Weight : | 412.53 (gm) |
| Area : | 2.464 (in$^2$) |
| Volume : | 9.464 (in$^3$) |

Modulus: psi

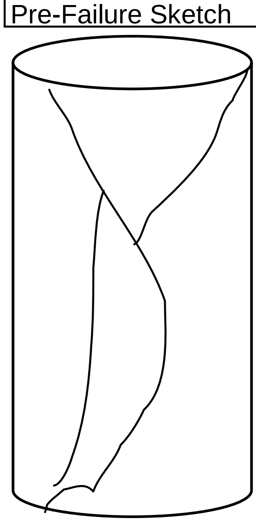
Poisson's:

| | |
|---|---|
| Fail Stress | 17,574 (psi) |
| | 121.2 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 43,300 (lbs) |
| Mode of Failure | Fracture |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

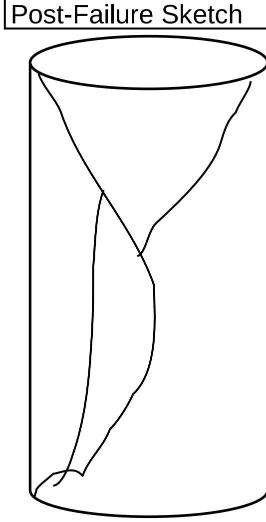
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | XX |
| Intact | |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.770 | Ht. 1 | 3.839 |
| Dia. 2 | 1.768 | Ht. 2 | 3.843 |
| Dia. 3 | 1.777 | Ht. 3 | 3.841 |
| Dia. 4 | 1.772 | Ht. 4 | 3.843 |
| Dia. 5 | 1.770 | Weight (gm) | 412.53 |
| Dia. 6 | 1.771 | Sample # | **06-001U** |

**Fail Load** 43300 lbs

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*TUCSON, ARIZONA USA*

## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **06-003U** |
| Rock Type | GRANITE |
| Density : | 165.8 (pcf) |
| | 2,655.7 (kg/m$^3$) |

Sample # **06-003U**

Fail Stress **17,803** psi

122.78 Mpa

Modulus psi

Poisson's

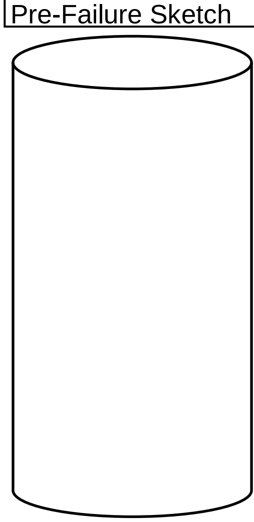
| **Sample Data :** | |
|---|---|
| Sample # : | **06-003U** |
| Rock Type: | GRANITE |
| Hole # : | C09-06 |
| Depth : | 108.35 |
| Alterations: | |
| Diameter : | 1.774 (in) |
| Height : | 3.309 (in) |
| Weight : | 355.87 (gm) |
| Area : | 2.472 (in$^2$) |
| Volume : | 8.177 (in$^3$) |

| | |
|---|---|
| Fail Stress | 17,803 (psi) |
| | 122.8 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 44,400 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 0.991 |

Worksheet

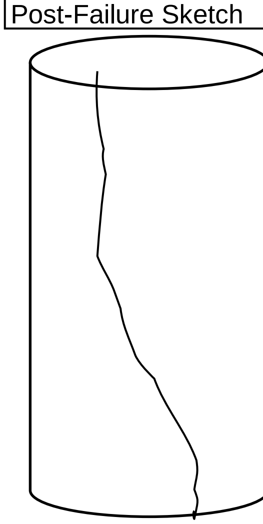
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.773 | Ht. 1 | 3.308 |
| Dia. 2 | 1.773 | Ht. 2 | 3.309 |
| Dia. 3 | 1.774 | Ht. 3 | 3.310 |
| Dia. 4 | 1.775 | Ht. 4 | 3.309 |
| Dia. 5 | 1.778 | Weight (gm) | 355.87 |
| Dia. 6 | 1.772 | Sample # | **06-003U** |

**Fail Load** 44400 lbs

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## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **06-004U** |
| Rock Type | GRANITE |
| Density : | 167.4 (pcf) |
| | 2,680.8 (kg/m$^3$) |

Sample # **06-004U**

Fail Stress **14,601** psi

100.70 Mpa

Modulus psi

Poisson's

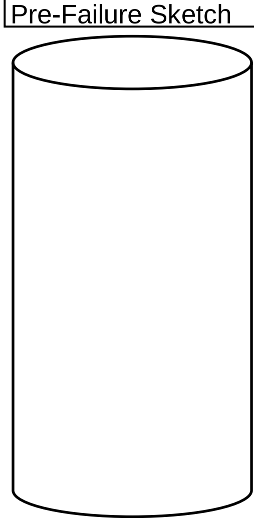
| **Sample Data :** | |
|---|---|
| Sample # : | **06-004U** |
| Rock Type: | GRANITE |
| Hole # : | C09-06 |
| Depth : | 138 |
| Alterations: | |
| Diameter : | 1.776 (in) |
| Height : | 2.154 (in) |
| Weight : | 234.59 (gm) |
| Area : | 2.479 (in$^2$) |
| Volume : | 5.340 (in$^3$) |

| | |
|---|---|
| Fail Stress | 14,601 (psi) |
| | 100.7 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 39,000 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 0.928 |

Worksheet

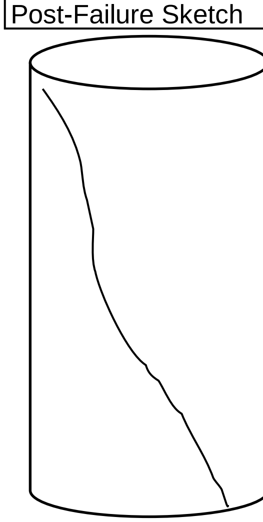
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.779 | Ht. 1 | 2.156 |
| Dia. 2 | 1.774 | Ht. 2 | 2.155 |
| Dia. 3 | 1.780 | Ht. 3 | 2.154 |
| Dia. 4 | 1.780 | Ht. 4 | 2.153 |
| Dia. 5 | 1.774 | Weight (gm) | 234.59 |
| Dia. 6 | 1.773 | Sample # | **06-004U** |

**Fail Load** 39000 lbs

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*TUCSON, ARIZONA USA*

## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022_006 |
| Date | 7/22/2009 |
| Technician | D.Streeter |

| | |
|---|---|
| Client | SRK |
| Location | MINTO |
| Sample # | **C09-07-01U** |
| Rock Type | GRANITE |
| Density : | 170.0 (pcf) |
| | 2,723.5 (kg/$m^3$) |

Sample # **C09-07-01U**

Fail Stress **24,982** psi

172.29 Mpa

Modulus psi

Poisson's

| **Sample Data :** | |
|---|---|
| Sample # : | **C09-07-01U** |
| Rock Type: | GRANITE |
| Hole # : | C09-07 |
| Depth : | 29.32-29.48 |
| Alterations: | |
| Diameter : | 1.772 (in) |
| Height : | 3.812 (in) |
| Weight : | 419.54 (gm) |
| Area : | 2.466 ($in^2$) |
| Volume : | 9.400 ($in^3$) |

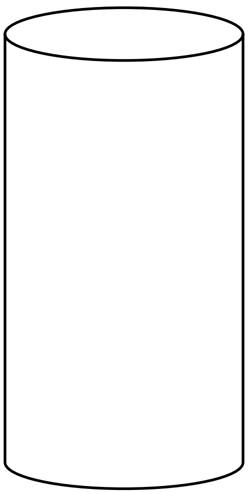
| | |
|---|---|
| Fail Stress | 24,982 (psi) |
| | 172.3 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 61,600 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

---

Worksheet

Pre-Failure Sketch



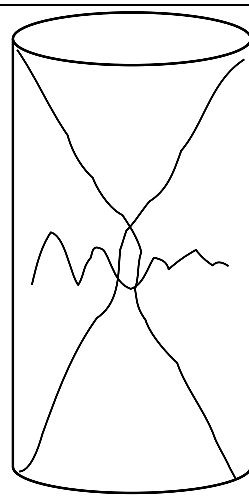
Mode of Failure :

Fracture

Intact XX

Both

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.771 | Ht. 1 | 3.812 |
| Dia. 2 | 1.770 | Ht. 2 | 3.811 |
| Dia. 3 | 1.770 | Ht. 3 | 3.813 |
| Dia. 4 | 1.779 | Ht. 4 | 3.814 |
| Dia. 5 | 1.771 | Weight (gm) | 419.54 |
| Dia. 6 | 1.771 | Sample # | C09-07-01U |

**Fail Load** 61600 lbs

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*TUCSON, ARIZONA USA*

| | |
|---|---|
| Project # | 2CM022_006 |
| Date | 7/22/2009 |
| Technician | D.Streeter |
| | |

## Uniaxial Compression Test Results

| | |
|---|---|
| Client | SRK |
| Location | MINTO |
| Sample # | **C09-07-03U** |
| Rock Type | GRANITE |
| Density : | 169.1 (pcf) |
| | 2,707.8 (kg/m$^3$) |

| | | |
|---|---|---|
| Sample # | **C09-07-03U** | |
| Fail Stress | **20,255** | psi |
| | 139.69 | Mpa |
| Modulus | | psi |
| Poisson's | | |

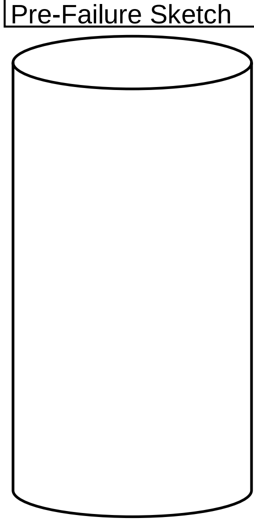
| | |
|---|---|
| Fail Stress | 20,255 (psi) |
| | 139.7 Mpa |

| **Sample Data :** | |
|---|---|
| Sample # : | **C09-07-03U** |
| Rock Type: | GRANITE |
| Hole # : | C09-07 |
| Depth : | 86.34-86.52 |
| Alterations: | |
| Diameter : | 1.775 (in) |
| Height : | 3.858 (in) |
| Weight : | 423.39 (gm) |
| Area : | 2.473 (in$^2$) |
| Volume : | 9.541 (in$^3$) |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 50,100 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

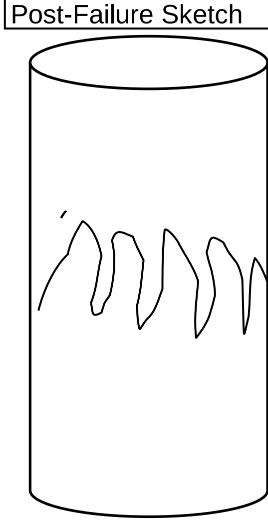
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.781 | Ht. 1 | 3.857 |
| Dia. 2 | 1.773 | Ht. 2 | 3.858 |
| Dia. 3 | 1.772 | Ht. 3 | 3.859 |
| Dia. 4 | 1.777 | Ht. 4 | 3.858 |
| Dia. 5 | 1.772 | Weight (gm) | 423.39 |
| Dia. 6 | 1.773 | Sample # | C09-07-03U |

| | | |
|---|---|---|
| **Fail Load** | 50100 | lbs |

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*TUCSON, ARIZONA USA*

## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022_006 |
| Date | 7/22/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | MINTO |
| Sample # | **C09-07-05U** |
| Rock Type | GRANITE |
| Density : | 167.6 (pcf) |
| | 2,685.0 (kg/m$^3$) |

| | | |
|---|---|---|
| Sample # | **C09-07-05U** | |
| Fail Stress | **18,078** | psi |
| | 124.68 | Mpa |
| Modulus | | psi |
| Poisson's | | |

| | |
|---|---|
| Fail Stress | 18,078 (psi) |
| | 124.7 Mpa |

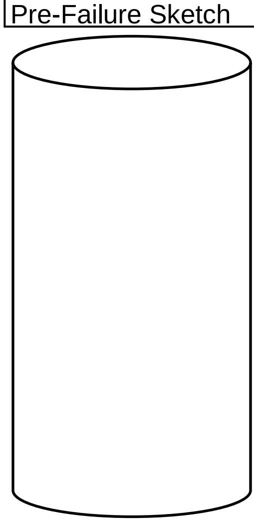
| **Sample Data :** | |
|---|---|
| Sample # : | **C09-07-05U** |
| Rock Type: | GRANITE |
| Hole # : | C09-07 |
| Depth : | 124.57-124.76 |
| Alterations: | |
| Diameter : | 1.766 (in) |
| Height : | 3.872 (in) |
| Weight : | 417.48 (gm) |
| Area : | 2.450 (in$^2$) |
| Volume : | 9.488 (in$^3$) |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 44,300 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

---

Worksheet

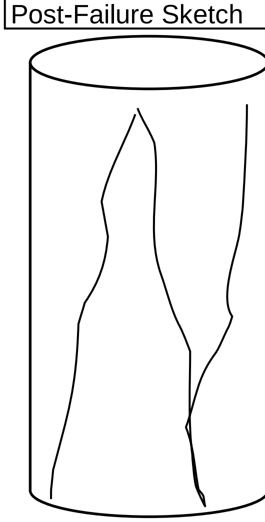
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.766 | Ht. 1 | 3.873 |
| Dia. 2 | 1.765 | Ht. 2 | 3.874 |
| Dia. 3 | 1.767 | Ht. 3 | 3.876 |
| Dia. 4 | 1.765 | Ht. 4 | 3.866 |
| Dia. 5 | 1.767 | Weight (gm) | 417.48 |
| Dia. 6 | 1.769 | Sample # | C09-07-05U |

| | | |
|---|---|---|
| **Fail Load** | 44300 | lbs |

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TUCSON, ARIZONA USA

| | | | | |
|---|---|---|---|---|
| Project # | 2CM022_006 | Uniaxial Compression Test Results | Client | SRK |
| Date | 7/22/2009 | | Location | MINTO |
| Technician | D.Streeter | | Sample # | **C09-08-01U** |

Sample # **C09-08-01U**

| | |
|---|---|
| Rock Type | GRANITE |
| Density : | 168.9 (pcf) |
| | 2,704.7 (kg/$m^3$) |

| | | |
|---|---|---|
| Fail Stress | **22,867** | psi |
| | 157.71 | Mpa |
| Modulus | | psi |
| Poisson's | | |

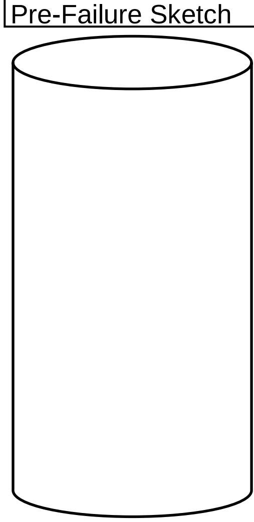
| | |
|---|---|
| Fail Stress | 22,867 (psi) |
| | 157.7 Mpa |

| **Sample Data :** | |
|---|---|
| Sample # : | **C09-08-01U** |
| Rock Type: | GRANITE |
| Hole # : | C09-08 |
| Depth : | 47.53-47.74 |
| Alterations: | |
| Diameter : | 1.769 (in) |
| Height : | 3.863 (in) |
| Weight : | 420.80 (gm) |
| Area : | 2.458 ($in^2$) |
| Volume : | 9.494 ($in^3$) |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 56,200 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

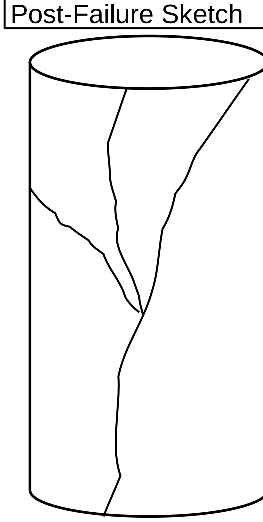
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.771 | Ht. 1 | 3.864 |
| Dia. 2 | 1.769 | Ht. 2 | 3.865 |
| Dia. 3 | 1.767 | Ht. 3 | 3.861 |
| Dia. 4 | 1.773 | Ht. 4 | 3.862 |
| Dia. 5 | 1.767 | Weight (gm) | 420.80 |
| Dia. 6 | 1.767 | Sample # | C09-08-01U |

**Fail Load** 56200 lbs

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| Project # | 2CM022_006 | Uniaxial Compression Test Results | Client | SRK |
|---|---|---|---|---|
| Date | 7/22/2009 | | Location | MINTO |
| Technician | D.Streeter | | Sample # | **C09-08-04U** |
| | | | Rock Type | GRANITE |
| | | | Density : | 168.5 (pcf) |
| | | | | 2,698.7 (kg/$m^3$) |

| | |
|---|---|
| Sample # | **C09-08-04U** |
| Fail Stress | **13,676** psi |
| | 94.31 Mpa |
| Modulus | psi |
| Poisson's | |

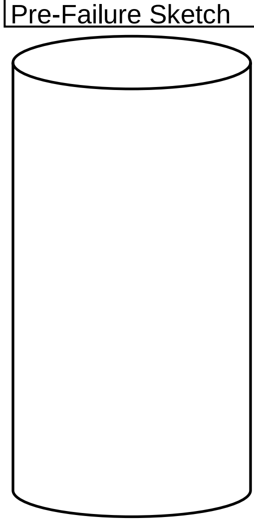
| Sample Data : | |
|---|---|
| Sample # : | **C09-08-04U** |
| Rock Type: | GRANITE |
| Hole # : | C09-08 |
| Depth : | 89.15-89.39 |
| Alterations: | |
| Diameter : | 1.774 (in) |
| Height : | 3.916 (in) |
| Weight : | 428.04 (gm) |
| Area : | 2.472 ($in^2$) |
| Volume : | 9.679 ($in^3$) |

| | |
|---|---|
| Fail Stress | 13,676 (psi) |
| | 94.3 Mpa |

| Test Data: | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 33,800 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

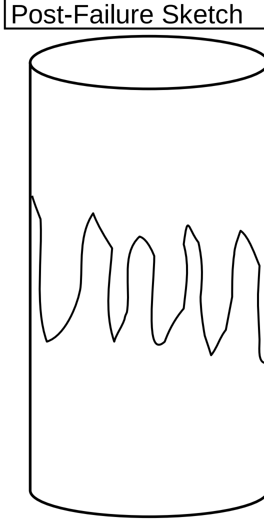
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.773 | Ht. 1 | 3.915 |
| Dia. 2 | 1.772 | Ht. 2 | 3.915 |
| Dia. 3 | 1.776 | Ht. 3 | 3.919 |
| Dia. 4 | 1.777 | Ht. 4 | 3.916 |
| Dia. 5 | 1.773 | Weight (gm) | 428.04 |
| Dia. 6 | 1.774 | Sample # | C09-08-04U |

**Fail Load** 33800 lbs

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## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022_006 |
| Date | 7/22/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | MINTO |
| Sample # | **C09-08-07U** |
| Rock Type | GRANITE |
| Density : | 167.9 (pcf) |
| | 2,689.5 (kg/m$^3$) |

Sample # **C09-08-07U**

Fail Stress **22,273** psi

153.60 Mpa

Modulus psi

Poisson's

| **Sample Data :** | |
|---|---|
| Sample # : | **C09-08-07U** |
| Rock Type: | GRANITE |
| Hole # : | C09-08 |
| Depth : | 129.4-129.65 |
| Alterations: | |
| Diameter : | 1.770 (in) |
| Height : | 3.884 (in) |
| Weight : | 421.15 (gm) |
| Area : | 2.460 (in$^2$) |
| Volume : | 9.555 (in$^3$) |

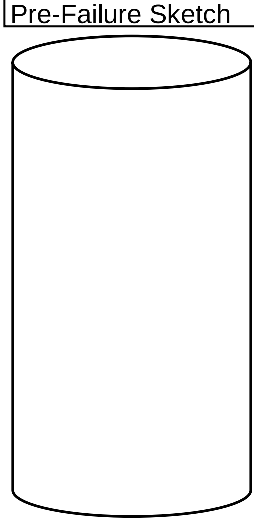
| | |
|---|---|
| Fail Stress | 22,273 (psi) |
| | 153.6 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 54,800 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

---

Worksheet

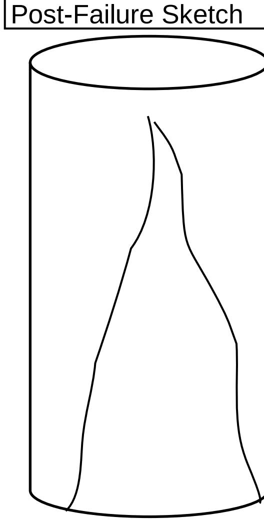
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.767 | Ht. 1 | 3.883 |
| Dia. 2 | 1.770 | Ht. 2 | 3.884 |
| Dia. 3 | 1.772 | Ht. 3 | 3.885 |
| Dia. 4 | 1.774 | Ht. 4 | 3.884 |
| Dia. 5 | 1.770 | Weight (gm) | 421.15 |
| Dia. 6 | 1.768 | Sample # | C09-08-07U |

**Fail Load** 54800 lbs

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## Uniaxial Compression Test Results

| | | | |
|---|---|---|---|
| Project # | 2CM022 | Client | SRK |
| Date | 5/4/2009 | Location | |
| Technician | D.Streeter | Sample # | **01-001E** |
| | | Rock Type | GRANITE |
| | | Density : | 166.3 (pcf) |
| | | | 2,663.7 (kg/m$^3$) |

| | | |
|---|---|---|
| Sample # | **01-001E** | |
| Fail Stress | **12,790** | psi |
| | 88.21 | Mpa |
| Modulus | **7.32E+06** | psi |
| Poisson's | 0.217 | |

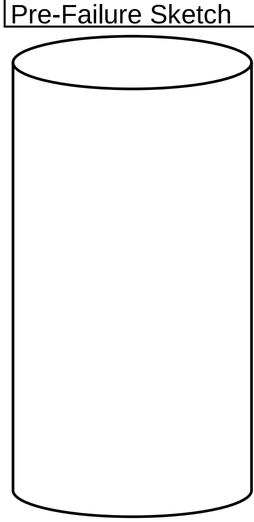
| Fail Stress | 12,790 (psi) |
|---|---|
| | 88.2 Mpa |

| **Sample Data :** | |
|---|---|
| Sample # : | **01-001E** |
| Rock Type: | GRANITE |
| Hole # : | C09-01 |
| Depth : | 32.1 |
| Alterations: | |
| Diameter : | 1.774 (in) |
| Height : | 3.830 (in) |
| Weight : | 413.02 (gm) |
| Area : | 2.471 (in$^2$) |
| Volume : | 9.462 (in$^3$) |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 31,600 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

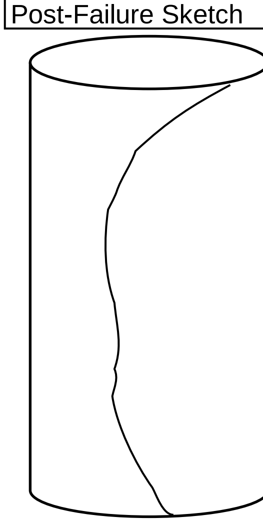
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.770 | Ht. 1 | 3.840 |
| Dia. 2 | 1.774 | Ht. 2 | 3.835 |
| Dia. 3 | 1.779 | Ht. 3 | 3.823 |
| Dia. 4 | 1.778 | Ht. 4 | 3.822 |
| Dia. 5 | 1.771 | Weight (gm) | 413.02 |
| Dia. 6 | 1.771 | Sample # | **01-001E** |

**Fail Load** 31600 lbs

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Uniaxial Compression Test Results | Client | SRK |
| Date | 5/4/2009 | | Location | |
| Technician | D.Streeter | | Sample # | **01-008E** |
| | | Sample # **01-008E** | Rock Type | GRANITE |
| | | | Density : | 169.4 (pcf) |
| | | | | 2,713.3 (kg/m$^3$) |

| | | |
|---|---|---|
| Sample # | **01-008E** | |
| Fail Stress | **23,878** | psi |
| | 164.68 | Mpa |
| Modulus | **9.65E+06** | psi |
| Poisson's | 0.302 | |

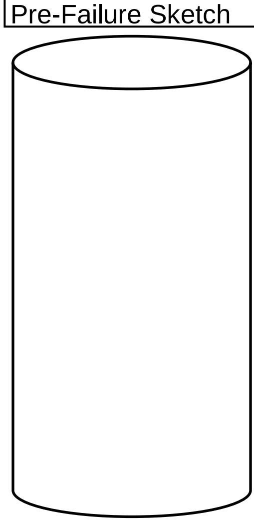
| **Sample Data :** | |
|---|---|
| Sample # : | **01-008E** |
| Rock Type: | GRANITE |
| Hole # : | C09-01 |
| Depth : | 220.3 |
| Alterations: | |
| Diameter : | 1.775 (in) |
| Height : | 4.038 (in) |
| Weight : | 444.35 (gm) |
| Area : | 2.475 (in$^2$) |
| Volume : | 9.993 (in$^3$) |

| | |
|---|---|
| Fail Stress | 23,878 (psi) |
| | 164.7 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 59,100 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

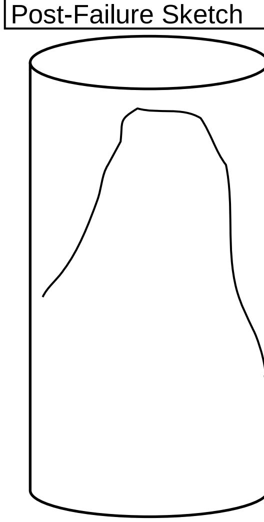
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.774 | Ht. 1 | 4.048 |
| Dia. 2 | 1.775 | Ht. 2 | 4.046 |
| Dia. 3 | 1.775 | Ht. 3 | 4.033 |
| Dia. 4 | 1.773 | Ht. 4 | 4.025 |
| Dia. 5 | 1.778 | Weight (gm) | 444.35 |
| Dia. 6 | 1.776 | Sample # | **01-008E** |

| | | |
|---|---|---|
| **Fail Load** | 59100 | lbs |

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## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **02-004E** |
| Rock Type | GRANITE |
| Density : | 166.6 (pcf) |
| | 2,668.0 (kg/m$^3$) |

| | | |
|---|---|---|
| Sample # | **02-004E** | |
| Fail Stress | **10,395** | psi |
| | 71.69 | Mpa |
| Modulus | **7.14E+06** | psi |
| Poisson's | 0.214 | |

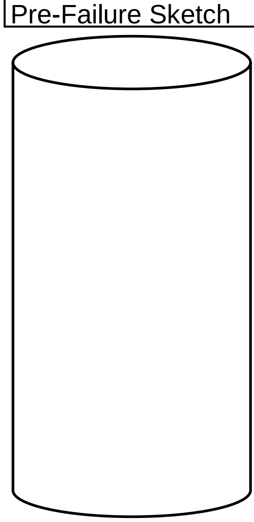
| **Sample Data :** | |
|---|---|
| Sample # : | **02-004E** |
| Rock Type: | GRANITE |
| Hole # : | C09-02 |
| Depth : | 122.67 |
| Alterations: | |
| Diameter : | 1.771 (in) |
| Height : | 3.905 (in) |
| Weight : | 420.48 (gm) |
| Area : | 2.463 (in$^2$) |
| Volume : | 9.617 (in$^3$) |

| | |
|---|---|
| Fail Stress | 10,395 (psi) |
| | 71.7 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 25,600 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

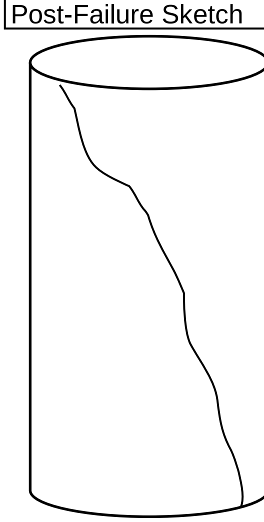
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.769 | Ht. 1 | 3.907 |
| Dia. 2 | 1.773 | Ht. 2 | 3.906 |
| Dia. 3 | 1.773 | Ht. 3 | 3.905 |
| Dia. 4 | 1.768 | Ht. 4 | 3.904 |
| Dia. 5 | 1.768 | Weight (gm) | 420.48 |
| Dia. 6 | 1.774 | Sample # | **02-004E** |

**Fail Load** 25600 lbs

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Uniaxial Compression Test Results | Client | SRK |
| Date | 5/4/2009 | | Location | |
| Technician | D.Streeter | | Sample # | **03-002E** |
| | | Sample # **03-002E** | Rock Type | LT GRANITE |
| | | | Density : | 164.2 (pcf) |
| | | | | 2,630.8 (kg/m$^3$) |

| | | |
|---|---|---|
| Fail Stress | **7,096** | psi |
| | 48.94 | Mpa |
| Modulus | **2.16E+06** | psi |
| Poisson's | 0.084 | |

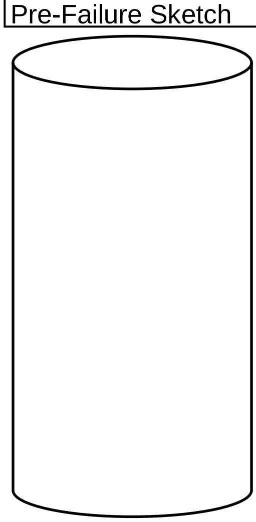
| | |
|---|---|
| Fail Stress | 7,096 (psi) |
| | 48.9 Mpa |

| **Sample Data :** | |
|---|---|
| Sample # : | **03-002E** |
| Rock Type: | LT GRANITE |
| Hole # : | C09-03 |
| Depth : | 38 |
| Alterations: | |
| Diameter : | 1.767 (in) |
| Height : | 3.699 (in) |
| Weight : | 391.01 (gm) |
| Area : | 2.452 (in$^2$) |
| Volume : | 9.070 (in$^3$) |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 17,400 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

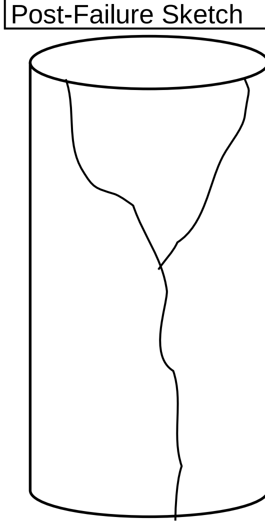
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.766 | Ht. 1 | 3.681 |
| Dia. 2 | 1.765 | Ht. 2 | 3.688 |
| Dia. 3 | 1.767 | Ht. 3 | 3.710 |
| Dia. 4 | 1.770 | Ht. 4 | 3.717 |
| Dia. 5 | 1.767 | Weight (gm) | 391.01 |
| Dia. 6 | 1.767 | Sample # | **03-002E** |

| | | |
|---|---|---|
| **Fail Load** | 17400 | lbs |

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*TUCSON, ARIZONA USA*

## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **03-007E** |
| Rock Type | GRANITE |
| Density : | 168.6 (pcf) |
| | 2,700.3 (kg/m$^3$) |

| | | |
|---|---|---|
| Sample # | **03-007E** | |
| Fail Stress | **15,136** | psi |
| | 104.39 | Mpa |
| Modulus | **6.86E+06** | psi |
| Poisson's | 0.228 | |

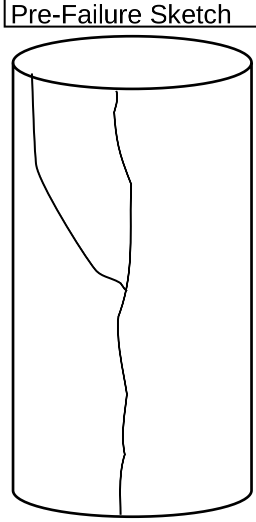
| | |
|---|---|
| Fail Stress | 15,136 (psi) |
| | 104.4 Mpa |

| **Sample Data :** | |
|---|---|
| Sample # : | **03-007E** |
| Rock Type: | GRANITE |
| Hole # : | C09-03 |
| Depth : | 161.03 |
| Alterations: | |
| Diameter : | 1.774 (in) |
| Height : | 3.890 (in) |
| Weight : | 425.37 (gm) |
| Area : | 2.471 (in$^2$) |
| Volume : | 9.613 (in$^3$) |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 37,400 (lbs) |
| Mode of Failure | Fracture |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

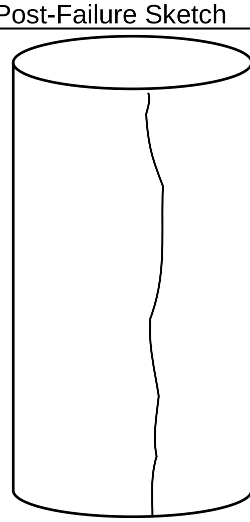
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | XX |
| Intact | |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.776 | Ht. 1 | 3.892 |
| Dia. 2 | 1.773 | Ht. 2 | 3.890 |
| Dia. 3 | 1.772 | Ht. 3 | 3.890 |
| Dia. 4 | 1.774 | Ht. 4 | 3.890 |
| Dia. 5 | 1.772 | Weight (gm) | 425.37 |
| Dia. 6 | 1.775 | Sample # | **03-007E** |

| | | |
|---|---|---|
| **Fail Load** | 37400 | lbs |

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*TUCSON, ARIZONA USA*

## Uniaxial Compression Test Results

| | |
|---|---|
| Project # | 2CM022 |
| Date | 5/4/2009 |
| Technician | D.Streeter |
| | |

| | |
|---|---|
| Client | SRK |
| Location | |
| Sample # | **05-005E** |
| Rock Type | PK GRANITE |
| Density : | 167.2 (pcf) |
| | 2,677.7 (kg/m$^3$) |

| | | |
|---|---|---|
| Sample # | **05-005E** | |
| Fail Stress | **12,573** | psi |
| | 86.71 | Mpa |
| Modulus | **7.82E+06** | psi |
| Poisson's | 0.262 | |

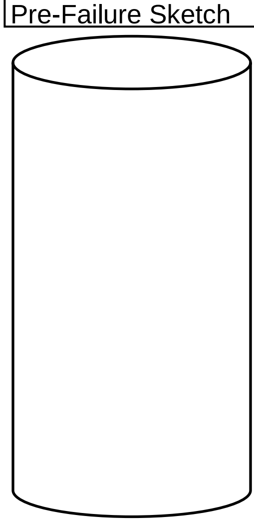
| | |
|---|---|
| Fail Stress | 12,573 (psi) |
| | 86.7 Mpa |

| **Sample Data :** | |
|---|---|
| Sample # : | **05-005E** |
| Rock Type: | PK GRANITE |
| Hole # : | C09-05 |
| Depth : | 150.11 |
| Alterations: | |
| Diameter : | 1.769 (in) |
| Height : | 3.879 (in) |
| Weight : | 418.29 (gm) |
| Area : | 2.458 (in$^2$) |
| Volume : | 9.532 (in$^3$) |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 30,900 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

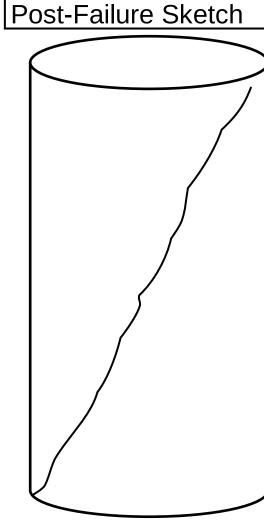
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.768 | Ht. 1 | 3.879 |
| Dia. 2 | 1.769 | Ht. 2 | 3.879 |
| Dia. 3 | 1.772 | Ht. 3 | 3.879 |
| Dia. 4 | 1.768 | Ht. 4 | 3.878 |
| Dia. 5 | 1.770 | Weight (gm) | 418.29 |
| Dia. 6 | 1.768 | Sample # | **05-005E** |

| | | |
|---|---|---|
| **Fail Load** | 30900 | lbs |

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## Uniaxial Compression Test Results

| | | | |
|---|---|---|---|
| Project # | 2CM022 | Client | SRK |
| Date | 5/4/2009 | Location | |
| Technician | D.Streeter | Sample # | **06-002E** |
| | | Rock Type | PK GRANITE |
| | | Density : | 165.6 (pcf) |
| | | | 2,652.7 (kg/m$^3$) |

| | | |
|---|---|---|
| Sample # | **06-002E** | |
| Fail Stress | **19,041** | psi |
| | 131.32 | Mpa |
| Modulus | **7.61E+06** | psi |
| Poisson's | 0.294 | |

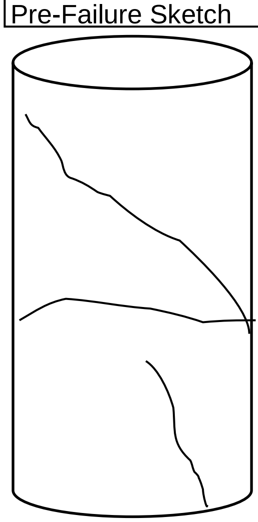
| **Sample Data :** | |
|---|---|
| Sample # : | **06-002E** |
| Rock Type: | PK GRANITE |
| Hole # : | C09-06 |
| Depth : | 71.22 |
| Alterations: | |
| Diameter : | 1.773 (in) |
| Height : | 3.942 (in) |
| Weight : | 423.02 (gm) |
| Area : | 2.468 (in$^2$) |
| Volume : | 9.731 (in$^3$) |

| | |
|---|---|
| Fail Stress | 19,041 (psi) |
| | 131.3 Mpa |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 47,000 (lbs) |
| Mode of Failure | Both |
| Test Duration : | (sec) |
| 2:1 Correction : | 1 |

Worksheet

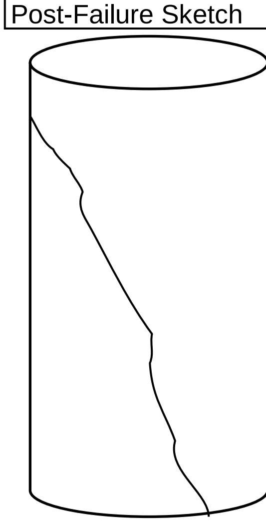
Pre-Failure Sketch



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | |
| Both | XX |

Post-Failure Sketch



| | | | |
|---|---|---|---|
| Dia. 1 | 1.777 | Ht. 1 | 3.940 |
| Dia. 2 | 1.772 | Ht. 2 | 3.943 |
| Dia. 3 | 1.772 | Ht. 3 | 3.946 |
| Dia. 4 | 1.774 | Ht. 4 | 3.941 |
| Dia. 5 | 1.770 | Weight (gm) | 423.02 |
| Dia. 6 | 1.773 | Sample # | **06-002E** |

| | | |
|---|---|---|
| **Fail Load** | 47000 | lbs |

# Triaxial Compressive Strength Testing

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Triaxial Compression Test Results | Client | SRK |
| Date | 5/7/2009 | | Location | |
| Technician | D.Streeter | **Failure Data:** | Sample # | **01-002T** |
| | | **Sample # 01-002T** | Rock Type | GRANITE |
| | | | Density : | 168.3 (pcf) |
| | | | | 2,696.0 (kg/m$^3$) |

**U.S. Standard**

| Sigma 3 (psi) | Sigma 1 (psi) | |
|---|---|---|
| 1,000 | 32,214 | **Peak** |
| | 0 | **Residuals** |
| | 0 | |
| | 0 | |
| | 0 | |

| **Sample Data :** | |
|---|---|
| Sample # : | 01-002T |
| Rock Type | GRANITE |
| Hole # : | C09-01 |
| Depth : | 59.88 |
| Alterations | |
| Diameter : | 1.773 (in) |
| Height : | 3.939 (in) |
| Weight : | 429.42 (gm) |
| Area : | 2.468 (in$^2$) |
| Volume : | 9.720 (in$^3$) |

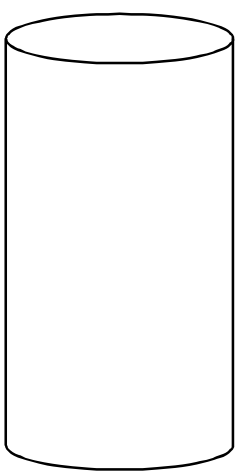
| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 79,500 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| | |

**Metric Standard**

| Sigma 3 (MPa) | Sigma 1 (MPa) | |
|---|---|---|
| 6.90 | 222.2 | **Peak** |
| #VALUE! | 0.0 | **Residuals** |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |

---

Worksheet



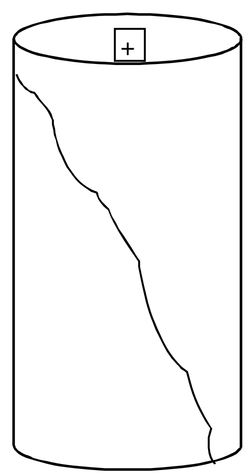
Pre-Failure Sketch

Mode of Failure :

Fracture [ ]

Intact [ XX ]

Both [ ]



Post-Failure

| | | | |
|---|---|---|---|
| Dia. 1 | 1.774 | Ht. 1 | 3.938 |
| Dia. 2 | 1.774 | Ht. 2 | 3.940 |
| Dia. 3 | 1.772 | Ht. 3 | 3.940 |
| Dia. 4 | 1.771 | Ht. 4 | 3.937 |
| Dia. 5 | 1.773 | Weight (gm) | 429.42 |
| Dia. 6 | 1.772 | Sample # | **01-002T** |

| **Sigma 3** (psi) | **Fail Load** gage (lbs) |
|---|---|
| 1,000 | 79,500 |
| | 0 |
| | 0 |
| | 0 |
| | 0 |

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Triaxial Compression Test Results | Client | SRK |
| Date | 5/7/2009 | | Location | |
| Technician | D.Streeter | **Failure Data:** | Sample # | **01-005T** |
| | | **Sample # 01-005T** | Rock Type | GRANITE |
| | | | Density : | 167.1 (pcf) |
| | | | | 2,677.3 (kg/$m^3$) |

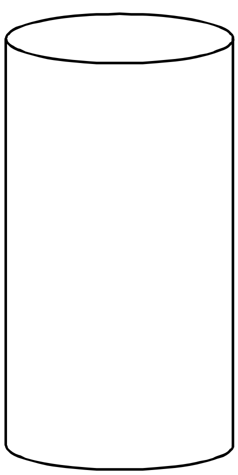
| **Sample Data :** | |
|---|---|
| Sample # : | 01-005T |
| Rock Type | GRANITE |
| Hole # : | C09-01 |
| Depth : | 153.3 |
| Alterations | |
| Diameter : | 1.776 (in) |
| Height : | 3.796 (in) |
| Weight : | 412.81 (gm) |
| Area : | 2.479 ($in^2$) |
| Volume : | 9.409 ($in^3$) |

| **U.S. Standard** | | |
|---|---|---|
| **Sigma 3** (psi) | **Sigma 1** (psi) | |
| 2,500 | 40,141 | **Peak** |
| | 0 | **Residuals** |
| | 0 | |
| | 0 | |
| | 0 | |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 99,500 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| | |

| **Metric Standard** | | |
|---|---|---|
| **Sigma 3** (MPa) | **Sigma 1** (MPa) | |
| 17.24 | 276.8 | **Peak** |
| #VALUE! | 0.0 | **Residuals** |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |

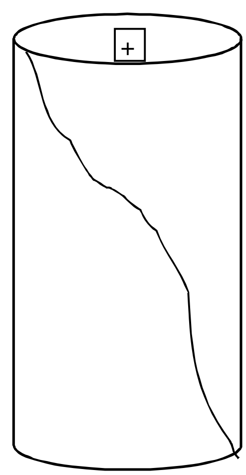
Worksheet



Pre-Failure Sketch

Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |



Post-Failure

| | | | |
|---|---|---|---|
| Dia. 1 | 1.775 | Ht. 1 | 3.797 |
| Dia. 2 | 1.775 | Ht. 2 | 3.796 |
| Dia. 3 | 1.779 | Ht. 3 | 3.795 |
| Dia. 4 | 1.775 | Ht. 4 | 3.796 |
| Dia. 5 | 1.779 | Weight (gm) | 412.81 |
| Dia. 6 | 1.777 | Sample # | **01-005T** |

| **Sigma 3** (psi) | **Fail Load** gage (lbs) |
|---|---|
| 2,500 | 99,500 |
| | 0 |
| | 0 |
| | 0 |
| | 0 |

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Triaxial Compression Test Results | Client | SRK |
| Date | 5/7/2009 | | Location | |
| Technician | D.Streeter | **Failure Data:** | Sample # | **02-005T** |
| | | **Sample # 02-005T** | Rock Type | GRANITE |
| | | | Density : | 167.8 (pcf) |
| | | | | 2,688.3 (kg/$m^3$) |

**U.S. Standard**

| Sigma 3 (psi) | Sigma 1 (psi) | |
|---|---|---|
| 1,500 | 31,004 | **Peak** |
| | 0 | **Residuals** |
| | 0 | |
| | 0 | |
| | 0 | |

| **Sample Data :** | |
|---|---|
| Sample # : | 02-005T |
| Rock Type | GRANITE |
| Hole # : | C09-02 |
| Depth : | 150.1 |
| Alterations | |
| Diameter : | 1.775 (in) |
| Height : | 3.349 (in) |
| Weight : | 365.03 (gm) |
| Area : | 2.474 ($in^2$) |
| Volume : | 8.286 ($in^3$) |

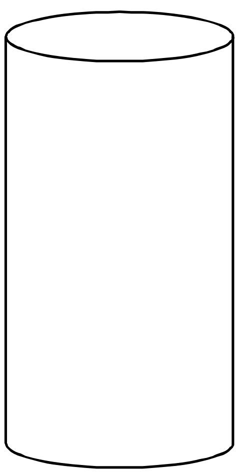
| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 76,700 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| | |

**Metric Standard**

| Sigma 3 (MPa) | Sigma 1 (MPa) | |
|---|---|---|
| 10.34 | 213.8 | **Peak** |
| #VALUE! | 0.0 | **Residuals** |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |

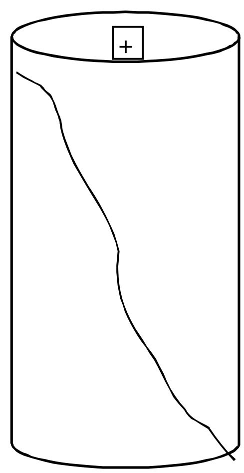
---

Worksheet



Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |



Pre-Failure Sketch

Post-Failure

| | | | |
|---|---|---|---|
| Dia. 1 | 1.774 | Ht. 1 | 3.350 |
| Dia. 2 | 1.774 | Ht. 2 | 3.350 |
| Dia. 3 | 1.775 | Ht. 3 | 3.349 |
| Dia. 4 | 1.773 | Ht. 4 | 3.350 |
| Dia. 5 | 1.779 | Weight (gm) | 365.03 |
| Dia. 6 | 1.774 | Sample # | **02-005T** |

| **Sigma 3** (psi) | **Fail Load** gage (lbs) |
|---|---|
| 1,500 | 76,700 |
| | 0 |
| | 0 |
| | 0 |
| | 0 |

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Triaxial Compression Test Results | Client | SRK |
| Date | 5/7/2009 | | Location | |
| Technician | D.Streeter | **Failure Data:** | Sample # | **02-007T** |
| | | **Sample # 02-007T** | Rock Type | GRANITE |
| | | | Density : | 168.2 (pcf) |
| | | | | 2,693.6 (kg/$m^3$) |

**U.S. Standard**

| Sigma 3 (psi) | Sigma 1 (psi) | |
|---|---|---|
| 2,000 | 42,659 | **Peak** |
| | 0 | Residuals |
| | 0 | |
| | 0 | |
| | 0 | |

| **Sample Data :** | |
|---|---|
| Sample # : | 02-007T |
| Rock Type | GRANITE |
| Hole # : | C09-02 |
| Depth : | 209.69 |
| Alterations | |
| Diameter : | 1.770 (in) |
| Height : | 3.926 (in) |
| Weight : | 426.55 (gm) |
| Area : | 2.461 ($in^2$) |
| Volume : | 9.663 ($in^3$) |

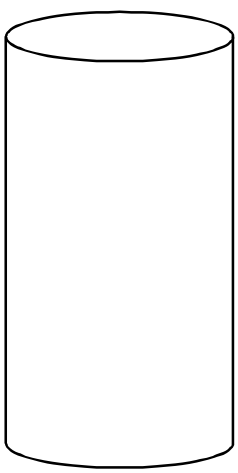
| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 105,000 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| | |

**Metric Standard**

| Sigma 3 (MPa) | Sigma 1 (MPa) | |
|---|---|---|
| 13.79 | 294.2 | **Peak** |
| #VALUE! | 0.0 | Residuals |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |

---

Worksheet



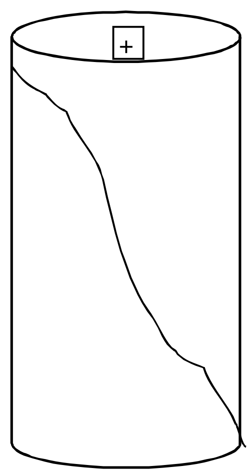
Pre-Failure Sketch

Mode of Failure :

Fracture [ ]

Intact [XX]

Both [ ]




Post-Failure

| | | | |
|---|---|---|---|
| Dia. 1 | 1.772 | Ht. 1 | 3.926 |
| Dia. 2 | 1.771 | Ht. 2 | 3.925 |
| Dia. 3 | 1.770 | Ht. 3 | 3.927 |
| Dia. 4 | 1.771 | Ht. 4 | 3.927 |
| Dia. 5 | 1.770 | Weight (gm) | 426.55 |
| Dia. 6 | 1.769 | Sample # | **02-007T** |

| **Sigma 3** (psi) | **Fail Load** gage (lbs) |
|---|---|
| 2,000 | 105,000 |
| | 0 |
| | 0 |
| | 0 |
| | 0 |

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Triaxial Compression Test Results | Client | SRK |
| Date | 5/7/2009 | | Location | |
| Technician | D.Streeter | **Failure Data:** | Sample # | **03-010T** |
| | | **Sample # 03-010T** | Rock Type | GRANITE |
| | | | Density : | 168.9 (pcf) |
| | | | | 2,706.0 (kg/m³) |

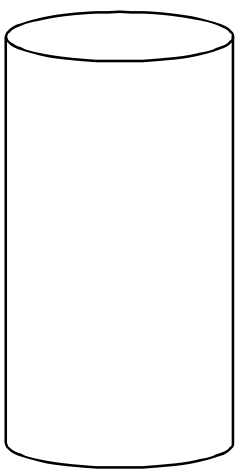
| **Sample Data :** | |
|---|---|
| Sample # : | 03-010T |
| Rock Type | GRANITE |
| Hole # : | C09-03 |
| Depth : | 250.17 |
| Alterations | |
| Diameter : | 1.771 (in) |
| Height : | 3.972 (in) |
| Weight : | 434.02 (gm) |
| Area : | 2.464 ($in^2$) |
| Volume : | 9.788 ($in^3$) |

| **U.S. Standard** | | |
|---|---|---|
| **Sigma 3** (psi) | **Sigma 1** (psi) | |
| 2,000 | 41,796 | **Peak** |
| | 0 | **Residuals** |
| | 0 | |
| | 0 | |
| | 0 | |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 103,000 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| | |

| **Metric Standard** | | |
|---|---|---|
| **Sigma 3** (MPa) | **Sigma 1** (MPa) | |
| 13.79 | 288.2 | **Peak** |
| #VALUE! | 0.0 | **Residuals** |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |

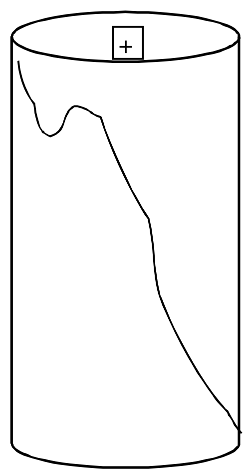
Worksheet



Pre-Failure Sketch

Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |



Post-Failure

| | | | |
|---|---|---|---|
| Dia. 1 | 1.771 | Ht. 1 | 3.971 |
| Dia. 2 | 1.770 | Ht. 2 | 3.972 |
| Dia. 3 | 1.776 | Ht. 3 | 3.974 |
| Dia. 4 | 1.770 | Ht. 4 | 3.971 |
| Dia. 5 | 1.771 | Weight (gm) | 434.02 |
| Dia. 6 | 1.772 | Sample # | **03-010T** |

| **Sigma 3** (psi) | **Fail Load** gage (lbs) |
|---|---|
| 2,000 | 103,000 |
| | 0 |
| | 0 |
| | 0 |
| | 0 |

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | Triaxial Compression Test Results | Client | SRK |
| Date | 5/7/2009 | | Location | |
| Technician | D.Streeter | **Failure Data:** | Sample # | **04-004T** |
| | | **Sample # 04-004T** | Rock Type | GRANITE |
| | | | Density : | 167.7 (pcf) |
| | | | | 2,687.0 ($kg/m^3$) |

**U.S. Standard**

| **Sigma 3** (psi) | **Sigma 1** (psi) | |
|---|---|---|
| 3,000 | 40,250 | **Peak** |
| | 0 | **Residuals** |
| | 0 | |
| | 0 | |
| | 0 | |

| **Sample Data :** | |
|---|---|
| Sample # : | 04-004T |
| Rock Type | GRANITE |
| Hole # : | C09-04 |
| Depth : | 123.25 |
| Alterations | |
| Diameter : | 1.768 (in) |
| Height : | 3.835 (in) |
| Weight : | 414.51 (gm) |
| Area : | 2.455 ($in^2$) |
| Volume : | 9.414 ($in^3$) |

| **Test Data:** | |
|---|---|
| Disp. Rate : | 0.0003 (in/sec) |
| Load Rate : | (lbs/sec) |
| Gage Reading : | 98,800 (lbs) |
| Mode of Failure | Intact |
| Test Duration : | (sec) |
| | |

**Metric Standard**

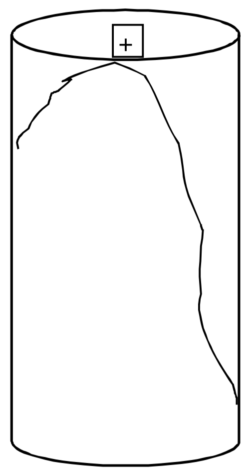
| **Sigma 3** (MPa) | **Sigma 1** (MPa) | |
|---|---|---|
| 20.69 | 277.6 | **Peak** |
| #VALUE! | 0.0 | **Residuals** |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |
| #VALUE! | 0.0 | |

Worksheet

Mode of Failure :

| | |
|---|---|
| Fracture | |
| Intact | XX |
| Both | |

Pre-Failure Sketch



Post-Failure

| | | | |
|---|---|---|---|
| Dia. 1 | 1.768 | Ht. 1 | 3.838 |
| Dia. 2 | 1.768 | Ht. 2 | 3.834 |
| Dia. 3 | 1.767 | Ht. 3 | 3.832 |
| Dia. 4 | 1.771 | Ht. 4 | 3.837 |
| Dia. 5 | 1.765 | Weight (gm) | 414.51 |
| Dia. 6 | 1.770 | Sample # | **04-004T** |

| **Sigma 3** (psi) | **Fail Load** gage (lbs) |
|---|---|
| 3,000 | 98,800 |
| | 0 |
| | 0 |
| | 0 |
| | 0 |

# Direct Shear Testing

# *University of Arizona*

*Geomechanical Laboratory*

*Tucson, Arizona USA*

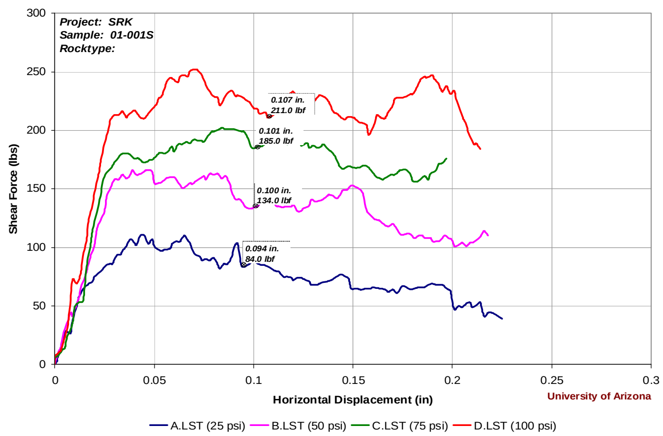
| | | | | |
|---|---|---|---|---|
| *Date* | 05/06/09 | Area & Load Data for SSDS | *Project #* | 2CM022 |
| *Technician* | D.STREETER | | *Client* | SRK |

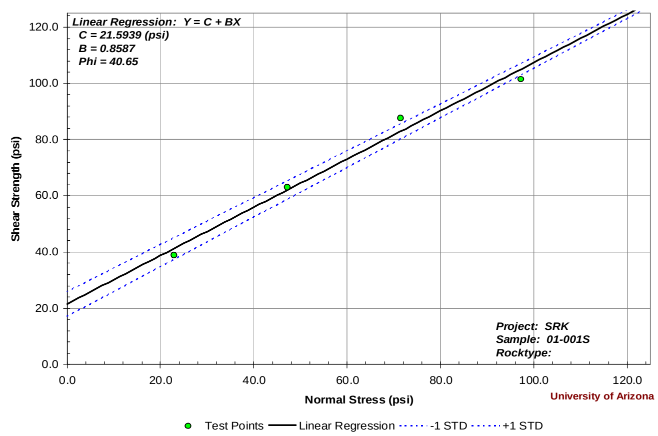
| *Sample Data* | | |
|---|---|---|
| *Location* | | |
| *Sample #* | **01-001S** | |
| *Rock Type* | | |
| *Drill Hole* | C09-01 | |
| *Depth* | 49.87 | |
| *Test Data* | | |
| *Shear Plane* | Joint | |
| *Surface Prep* | Insitu | |
| *Shape* | Digitized/Circular | |
| *Test Speed* | 0.025 | *in/min* |
| *Area* | 2.365 | *$in^2$* |
| *Diameter* | 1.735 | *in* |
| *Radius* | 0.868 | *in* |
| *Tilt Correction* | 4.778 | *deg* |

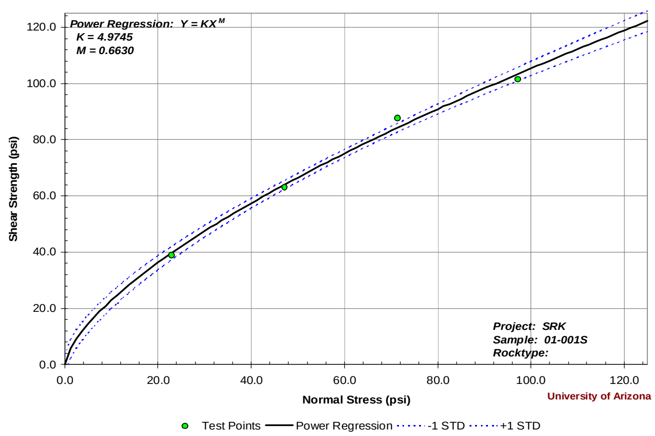
| *Trace Information* | | | |
|---|---|---|---|
| *Normal Stress (psi)* | *Normal Load (Lbs) / (kgs)* | | *Filename (.dat)* |
| 25 | 59 | 26.8 | 01-001S A |
| 50 | 118 | 53.6 | 01-001S B |
| 75 | 177 | 80.5 | 01-001S C |
| 100 | 237 | 107.3 | 01-001S D |

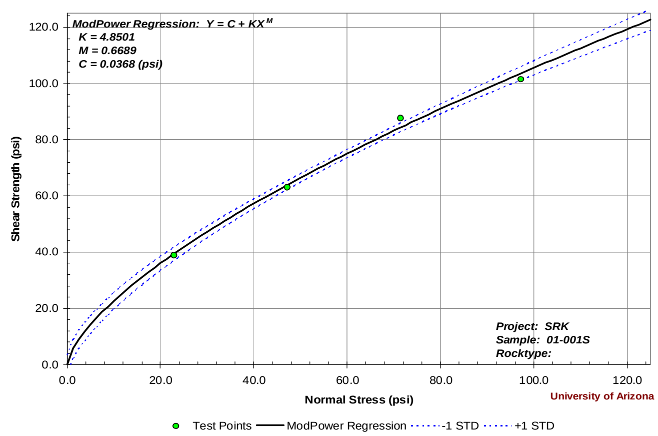
| *Results* | | |
|---|---|---|
| *Linear: Y = BX + C* | | |
| *Friction Angle* | 40.65 | *deg* |
| *Cohesion* | 21.59 | *psi* |
| *Power: $Y = KX^M$* | | |
| *K* | 4.9745 | *(for X in psi)* |
| *M* | 0.6630 | |
| *Modified Power: $Y = KX^M + C$* | | |
| *K* | 4.8501 | *(for X in psi)* |
| *M* | 0.6689 | |
| *C* | 0.0368 | *psi* |

*Plot of Raw Trace Data*




Notes :

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# *University of Arizona*

*Geomechanical Laboratory*

*Tucson, Arizona USA*

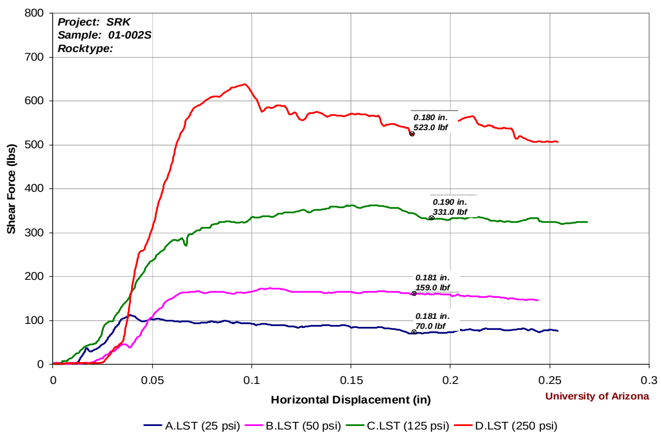
| *Date* | 05/05/09 | Area & Load Data for SSDS | *Project #* | 2CM022 |
|---|---|---|---|---|
| *Technician* | D.STREETER | | *Client* | SRK |

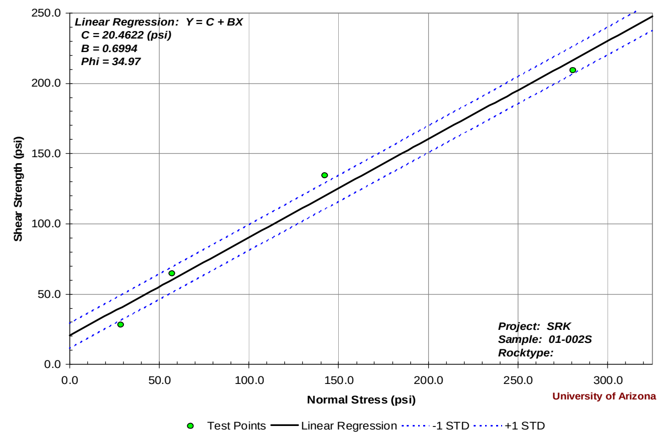
| *Sample Data* | | |
|---|---|---|
| *Location* | | |
| *Sample #* | **01-002S** | |
| *Rock Type* | | |
| *Drill Hole* | C09-01 | |
| *Depth* | 103 | |
| *Test Data* | | |
| *Shear Plane* | Joint | |
| *Surface Prep* | Insitu | |
| *Shape* | Area/Circular | |
| *Test Speed* | 0.025 | *in/min* |
| *Area* | 2.596 | *in*$^2$ |
| *Diameter* | 1.818 | *in* |
| *Radius* | 0.909 | *in* |
| *Tilt Correction* | -2.066 | *deg* |

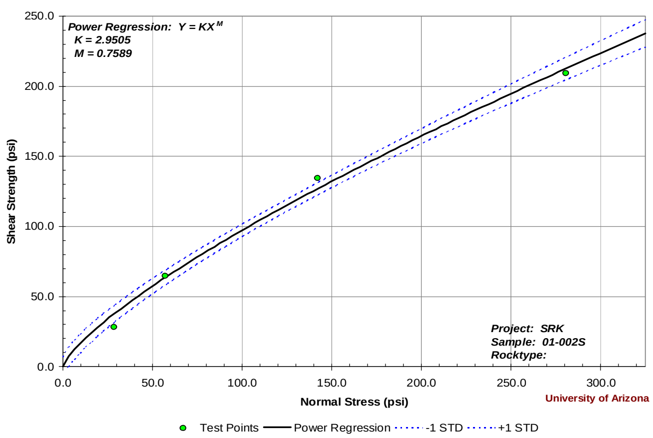
| *Trace Information* | | | |
|---|---|---|---|
| *Normal Stress (psi)* | *Normal Load (Lbs) / (kgs)* | | *Filename (.dat)* |
| 25 | 65 | 29.4 | 01-002S A |
| 50 | 130 | 58.9 | 01-002S B |
| 125 | 325 | 147.2 | 01-002S C |
| 250 | 649 | 294.4 | 01-002S D |

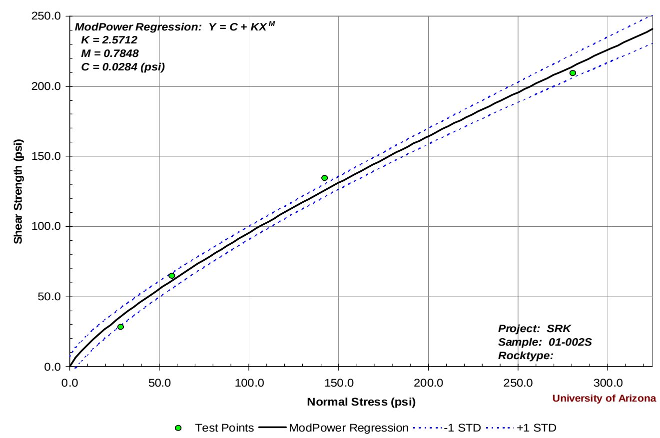
| *Results* | | |
|---|---|---|
| *Linear: Y = BX + C* | | |
| *Friction Angle* | 34.97 | *deg* |
| *Cohesion* | 20.46 | *psi* |
| *Power: Y = KX$^M$* | | |
| *K* | 2.9505 | *(for X in psi)* |
| *M* | 0.7589 | |
| *Modified Power: Y = KX$^M$ + C* | | |
| *K* | 2.5712 | *(for X in psi)* |
| *M* | 0.7848 | |
| *C* | 0.0284 | *psi* |

*Plot of Raw Trace Data*




Notes :

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**Sample: 01-002S**

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*Tucson, Arizona USA*

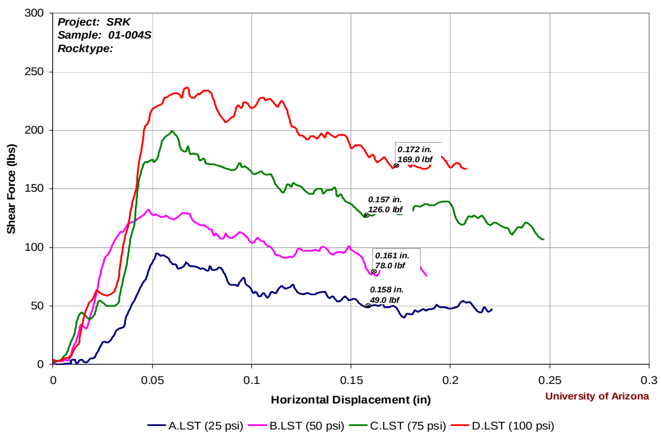
| | | | | |
|---|---|---|---|---|
| *Date* | 05/06/09 | Area & Load Data for SSDS | *Project #* | 2CM022 |
| *Technician* | D.STREETER | | *Client* | SRK |

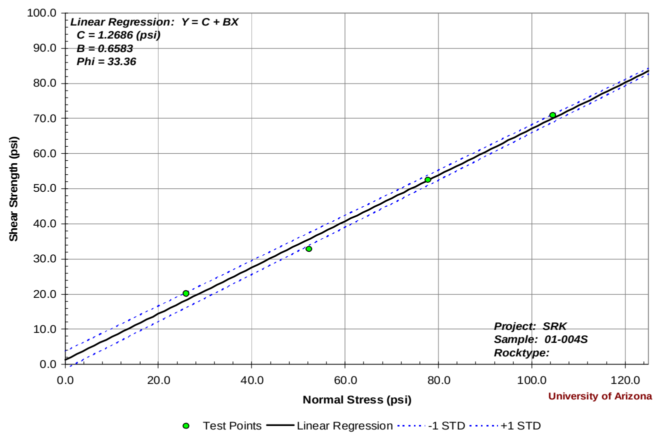
| *Sample Data* | | |
|---|---|---|
| *Location* | | |
| *Sample #* | **01-004S** | |
| *Rock Type* | | |
| *Drill Hole* | C09-01 | |
| *Depth* | 212.15 | |
| *Test Data* | | |
| *Shear Plane* | Joint | |
| *Surface Prep* | Insitu | |
| *Shape* | Digitized/Circular | |
| *Test Speed* | 0.025 | *in/min* |
| *Area* | 2.835 | $in^2$ |
| *Diameter* | 1.900 | *in* |
| *Radius* | 0.950 | *in* |
| *Tilt Correction* | 3.425 | *deg* |

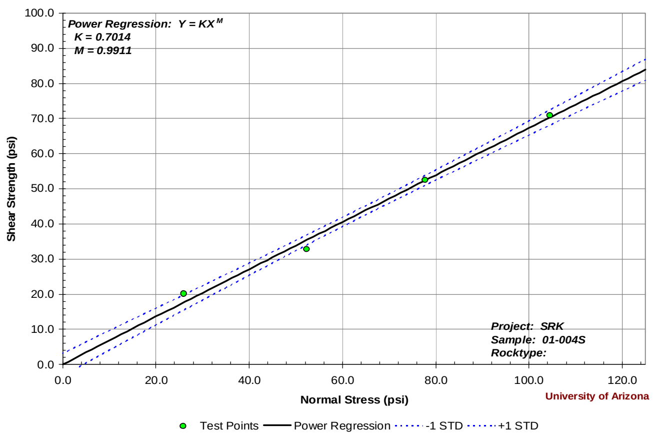
| *Trace Information* | | | |
|---|---|---|---|
| *Normal Stress (psi)* | *Normal Load (Lbs) / (kgs)* | | *Filename (.dat)* |
| 25 | 71 | 32.1 | 01-004S A |
| 50 | 142 | 64.3 | 01-004S B |
| 75 | 213 | 96.4 | 01-004S C |
| 100 | 284 | 128.6 | 01-004S D |

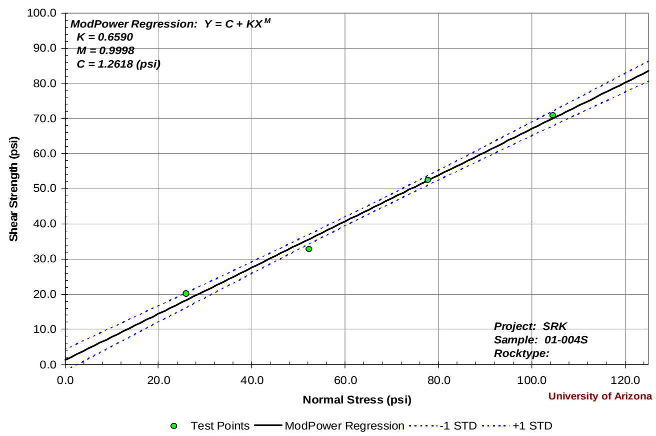
| *Results* | | |
|---|---|---|
| *Linear: Y = BX + C* | | |
| *Friction Angle* | 33.36 | *deg* |
| *Cohesion* | 1.27 | *psi* |
| *Power: $Y = KX^M$* | | |
| *K* | 0.7014 | *(for X in psi)* |
| *M* | 0.9911 | |
| *Modified Power: $Y = KX^M + C$* | | |
| *K* | 0.6590 | *(for X in psi)* |
| *M* | 0.9998 | |
| *C* | 1.2618 | *psi* |

*Plot of Raw Trace Data*




Notes :

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*Tucson, Arizona USA*

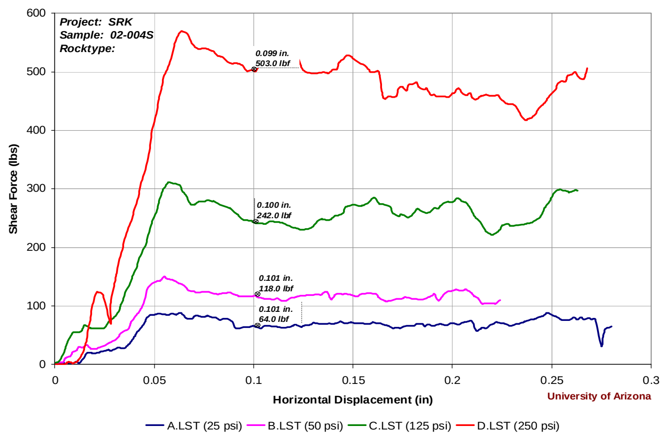
| | | | | |
|---|---|---|---|---|
| *Date* | 05/06/09 | Area & Load Data for SSDS | *Project #* | 2CM022 |
| *Technician* | D.STREETER | | *Client* | SRK |

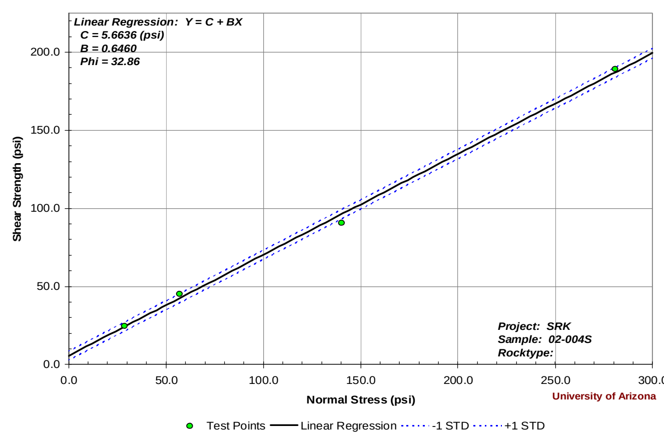
| *Sample Data* | | |
|---|---|---|
| *Location* | | |
| *Sample #* | **02-004S** | |
| *Rock Type* | | |
| *Drill Hole* | C09-02 | |
| *Depth* | 211.14 | |
| *Test Data* | | |
| *Shear Plane* | Joint | |
| *Surface Prep* | Insitu | |
| *Shape* | Digitized/Circular | |
| *Test Speed* | 0.025 | *in/min* |
| *Area* | 2.440 | *in$^2$* |
| *Diameter* | 1.763 | *in* |
| *Radius* | 0.881 | *in* |
| *Tilt Correction* | -5.294 | *deg* |

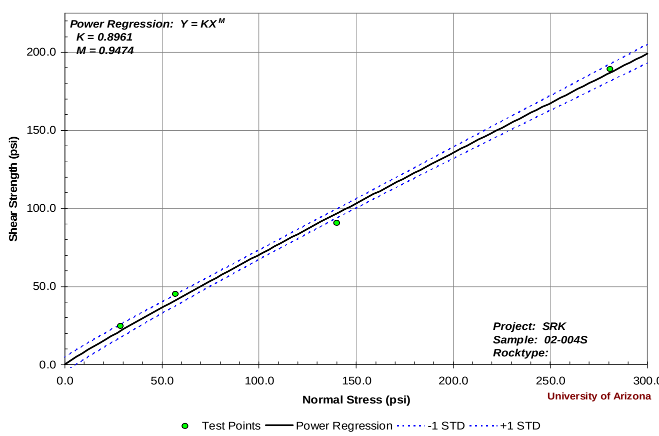
| *Trace Information* | | | |
|---|---|---|---|
| *Normal Stress (psi)* | *Normal Load (Lbs) / (kgs)* | | *Filename (.dat)* |
| 25 | 61 | 27.7 | 02-004S A |
| 50 | 122 | 55.3 | 02-004S B |
| 125 | 305 | 138.3 | 02-004S C |
| 250 | 610 | 276.7 | 02-004S D |
| | | | |
| | | | |
| | | | |
| | | | |

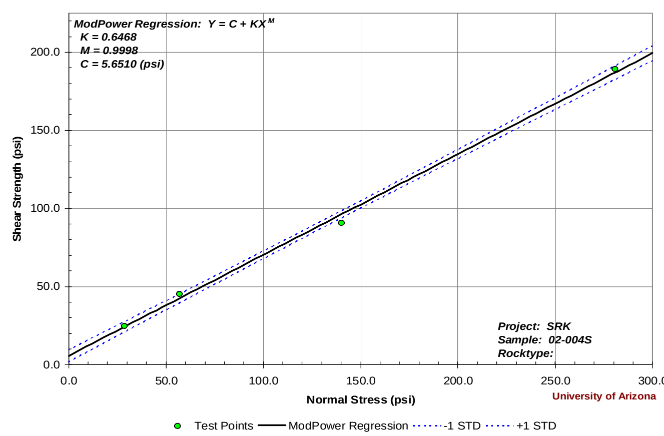
| *Results* | | |
|---|---|---|
| *Linear: Y = BX + C* | | |
| *Friction Angle* | 32.86 | *deg* |
| *Cohesion* | 5.66 | *psi* |
| *Power: $Y = KX^M$* | | |
| *K* | 0.8961 | *(for X in psi)* |
| *M* | 0.9474 | |
| *Modified Power: $Y = KX^M + C$* | | |
| *K* | 0.6468 | *(for X in psi)* |
| *M* | 0.9998 | |
| *C* | 5.6510 | *psi* |

*Plot of Raw Trace Data*




Notes :

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**Sample: 02-004S**

**University of Arizona**

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*Tucson, Arizona USA*

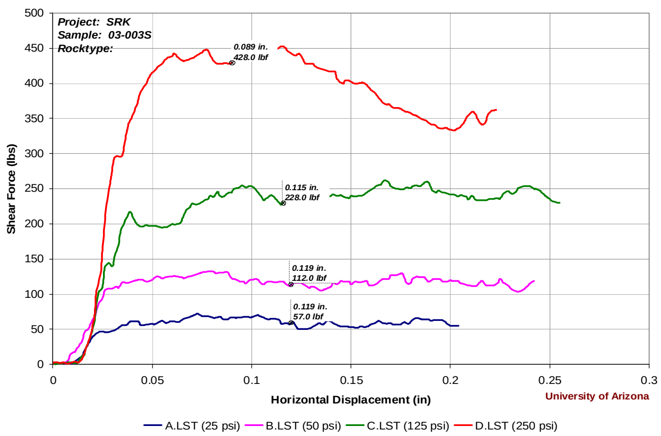
| | | | | |
|---|---|---|---|---|
| Date | 05/05/09 | Area & Load Data for SSDS | Project # | 2CM022 |
| Technician | D.STREETER | | Client | SRK |

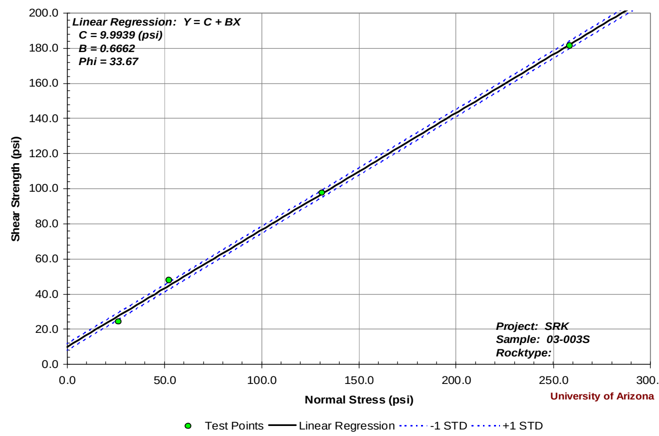
| Sample Data | | |
|---|---|---|
| Location | | |
| Sample # | **03-003S** | |
| Rock Type | | |
| Drill Hole | C09-03 | |
| Depth | 162.55 | |
| Test Data | | |
| Shear Plane | Joint | |
| Surface Prep | Insitu | |
| Shape | Digitized/Circular | |
| Test Speed | 0.025 | in/min |
| Area | 2.501 | $in^2$ |
| Diameter | 1.784 | in |
| Radius | 0.892 | in |
| Tilt Correction | 0.687 | deg |

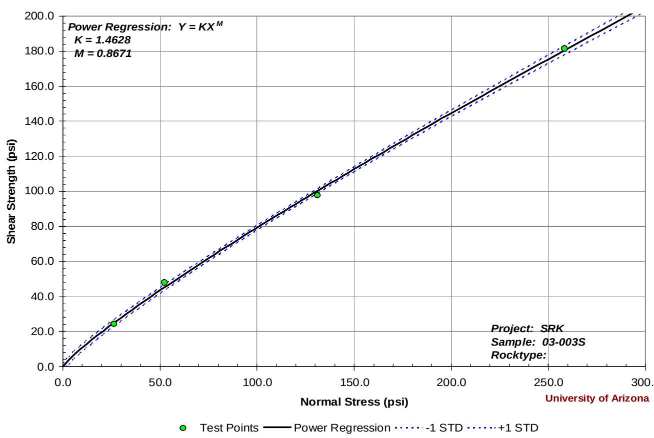
| Trace Information | | | |
|---|---|---|---|
| Normal Stress (psi) | Normal Load (Lbs) / (kgs) | | Filename (.dat) |
| 25 | 63 | 28.4 | 03-003S A |
| 50 | 125 | 56.7 | 03-003S B |
| 125 | 313 | 141.8 | 03-003S C |
| 250 | 625 | 283.6 | 03-003S D |

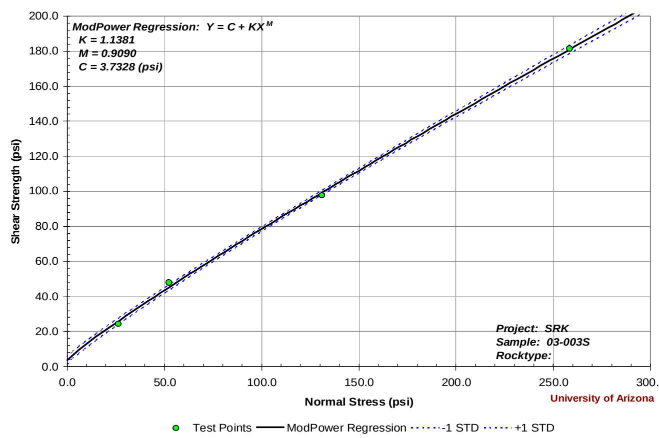
| Results | | |
|---|---|---|
| Linear: Y = BX + C | | |
| Friction Angle | 33.67 | deg |
| Cohesion | 9.99 | psi |
| Power: $Y = KX^M$ | | |
| K | 1.4628 | (for X in psi) |
| M | 0.8671 | |
| Modified Power: $Y = KX^M + C$ | | |
| K | 1.1381 | (for X in psi) |
| M | 0.9090 | |
| C | 3.7328 | psi |

## Plot of Raw Trace Data




Notes :

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**Sample: 03-003S**

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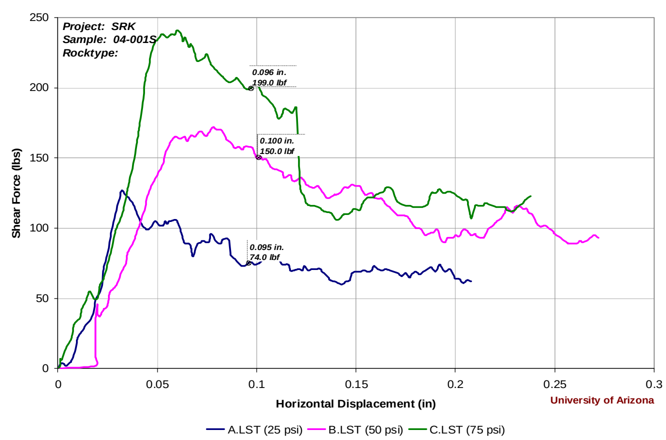
| | | | | |
|---|---|---|---|---|
| *Date* | 05/05/09 | Area & Load Data for SSDS | *Project #* | 2CM022 |
| *Technician* | D.STREETER | | *Client* | SRK |

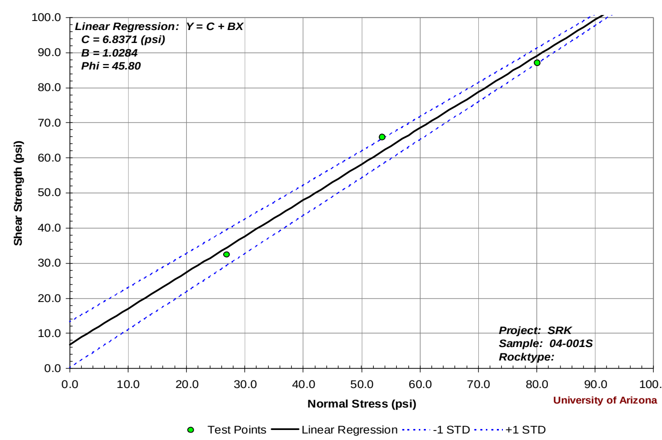
| *Sample Data* | | |
|---|---|---|
| *Location* | | |
| *Sample #* | **04-001S** | |
| *Rock Type* | | |
| *Drill Hole* | C09-04 | |
| *Depth* | 52.02 | |
| *Test Data* | | |
| *Shear Plane* | Joint | |
| *Surface Prep* | Insitu | |
| *Shape* | Digitized/Circular | |
| *Test Speed* | 0.025 | *in/min* |
| *Area* | 2.347 | *$in^2$* |
| *Diameter* | 1.729 | *in* |
| *Radius* | 0.864 | *in* |
| *Tilt Correction* | -1.082 | *deg* |

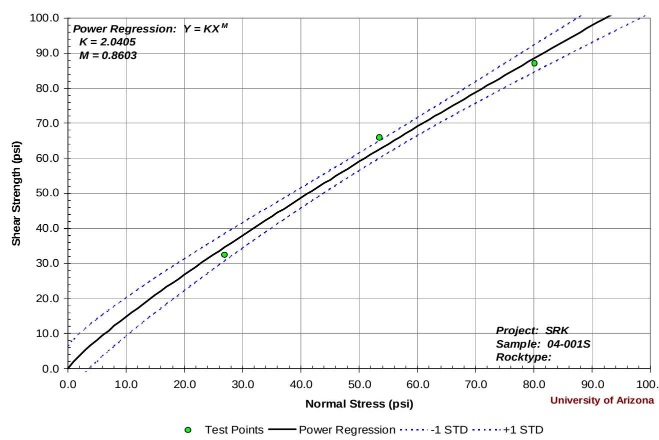
| *Trace Information* | | | |
|---|---|---|---|
| *Normal Stress (psi)* | *Normal Load (Lbs) / (kgs)* | | *Filename (.dat)* |
| 25 | 59 | 26.6 | 04-001S A |
| 50 | 117 | 53.2 | 04-001S B |
| 75 | 176 | 79.8 | 04-001S C |
| 100 | 235 | 106.5 | 04-001S D |

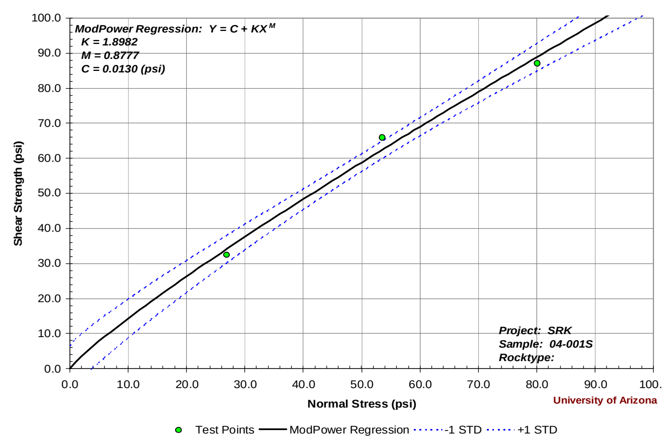
| *Results* | | |
|---|---|---|
| *Linear: Y = BX + C* | | |
| *Friction Angle* | 45.80 | *deg* |
| *Cohesion* | 6.84 | *psi* |
| *Power: $Y = KX^M$* | | |
| *K* | 2.0405 | *(for X in psi)* |
| *M* | 0.8603 | |
| *Modified Power: $Y = KX^M + C$* | | |
| *K* | 1.8982 | *(for X in psi)* |
| *M* | 0.8777 | |
| *C* | 0.0130 | *psi* |

*Plot of Raw Trace Data*




Notes :

Trace four was not plotted as sample broke during end of trace three

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*Tucson, Arizona USA*

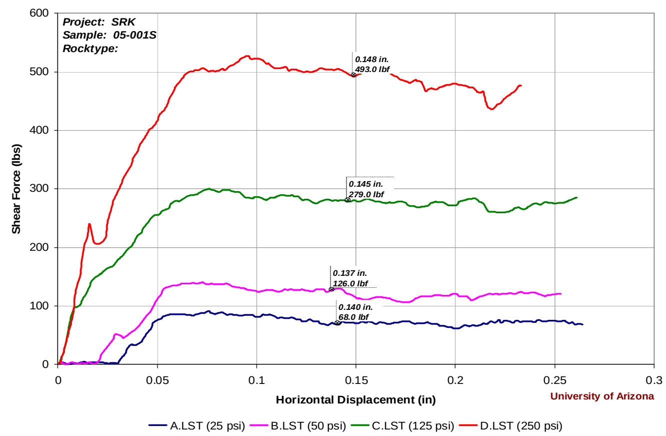
| | | | | |
|---|---|---|---|---|
| *Date* | 05/05/09 | Area & Load Data for SSDS | *Project #* | 2CM022 |
| *Technician* | D.STREETER | | *Client* | SRK |

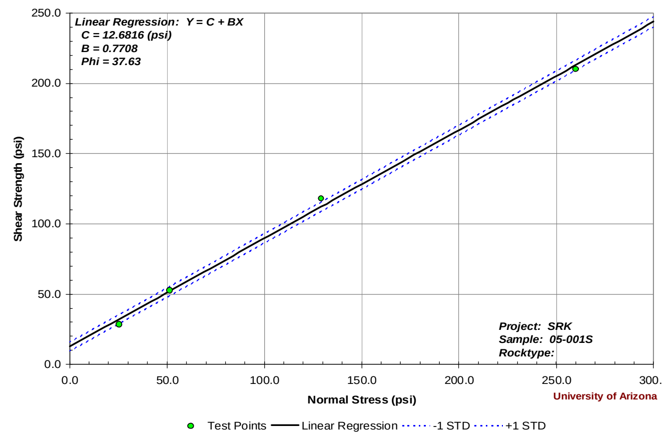
| *Sample Data* | | |
|---|---|---|
| *Location* | | |
| *Sample #* | **05-001S** | |
| *Rock Type* | | |
| *Drill Hole* | C09-05 | |
| *Depth* | 61.07 | |
| *Test Data* | | |
| *Shear Plane* | Joint | |
| *Surface Prep* | Insitu | |
| *Shape* | Digitized/Circular | |
| *Test Speed* | 0.025 | *in/min* |
| *Area* | 2.654 | *$in^2$* |
| *Diameter* | 1.838 | *in* |
| *Radius* | 0.919 | *in* |
| *Tilt Correction* | 2.443 | *deg* |

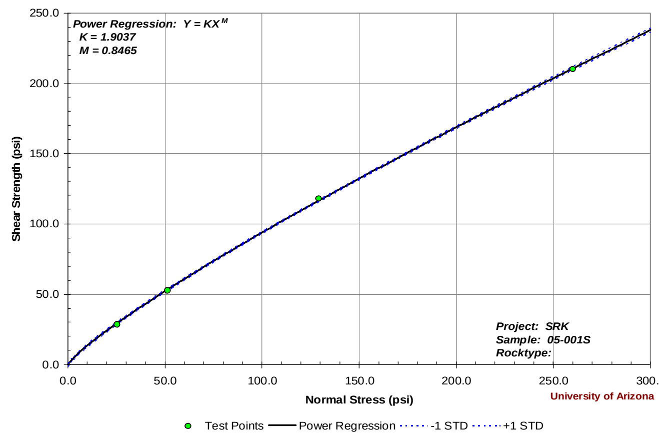
| *Trace Information* | | | |
|---|---|---|---|
| *Normal Stress (psi)* | *Normal Load (Lbs) / (kgs)* | | *Filename (.dat)* |
| 25 | 66 | 30.1 | 05-001S A |
| 50 | 133 | 60.2 | 05-001S B |
| 125 | 332 | 150.5 | 05-001S C |
| 250 | 664 | 301.0 | 05-001S D |
| | | | |
| | | | |
| | | | |
| | | | |

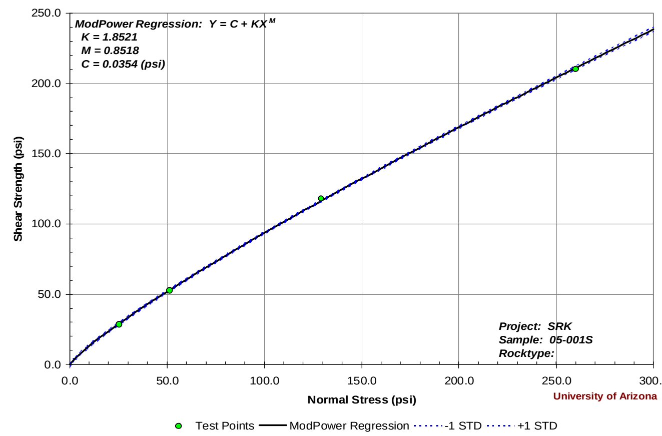
| *Results* | | |
|---|---|---|
| *Linear: Y = BX + C* | | |
| *Friction Angle* | 37.63 | *deg* |
| *Cohesion* | 12.68 | *psi* |
| *Power: $Y = KX^M$* | | |
| *K* | 1.9037 | *(for X in psi)* |
| *M* | 0.8465 | |
| *Modified Power: $Y = KX^M + C$* | | |
| *K* | 1.8521 | *(for X in psi)* |
| *M* | 0.8518 | |
| *C* | 0.0354 | *psi* |

*Plot of Raw Trace Data*




Notes :

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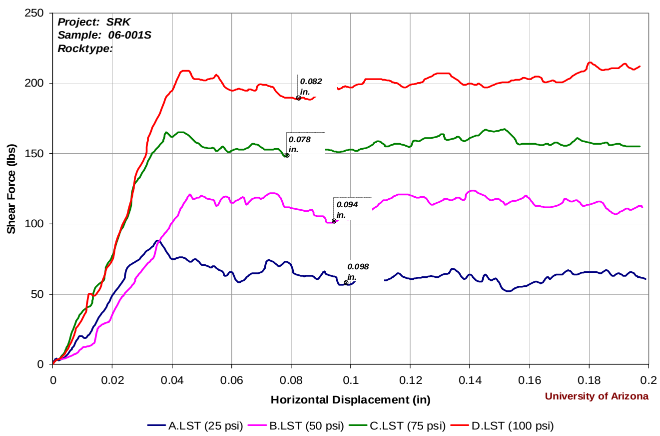
| | | | | |
|---|---|---|---|---|
| *Date* | 05/06/09 | Area & Load Data for SSDS | *Project #* | 2CM022 |
| *Technician* | D.STREETER | | *Client* | SRK |

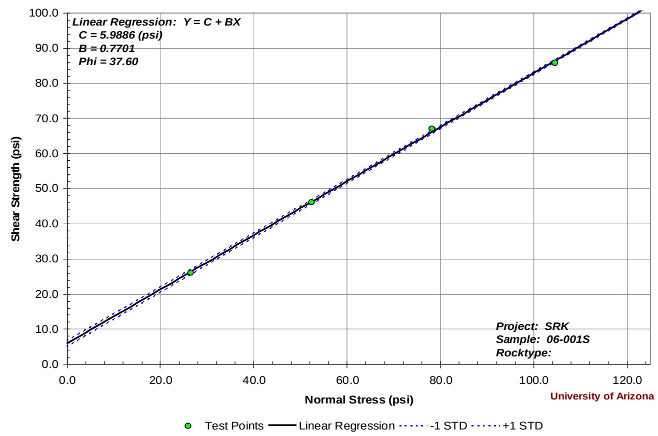
| *Sample Data* | | |
|---|---|---|
| *Location* | | |
| *Sample #* | **06-001S** | |
| *Rock Type* | | |
| *Drill Hole* | C09-06 | |
| *Depth* | 51.94 | |
| *Test Data* | | |
| *Shear Plane* | Joint | |
| *Surface Prep* | Insitu | |
| *Shape* | Digitized/Circular | |
| *Test Speed* | 0.025 | *in/min* |
| *Area* | 2.269 | *in²* |
| *Diameter* | 1.700 | *in* |
| *Radius* | 0.850 | *in* |
| *Tilt Correction* | -0.374 | *deg* |

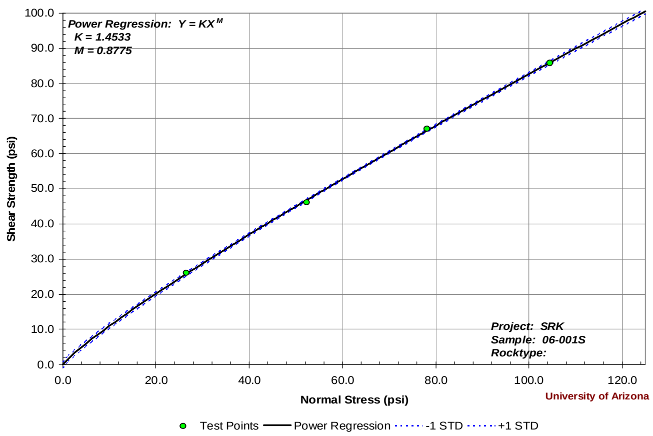
| *Trace Information* | | | | |
|---|---|---|---|---|
| *Normal Stress (psi)* | *Normal Load (Lbs) / (kgs)* | | *Filename (.dat)* | |
| 25 | 57 | 25.7 | 06-001S | A |
| 50 | 113 | 51.5 | 06-001S | B |
| 75 | 170 | 77.2 | 06-001S | C |
| 100 | 227 | 102.9 | 06-001S | D |

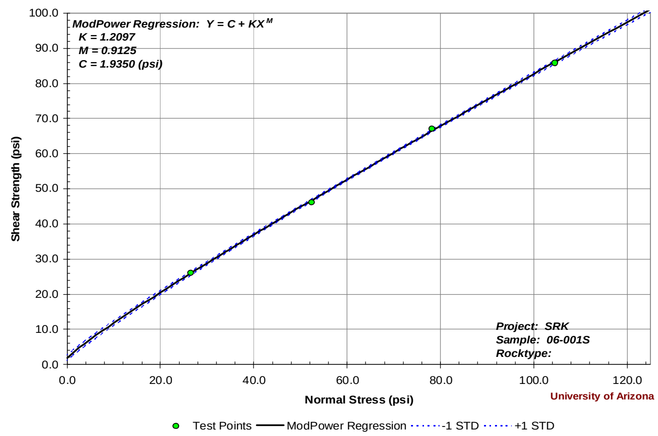
| *Results* | | |
|---|---|---|
| *Linear: Y = BX + C* | | |
| *Friction Angle* | 37.60 | *deg* |
| *Cohesion* | 5.99 | *psi* |
| *Power: $Y = KX^M$* | | |
| *K* | 1.4533 | *(for X in psi)* |
| *M* | 0.8775 | |
| *Modified Power: $Y = KX^M + C$* | | |
| *K* | 1.2097 | *(for X in psi)* |
| *M* | 0.9125 | |
| *C* | 1.9350 | *psi* |

*Plot of Raw Trace Data*




Notes :

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# *University of Arizona*

*Geomechanical Laboratory*

*Tucson, Arizona USA*

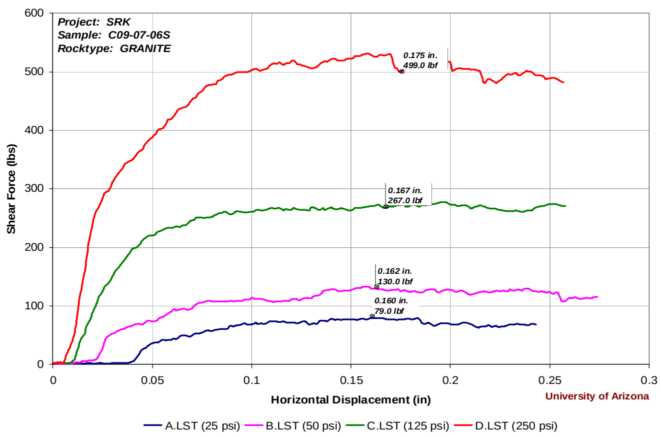
| | | | | |
|---|---|---|---|---|
| *Date* | 07/21/09 | Area & Load Data for SSDS | *Project #* | 2CM022_006 |
| *Technician* | D.STREETER | | *Client* | SRK |

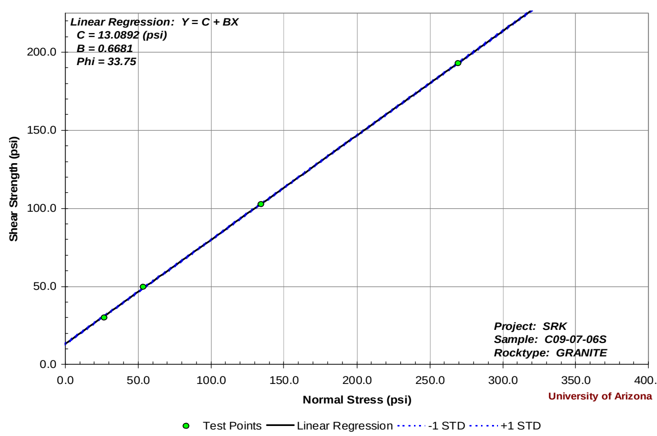
| *Sample Data* | | |
|---|---|---|
| *Location* | MINTO | |
| *Sample #* | **C09-07-06S** | |
| *Rock Type* | GRANITE | |
| *Drill Hole* | C09-07 | |
| *Depth* | 137.2-137.3 | |
| *Test Data* | | |
| *Shear Plane* | Joint | |
| *Surface Prep* | Insitu | |
| *Shape* | Digitized/Circular | |
| *Test Speed* | 0.025 | *in/min* |
| *Area* | 2.877 | *in$^2$* |
| *Diameter* | 1.914 | *in* |
| *Radius* | 0.957 | *in* |
| *Tilt Correction* | 0.865 | *deg* |

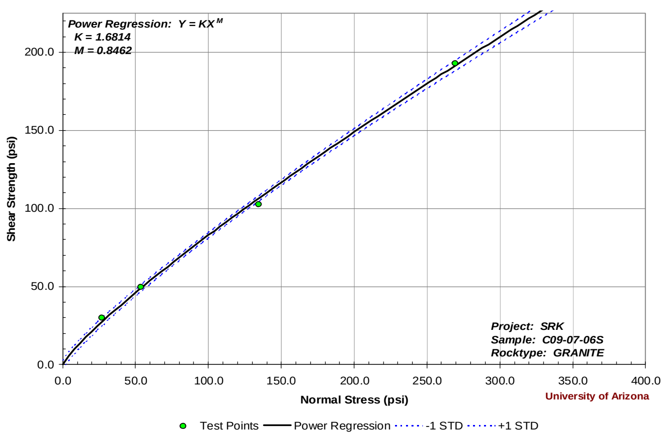
| *Trace Information* | | | | |
|---|---|---|---|---|
| *Normal Stress (psi)* | *Normal Load (Lbs) / (kgs)* | | *Filename (.dat)* | |
| 25 | 72 | 32.6 | C09-07-06S | A |
| 50 | 144 | 65.2 | C09-07-06S | B |
| 125 | 360 | 163.1 | C09-07-06S | C |
| 250 | 719 | 326.2 | C09-07-06S | D |

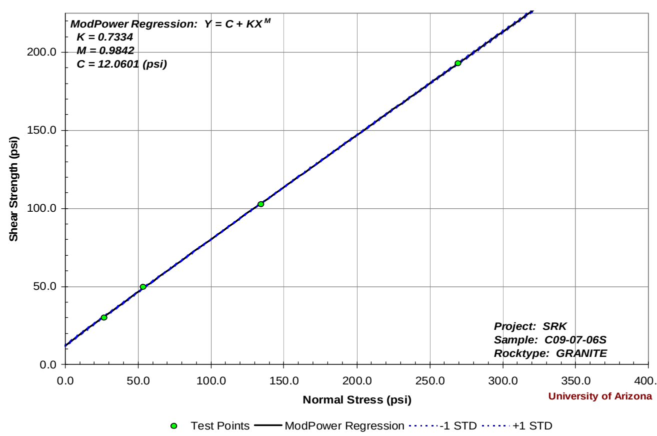
| *Results* | | |
|---|---|---|
| *Linear: Y = BX + C* | | |
| *Friction Angle* | 33.75 | *deg* |
| *Cohesion* | 13.09 | *psi* |
| *Power: Y = KX$^M$* | | |
| *K* | 1.6814 | *(for X in psi)* |
| *M* | 0.8462 | |
| *Modified Power: Y = KX$^M$ + C* | | |
| *K* | 0.7334 | *(for X in psi)* |
| *M* | 0.9842 | |
| *C* | 12.0601 | *psi* |

*Plot of Raw Trace Data*




Notes :

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# University of Arizona

*Geomechanical Laboratory*

*Tucson, Arizona USA*

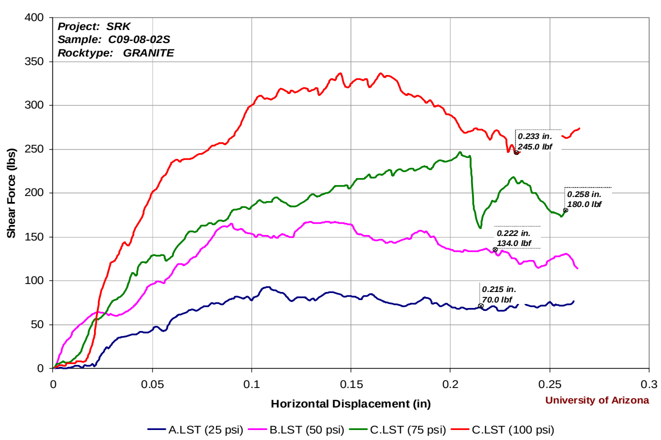
| | | | | |
|---|---|---|---|---|
| *Date* | 07/21/09 | Area & Load Data for SSDS | *Project #* | 2CM022_006 |
| *Technician* | D.STREETER | | *Client* | SRK |

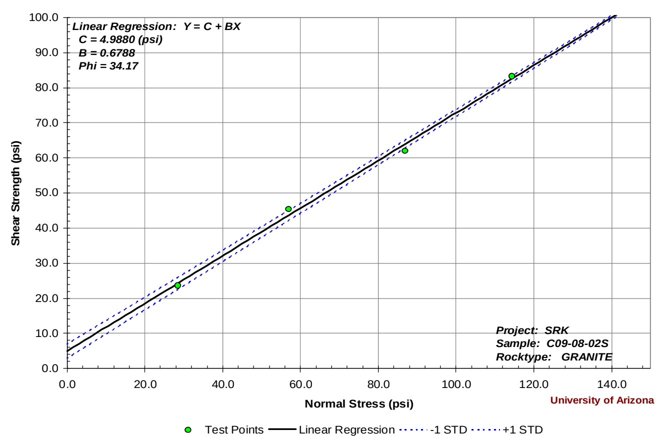
| *Sample Data* | | |
|---|---|---|
| *Location* | MINTO | |
| *Sample #* | **C09-08-02S** | |
| *Rock Type* | GRANITE | |
| *Drill Hole* | C09-08 | |
| *Depth* | 54.9-55.07 | |
| *Test Data* | | |
| *Shear Plane* | Joint | |
| *Surface Prep* | Insitu | |
| *Shape* | Digitized/Circular | |
| *Test Speed* | 0.025 | *in/min* |
| *Area* | 3.203 | *$in^2$* |
| *Diameter* | 2.019 | *in* |
| *Radius* | 1.010 | *in* |
| *Tilt Correction* | -1.338 | *deg* |

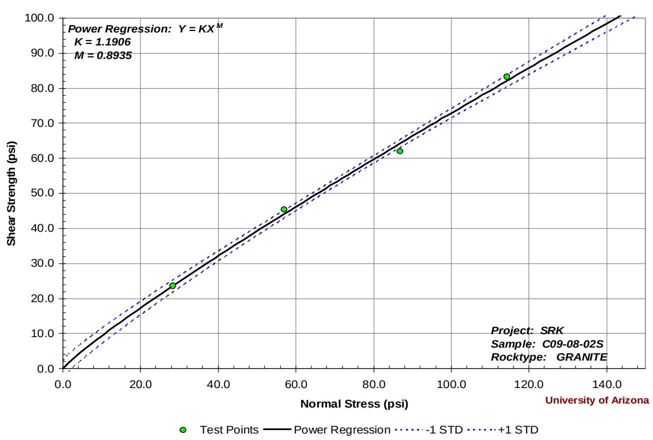
| *Trace Information* | | | | |
|---|---|---|---|---|
| *Normal Stress (psi)* | *Normal Load (Lbs) / (kgs)* | | *Filename (.dat)* | |
| 25 | 80 | 36.3 | C09-08-02S | A |
| 50 | 160 | 72.6 | C09-08-02S | B |
| 75 | 240 | 109.0 | C09-08-02S | C |
| 100 | 320 | 145.3 | C09-08-02S | D |

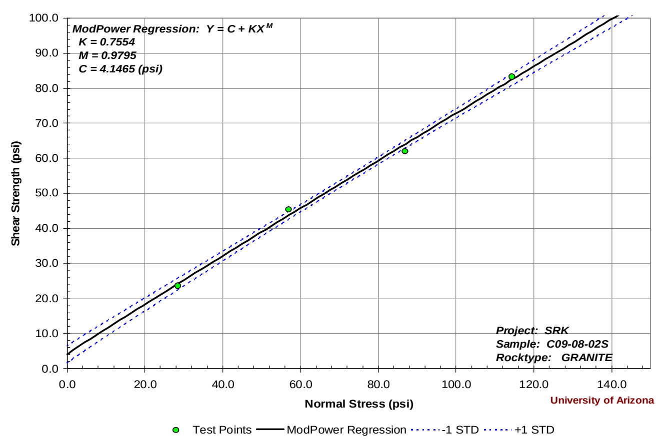
| *Results* | | |
|---|---|---|
| *Linear: Y = BX + C* | | |
| *Friction Angle* | 34.17 | *deg* |
| *Cohesion* | 4.99 | *psi* |
| *Power: $Y = KX^M$* | | |
| *K* | 1.1906 | *(for X in psi)* |
| *M* | 0.8935 | |
| *Modified Power: $Y = KX^M + C$* | | |
| *K* | 0.7554 | *(for X in psi)* |
| *M* | 0.9795 | |
| *C* | 4.1465 | *psi* |

*Plot of Raw Trace Data*




Notes :

Large Drop in shear load for trace three was due to plaster on top mold cracking and relieving stress.

# Brazilian Disk Tension Testing

# CALL & NICHOLAS, INC.

*GEOMECHANICAL LABORATORY*

*TUCSON, ARIZONA USA*

| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | **Brazilian Disk Test Results** | Client | SRK |
| Date | 5/4/2009 | | Location | |
| Technician | D.Streeter | | Sample # | **02-005B** |
| | | Sample # **02-005B** | Rock Type | GRANITE |

T psi **1,566** psi

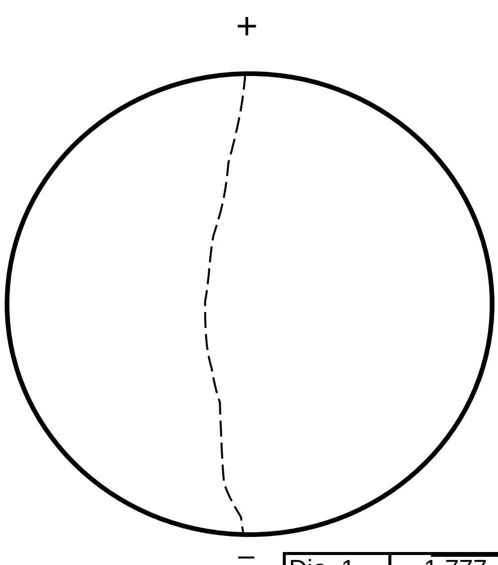
10.80 Mpa

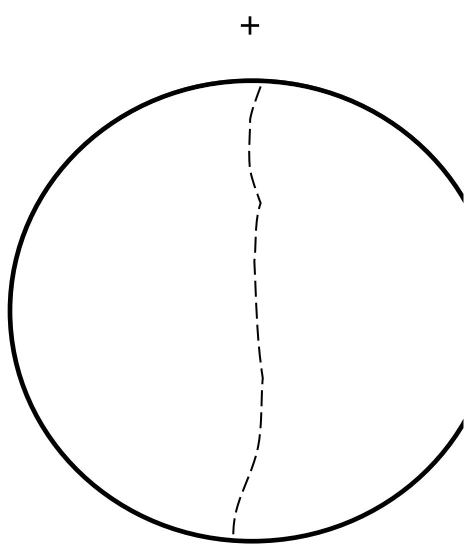
| | | | | |
|---|---|---|---|---|
| **Sample Data :** | | | T psi | 1,566 (psi) |
| Sample # : | **02-005B** | T= Indirect tensile strength | | 10.8 Mpa |
| Rock Type: | GRANITE | | | |
| Hole # : | C09-02 | | **Test Data:** | |
| Depth : | 150.1 | | Disp. Rate : | |
| Alterations: | | | Load Rate : | 54 (lbs/sec) |
| Diameter : | 1.776 (in) | | Gage Reading : | 4,620 (lbs) |
| Length: | 1.058 (in) | | | |

Worksheet

Pre-Failure Sketch w

Post-Failure Sketch





Pre-existing Weakness Plane

Post Failure Fracture

| | | | | | | |
|---|---|---|---|---|---|---|
| Dia. 1 | 1.777 | Ht. 1 | 1.061 | | | |
| Dia. 2 | 1.777 | Ht. 2 | 1.069 | **Fail Load** | 4620 | lbs Force |
| Dia. 3 | 1.773 | Ht. 3 | 1.045 | | | |

| | |
|---|---|
| Sample # | **02-005B** |

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*TUCSON, ARIZONA USA*

## Brazilian Disk Test Results

| | | | |
|---|---|---|---|
| Project # | 2CM022 | Client | SRK |
| Date | 5/4/2009 | Location | |
| Technician | D.Streeter | Sample # | **02-009B** |
| | | Rock Type | GRANITE |

Sample # **02-009B**

T psi **1,357** psi

9.36 Mpa

T= Indirect tensile strength

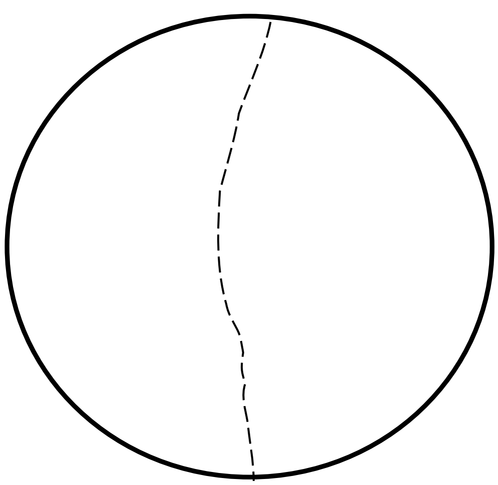
| Sample Data : | |
|---|---|
| Sample # : | **02-009B** |
| Rock Type: | GRANITE |
| Hole # : | C09-02 |
| Depth : | 271.9 |
| Alterations: | |
| Diameter : | 1.773 (in) |
| Length: | 1.022 (in) |

| | |
|---|---|
| T psi | 1,357 (psi) |
| | 9.4 Mpa |

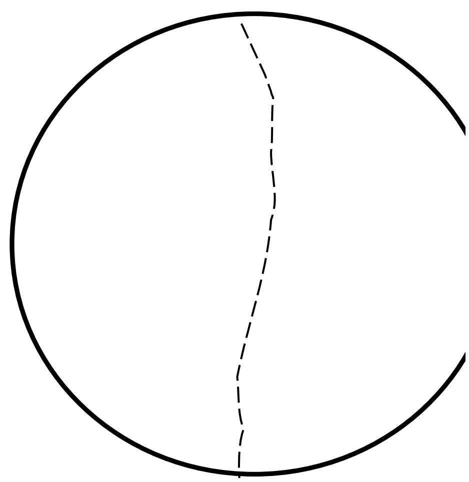
| Test Data: | |
|---|---|
| Disp. Rate : | |
| Load Rate : | 52 (lbs/sec) |
| Gage Reading : | 3,860 (lbs) |

Worksheet

Pre-Failure Sketch w




Post-Failure Sketch




Pre-existing Weakness Plane

Post Failure Fracture

| | | | |
|---|---|---|---|
| Dia. 1 | 1.775 | Ht. 1 | 1.028 |
| Dia. 2 | 1.774 | Ht. 2 | 1.026 |
| Dia. 3 | 1.772 | Ht. 3 | 1.012 |

**Fail Load** 3860 lbs Force

| | |
|---|---|
| Sample # | **02-009B** |

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | **Brazilian Disk Test Results** | Client | SRK |
| Date | 5/4/2009 | | Location | |
| Technician | D.Streeter | | Sample # | **03-007B** |
| | | Sample # **03-007B** | Rock Type | GRANITE |

T psi **1,107** psi

7.63 Mpa

| **Sample Data :** | |
|---|---|
| Sample # : | **03-007B** |
| Rock Type: | GRANITE |
| Hole # : | C09-03 |
| Depth : | 161.03 |
| Alterations: | |
| Diameter : | 1.773 (in) |
| Length: | 0.932 (in) |

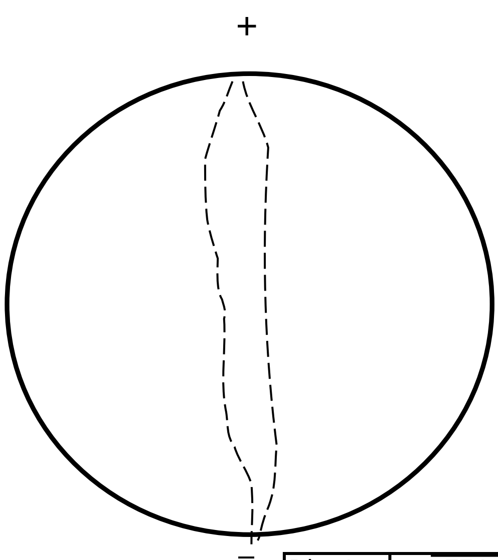
T= Indirect tensile strength

| | |
|---|---|
| T psi | 1,107 (psi) |
| | 7.6 Mpa |

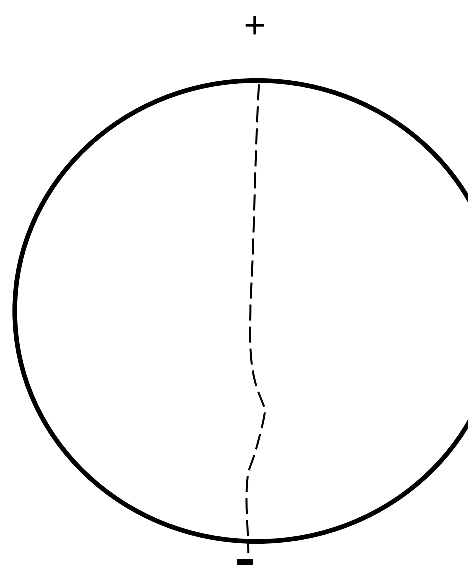
| **Test Data:** | |
|---|---|
| Disp. Rate : | |
| Load Rate : | 47 (lbs/sec) |
| Gage Reading : | 2,870 (lbs) |

Worksheet

Pre-Failure Sketch w




Post-Failure Sketch




Pre-existing Weakness Plane

Post Failure Fracture

| | | | |
|---|---|---|---|
| Dia. 1 | 1.774 | Ht. 1 | 0.934 |
| Dia. 2 | 1.773 | Ht. 2 | 0.928 |
| Dia. 3 | 1.772 | Ht. 3 | 0.933 |

**Fail Load** 2870 lbs Force

Sample # **03-007B**

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## Brazilian Disk Test Results

| | | | |
|---|---|---|---|
| Project # | 2CM022 | Client | SRK |
| Date | 5/4/2009 | Location | |
| Technician | D.Streeter | Sample # | **05-005B** |
| | | Rock Type | GRANITE |

Sample # **05-005B**

T psi **1,045** psi

7.20 Mpa

| Sample Data : | |
|---|---|
| Sample # : | **05-005B** |
| Rock Type: | GRANITE |
| Hole # : | C09-05 |
| Depth : | 150.11 |
| Alterations: | |
| Diameter : | 1.772 (in) |
| Length: | 0.922 (in) |

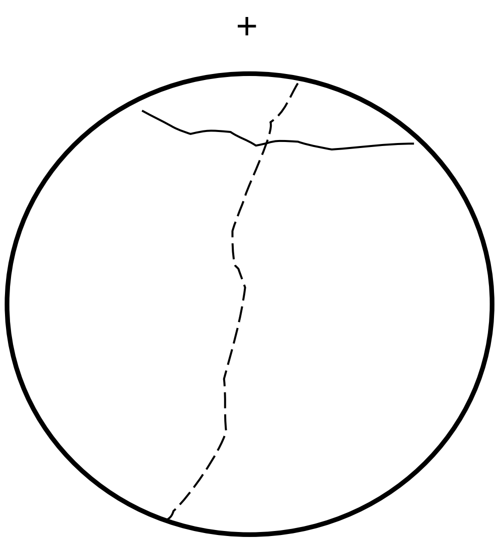
T= Indirect tensile strength

| | |
|---|---|
| T psi | 1,045 (psi) |
| | 7.2 Mpa |

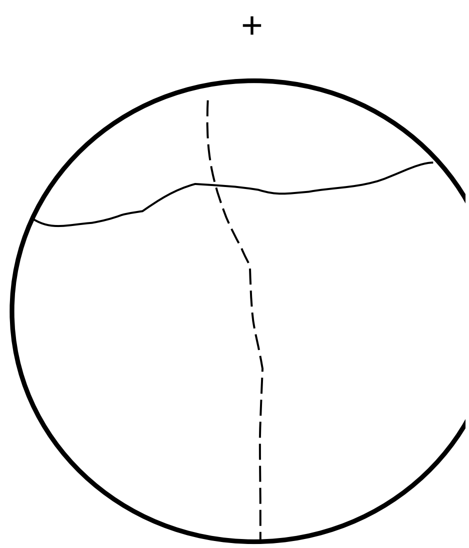
| Test Data: | |
|---|---|
| Disp. Rate : | |
| Load Rate : | 47 (lbs/sec) |
| Gage Reading : | 2,680 (lbs) |

Worksheet

Pre-Failure Sketch w



Post-Failure Sketch



Pre-existing Weakness Plane

Post Failure Fracture

| | | | |
|---|---|---|---|
| Dia. 1 | 1.775 | Ht. 1 | 0.927 |
| Dia. 2 | 1.770 | Ht. 2 | 0.925 |
| Dia. 3 | 1.772 | Ht. 3 | 0.915 |

**Fail Load** 2680 lbs Force

| | |
|---|---|
| Sample # | **05-005B** |

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| | | | | |
|---|---|---|---|---|
| Project # | 2CM022 | **Brazilian Disk Test Results** | Client | SRK |
| Date | 5/4/2009 | | Location | |
| Technician | D.Streeter | | Sample # | **06-001B** |
| | | Sample # **06-001B** | Rock Type | GRANITE |

T psi **1,291** psi

8.90 Mpa

| **Sample Data :** | |
|---|---|
| Sample # : | **06-001B** |
| Rock Type: | GRANITE |
| Hole # : | C09-06 |
| Depth : | 37.2 |
| Alterations: | |
| Diameter : | 1.772 (in) |
| Length: | 0.980 (in) |

T= Indirect tensile strength

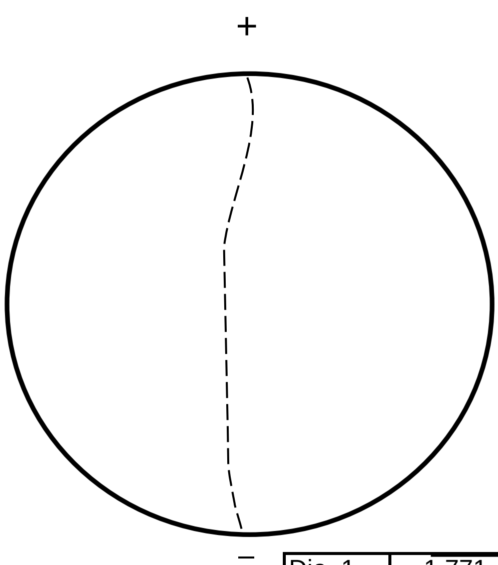
| | |
|---|---|
| T psi | 1,291 (psi) |
| | 8.9 Mpa |

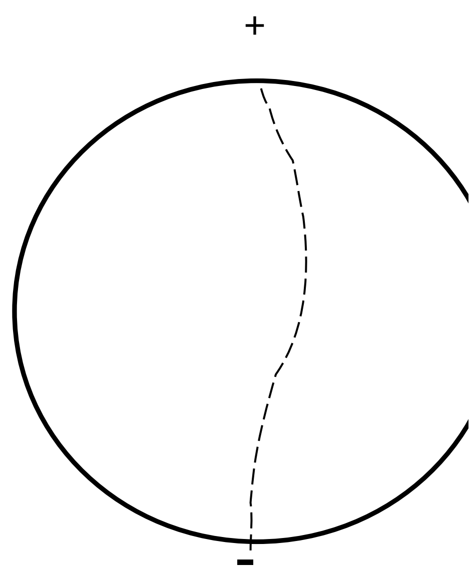
| **Test Data:** | |
|---|---|
| Disp. Rate : | |
| Load Rate : | 50 (lbs/sec) |
| Gage Reading : | 3,520 (lbs) |

Worksheet

Pre-Failure Sketch w

Post-Failure Sketch




Pre-existing Weakness Plane

Post Failure Fracture

| | | | |
|---|---|---|---|
| Dia. 1 | 1.771 | Ht. 1 | 0.986 |
| Dia. 2 | 1.772 | Ht. 2 | 0.980 |
| Dia. 3 | 1.773 | Ht. 3 | 0.975 |

**Fail Load** 3520 lbs Force

| | |
|---|---|
| Sample # | **06-001B** |

# Appendix C
# Environmental

| MineArea | Waste Class | SAMPLE | HOLE ID | FROM (m) | TO (m) |
|---|---|---|---|---|---|
| Ridgetop | Bulk Waste | RT_ABA-1 | 07SWC254 | 2.0 | 8.5 |
| Ridgetop | Bulk Waste | RT_ABA-2 | 07SWC254 | 20.0 | 27.2 |
| Ridgetop | Bulk Waste | RT_ABA-3 | 07SWC251 | 19.6 | 26.5 |
| Ridgetop | Bulk Waste | RT_ABA-6 | 07SWC251 | 66.1 | 72.1 |
| Ridgetop | Bulk Waste | RT_ABA-7 | 07SWC250 | 1.8 | 6.7 |
| Ridgetop | Bulk Waste | RT_ABA-8 | 07SWC250 | 14.5 | 20.5 |
| Ridgetop | Bulk Waste | RT_ABA-9 | 07SWC250 | 30.7 | 36.4 |
| Ridgetop | Bulk Waste | RT_ABA-10 | 07SWC246 | 1.5 | 6.8 |
| Ridgetop | Bulk Waste | RT_ABA-11 | 07SWC244 | 2.0 | 6.8 |
| Ridgetop | Bulk Waste | RT_ABA-12 | 07SWC183 | 5.9 | 10.9 |
| Ridgetop | Bulk Waste | RT_ABA-14 | 07SWC183 | 23.6 | 30.1 |
| Ridgetop | Bulk Waste | RT_ABA-15 | 07SWC182 | 4.9 | 12.2 |
| Ridgetop | Bulk Waste | RT_ABA-16 | 07SWC182 | 25.6 | 31.3 |
| Ridgetop | Bulk Waste | RT_ABA-17 | 07SWC182 | 32.8 | 38.7 |
| Ridgetop | Bulk Waste | RT_ABA-18 | 07SWC181 | 3.1 | 9.2 |
| Ridgetop | Bulk Waste | RT_ABA-19 | 07SWC181 | 9.2 | 14.7 |
| Ridgetop | Min Waste | RT_ABA-5 | 07SWC251 | 48.4 | 53.2 |
| Ridgetop | Min Waste | RT_ABA-20 | 07SWC181 | 14.7 | 20.6 |
| Ridgetop | Ore | RT_ABA-4 | 07SWC251 | 40.2 | 45.8 |
| Ridgetop | Ore | RT_ABA-13 | 07SWC183 | 15.9 | 21.1 |
| Minto North | Hanging wall | G0755001 | 09SWC393 | 6.1 | 11.9 |
| Minto North | Hanging wall | G0755002 | 09SWC393 | 12.3 | 18.3 |
| Minto North | Hanging wall | G0755003 | 09SWC393 | 31.0 | 37.0 |
| Minto North | Hanging wall | G0755004 | 09SWC393 | 45.5 | 51.5 |
| Minto North | Hanging wall | G0755006 | 09SWC404 | 10.8 | 16.8 |
| Minto North | Hanging wall | G0755007 | 09SWC404 | 34.6 | 40.6 |
| Minto North | Hanging wall | G0755008 | 09SWC404 | 52.2 | 58.2 |
| Minto North | Hanging wall | G0755010 | 09SWC408 | 21.0 | 27.0 |
| Minto North | Hanging wall | G0755011 | 09SWC408 | 46.9 | 52.9 |
| Minto North | Hanging wall | G0755012 | 09SWC408 | 65.8 | 71.8 |
| Minto North | Hanging wall | G0755014 | 09SWC 459 | 3.1 | 9.1 |
| Minto North | Hanging wall | G0755015 | 09SWC 459 | 30.0 | 36.0 |
| Minto North | Hanging wall | G0755016 | 09SWC 459 | 59.4 | 65.4 |
| Minto North | Hanging wall | G0755017 | 09SWC 459 | 75.4 | 81.4 |
| Minto North | Hanging wall | G0755019 | 09SWC465 | 9.1 | 15.1 |
| Minto North | Hanging wall | G0755020 | 09SWC465 | 30.0 | 36.0 |
| Minto North | Hanging wall | G0755021 | 09SWC465 | 49.9 | 55.9 |
| Minto North | Hanging wall | G0755022 | 09SWC465 | 73.8 | 79.8 |
| Minto North | Footwall | G0755023 | 09SWC465 | 88.5 | 94.5 |
| Minto North | Footwall | G0755005 | 09SWC393 | 80.4 | 86.4 |
| Minto North | Footwall | G0755009 | 09SWC404 | 86.4 | 92.4 |
| Minto North | Footwall | G0755018 | 09SWC 459 | 109.2 | 115.2 |
| Minto North | Footwall | G0755013 | 09SWC408 | 101.9 | 107.9 |
| Area2 | Bulk Waste | B463023 | 06SWC075 | 101.0 | 104.0 |
| Area2 | Bulk Waste | B464081 | 06SWC107 | 85.0 | 87.0 |
| Area2 | Bulk Waste | B464533 | 06SWC111 | 54.0 | 55.5 |
| Area2 | Bulk Waste | B464552 | 06SWC111 | 77.9 | 80.9 |
| Area2 | Bulk Waste | B792721 | 06SWC115 | 18.3 | 19.8 |
| Area2 | Bulk Waste | B792785 | 06SWC115 | 88.6 | 91.6 |
| Area2 | Bulk Waste | B793427 | 06SWC103 | 50.2 | 51.6 |
| Area2 | Bulk Waste | B793608 | 06SWC105 | 11.0 | 12.8 |
| Area2 | Bulk Waste | B793671 | 06SWC105 | 80.5 | 81.8 |
| Area2 | Bulk Waste | B795306 | 06SWC118 | 56.3 | 57.3 |

| MineArea | Waste Class | SAMPLE | HOLE ID | FROM (m) | TO (m) |
|---|---|---|---|---|---|
| Area2 | Bulk Waste | B795352 | 06SWC118 | 116.0 | 119.0 |
| Area2 | Bulk Waste | B795536 | 06SWC122 | 34.4 | 35.4 |
| Area2 | Bulk Waste | B797575 | 06SWC125 | 100.8 | 102.8 |
| Area2 | Bulk Waste | C441551 | 06SWC145 | 14.0 | 15.0 |
| Area2 | Bulk Waste | C441560 | 06SWC145 | 72.1 | 74.1 |
| Area2 | Bulk Waste | C441593 | 06SWC145 | 110.3 | 112.0 |
| Area2 | Bulk Waste | C441609 | 06SWC145 | 128.4 | 129.4 |
| Area2 | Bulk Waste | C486016 | 06SWC143 | 19.7 | 22.7 |
| Area2 | Bulk Waste | C486040 | 06SWC143 | 48.5 | 50.0 |
| Area2 | Bulk Waste | C486396 | 06SWC152 | 78.7 | 81.7 |
| Area2 | Bulk Waste | C486413 | 06SWC152 | 100.1 | 101.6 |
| Area2 | Bulk Waste | C486761 | 06SWC156 | 46.9 | 48.4 |
| Area2 | Bulk Waste | C486805 | 06SWC156 | 134.9 | 136.4 |
| Area2_SW | Bulk Waste | A2_ABA-1 | 07SWC201 | 43.3 | 49.0 |
| Area2_SW | Bulk Waste | A2_ABA-8 | 07SWC257 | 33.3 | 38.5 |
| Area2_SW | Bulk Waste | A2_ABA-9 | 07SWC257 | 38.5 | 76.0 |
| Area2_SW | Bulk Waste | A2_ABA-11 | 07SWC263 | 21.9 | 28.4 |
| Area2_SW | Bulk Waste | A2_ABA-12 | 07SWC263 | 36.7 | 42.2 |
| Area2_SW | Bulk Waste | A2_ABA-13 | 07SWC263 | 42.2 | 48.9 |
| Area2_SW | Bulk Waste | A2_ABA-15 | 07SWC267 | 30.1 | 36.3 |
| Area2_SW | Bulk Waste | A2_ABA-16 | 07SWC267 | 36.3 | 41.8 |
| Area2_SW | Bulk Waste | A2_ABA-17 | 07SWC267 | 41.8 | 74.3 |
| Area2_SW | Bulk Waste | A2_ABA-18 | 07SWC267 | 74.3 | 80.8 |
| Area2_SW | Bulk Waste | A2_ABA-20 | 07SWC267 | 88.5 | 95.0 |
| Area2 | Min Waste | B462373 | 06SWC075 | 26.0 | 27.0 |
| Area2 | Min Waste | B464032 | 06SWC107 | 19.8 | 21.4 |
| Area2 | Min Waste | B464062 | 06SWC107 | 56.9 | 59.0 |
| Area2 | Min Waste | B793448 | 06SWC103 | 71.9 | 73.3 |
| Area2 | Min Waste | B793458 | 06SWC103 | 92.0 | 93.0 |
| Area2 | Min Waste | B793635 | 06SWC105 | 42.7 | 43.1 |
| Area2 | Min Waste | B797524 | 06SWC125 | 17.3 | 18.8 |
| Area2 | Min Waste | B797615 | 06SWC125 | 147.5 | 149.0 |
| Area2 | Min Waste | C441666 | 06SWC146 | 148.7 | 150.2 |
| Area2 | Min Waste | C486779 | 06SWC156 | 84.5 | 86.0 |
| Area2_SW | Min Waste | A2_ABA-3 | 07SWC201 | 75.9 | 81.9 |
| Area2_SW | Min Waste | A2_ABA-4 | 07SWC201 | 83.7 | 89.7 |
| Area2_SW | Min Waste | A2_ABA-5 | 07SWC220 | 67.4 | 73.4 |
| Area2_SW | Min Waste | A2_ABA-6 | 07SWC220 | 73.4 | 79.4 |
| Area2_SW | Min Waste | A2_ABA-7 | 07SWC220 | 79.4 | 85.4 |
| Area2_SW | Min Waste | A2_ABA-10 | 07SWC263 | 3.1 | 21.9 |
| Area2_SW | Min Waste | A2_ABA-19 | 07SWC267 | 80.8 | 87.1 |
| Area2 | Ore | B464503 | 06SWC111 | 10.9 | 12.4 |
| Area2 | Ore | B792761 | 06SWC115 | 63.3 | 64.5 |
| Area2 | Ore | B792807 | 06SWC115 | 129.1 | 130.5 |
| Area2_SW | Ore | A2_ABA-2 | 07SWC201 | 53.1 | 57.9 |
| Area2_SW | Ore | A2_ABA-14 | 07SWC267 | 22.8 | 29.1 |

| MineArea | Waste Class | SAMPLE | pH | FIZZ RATING | MPA | NNP | AP | NP | Ratio (NP:MPA) | Ratio (NP:AP) | Total S (%) | Sulphate-S (%) | Sulphide-S (%) | TIC as C (%) | TIC as CO2 (%) | TIC-NP | Ratio (TIC-NP:AP) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ridgetop | Bulk Waste | RT_ABA-1 | 8.4 | 1 | 0.3 | 9 | 0.3 | 9 | 28.8 | 28.8 | 0.01 | <0.01 | 0.01 | <0.05 | <0.2 | 4.2 | 13.3 |
| Ridgetop | Bulk Waste | RT_ABA-2 | 8.9 | 2 | 0.3 | 19 | 0.3 | 19 | 60.8 | 60.8 | 0.01 | <0.01 | 0.01 | 0.16 | 0.6 | 13.3 | 42.7 |
| Ridgetop | Bulk Waste | RT_ABA-3 | 8.8 | 2 | 0.6 | 21 | 0.6 | 22 | 35.2 | 35.2 | 0.02 | <0.01 | 0.02 | 0.17 | 0.6 | 14.2 | 22.7 |
| Ridgetop | Bulk Waste | RT_ABA-6 | 8.7 | 2 | 0.3 | 33 | 0.3 | 33 | 105.6 | 105.6 | 0.01 | <0.01 | 0.01 | 0.39 | 1.4 | 32.5 | 104.0 |
| Ridgetop | Bulk Waste | RT_ABA-7 | 8.4 | 2 | 0.3 | 57 | 0.3 | 57 | 182.4 | 182.4 | 0.01 | <0.01 | 0.01 | 0.63 | 2.3 | 52.5 | 168.0 |
| Ridgetop | Bulk Waste | RT_ABA-8 | 8.5 | 2 | 0.3 | 58 | 0.3 | 58 | 185.6 | 185.6 | 0.01 | <0.01 | 0.01 | 0.6 | 2.2 | 50.0 | 160.0 |
| Ridgetop | Bulk Waste | RT_ABA-9 | 9.1 | 2 | 0.9 | 54 | 0.9 | 55 | 58.7 | 58.7 | 0.03 | <0.01 | 0.03 | 0.75 | 2.8 | 62.5 | 66.7 |
| Ridgetop | Bulk Waste | RT_ABA-10 | 8.6 | 2 | 0.9 | 22 | 0.9 | 23 | 24.5 | 24.5 | 0.03 | <0.01 | 0.03 | 0.23 | 0.8 | 19.2 | 20.4 |
| Ridgetop | Bulk Waste | RT_ABA-11 | 8.5 | 1 | 0.3 | 13 | 0.3 | 13 | 41.6 | 41.6 | 0.01 | <0.01 | 0.01 | <0.05 | <0.2 | 4.2 | 13.3 |
| Ridgetop | Bulk Waste | RT_ABA-12 | 8.8 | 1 | 0.3 | 15 | 0.3 | 15 | 48.0 | 48.0 | 0.01 | <0.01 | 0.01 | <0.05 | <0.2 | 4.2 | 13.3 |
| Ridgetop | Bulk Waste | RT_ABA-14 | 9 | 2 | 0.6 | 25 | 0.6 | 26 | 41.6 | 41.6 | 0.02 | <0.01 | 0.02 | 0.29 | 1.1 | 24.2 | 38.7 |
| Ridgetop | Bulk Waste | RT_ABA-15 | 8.7 | 2 | 0.9 | 44 | 0.3 | 45 | 48.0 | 144.0 | 0.03 | 0.03 | <0.01 | 0.46 | 1.7 | 38.3 | 122.7 |
| Ridgetop | Bulk Waste | RT_ABA-16 | 8.4 | 3 | 1.9 | 101 | 1.6 | 103 | 54.9 | 65.9 | 0.06 | 0.01 | 0.05 | 1.63 | 6 | 135.8 | 86.9 |
| Ridgetop | Bulk Waste | RT_ABA-17 | 8.8 | 3 | 0.9 | 111 | 0.9 | 112 | 119.5 | 119.5 | 0.03 | <0.01 | 0.03 | 1.38 | 5 | 115.0 | 122.7 |
| Ridgetop | Bulk Waste | RT_ABA-18 | 8.5 | 2 | 0.3 | 34 | 0.3 | 34 | 108.8 | 108.8 | 0.01 | <0.01 | 0.01 | 0.28 | 1 | 23.3 | 74.7 |
| Ridgetop | Bulk Waste | RT_ABA-19 | 8.5 | 2 | <0.3 | 25 | 0.3 | 25 | 160.0 | 80.0 | 0.01 | <0.01 | <0.01 | 0.21 | 0.8 | 17.5 | 56.0 |
| Ridgetop | Min Waste | RT_ABA-5 | 8.9 | 2 | 11.3 | 36 | 10.9 | 47 | 4.2 | 4.3 | 0.36 | 0.01 | 0.35 | 0.67 | 2.4 | 55.8 | 5.1 |
| Ridgetop | Min Waste | RT_ABA-20 | 8.9 | 2 | 3.1 | 28 | 3.1 | 31 | 9.9 | 9.9 | 0.1 | <0.01 | 0.1 | 0.32 | 1.2 | 26.7 | 8.5 |
| Ridgetop | Ore | RT_ABA-4 | 8.5 | 2 | 3.8 | 37 | 3.8 | 41 | 10.9 | 10.9 | 0.12 | <0.01 | 0.12 | 0.49 | 1.8 | 40.8 | 10.9 |
| Ridgetop | Ore | RT_ABA-13 | 8.2 | 2 | 9.4 | 11 | 9.4 | 20 | 2.1 | 2.1 | 0.3 | <0.01 | 0.3 | 0.23 | 0.8 | 19.2 | 2.0 |
| Minto North | Hanging wall | G0755001 | 8.41 | Slight | <0.6 | 21.5 | 0.6 | 21.5 | 35.8 | 35.8 | <0.02 | <0.01 | <0.02 | 0.12 | 0.43 | 9.8 | 16.3 |
| Minto North | Hanging wall | G0755002 | 8.58 | Slight | <0.6 | 25.3 | 0.6 | 25.3 | 42.2 | 42.2 | <0.02 | <0.01 | <0.02 | 0.12 | 0.45 | 10.2 | 17.0 |
| Minto North | Hanging wall | G0755003 | 8.8 | Slight | <0.6 | 25.4 | 0.6 | 25.4 | 42.3 | 42.3 | <0.02 | <0.01 | <0.02 | 0.14 | 0.51 | 11.6 | 19.3 |
| Minto North | Hanging wall | G0755004 | 9.18 | Slight | <0.6 | 19.9 | 0.6 | 19.9 | 33.2 | 33.2 | <0.02 | <0.01 | <0.02 | 0.07 | 0.26 | 5.9 | 9.8 |
| Minto North | Hanging wall | G0755006 | 8.7 | Slight | <0.6 | 33 | 0.6 | 33 | 55.0 | 55.0 | <0.02 | <0.01 | <0.02 | 0.22 | 0.81 | 18.4 | 30.7 |
| Minto North | Hanging wall | G0755007 | 8.73 | Slight | <0.6 | 32 | 0.6 | 32 | 53.3 | 53.3 | <0.02 | <0.01 | <0.02 | 0.17 | 0.64 | 14.5 | 24.2 |
| Minto North | Hanging wall | G0755008 | 8.61 | Slight | <0.6 | 33.3 | 0.6 | 33.3 | 55.5 | 55.5 | <0.02 | <0.01 | <0.02 | 0.24 | 0.89 | 20.2 | 33.7 |
| Minto North | Hanging wall | G0755010 | 8.85 | None | <0.6 | 12.2 | 0.6 | 12.2 | 20.3 | 20.3 | <0.02 | <0.01 | <0.02 | <0.05 | <0.02 | <0.5 | 0.8 |
| Minto North | Hanging wall | G0755011 | 8.63 | Slight | <0.6 | 26.3 | 0.6 | 26.3 | 43.8 | 43.8 | <0.02 | <0.01 | <0.02 | 0.17 | 0.62 | 14.1 | 23.5 |
| Minto North | Hanging wall | G0755012 | 8.88 | Slight | <0.6 | 26 | 0.6 | 26 | 43.3 | 43.3 | <0.02 | <0.01 | <0.02 | 0.11 | 0.4 | 9.1 | 15.2 |
| Minto North | Hanging wall | G0755014 | 9.02 | None | <0.6 | 12.3 | 0.6 | 12.3 | 20.5 | 20.5 | <0.02 | <0.01 | <0.02 | 0.01 | 0.03 | 0.7 | 1.1 |
| Minto North | Hanging wall | G0755015 | 8.95 | None | <0.6 | 15.1 | 0.6 | 15.1 | 25.2 | 25.2 | <0.02 | <0.01 | <0.02 | 0.03 | 0.12 | 2.7 | 4.5 |
| Minto North | Hanging wall | G0755016 | 8.89 | Slight | <0.6 | 27.3 | 0.6 | 27.3 | 45.5 | 45.5 | <0.02 | <0.01 | <0.02 | 0.14 | 0.53 | 12.0 | 20.1 |
| Minto North | Hanging wall | G0755017 | 8.92 | Slight | <0.6 | 24.7 | 0.6 | 24.7 | 41.2 | 41.2 | <0.02 | <0.01 | <0.02 | 0.10 | 0.37 | 8.4 | 14.0 |
| Minto North | Hanging wall | G0755019 | 8.77 | Slight | <0.6 | 24 | 0.6 | 24 | 40.0 | 40.0 | <0.02 | <0.01 | <0.02 | 0.10 | 0.36 | 8.2 | 13.6 |
| Minto North | Hanging wall | G0755020 | 8.91 | Slight | <0.6 | 23.8 | 0.6 | 23.8 | 39.7 | 39.7 | <0.02 | <0.01 | <0.02 | 0.14 | 0.5 | 11.4 | 18.9 |
| Minto North | Hanging wall | G0755021 | 8.65 | Slight | <0.6 | 30.8 | 0.6 | 30.8 | 51.3 | 51.3 | <0.02 | <0.01 | <0.02 | 0.30 | 1.1 | 25.0 | 41.7 |
| Minto North | Hanging wall | G0755022 | 8.9 | Slight | <0.6 | 25.4 | 0.6 | 25.4 | 42.3 | 42.3 | <0.02 | <0.01 | <0.02 | 0.16 | 0.6 | 13.6 | 22.7 |
| Minto North | Footwall | G0755023 | 8.42 | Slight | <0.6 | 37.5 | 0.6 | 37.5 | 62.5 | 62.5 | <0.02 | <0.01 | <0.02 | 0.33 | 1.21 | 27.5 | 45.8 |
| Minto North | Footwall | G0755005 | 8.82 | Slight | <0.6 | 31.5 | 0.6 | 31.5 | 52.5 | 52.5 | <0.02 | <0.01 | <0.02 | 0.21 | 0.76 | 17.3 | 28.8 |
| Minto North | Footwall | G0755009 | 9.2 | Slight | <0.6 | 24.8 | 0.6 | 24.8 | 41.3 | 41.3 | <0.02 | <0.01 | <0.02 | 0.09 | 0.32 | 7.3 | 12.1 |
| Minto North | Footwall | G0755018 | 9.28 | Slight | <0.6 | 25.3 | 0.6 | 25.3 | 42.2 | 42.2 | <0.02 | <0.01 | <0.02 | 0.08 | 0.3 | 6.8 | 11.4 |
| Minto North | Footwall | G0755013 | 8.9 | Slight | <0.6 | 23.3 | 0.6 | 23.3 | 38.8 | 38.8 | <0.02 | <0.01 | <0.02 | 0.09 | 0.34 | 7.7 | 12.9 |

| MineArea | Waste Class | SAMPLE | pH | FIZZ RATING | MPA | NNP | AP | NP | Ratio (NP:MPA) | Ratio (NP:AP) | Total S (%) | Sulphate-S (%) | Sulphide-S (%) | TIC as C (%) | TIC as CO2 (%) | TIC-NP | Ratio (TIC-NP:AP) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Area2 | Bulk Waste | B463023 | 9.7 | 2 | | 24 | 0.3 | 24 | 76.8 | 76.8 | 0.01 | <0.01 | 0.01 | 0.13 | 0.5 | 11.4 | 36.3 |
| Area2 | Bulk Waste | B464081 | 9.5 | 2 | | 21 | 0.3 | 21 | 67.2 | 67.2 | 0.01 | <0.01 | 0.01 | 0.1 | 0.4 | 9.1 | 29.1 |
| Area2 | Bulk Waste | B464533 | 9.5 | 2 | | 16 | 0.3 | 16 | 51.2 | 51.2 | <0.01 | <0.01 | 0.01 | 0.19 | 0.7 | 15.9 | 50.8 |
| Area2 | Bulk Waste | B464552 | 9.4 | 2 | | 31 | 0.3 | 31 | 99.2 | 99.2 | <0.01 | <0.01 | 0.01 | 0.3 | 1.1 | 25.0 | 79.9 |
| Area2 | Bulk Waste | B792721 | 9 | 2 | | 30 | 0.3 | 30 | 96.0 | 96.0 | 0.01 | <0.01 | 0.01 | 0.24 | 0.9 | 20.4 | 65.4 |
| Area2 | Bulk Waste | B792785 | 9.1 | 2 | | 29 | 0.3 | 29 | 92.8 | 92.8 | 0.01 | <0.01 | 0.01 | 0.26 | 1 | 22.7 | 72.6 |
| Area2 | Bulk Waste | B793427 | 9.5 | 2 | | 26 | 0.3 | 26 | 83.2 | 83.2 | 0.01 | <0.01 | 0.01 | 0.54 | 2 | 45.4 | 145.3 |
| Area2 | Bulk Waste | B793608 | 9.2 | 2 | | 29 | 0.3 | 29 | 92.8 | 92.8 | <0.01 | <0.01 | 0.01 | 0.23 | 0.8 | 18.2 | 58.1 |
| Area2 | Bulk Waste | B793671 | 9.6 | 2 | | 28 | 0.9 | 29 | 30.9 | 30.9 | 0.03 | <0.01 | 0.03 | 0.18 | 0.7 | 15.9 | 16.9 |
| Area2 | Bulk Waste | B795306 | 9.5 | 2 | | 17 | 0.6 | 18 | 28.8 | 28.8 | 0.02 | <0.01 | 0.02 | 0.12 | 0.5 | 11.4 | 18.2 |
| Area2 | Bulk Waste | B795352 | 9.4 | 2 | | 23 | 0.3 | 23 | 73.6 | 73.6 | <0.01 | <0.01 | 0.01 | 0.18 | 0.6 | 13.6 | 43.6 |
| Area2 | Bulk Waste | B795536 | 9.1 | 3 | | 47 | 0.9 | 48 | 51.2 | 51.2 | 0.03 | <0.01 | 0.03 | 0.47 | 1.7 | 38.6 | 41.2 |
| Area2 | Bulk Waste | B797575 | 9.1 | 2 | | 54 | 0.3 | 54 | 172.8 | 172.8 | 0.01 | <0.01 | 0.01 | 0.59 | 2.2 | 49.9 | 159.8 |
| Area2 | Bulk Waste | C441551 | 9.3 | 1 | | 14 | 0.6 | 15 | 24.0 | 24.0 | 0.02 | <0.01 | 0.02 | 0.1 | 0.4 | 9.1 | 14.5 |
| Area2 | Bulk Waste | C441560 | 9.6 | 2 | | 25 | 0.3 | 25 | 80.0 | 80.0 | <0.01 | <0.01 | 0.01 | 0.19 | 0.7 | 15.9 | 50.8 |
| Area2 | Bulk Waste | C441593 | 9.4 | 2 | | 23 | 3.4 | 26 | 7.6 | 7.6 | 0.11 | <0.01 | 0.11 | 0.21 | 0.8 | 18.2 | 5.3 |
| Area2 | Bulk Waste | C441609 | 9.2 | 2 | | 24 | 0.3 | 24 | 76.8 | 76.8 | 0.01 | <0.01 | 0.01 | 0.17 | 0.6 | 13.6 | 43.6 |
| Area2 | Bulk Waste | C486016 | 9.6 | 2 | | 18 | 0.3 | 18 | 57.6 | 57.6 | <0.01 | <0.01 | 0.01 | 0.11 | 0.4 | 9.1 | 29.1 |
| Area2 | Bulk Waste | C486040 | 9.4 | 2 | | 24 | 0.3 | 24 | 76.8 | 76.8 | 0.01 | 0.01 | 0.01 | 0.2 | 0.7 | 15.9 | 50.8 |
| Area2 | Bulk Waste | C486396 | 9.4 | 2 | | 24 | 1.9 | 26 | 13.9 | 13.9 | 0.06 | <0.01 | 0.06 | 0.2 | 0.7 | 15.9 | 8.5 |
| Area2 | Bulk Waste | C486413 | 9.2 | 2 | | 19 | 1.6 | 21 | 13.4 | 13.4 | 0.07 | 0.02 | 0.05 | 0.11 | 0.4 | 9.1 | 5.8 |
| Area2 | Bulk Waste | C486761 | 8.9 | 2 | | 34 | 0.3 | 34 | 108.8 | 108.8 | 0.01 | <0.01 | 0.01 | 0.27 | 1 | 22.7 | 72.6 |
| Area2 | Bulk Waste | C486805 | 8.8 | 2 | | 56 | 0.3 | 56 | 179.2 | 179.2 | 0.01 | <0.01 | 0.01 | 0.63 | 2.3 | 52.2 | 167.1 |
| Area2_SW | Bulk Waste | A2_ABA-1 | 8.3 | 2 | 0.3 | 38 | 0.3 | 38 | 121.6 | 121.6 | 0.01 | <0.01 | 0.01 | 0.35 | 1.3 | 29.2 | 93.3 |
| Area2_SW | Bulk Waste | A2_ABA-8 | 8.9 | 2 | 0.3 | 25 | 0.3 | 25 | 80.0 | 80.0 | 0.01 | <0.01 | 0.01 | 0.16 | 0.6 | 13.3 | 42.7 |
| Area2_SW | Bulk Waste | A2_ABA-9 | 9 | 2 | 2.2 | 26 | 2.2 | 28 | 12.8 | 12.8 | 0.07 | <0.01 | 0.07 | 0.28 | 1 | 23.3 | 10.7 |
| Area2_SW | Bulk Waste | A2_ABA-11 | 9.1 | 2 | 0.3 | 40 | 0.3 | 40 | 128.0 | 128.0 | 0.01 | <0.01 | 0.01 | 0.45 | 1.7 | 37.5 | 120.0 |
| Area2_SW | Bulk Waste | A2_ABA-12 | 9.2 | 2 | 0.3 | 21 | 0.3 | 21 | 67.2 | 67.2 | 0.01 | <0.01 | 0.01 | 0.11 | 0.4 | 9.2 | 29.3 |
| Area2_SW | Bulk Waste | A2_ABA-13 | 9.2 | 2 | 0.3 | 23 | 0.3 | 23 | 73.6 | 73.6 | 0.01 | <0.01 | 0.01 | 0.29 | 1.1 | 24.2 | 77.3 |
| Area2_SW | Bulk Waste | A2_ABA-15 | 8.3 | 1 | <0.3 | 10 | 0.3 | 10 | 64.0 | 32.0 | <0.01 | <0.01 | <0.01 | <0.05 | <0.2 | 4.2 | 13.3 |
| Area2_SW | Bulk Waste | A2_ABA-16 | 8.5 | 2 | <0.3 | 26 | 0.3 | 26 | 166.4 | 83.2 | <0.01 | <0.01 | <0.01 | 0.23 | 0.9 | 19.2 | 61.3 |
| Area2_SW | Bulk Waste | A2_ABA-17 | 8.8 | 2 | <0.3 | 22 | 0.3 | 22 | 140.8 | 70.4 | <0.01 | <0.01 | <0.01 | 0.13 | 0.5 | 10.8 | 34.7 |
| Area2_SW | Bulk Waste | A2_ABA-18 | 8.3 | 2 | 0.6 | 72 | 0.6 | 73 | 116.8 | 116.8 | 0.02 | <0.01 | 0.02 | 0.78 | 2.8 | 65.0 | 104.0 |
| Area2_SW | Bulk Waste | A2_ABA-20 | 8.1 | 2 | 2.5 | 68 | 2.5 | 70 | 28.0 | 28.0 | 0.08 | <0.01 | 0.08 | 0.72 | 2.7 | 60.0 | 24.0 |
| Area2 | Min Waste | B462373 | 8.7 | 2 | | 35 | 19.7 | 55 | 2.8 | 2.8 | 0.63 | <0.01 | 0.63 | 0.64 | 2.3 | 52.2 | 2.7 |
| Area2 | Min Waste | B464032 | 6.2 | 1 | | -16 | 12.5 | 7 | 0.6 | 0.6 | 0.74 | 0.34 | 0.4 | <0.05 | 0.2 | 4.5 | 0.4 |
| Area2 | Min Waste | B464062 | 9 | 3 | | 74 | 7.2 | 81 | 11.3 | 11.3 | 0.23 | <0.01 | 0.23 | 0.95 | 3.5 | 79.5 | 11.1 |
| Area2 | Min Waste | B793448 | 9.1 | 2 | | 32 | 14.7 | 47 | 3.2 | 3.2 | 0.47 | <0.01 | 0.47 | 0.13 | 0.5 | 11.4 | 0.8 |
| Area2 | Min Waste | B793458 | 9.8 | 2 | | 19 | 0.3 | 19 | 60.8 | 60.8 | <0.01 | <0.01 | 0.01 | 0.63 | 2.3 | 52.2 | 167.1 |
| Area2 | Min Waste | B793635 | 9.6 | 2 | | 13 | 6.3 | 19 | 3.0 | 3.0 | 0.2 | <0.01 | 0.2 | 0.2 | 0.7 | 15.9 | 2.5 |
| Area2 | Min Waste | B797524 | 8.8 | 1 | | 15 | 0.3 | 15 | 48.0 | 48.0 | 0.01 | <0.01 | 0.01 | 0.1 | 0.4 | 9.1 | 29.1 |
| Area2 | Min Waste | B797615 | 9.4 | 2 | | 9 | 9.7 | 19 | 2.0 | 1.96 | 0.31 | <0.01 | 0.31 | 0.11 | 0.4 | 9.1 | 0.9 |
| Area2 | Min Waste | C441666 | 9.4 | 2 | | 24 | 1.3 | 25 | 20.0 | 20.0 | 0.04 | <0.01 | 0.04 | 0.18 | 0.7 | 15.9 | 12.7 |
| Area2 | Min Waste | C486779 | 9.1 | 2 | | 26 | 4.4 | 30 | 6.9 | 6.9 | 0.14 | <0.01 | 0.14 | 0.27 | 1 | 22.7 | 5.2 |
| Area2_SW | Min Waste | A2_ABA-3 | 8.1 | 2 | 10.3 | 21 | 10.0 | 31 | 3.0 | 3.1 | 0.33 | 0.01 | 0.32 | 0.24 | 0.9 | 20.0 | 2.0 |
| Area2_SW | Min Waste | A2_ABA-4 | 8.4 | 2 | 7.5 | 21 | 7.5 | 28 | 3.7 | 3.7 | 0.24 | <0.01 | 0.24 | 0.2 | 0.8 | 16.7 | 2.2 |
| Area2_SW | Min Waste | A2_ABA-5 | 8.4 | 2 | 10.6 | 61 | 10.6 | 72 | 6.8 | 6.8 | 0.34 | <0.01 | 0.34 | 0.64 | 2.4 | 53.3 | 5.0 |
| Area2_SW | Min Waste | A2_ABA-6 | 8.5 | 2 | 7.8 | 39 | 7.8 | 47 | 6.0 | 6.0 | 0.25 | <0.01 | 0.25 | 0.66 | 2.4 | 55.0 | 7.0 |
| Area2_SW | Min Waste | A2_ABA-7 | 8.4 | 2 | 12.8 | 20 | 12.8 | 33 | 2.6 | 2.6 | 0.41 | <0.01 | 0.41 | 0.45 | 1.7 | 37.5 | 2.9 |
| Area2_SW | Min Waste | A2_ABA-10 | 8.9 | 2 | 2.2 | 26 | 2.2 | 28 | 12.8 | 12.8 | 0.07 | <0.01 | 0.07 | 0.3 | 1.1 | 25.0 | 11.4 |
| Area2_SW | Min Waste | A2_ABA-19 | 8.3 | 2 | 2.5 | 38 | 2.5 | 40 | 16.0 | 16.0 | 0.08 | <0.01 | 0.08 | 0.34 | 1.3 | 28.3 | 11.3 |
| Area2 | Ore | B464503 | 8.4 | 2 | | 13 | 0.3 | 13 | 41.6 | 41.6 | <0.01 | <0.01 | 0.01 | 0.14 | 0.5 | 11.4 | 36.3 |
| Area2 | Ore | B792761 | 8.1 | 3 | | 113 | 136.6 | 250 | 1.8 | 1.8 | 4.37 | <0.01 | 4.37 | 3.48 | 12.8 | 290.6 | 2.1 |
| Area2 | Ore | B792807 | 8.8 | 2 | | 11 | 24.1 | 35 | 1.5 | 1.5 | 0.77 | <0.01 | 0.77 | 0.32 | 1.2 | 27.2 | 1.1 |
| Area2_SW | Ore | A2_ABA-2 | 8 | 3 | 26.3 | 78 | 25.0 | 104 | 4.0 | 4.2 | 0.84 | 0.04 | 0.8 | 1.05 | 3.9 | 87.5 | 3.5 |
| Area2_SW | Ore | A2_ABA-14 | 7.6 | 1 | 4.1 | 5 | 4.1 | 9 | 2.2 | 2.2 | 0.13 | <0.01 | 0.13 | <0.05 | <0.2 | 4.2 | 1.0 |

| MineArea | Waste Class | SAMPLE | Total Cu% Av | NS Cu % | Au ppm | Ag ppm ICP | Al_%ICP | As_ppmICP | B_ppmICP | Ba_ppmICP | Be_ppmICP | Bi_ppmICP | Ca_%ICP | Cd_ppmICP | Co_ppmICP | Cr_ppmICP | Cu_ppmICP | Fe_%ICP | Ga_ppmICP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Crustal average for granitic rocks* | | | 0.4 to 0.5 | 7.2 to 8.8 | 1.4 to 1.9 | 9 to 10 | 420 to 1600 | 1 to 3 | ..01 | 0.5 to 2.5 | 0.13 | 1 to 7 | 2 to 22 | 5 to 30 | 1.4 to 3.7 | 17 to 30 |
| | | | *Source: Price, W.A. (1997). Draft guidelines and recommended methods for the prediction of metal leaching and acid rock drainage at minesites in BC | | | | | | | | | | | | | | | | |
| Ridgetop | Bulk Waste | RT_ABA-1 | 0.015 | 0.005 | 0.005 | 0.3 | 1.9 | 15 | 5 | 693 | 0.44 | 1 | 0.4 | 0.31 | 10 | 6 | 152 | 4.8 | 10 |
| Ridgetop | Bulk Waste | RT_ABA-2 | 0.010 | 0.005 | 0.008 | 0.2 | 1.6 | 7 | 7 | 550 | 0.25 | 2 | 0.8 | 0.25 | 7 | 7 | 94 | 2.6 | 10 |
| Ridgetop | Bulk Waste | RT_ABA-3 | 0.125 | 0.049 | 0.006 | 0.2 | 1.6 | 3 | 5 | 511 | 0.25 | 2 | 0.8 | 0.25 | 8 | 7 | 1297 | 3.0 | 10 |
| Ridgetop | Bulk Waste | RT_ABA-6 | 0.005 | 0.001 | 0.003 | 0.1 | 0.8 | 1 | 5 | 198 | 0.25 | 1 | 0.8 | 0.25 | 6 | 5 | 23 | 2.2 | 5 |
| Ridgetop | Bulk Waste | RT_ABA-7 | 0.021 | 0.011 | 0.040 | 0.2 | 1.5 | 5 | 8 | 361 | 0.61 | 1 | 2.2 | 0.25 | 7 | 5 | 279 | 2.5 | 9 |
| Ridgetop | Bulk Waste | RT_ABA-8 | 0.095 | 0.067 | 0.064 | 0.1 | 1.8 | 4 | 7 | 297 | 0.61 | 1 | 2.2 | 0.65 | 8 | 6 | 1017 | 3.3 | 10 |
| Ridgetop | Bulk Waste | RT_ABA-9 | 0.050 | 0.012 | 0.004 | 0.1 | 1.7 | 3 | 7 | 212 | 0.45 | 2 | 1.5 | 0.71 | 7 | 4 | 530 | 3.9 | 10 |
| Ridgetop | Bulk Waste | RT_ABA-10 | 0.310 | 0.204 | 0.045 | 0.3 | 1.4 | 3 | 5 | 307 | 0.25 | 1 | 0.8 | 0.25 | 5 | 14 | 3068 | 2.5 | 10 |
| Ridgetop | Bulk Waste | RT_ABA-11 | 0.027 | 0.007 | 0.005 | 0.2 | 1.1 | 50 | 5 | 376 | 0.25 | 1 | 0.3 | 0.25 | 7 | 6 | 124 | 2.5 | 8 |
| Ridgetop | Bulk Waste | RT_ABA-12 | 0.024 | 0.012 | 0.004 | 1.0 | 1.2 | 6 | 5 | 422 | 0.25 | 1 | 0.3 | 0.25 | 6 | 6 | 205 | 2.3 | 8 |
| Ridgetop | Bulk Waste | RT_ABA-14 | 0.015 | 0.002 | 0.005 | 0.3 | 1.1 | 10 | 5 | 447 | 0.25 | 1 | 0.8 | 0.25 | 5 | 7 | 126 | 2.2 | 5 |
| Ridgetop | Bulk Waste | RT_ABA-15 | 0.012 | 0.006 | 0.003 | 0.2 | 0.6 | 6 | 10 | 1165 | 0.56 | 1 | 1.8 | 0.25 | 6 | 2 | 139 | 1.9 | 5 |
| Ridgetop | Bulk Waste | RT_ABA-16 | 0.120 | 0.070 | 0.002 | 0.6 | 0.7 | 11 | 10 | 963 | 0.84 | 1 | 3.2 | 0.25 | 8 | 2 | 1255 | 2.8 | 5 |
| Ridgetop | Bulk Waste | RT_ABA-17 | 0.039 | 0.019 | 0.002 | 0.2 | 0.7 | 1 | 8 | 679 | 0.52 | 1 | 2.9 | 0.33 | 7 | 3 | 380 | 2.6 | 5 |
| Ridgetop | Bulk Waste | RT_ABA-18 | 0.027 | 0.011 | 0.003 | 0.2 | 1.3 | 5 | 5 | 258 | 0.31 | 1 | 1.3 | 0.25 | 9 | 4 | 271 | 3.3 | 8 |
| Ridgetop | Bulk Waste | RT_ABA-19 | 0.005 | 0.002 | 0.003 | 0.3 | 1.1 | 3 | 5 | 168 | 0.25 | 1 | 1.0 | 0.25 | 6 | 5 | 37 | 2.4 | 7 |
| Ridgetop | Min Waste | RT_ABA-5 | 0.079 | 0.011 | 0.003 | 0.4 | 1.5 | 4 | 5 | 322 | 0.25 | 1 | 1.2 | 0.59 | 10 | 5 | 884 | 4.2 | 10 |
| Ridgetop | Min Waste | RT_ABA-20 | 0.451 | 0.276 | 0.140 | 1.5 | 1.0 | 1 | 5 | 154 | 0.29 | 1 | 1.1 | 0.36 | 5 | 5 | 4514 | 2.4 | 6 |
| Ridgetop | Ore | RT_ABA-4 | 0.665 | 0.243 | 0.057 | 0.4 | 1.0 | 2 | 6 | 237 | 0.39 | 1 | 1.2 | 0.25 | 6 | 3 | 6889 | 2.4 | 7 |
| Ridgetop | Ore | RT_ABA-13 | 0.708 | 0.335 | 0.047 | 1.5 | 1.1 | 32 | 6 | 394 | 0.25 | 2 | 0.7 | 0.43 | 8 | 5 | 7085 | 2.7 | 6 |
| Minto North | Hanging wall | G0755001 | #N/A | #N/A | 0.001 | <0.1 | 1.0 | <0.5 | <0.1 | 235 | #N/A | <0.1 | 0.7 | <0.1 | 6 | 56 | 2 | 1.9 | 4 |
| Minto North | Hanging wall | G0755002 | #N/A | #N/A | 0.001 | <0.1 | 1.1 | 1 | <0.1 | 175 | #N/A | <0.1 | 0.9 | <0.1 | 6 | 59 | 4 | 2.0 | 5 |
| Minto North | Hanging wall | G0755003 | #N/A | #N/A | 0.001 | <0.1 | 1.0 | <0.5 | <0.1 | 196 | #N/A | <0.1 | 0.8 | <0.1 | 6 | 61 | 4 | 2.0 | 4 |
| Minto North | Hanging wall | G0755004 | #N/A | #N/A | 0.001 | <0.1 | 1.1 | 1 | <0.1 | 54 | #N/A | <0.1 | 1.1 | <0.1 | 6 | 42 | 3 | 1.8 | 5 |
| Minto North | Hanging wall | G0755006 | #N/A | #N/A | 0.003 | <0.1 | 1.1 | 1 | <0.1 | 54 | #N/A | <0.1 | 1.1 | <0.1 | 6 | 42 | 3 | 1.8 | 5 |
| Minto North | Hanging wall | G0755007 | #N/A | #N/A | 0.001 | <0.1 | 1.2 | 1 | <0.1 | 60 | #N/A | <0.1 | 1.0 | <0.1 | 7 | 48 | 4 | 2.0 | 6 |
| Minto North | Hanging wall | G0755008 | #N/A | #N/A | 0.001 | <0.1 | 0.9 | 1 | <0.1 | 128 | #N/A | <0.1 | 1.0 | <0.1 | 6 | 58 | 6 | 1.8 | 4 |
| Minto North | Hanging wall | G0755010 | #N/A | #N/A | 0.001 | 0.1 | 1.0 | 1 | <0.1 | 161 | #N/A | <0.1 | 0.6 | <0.1 | 6 | 59 | 8 | 1.9 | 4 |
| Minto North | Hanging wall | G0755011 | #N/A | #N/A | 0.001 | <0.1 | 1.1 | 1 | <0.1 | 110 | #N/A | <0.1 | 1.1 | <0.1 | 6 | 48 | 5 | 2.0 | 5 |
| Minto North | Hanging wall | G0755012 | #N/A | #N/A | 0.001 | <0.1 | 1.1 | 1 | <0.1 | 150 | #N/A | <0.1 | 0.9 | <0.1 | 6 | 58 | 4 | 2.0 | 5 |
| Minto North | Hanging wall | G0755014 | #N/A | #N/A | 0.001 | <0.1 | 1.0 | <0.5 | <0.1 | 177 | #N/A | <0.1 | 0.4 | <0.1 | 5 | 65 | 3 | 1.9 | 4 |
| Minto North | Hanging wall | G0755015 | #N/A | #N/A | 0.001 | <0.1 | 1.0 | 3 | <0.1 | 108 | #N/A | <0.1 | 0.7 | <0.1 | 6 | 60 | 3 | 2.0 | 5 |
| Minto North | Hanging wall | G0755016 | #N/A | #N/A | 0.001 | <0.1 | 1.0 | 1 | <0.1 | 127 | #N/A | <0.1 | 0.9 | <0.1 | 6 | 57 | 2 | 2.0 | 5 |
| Minto North | Hanging wall | G0755017 | #N/A | #N/A | 0.001 | <0.1 | 1.1 | <0.5 | <0.1 | 230 | #N/A | <0.1 | 0.7 | <0.1 | 6 | 60 | 78 | 2.1 | 5 |
| Minto North | Hanging wall | G0755019 | #N/A | #N/A | 0.001 | <0.1 | 0.9 | 1 | <0.1 | 92 | #N/A | <0.1 | 0.9 | <0.1 | 6 | 55 | 2 | 1.8 | 5 |
| Minto North | Hanging wall | G0755020 | #N/A | #N/A | 0.001 | <0.1 | 1.2 | <0.5 | <0.1 | 334 | #N/A | <0.1 | 0.6 | <0.1 | 6 | 59 | 3 | 2.3 | 5 |
| Minto North | Hanging wall | G0755021 | #N/A | #N/A | 0.001 | <0.1 | 0.9 | <0.5 | <0.1 | 234 | #N/A | <0.1 | 1.0 | <0.1 | 6 | 63 | 3 | 2.0 | 4 |
| Minto North | Hanging wall | G0755022 | #N/A | #N/A | 0.001 | <0.1 | 1.2 | <0.5 | <0.1 | 296 | #N/A | <0.1 | 0.7 | <0.1 | 6 | 64 | 8 | 2.4 | 5 |
| Minto North | Footwall | G0755023 | #N/A | #N/A | 0.007 | <0.1 | 1.1 | 1 | <0.1 | 109 | #N/A | <0.1 | 1.4 | 0.20 | 5 | 54 | 232 | 2.1 | 5 |
| Minto North | Footwall | G0755005 | #N/A | #N/A | 0.003 | <0.1 | 1.0 | <0.5 | <0.1 | 153 | #N/A | <0.1 | 1.0 | <0.1 | 6 | 58 | 48 | 1.9 | 4 |
| Minto North | Footwall | G0755009 | #N/A | #N/A | 0.003 | <0.1 | 1.0 | 1 | <0.1 | 187 | #N/A | <0.1 | 0.6 | <0.1 | 6 | 59 | 179 | 2.1 | 4 |
| Minto North | Footwall | G0755018 | #N/A | #N/A | 0.001 | <0.1 | 1.1 | 1 | <0.1 | 107 | #N/A | <0.1 | 0.8 | <0.1 | 6 | 54 | 6 | 2.0 | 6 |
| Minto North | Footwall | G0755013 | #N/A | #N/A | 0.001 | <0.1 | 1.1 | 1 | <0.1 | 173 | #N/A | <0.1 | 0.7 | <0.1 | 6 | 61 | 9 | 2.1 | 5 |

| MineArea | Waste Class | SAMPLE | Total Cu% Av | NS Cu % | Au ppm | Ag ppm ICP | Al_%ICP | As_ppmICP | B_ppmICP | Ba_ppmICP | Be_ppmICP | Bi_ppmICP | Ca_%ICP | Cd_ppmICP | Co_ppmICP | Cr_ppmICP | Cu_ppmICP | Fe_%ICP | Ga_ppmICP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Crustal average for granitic rocks* | | | 0.4 to 0.5 | 7.2 to 8.8 | 1.4 to 1.9 | 9 to 10 | 420 to 1600 | 1 to 3 | ..01 | 0.5 to 2.5 | 0.13 | 1 to 7 | 2 to 22 | 5 to 30 | 1.4 to 3.7 | 17 to 30 |
| | | | *Source: Price, W.A. (1997). Draft guidelines and recommended methods for the prediction of metal leaching and acid rock drainage at minesites in BC | | | | | | | | | | | | | | | | |
| Area2 | Bulk Waste | B463023 | 0.005 | 0.001 | 0.025 | 0.1 | 1.7 | 3 | 5 | 200 | 0.25 | 1 | 1.4 | 0.25 | 7 | 13 | 44 | 2.9 | 10 |
| Area2 | Bulk Waste | B464081 | 0.005 | 0.001 | 0.003 | 0.1 | 1.4 | 1 | 5 | 160 | 0.25 | 2 | 1.0 | 0.25 | 6 | 36 | 5 | 2.3 | 10 |
| Area2 | Bulk Waste | B464533 | 0.005 | 0.001 | 0.003 | 0.1 | 1.2 | 3 | 5 | 410 | 0.25 | 1 | 0.6 | 0.25 | 5 | 8 | 10 | 2.2 | 5 |
| Area2 | Bulk Waste | B464552 | 0.010 | 0.001 | 0.003 | 0.1 | 1.5 | 1 | 5 | 380 | 0.25 | 1 | 1.2 | 0.25 | 5 | 7 | 13 | 2.7 | 10 |
| Area2 | Bulk Waste | B792721 | 0.005 | 0.001 | 0.003 | 0.5 | 1.4 | 1 | 5 | 290 | 0.25 | 1 | 1.1 | 0.25 | 6 | 5 | 7 | 2.6 | 10 |
| Area2 | Bulk Waste | B792785 | 0.005 | 0.001 | 0.003 | 0.1 | 1.2 | 1 | 5 | 170 | 0.25 | 1 | 1.1 | 0.25 | 6 | 107 | 9 | 2.7 | 10 |
| Area2 | Bulk Waste | B793427 | 0.005 | 0.001 | 0.003 | 0.1 | 1.9 | 1 | 5 | 630 | 0.25 | 1 | 0.9 | 0.25 | 7 | 5 | 7 | 3.0 | 10 |
| Area2 | Bulk Waste | B793608 | 0.010 | 0.002 | 0.003 | 0.7 | 1.5 | 6 | 5 | 400 | 0.25 | 1 | 1.1 | 0.25 | 7 | 5 | 32 | 3.0 | 10 |
| Area2 | Bulk Waste | B793671 | 0.040 | 0.005 | 0.023 | 0.3 | 1.7 | 5 | 5 | 220 | 0.25 | 1 | 1.2 | 0.25 | 8 | 10 | 440 | 3.0 | 10 |
| Area2 | Bulk Waste | B795306 | 0.030 | 0.001 | 0.003 | 0.1 | 1.6 | 1 | 5 | 460 | 0.25 | 2 | 0.7 | 0.25 | 4 | 3 | 40 | 2.6 | 10 |
| Area2 | Bulk Waste | B795352 | 0.005 | 0.001 | 0.003 | 0.1 | 0.7 | 1 | 5 | 130 | 0.25 | 1 | 0.7 | 0.25 | 2 | 5 | 5 | 1.6 | 5 |
| Area2 | Bulk Waste | B795536 | 0.010 | 0.001 | 0.003 | 0.3 | 0.7 | 1 | 5 | 80 | 0.25 | 1 | 1.1 | 0.25 | 5 | 4 | 59 | 1.8 | 5 |
| Area2 | Bulk Waste | B797575 | 0.010 | 0.001 | 0.006 | 0.1 | 1.2 | 1 | 5 | 180 | 0.25 | 1 | 1.6 | 0.25 | 5 | 10 | 1 | 2.3 | 5 |
| Area2 | Bulk Waste | C441551 | 0.005 | 0.001 | 0.003 | 0.1 | 1.6 | 1 | 5 | 440 | 0.25 | 1 | 0.5 | 0.25 | 6 | 8 | 17 | 2.8 | 10 |
| Area2 | Bulk Waste | C441560 | 0.005 | 0.001 | 0.003 | 0.1 | 1.4 | 1 | 5 | 520 | 0.25 | 1 | 0.9 | 0.25 | 6 | 7 | 1 | 2.8 | 10 |
| Area2 | Bulk Waste | C441593 | 0.005 | 0.001 | 0.003 | 0.1 | 1.5 | 1 | 5 | 490 | 0.25 | 1 | 0.8 | 0.25 | 5 | 6 | 97 | 2.5 | 10 |
| Area2 | Bulk Waste | C441609 | 0.005 | 0.001 | 0.003 | 0.1 | 1.5 | 1 | 5 | 360 | 0.25 | 1 | 1.1 | 0.25 | 7 | 4 | 54 | 2.5 | 10 |
| Area2 | Bulk Waste | C486016 | 0.005 | 0.001 | 0.003 | 0.1 | 1.2 | 2 | 5 | 280 | 0.25 | 1 | 0.7 | 0.25 | 5 | 10 | 10 | 2.2 | 5 |
| Area2 | Bulk Waste | C486040 | 0.005 | 0.001 | 0.003 | 0.1 | 1.2 | 1 | 5 | 250 | 0.25 | 1 | 0.9 | 0.50 | 7 | 5 | 7 | 2.4 | 5 |
| Area2 | Bulk Waste | C486396 | 0.010 | 0.001 | 0.003 | 0.1 | 1.4 | 1 | 5 | 410 | 0.25 | 1 | 0.8 | 0.25 | 5 | 6 | 120 | 2.2 | 10 |
| Area2 | Bulk Waste | C486413 | 0.005 | 0.001 | 0.003 | 0.1 | 1.7 | 1 | 5 | 400 | 0.25 | 1 | 0.8 | 0.25 | 6 | 7 | 17 | 2.7 | 10 |
| Area2 | Bulk Waste | C486761 | 0.005 | 0.001 | 0.003 | 0.1 | 1.7 | 1 | 5 | 190 | 0.50 | 1 | 1.8 | 0.25 | 6 | 3 | 32 | 3.1 | 10 |
| Area2 | Bulk Waste | C486805 | 0.005 | 0.001 | 0.003 | 0.1 | 1.1 | 1 | 5 | 370 | 0.25 | 1 | 1.9 | 0.25 | 5 | 5 | 7 | 2.3 | 5 |
| Area2_SW | Bulk Waste | A2_ABA-1 | 0.005 | 0.001 | 0.003 | 0.1 | 1.6 | 2 | 5 | 97 | 0.29 | 1 | 1.5 | 0.25 | 5 | 7 | 24 | 2.5 | 10 |
| Area2_SW | Bulk Waste | A2_ABA-8 | 0.005 | 0.001 | 0.006 | 0.1 | 1.3 | 3 | 5 | 251 | 0.25 | 1 | 0.9 | 0.25 | 6 | 6 | 9 | 2.6 | 8 |
| Area2_SW | Bulk Waste | A2_ABA-9 | 0.086 | 0.008 | 0.014 | 0.3 | 1.3 | 4 | 5 | 246 | 0.25 | 1 | 1.0 | 0.25 | 6 | 6 | 821 | 2.4 | 8 |
| Area2_SW | Bulk Waste | A2_ABA-11 | 0.005 | 0.000 | 0.002 | 0.1 | 1.2 | 1 | 5 | 326 | 0.25 | 1 | 1.2 | 0.25 | 6 | 7 | 6 | 2.4 | 10 |
| Area2_SW | Bulk Waste | A2_ABA-12 | 0.006 | 0.001 | 0.003 | 0.1 | 1.4 | 2 | 5 | 167 | 0.25 | 1 | 0.9 | 0.25 | 7 | 7 | 21 | 2.7 | 10 |
| Area2_SW | Bulk Waste | A2_ABA-13 | 0.005 | 0.001 | 0.003 | 0.1 | 1.2 | 2 | 5 | 274 | 0.25 | 1 | 0.8 | 0.25 | 5 | 6 | 3 | 2.2 | 9 |
| Area2_SW | Bulk Waste | A2_ABA-15 | 0.006 | 0.002 | 0.003 | 0.1 | 1.2 | 2 | 5 | 118 | 0.34 | 1 | 0.5 | 0.25 | 5 | 4 | 44 | 2.0 | 10 |
| Area2_SW | Bulk Waste | A2_ABA-16 | 0.035 | 0.014 | 0.007 | 0.2 | 1.2 | 4 | 5 | 75 | 0.32 | 1 | 1.3 | 0.25 | 5 | 4 | 344 | 2.3 | 10 |
| Area2_SW | Bulk Waste | A2_ABA-17 | 0.041 | 0.029 | 0.016 | 0.2 | 1.4 | 3 | 5 | 90 | 0.39 | 1 | 1.1 | 0.25 | 6 | 5 | 387 | 2.5 | 9 |
| Area2_SW | Bulk Waste | A2_ABA-18 | 0.053 | 0.008 | 0.009 | 0.3 | 2.8 | 23 | 8 | 66 | 0.85 | 1 | 3.5 | 0.25 | 8 | 17 | 516 | 2.2 | 11 |
| Area2_SW | Bulk Waste | A2_ABA-20 | 0.101 | 0.003 | 0.015 | 0.1 | 3.0 | 2 | 8 | 45 | 0.56 | 1 | 3.7 | 0.25 | 5 | 2 | 1069 | 1.9 | 10 |
| Area2 | Min Waste | B462373 | 0.430 | 0.118 | 0.025 | 0.9 | 2.2 | 1 | 5 | 230 | 0.25 | 2 | 0.9 | 0.25 | 23 | 5 | 4330 | 5.4 | 10 |
| Area2 | Min Waste | B464032 | 0.160 | 0.081 | 0.003 | 1.0 | 1.3 | 1 | 5 | 200 | 0.25 | 1 | 0.5 | 0.25 | 21 | 28 | 1430 | 3.4 | 10 |
| Area2 | Min Waste | B464062 | 0.030 | 0.004 | 0.003 | 0.2 | 1.0 | 1 | 5 | 320 | 0.25 | 1 | 2.2 | 0.25 | 10 | 46 | 298 | 3.5 | 10 |
| Area2 | Min Waste | B793448 | 0.290 | 0.011 | 0.030 | 0.5 | 1.7 | 1 | 5 | 230 | 0.25 | 1 | 1.9 | 0.25 | 9 | 5 | 2790 | 3.4 | 10 |
| Area2 | Min Waste | B793458 | 0.050 | 0.004 | 0.022 | 0.1 | 1.4 | 1 | 5 | 270 | 0.25 | 1 | 0.6 | 0.25 | 5 | 20 | 359 | 2.5 | 5 |
| Area2 | Min Waste | B793635 | 0.020 | 0.002 | 0.003 | 0.1 | 1.0 | 1 | 5 | 250 | 0.25 | 1 | 0.4 | 0.25 | 5 | 4 | 230 | 1.7 | 5 |
| Area2 | Min Waste | B797524 | 0.050 | 0.023 | 0.006 | 0.2 | 2.0 | 5 | 5 | 360 | 0.25 | 1 | 0.5 | 0.25 | 7 | 5 | 497 | 3.8 | 10 |
| Area2 | Min Waste | B797615 | 0.340 | 0.010 | 0.028 | 0.5 | 1.4 | 1 | 5 | 150 | 0.25 | 1 | 0.7 | 0.25 | 5 | 6 | 3190 | 2.1 | 5 |
| Area2 | Min Waste | C441666 | 0.060 | 0.011 | 0.050 | 0.4 | 1.5 | 3 | 5 | 220 | 0.25 | 1 | 0.8 | 0.25 | 7 | 7 | 597 | 2.5 | 10 |
| Area2 | Min Waste | C486779 | 0.050 | 0.002 | 0.003 | 0.2 | 1.1 | 3 | 5 | 40 | 0.25 | 1 | 1.2 | 0.25 | 5 | 4 | 480 | 2.0 | 10 |
| Area2_SW | Min Waste | A2_ABA-3 | 0.280 | 0.010 | 0.033 | 0.5 | 2.0 | 5 | 5 | 173 | 0.38 | 1 | 1.4 | 0.25 | 6 | 9 | 2595 | 2.5 | 10 |
| Area2_SW | Min Waste | A2_ABA-4 | 0.225 | 0.009 | 0.027 | 0.5 | 1.6 | 2 | 5 | 245 | 0.25 | 1 | 1.0 | 0.25 | 6 | 11 | 2072 | 2.7 | 10 |
| Area2_SW | Min Waste | A2_ABA-5 | 0.383 | 0.008 | 0.066 | 0.7 | 1.4 | 3 | 5 | 220 | 0.25 | 1 | 1.7 | 0.31 | 7 | 5 | 3858 | 2.8 | 10 |
| Area2_SW | Min Waste | A2_ABA-6 | 0.333 | 0.015 | 0.072 | 0.8 | 1.4 | 2 | 5 | 295 | 0.25 | 1 | 1.4 | 0.25 | 7 | 5 | 3248 | 2.8 | 9 |
| Area2_SW | Min Waste | A2_ABA-7 | 0.525 | 0.014 | 0.128 | 1.3 | 1.4 | 1 | 5 | 293 | 0.25 | 1 | 1.1 | 0.31 | 7 | 6 | 5255 | 3.4 | 9 |
| Area2_SW | Min Waste | A2_ABA-10 | 0.133 | 0.028 | 0.036 | 0.7 | 1.0 | 2 | 5 | 373 | 0.25 | 1 | 0.9 | 0.27 | 6 | 5 | 1301 | 2.0 | 9 |
| Area2_SW | Min Waste | A2_ABA-19 | 0.144 | 0.012 | 0.021 | 0.1 | 2.5 | 5 | 5 | 58 | 0.31 | 1 | 2.2 | 0.25 | 5 | 3 | 1470 | 1.9 | 10 |
| Area2 | Ore | B464503 | 1.290 | 1.125 | 0.153 | 2.4 | 1.0 | 1 | 5 | 90 | 0.25 | 1 | 0.3 | 0.25 | 5 | 4 | 10000 | 2.4 | 5 |
| Area2 | Ore | B792761 | 0.450 | 0.035 | 0.007 | 1.2 | 0.3 | 7 | 5 | 50 | 0.60 | 1 | 5.5 | 0.25 | 71 | 48 | 4190 | 8.8 | 5 |
| Area2 | Ore | B792807 | 1.220 | 0.056 | 0.446 | 3.7 | 1.3 | 1 | 5 | 70 | 0.25 | 2 | 1.1 | 0.25 | 6 | 100 | 10000 | 6.4 | 10 |
| Area2_SW | Ore | A2_ABA-2 | 0.937 | 0.040 | 0.159 | 3.0 | 1.3 | 4 | 5 | 116 | 0.25 | 2 | 3.7 | 0.46 | 5 | 6 | 3952 | 2.3 | 6 |
| Area2_SW | Ore | A2_ABA-14 | 0.859 | 0.460 | 0.119 | 1.0 | 3.2 | 7 | 5 | 177 | 0.38 | 2 | 1.2 | 0.25 | 13 | 5 | -2524 | 5.4 | 10 |

| MineArea | Waste Class | SAMPLE | Hg_ppmICP | K_%ICP | La_ppmICP | Mg_%ICP | Mn_ppmICP | Mo_ppmICP | Na_%ICP | Ni_ppmICP | P_ppmICP | Pb_ppmICP | S_%ICP | Sb_ppmICP | Sc_ppmICP | Sr_ppmICP | Ti_%ICP | Tl_ppmICP | U_ppmICP | V_ppmICP | W_ppmICP | Zn_ppmICP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | 0.08 | 0.48 to 4.2 | 45 to 70 | 0.16 to 0.94 | 390 to 850 | 0.6 to 1.3 | 2.6 to 4.0 | 4 to 15 | 600 to 920 | 12 to 19 | 0.03 | 0.2 | 3 to 14 | 100 to 440 | 0.12 to 0.35 | 0.72 to 2.3 | 3 | 30 to 88 | 1.3 to 2.2 | 39 to 130 |
| Ridgetop | Bulk Waste | RT_ABA-1 | 1 | 1.3 | 12 | 0.9 | 1304 | 2.4 | 0.05 | 6 | 1027 | 10 | 0.01 | 1.0 | 7 | 37 | 0.23 | 6 | 5 | 103 | 5 | 231 |
| Ridgetop | Bulk Waste | RT_ABA-2 | 1 | 1.1 | 5 | 0.9 | 388 | 0.5 | 0.08 | 3 | 1158 | 5 | 0.01 | 1.2 | 3 | 57 | 0.23 | 5 | 5 | 78 | 5 | 62 |
| Ridgetop | Bulk Waste | RT_ABA-3 | 1 | 1.2 | 5 | 0.8 | 545 | 1.0 | 0.08 | 1 | 1041 | 7 | 0.02 | 1.2 | 6 | 39 | 0.23 | 5 | 5 | 84 | 5 | 73 |
| Ridgetop | Bulk Waste | RT_ABA-6 | 1 | 0.5 | 13 | 0.5 | 383 | 0.6 | 0.04 | 1 | 592 | 5 | 0.01 | 1.3 | 2 | 35 | 0.09 | 5 | 5 | 43 | 5 | 62 |
| Ridgetop | Bulk Waste | RT_ABA-7 | 1 | 0.7 | 12 | 0.6 | 866 | 3.9 | 0.07 | 5 | 764 | 10 | 0.01 | 1.0 | 7 | 77 | 0.12 | 6 | 5 | 83 | 5 | 84 |
| Ridgetop | Bulk Waste | RT_ABA-8 | 0 | 1.0 | 17 | 1.0 | 674 | 2.7 | 0.07 | 5 | 1032 | 7 | 0.01 | 1.0 | 9 | 79 | 0.20 | 10 | 5 | 112 | 5 | 82 |
| Ridgetop | Bulk Waste | RT_ABA-9 | 1 | 1.0 | 17 | 1.2 | 1926 | 0.8 | 0.08 | 5 | 937 | 19 | 0.04 | 1.3 | 8 | 93 | 0.15 | 7 | 5 | 86 | 5 | 523 |
| Ridgetop | Bulk Waste | RT_ABA-10 | 1 | 0.8 | 10 | 0.7 | 265 | 1.4 | 0.07 | 2 | 845 | 4 | 0.03 | 1.5 | 6 | 53 | 0.17 | 5 | 5 | 83 | 5 | 50 |
| Ridgetop | Bulk Waste | RT_ABA-11 | 1 | 0.6 | 11 | 0.4 | 477 | 1.0 | 0.08 | 1 | 716 | 4 | 0.01 | 1.0 | 6 | 28 | 0.11 | 5 | 5 | 63 | 5 | 78 |
| Ridgetop | Bulk Waste | RT_ABA-12 | 1 | 0.8 | 7 | 0.5 | 434 | 1.0 | 0.08 | 2 | 634 | 35 | 0.01 | 1.0 | 5 | 34 | 0.13 | 5 | 5 | 55 | 5 | 54 |
| Ridgetop | Bulk Waste | RT_ABA-14 | 1 | 0.8 | 9 | 0.5 | 462 | 0.6 | 0.08 | 2 | 554 | 18 | 0.01 | 2.6 | 6 | 41 | 0.14 | 5 | 5 | 52 | 5 | 75 |
| Ridgetop | Bulk Waste | RT_ABA-15 | 1 | 0.3 | 8 | 0.1 | 478 | 1.7 | 0.05 | 1 | 854 | 6 | 0.04 | 1.0 | 1 | 35 | 0.01 | 5 | 5 | 21 | 5 | 63 |
| Ridgetop | Bulk Waste | RT_ABA-16 | 0 | 0.4 | 12 | 0.9 | 1111 | 16.1 | 0.04 | 2 | 797 | 13 | 0.08 | 1.3 | 2 | 84 | 0.00 | 5 | 7 | 30 | 5 | 143 |
| Ridgetop | Bulk Waste | RT_ABA-17 | 1 | 0.3 | 10 | 0.8 | 970 | 3.3 | 0.05 | 2 | 620 | 8 | 0.04 | 1.2 | 2 | 64 | 0.00 | 5 | 5 | 31 | 5 | 100 |
| Ridgetop | Bulk Waste | RT_ABA-18 | 1 | 0.8 | 18 | 0.6 | 1478 | 0.5 | 0.05 | 2 | 1041 | 16 | 0.02 | 1.0 | 7 | 49 | 0.13 | 5 | 5 | 86 | 5 | 235 |
| Ridgetop | Bulk Waste | RT_ABA-19 | 1 | 0.4 | 12 | 0.5 | 575 | 0.5 | 0.07 | 2 | 700 | 5 | 0.01 | 1.0 | 7 | 97 | 0.06 | 5 | 5 | 46 | 5 | 70 |
| Ridgetop | Min Waste | RT_ABA-5 | 1 | 1.2 | 20 | 1.2 | 1655 | 1.0 | 0.05 | 1 | 1198 | 15 | 0.41 | 1.0 | 8 | 55 | 0.21 | 5 | 5 | 105 | 5 | 406 |
| Ridgetop | Min Waste | RT_ABA-20 | 1 | 0.4 | 10 | 0.5 | 499 | 1.3 | 0.07 | 2 | 824 | 4 | 0.11 | 1.2 | 7 | 84 | 0.06 | 5 | 5 | 49 | 5 | 68 |
| Ridgetop | Ore | RT_ABA-4 | 1 | 0.8 | 12 | 0.7 | 318 | 1.5 | 0.03 | 1 | 886 | 7 | 0.09 | 1.3 | 6 | 63 | 0.15 | 5 | 5 | 98 | 5 | 67 |
| Ridgetop | Ore | RT_ABA-13 | 1 | 0.6 | 6 | 0.5 | 435 | 29.3 | 0.08 | 3 | 901 | 4 | 0.35 | 2.0 | 4 | 64 | 0.08 | 5 | 5 | 79 | 5 | 95 |
| Minto North | Hanging wall | G0755001 | <0.01 | 0.5 | 6 | 0.6 | 437 | 0.2 | 0.08 | 4 | 0 | 2 | <0.05 | <0.1 | 2 | 71 | 0.10 | 0 | 0 | 41 | <0.1 | 60 |
| Minto North | Hanging wall | G0755002 | <0.01 | 0.3 | 8 | 0.7 | 483 | 0.2 | 0.09 | 4 | 0 | 2 | <0.05 | <0.1 | 3 | 44 | 0.10 | <0.1 | 0 | 41 | <0.1 | 60 |
| Minto North | Hanging wall | G0755003 | <0.01 | 0.5 | 9 | 0.6 | 473 | 0.2 | 0.07 | 4 | 0 | 2 | <0.05 | <0.1 | 3 | 71 | 0.10 | 0 | 0 | 41 | <0.1 | 63 |
| Minto North | Hanging wall | G0755004 | <0.01 | 0.1 | 8 | 0.6 | 442 | 0.2 | 0.04 | 2 | 0 | 3 | <0.05 | <0.1 | 3 | 105 | 0.04 | <0.1 | 0 | 33 | <0.1 | 47 |
| Minto North | Hanging wall | G0755006 | <0.01 | 0.1 | 8 | 0.6 | 442 | 0.2 | 0.04 | 2 | 0 | 3 | <0.05 | <0.1 | 3 | 105 | 0.04 | <0.1 | 0 | 33 | <0.1 | 47 |
| Minto North | Hanging wall | G0755007 | <0.01 | 0.1 | 7 | 0.8 | 484 | 0.2 | 0.04 | 3 | 0 | 3 | <0.05 | <0.1 | 3 | 90 | 0.06 | <0.1 | 0 | 36 | <0.1 | 61 |
| Minto North | Hanging wall | G0755008 | <0.01 | 0.2 | 8 | 0.5 | 415 | 0.2 | 0.04 | 3 | 0 | 2 | <0.05 | <0.1 | 2 | 131 | 0.03 | <0.1 | 0 | 27 | <0.1 | 55 |
| Minto North | Hanging wall | G0755010 | <0.01 | 0.4 | 7 | 0.6 | 429 | 0.3 | 0.07 | 3 | 0 | 2 | <0.05 | <0.1 | 3 | 68 | 0.11 | <0.1 | 0 | 43 | <0.1 | 48 |
| Minto North | Hanging wall | G0755011 | <0.01 | 0.2 | 7 | 0.7 | 449 | 0.2 | 0.05 | 3 | 0 | 3 | <0.05 | <0.1 | 3 | 140 | 0.06 | <0.1 | 0 | 38 | <0.1 | 58 |
| Minto North | Hanging wall | G0755012 | <0.01 | 0.3 | 10 | 0.7 | 463 | 0.2 | 0.06 | 3 | 0 | 3 | <0.05 | <0.1 | 3 | 84 | 0.10 | <0.1 | 0 | 41 | <0.1 | 62 |
| Minto North | Hanging wall | G0755014 | <0.01 | 0.5 | 7 | 0.6 | 405 | 0.2 | 0.08 | 3 | 0 | 1 | <0.05 | <0.1 | 2 | 35 | 0.11 | <0.1 | 0 | 43 | <0.1 | 41 |
| Minto North | Hanging wall | G0755015 | <0.01 | 0.3 | 7 | 0.7 | 462 | 0.3 | 0.07 | 3 | 0 | 2 | <0.05 | <0.1 | 3 | 39 | 0.11 | <0.1 | 0 | 46 | <0.1 | 52 |
| Minto North | Hanging wall | G0755016 | <0.01 | 0.3 | 7 | 0.7 | 457 | 0.1 | 0.06 | 3 | 0 | 3 | <0.05 | <0.1 | 3 | 108 | 0.07 | <0.1 | 2 | 39 | <0.1 | 58 |
| Minto North | Hanging wall | G0755017 | <0.01 | 0.6 | 9 | 0.7 | 525 | 0.2 | 0.06 | 3 | 0 | 3 | <0.05 | <0.1 | 3 | 70 | 0.11 | 0 | 0 | 44 | <0.1 | 62 |
| Minto North | Hanging wall | G0755019 | <0.01 | 0.2 | 7 | 0.6 | 424 | 0.2 | 0.06 | 3 | 0 | 2 | <0.05 | <0.1 | 2 | 63 | 0.07 | <0.1 | 0 | 37 | <0.1 | 47 |
| Minto North | Hanging wall | G0755020 | <0.01 | 0.8 | 7 | 0.7 | 497 | 0.2 | 0.06 | 3 | 0 | 1 | <0.05 | <0.1 | 4 | 87 | 0.14 | 0 | 0 | 52 | <0.1 | 67 |
| Minto North | Hanging wall | G0755021 | <0.01 | 0.5 | 6 | 0.6 | 486 | 2.3 | 0.05 | 5 | 0 | 1 | <0.05 | <0.1 | 4 | 63 | 0.09 | 0 | 0 | 50 | <0.1 | 54 |
| Minto North | Hanging wall | G0755022 | <0.01 | 0.8 | 12 | 0.8 | 616 | 0.2 | 0.05 | 4 | 0 | 2 | <0.05 | <0.1 | 3 | 59 | 0.13 | 0 | 0 | 50 | <0.1 | 86 |
| Minto North | Footwall | G0755023 | <0.01 | 0.2 | 12 | 0.6 | 465 | 0.2 | 0.04 | 3 | 0 | 4 | <0.05 | <0.1 | 1 | 112 | 0.01 | <0.1 | 0 | 25 | <0.1 | 77 |
| Minto North | Footwall | G0755005 | <0.01 | 0.3 | 12 | 0.6 | 475 | 0.2 | 0.05 | 3 | 0 | 2 | <0.05 | <0.1 | 3 | 63 | 0.07 | 0 | 0 | 37 | <0.1 | 54 |
| Minto North | Footwall | G0755009 | <0.01 | 0.6 | 13 | 0.7 | 440 | 8.8 | 0.06 | 3 | 0 | 1 | <0.05 | <0.1 | 3 | 44 | 0.13 | 0 | 1 | 49 | <0.1 | 58 |
| Minto North | Footwall | G0755018 | <0.01 | 0.3 | 7 | 0.7 | 491 | 0.2 | 0.06 | 3 | 0 | 2 | <0.05 | <0.1 | 2 | 46 | 0.11 | <0.1 | 0 | 42 | <0.1 | 64 |
| Minto North | Footwall | G0755013 | <0.01 | 0.5 | 7 | 0.7 | 467 | 0.2 | 0.06 | 3 | 0 | 2 | <0.05 | <0.1 | 2 | 54 | 0.12 | <0.1 | 0 | 45 | <0.1 | 66 |

| MineArea | Waste Class | SAMPLE | Hg_ppmICP | K_%ICP | La_ppmICP | Mg_%ICP | Mn_ppmICP | Mo_ppmICP | Na_%ICP | Ni_ppmICP | P_ppmICP | Pb_ppmICP | S_%ICP | Sb_ppmICP | Sc_ppmICP | Sr_ppmICP | Ti_%ICP | Tl_ppmICP | U_ppmICP | V_ppmICP | W_ppmICP | Zn_ppmICP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | 0.08 | 0.48 to 4.2 | 45 to 70 | 0.16 to 0.94 | 390 to 850 | 0.6 to 1.3 | 2.6 to 4.0 | 4 to 15 | 600 to 920 | 12 to 19 | 0.03 | 0.2 | 3 to 14 | 100 to 440 | 0.12 to 0.35 | 0.72 to 2.3 | 3 | 30 to 88 | 1.3 to 2.2 | 39 to 130 |
| Area2 | Bulk Waste | B463023 | 1 | 0.5 | 10 | 1.0 | 738 | 0.5 | 0.21 | 2 | 1010 | 5 | 0.01 | 1.0 | 10 | 110 | 0.17 | 5 | 5 | 68 | 5 | 77 |
| Area2 | Bulk Waste | B464081 | 1 | 0.3 | 10 | 0.8 | 526 | 0.5 | 0.14 | 3 | 740 | 4 | 0.02 | 1.0 | 4 | 86 | 0.15 | 5 | 5 | 53 | 5 | 58 |
| Area2 | Bulk Waste | B464533 | 1 | 0.7 | 5 | 0.5 | 378 | 1 | 0.11 | 3 | 390 | 4 | 0.01 | 1.0 | 4 | 48 | 0.13 | 5 | 5 | 44 | 5 | 45 |
| Area2 | Bulk Waste | B464552 | 1 | 0.7 | 10 | 0.7 | 608 | 0.5 | 0.09 | 2 | 710 | 3 | 0.01 | 1.0 | 3 | 66 | 0.16 | 10 | 5 | 61 | 5 | 72 |
| Area2 | Bulk Waste | B792721 | 1 | 0.4 | 10 | 0.7 | 665 | 0.5 | 0.09 | 3 | 720 | 1 | 0.01 | 1.0 | 6 | 50 | 0.07 | 5 | 5 | 52 | 5 | 63 |
| Area2 | Bulk Waste | B792785 | 1 | 0.3 | 10 | 0.6 | 496 | 0.5 | 0.09 | 4 | 640 | 1 | 0.01 | 1.0 | 3 | 74 | 0.06 | 5 | 5 | 43 | 5 | 69 |
| Area2 | Bulk Waste | B793427 | 1 | 1.2 | 10 | 0.9 | 534 | 0.5 | 0.13 | 2 | 1030 | 2 | 0.02 | 1.0 | 6 | 78 | 0.23 | 5 | 5 | 75 | 5 | 74 |
| Area2 | Bulk Waste | B793608 | 1 | 0.7 | 40 | 0.7 | 621 | 0.5 | 0.10 | 1 | 1040 | 2 | 0.01 | 2.0 | 3 | 57 | 0.12 | 5 | 5 | 65 | 5 | 73 |
| Area2 | Bulk Waste | B793671 | 1 | 0.7 | 20 | 1.0 | 933 | 1 | 0.14 | 2 | 1130 | 3 | 0.04 | 1.0 | 4 | 118 | 0.15 | 5 | 5 | 69 | 5 | 112 |
| Area2 | Bulk Waste | B795306 | 1 | 1.1 | 10 | 0.8 | 390 | 0.5 | 0.09 | 2 | 860 | 4 | 0.02 | 1.0 | 6 | 50 | 0.20 | 5 | 5 | 59 | 5 | 45 |
| Area2 | Bulk Waste | B795352 | 1 | 0.2 | 10 | 0.4 | 380 | 0.5 | 0.07 | 1 | 400 | 2 | 0.01 | 1.0 | 2 | 73 | 0.04 | 5 | 5 | 25 | 5 | 40 |
| Area2 | Bulk Waste | B795536 | 1 | 0.2 | 10 | 0.5 | 298 | 1 | 0.06 | 1 | 1160 | 12 | 0.03 | 1.0 | 4 | 85 | 0.03 | 5 | 5 | 51 | 5 | 48 |
| Area2 | Bulk Waste | B797575 | 1 | 0.3 | 10 | 0.7 | 594 | 0.5 | 0.08 | 1 | 720 | 1 | 0.01 | 2.0 | 5 | 110 | 0.06 | 10 | 5 | 43 | 5 | 65 |
| Area2 | Bulk Waste | C441551 | 1 | 1.0 | 20 | 0.8 | 550 | 0.5 | 0.04 | 3 | 780 | 2 | 0.01 | 1.0 | 4 | 29 | 0.19 | 5 | 5 | 66 | 5 | 66 |
| Area2 | Bulk Waste | C441560 | 1 | 0.9 | 20 | 0.9 | 906 | 0.5 | 0.05 | 1 | 830 | 2 | 0.01 | 1.0 | 8 | 58 | 0.16 | 5 | 5 | 65 | 5 | 62 |
| Area2 | Bulk Waste | C441593 | 1 | 0.9 | 5 | 0.7 | 408 | 0.5 | 0.05 | 2 | 620 | 5 | 0.06 | 1.0 | 2 | 130 | 0.19 | 5 | 5 | 60 | 5 | 50 |
| Area2 | Bulk Waste | C441609 | 1 | 1.0 | 10 | 0.8 | 524 | 0.5 | 0.04 | 3 | 1150 | 2 | 0.01 | 1.0 | 3 | 281 | 0.17 | 5 | 5 | 60 | 5 | 72 |
| Area2 | Bulk Waste | C486016 | 1 | 0.6 | 10 | 0.6 | 546 | 1 | 0.14 | 3 | 580 | 2 | 0.01 | 1.0 | 5 | 56 | 0.12 | 5 | 5 | 47 | 5 | 66 |
| Area2 | Bulk Waste | C486040 | 1 | 0.6 | 10 | 0.7 | 647 | 0.5 | 0.08 | 1 | 730 | 3 | 0.01 | 1.0 | 5 | 55 | 0.12 | 5 | 5 | 52 | 5 | 70 |
| Area2 | Bulk Waste | C486396 | 1 | 0.7 | 10 | 0.6 | 382 | 0.5 | 0.10 | 1 | 640 | 3 | 0.09 | 1.0 | 5 | 40 | 0.11 | 5 | 5 | 45 | 5 | 42 |
| Area2 | Bulk Waste | C486413 | 1 | 0.9 | 5 | 0.9 | 501 | 0.5 | 0.08 | 2 | 1190 | 1 | 0.06 | 1.0 | 2 | 2130 | 0.18 | 5 | 10 | 61 | 5 | 68 |
| Area2 | Bulk Waste | C486761 | 1 | 0.2 | 10 | 0.8 | 641 | 0.5 | 0.14 | 2 | 1230 | 5 | 0.02 | 1.0 | 7 | 117 | 0.07 | 5 | 5 | 68 | 5 | 64 |
| Area2 | Bulk Waste | C486805 | 1 | 0.5 | 10 | 0.5 | 822 | 0.5 | 0.07 | 1 | 630 | 4 | 0.01 | 1.0 | 7 | 141 | 0.08 | 5 | 5 | 46 | 5 | 77 |
| Area2_SW | Bulk Waste | A2_ABA-1 | 1 | 0.2 | 16 | 0.8 | 513 | 0.5 | 0.07 | 2 | 735 | 6 | 0.01 | 1.2 | 3 | 70 | 0.02 | 0 | 5 | 44 | 5 | 59 |
| Area2_SW | Bulk Waste | A2_ABA-8 | 1 | 0.6 | 10 | 0.7 | 544 | 0.5 | 0.12 | 3 | 786 | 4 | 0.01 | 2.1 | 4 | 57 | 0.13 | 0 | 6 | 62 | 5 | 72 |
| Area2_SW | Bulk Waste | A2_ABA-9 | 1 | 0.7 | 13 | 0.7 | 526 | 0.8 | 0.13 | 2 | 744 | 4 | 0.08 | 2.3 | 5 | 71 | 0.12 | 0 | 6 | 58 | 5 | 66 |
| Area2_SW | Bulk Waste | A2_ABA-11 | 1 | 0.7 | 22 | 0.6 | 473 | 0.8 | 0.10 | 2 | 665 | 3 | 0.01 | 1.0 | 5 | 65 | 0.12 | 0 | 5 | 57 | 5 | 66 |
| Area2_SW | Bulk Waste | A2_ABA-12 | 1 | 0.4 | 10 | 0.9 | 550 | 0.6 | 0.11 | 2 | 819 | 6 | 0.01 | 1.0 | 5 | 65 | 0.15 | 0 | 5 | 62 | 5 | 72 |
| Area2_SW | Bulk Waste | A2_ABA-13 | 1 | 0.5 | 13 | 0.6 | 431 | 0.6 | 0.10 | 1 | 598 | 4 | 0.01 | 1.0 | 2 | 52 | 0.10 | 0 | 5 | 50 | 5 | 59 |
| Area2_SW | Bulk Waste | A2_ABA-15 | 1 | 0.1 | 9 | 0.5 | 392 | 1 | 0.06 | 2 | 539 | 5 | 0.01 | 1.0 | 2 | 52 | 0.05 | 0 | 5 | 33 | 5 | 58 |
| Area2_SW | Bulk Waste | A2_ABA-16 | 1 | 0.1 | 12 | 0.6 | 544 | 0.8 | 0.06 | 2 | 758 | 3 | 0.01 | 1.0 | 3 | 53 | 0.06 | 0 | 5 | 44 | 5 | 77 |
| Area2_SW | Bulk Waste | A2_ABA-17 | 1 | 0.2 | 10 | 0.7 | 536 | 0.5 | 0.08 | 2 | 705 | 5 | 0.01 | 1.0 | 4 | 69 | 0.13 | 0 | 5 | 54 | 5 | 78 |
| Area2_SW | Bulk Waste | A2_ABA-18 | 1 | 0.1 | 15 | 1.0 | 489 | 20 | 0.13 | 28 | 753 | 13 | 0.06 | 1.0 | 4 | 283 | 0.03 | 0 | 5 | 56 | 5 | 75 |
| Area2_SW | Bulk Waste | A2_ABA-20 | 1 | 0.2 | 13 | 0.7 | 425 | 15 | 0.06 | 2 | 786 | 14 | 0.18 | 1.2 | 2 | 395 | 0.01 | 0 | 5 | 28 | 5 | 59 |
| Area2 | Min Waste | B462373 | 1 | 0.5 | 10 | 1.2 | 765 | 39 | 0.05 | 3 | 90 | 3 | 1.36 | 1.0 | 5 | 32 | 0.09 | 5 | 5 | 116 | 5 | 86 |
| Area2 | Min Waste | B464032 | 1 | 0.5 | 10 | 0.6 | 693 | 2 | 0.08 | 2 | 830 | 4 | 0.87 | 1.0 | 5 | 34 | 0.08 | 5 | 5 | 58 | 5 | 65 |
| Area2 | Min Waste | B464062 | 1 | 0.4 | 10 | 0.5 | 584 | 1 | 0.06 | 3 | 630 | 3 | 0.15 | 1.0 | 2 | 96 | 0.07 | 5 | 5 | 81 | 5 | 57 |
| Area2 | Min Waste | B793448 | 1 | 1.1 | 20 | 1.0 | 538 | 13 | 0.10 | 2 | 1120 | 4 | 0.61 | 1.0 | 10 | 104 | 0.20 | 5 | 5 | 90 | 5 | 74 |
| Area2 | Min Waste | B793458 | 1 | 0.9 | 10 | 0.7 | 479 | 1 | 0.09 | 3 | 660 | 2 | 0.03 | 1.0 | 2 | 51 | 0.16 | 5 | 5 | 53 | 5 | 58 |
| Area2 | Min Waste | B793635 | 1 | 0.6 | 20 | 0.5 | 312 | 1 | 0.08 | 1 | 160 | 3 | 0.09 | 2.0 | 3 | 33 | 0.13 | 5 | 5 | 46 | 5 | 46 |
| Area2 | Min Waste | B797524 | 1 | 1.0 | 20 | 0.9 | 591 | 1 | 0.06 | 2 | 1210 | 4 | 0.01 | 1.0 | 8 | 28 | 0.18 | 5 | 5 | 88 | 5 | 97 |
| Area2 | Min Waste | B797615 | 1 | 0.9 | 20 | 1.0 | 462 | 59 | 0.09 | 3 | 1150 | 3 | 0.30 | 1.0 | 7 | 54 | 0.16 | 5 | 5 | 89 | 5 | 101 |
| Area2 | Min Waste | C441666 | 1 | 0.5 | 10 | 0.7 | 535 | 1 | 0.13 | 2 | 740 | 3 | 0.04 | 1.0 | 1 | 88 | 0.13 | 5 | 5 | 52 | 5 | 64 |
| Area2 | Min Waste | C486779 | 1 | 0.1 | 5 | 0.7 | 298 | 1 | 0.06 | 1 | 820 | 6 | 0.34 | 1.0 | 3 | 36 | 0.01 | 5 | 5 | 34 | 5 | 43 |
| Area2_SW | Min Waste | A2_ABA-3 | 1 | 0.3 | 15 | 1.0 | 379 | 81 | 0.07 | 4 | 1073 | 7 | 0.34 | 1.8 | 3 | 136 | 0.04 | 0 | 5 | 56 | 5 | 73 |
| Area2_SW | Min Waste | A2_ABA-4 | 0 | 0.5 | 20 | 0.9 | 338 | 11 | 0.08 | 1 | 1082 | 4 | 0.25 | 1.0 | 5 | 104 | 0.09 | 0 | 5 | 69 | 5 | 66 |
| Area2_SW | Min Waste | A2_ABA-5 | 1 | 0.9 | 23 | 1.0 | 429 | 9 | 0.08 | 1 | 1080 | 5 | 0.42 | 1.0 | 9 | 80 | 0.13 | 0 | 5 | 78 | 5 | 82 |
| Area2_SW | Min Waste | A2_ABA-6 | 1 | 0.8 | 13 | 0.9 | 436 | 14 | 0.09 | 2 | 858 | 5 | 0.30 | 1.0 | 6 | 73 | 0.14 | 0 | 5 | 69 | 5 | 76 |
| Area2_SW | Min Waste | A2_ABA-7 | 1 | 0.9 | 18 | 0.8 | 458 | 46 | 0.08 | 1 | 1040 | 2 | 0.48 | 1.0 | 6 | 46 | 0.16 | 0 | 5 | 79 | 5 | 81 |
| Area2_SW | Min Waste | A2_ABA-10 | 1 | 0.6 | 7 | 0.5 | 436 | 1 | 0.08 | 1 | 718 | 3 | 0.08 | 1.0 | 4 | 58 | 0.10 | 0 | 5 | 52 | 5 | 76 |
| Area2_SW | Min Waste | A2_ABA-19 | 1 | 0.2 | 14 | 0.7 | 413 | 16 | 0.07 | 2 | 790 | 5 | 0.17 | 1.0 | 3 | 198 | 0.01 | 0 | 5 | 32 | 5 | 63 |
| Area2 | Ore | B464503 | 1 | 0.6 | 10 | 0.5 | 673 | 1 | 0.05 | 1 | 910 | 4 | 0.01 | 1.0 | 7 | 42 | 0.11 | 5 | 5 | 71 | 5 | 63 |
| Area2 | Ore | B792761 | 1 | 0.1 | 70 | 1.3 | 1760 | 846 | 0.04 | 5 | 420 | 11 | 4.50 | 5.0 | 8 | 173 | 0.01 | 5 | 5 | 81 | 5 | 65 |
| Area2 | Ore | B792807 | 1 | 0.4 | 5 | 0.8 | 1090 | 1 | 0.04 | 6 | 870 | 6 | 0.90 | 1.0 | 5 | 40 | 0.07 | 5 | 5 | 80 | 5 | 102 |
| Area2_SW | Ore | A2_ABA-2 | 1 | 0.3 | 36 | 0.7 | 483 | 116 | 0.04 | 2 | 1380 | 6 | 0.89 | 1.6 | 5 | 103 | 0.05 | 0 | 5 | 74 | 5 | 77 |
| Area2_SW | Ore | A2_ABA-14 | 1 | 0.2 | 10 | 1.0 | 335 | 14 | 0.03 | 3 | 786 | 6 | 0.17 | 1.0 | 4 | 145 | 0.08 | 0 | 5 | 85 | 5 | 73 |

| Mine Area | Notes | Sample ID | Paste pH Std. Units | CO2 % CO2 | Equiv. CaCO3 kg CaCO3/t | Total S % S | Sulphate % S | Sulphur Diff. % S | AP kg CaCO3/t | NP kg CaCO3/t | Net NP kg CaCO3/t | NP/AP Ratio | Fizz Test Visual |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | LOD | 0.01 | 0.02 | #N/A | 0.02 | 0.01 | #N/A | #N/A | 0.2 | #N/A | #N/A | #N/A |
| | | Method Code | Sobek | HCl Leach | Calc. | Leco | HCl Leach | Calc. | Calc. | Modified NP | Calc. | Calc. | Sobek |
| Minto North | Locked cycle tails, G&T report KM2420 | 2420-10 Cu (Rougher + Cleaner Tails) Composite | 8.31 | 0.64 | 15 | <0.02 | <0.01 | <0.02 | 0.6 | 19.2 | 19 | 32 | Slight |
| Area 118- Lower | Locked cycle tails, G&T report KM2351 | Composite 2351-33 | 8.39 | 1.69 | 38 | 0.04 | 0.01 | 0.03 | 0.9 | 30.1 | 29 | 32 | Slight |
| Area 118- Upper | Locked cycle tails, G&T report KM2351 | Composite 2351-34 | 8.62 | 1.98 | 45 | 0.08 | <0.01 | 0.08 | 2.5 | 38.2 | 36 | 15 | Slight |
| Ridgetop East- Lower | Locked cycle tails, G&T report KM2351 | Composite 2351-35 | 8.58 | 2.55 | 58 | 0.04 | <0.01 | 0.04 | 1.3 | 32.9 | 32 | 26 | Slight |
| Ridgetop East- Upper | Locked cycle tails, G&T report KM2351 | Composite 2351-36 | 9.01 | 1.87 | 43 | <0.02 | <0.01 | <0.02 | 0.6 | 37.2 | 37 | 62 | Slight |
| Ridgetop East- Upper | Locked cycle tails, G&T report KM2351. | Composite 2351-37 | 9.26 | 1.9 | 43 | 0.04 | <0.01 | 0.04 | 1.3 | 35.2 | 34 | 28 | Slight |
| Area 2 | K1 Zone | KM 1966-13,22 MINTO K1 | 7.9 | 2 | 45 | 0.12 | <0.01 | 0.12 | 3.8 | 57 | 53 | 15 | Slight |
| Area 2 | L+M Zone | KM 1966-29,30 MINTO L+M ZONE | 8.3 | 0.7 | 16 | 0.09 | 0.01 | 0.08 | 2.5 | 31 | 29 | 12 | Slight |
| Area 2 | N Zone | KM 1966-28 MINTO N ZONE | 8.2 | 0.2 | 5 | 0.08 | 0.02 | 0.06 | 1.9 | 22 | 20 | 12 | Slight |
| Area 2 | M Zone | KM 1966-29 MINTO M ZONE | 8.4 | 0.5 | 11 | 0.06 | 0.01 | 0.05 | 1.6 | 27 | 25 | 17 | Slight |
| Area 2 | L Zone | KM 1966-30 MINTO L ZONE | 8.2 | 0.6 | 14 | 0.1 | <0.01 | 0.1 | 3.1 | 25 | 22 | 8.0 | Slight |
| Area 2 | O Zone | KM 1966-40 MINTO O ZONE | 8.1 | 1.4 | 32 | 0.1 | <0.01 | 0.1 | 3.1 | 43 | 40 | 14 | Slight |
| Area 2 | P Zone | KM 1966-41 MINTO P ZONE | 7.8 | <0.2 | 4.5 | 0.52 | 0.37 | 0.15 | 4.7 | 18 | 13 | 3.8 | Slight |
| Area 2 | Q Zone | KM 1966-42 MINTO Q ZONE | 7.9 | 0.2 | 4.5 | 0.28 | 0.13 | 0.15 | 4.7 | 28 | 23 | 6.0 | Slight |

| Mine Area | Notes | Sample ID | Mo ppm | Cu ppm | Pb ppm | Zn ppm | Ag ppm | Ni ppm | Co ppm | Mn ppm | Fe % | As ppm | U ppm | Au ppb | Th ppm | Sr ppm | Cd ppm | Sb ppm | Bi ppm | V ppm | Ca % |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | LOD<br>Method Code | 0.1<br>1DX | 0.1<br>1DX | 0.1<br>1DX | 1<br>1DX | 0.1<br>1DX | 0.1<br>1DX | 0.1<br>1DX | 1<br>1DX | 0.01<br>1DX | 0.5<br>1DX | 0.1<br>1DX | 0.5<br>1DX | 0.1<br>1DX | 1<br>1DX | 0.1<br>1DX | 0.1<br>1DX | 0.1<br>1DX | 2<br>1DX | 0.01<br>1DX |
| Minto North | Locked cycle tails, G&T report KM2420 | 2420-10 Cu (Rougher + Cleaner Tails) Composite | 0.7 | 453 | 4 | 117 | 0.5 | 8 | 7 | 542 | 4.6 | 1.3 | 0.2 | 115.2 | 3.1 | 28 | 0.2 | <0.1 | 0.1 | 74 | 0.51 |
| Area 118- Lower | Locked cycle tails, G&T report KM2351 | Composite 2351-33 | 0.4 | 591 | 2 | 151 | 0.5 | 5 | 9 | 794 | 4.5 | <0.5 | 0.1 | 88.9 | 2 | 34 | 0.8 | <0.1 | 0.1 | 72 | 0.91 |
| Area 118- Upper | Locked cycle tails, G&T report KM2351 | Composite 2351-34 | 2.6 | 515 | 3 | 109 | 0.3 | 9 | 7 | 651 | 3.0 | 0.6 | 0.7 | 45 | 4.7 | 57 | 0.4 | <0.1 | <0.1 | 77 | 1.12 |
| Ridgetop East- Lower | Locked cycle tails, G&T report KM2351 | Composite 2351-35 | 0.7 | 384 | 3 | 129 | 0.3 | 5 | 7 | 582 | 4.5 | 1.1 | 0.2 | 66.9 | 2.8 | 25 | 0.6 | <0.1 | <0.1 | 67 | 0.87 |
| Ridgetop East- Upper | Locked cycle tails, G&T report KM2351 | Composite 2351-36 | 2.1 | 1333 | 3 | 69 | 0.2 | 3 | 5 | 396 | 2.5 | 1.1 | 0.3 | 35.9 | 3 | 28 | 0.2 | <0.1 | 0.2 | 63 | 0.84 |
| Ridgetop East- Upper | Locked cycle tails, G&T report KM2351. I | Composite 2351-37 | 2.5 | 1340 | 4 | 68 | 0.2 | 3 | 5 | 379 | 2.3 | 0.8 | 0.3 | 37.2 | 2.7 | 25 | 0.2 | <0.1 | 0.2 | 60 | 0.8 |
| Area 2 | K1 Zone | KM 1966-13,22 MINTO K1 | 7.0 | 249 | 14 | 70 | 0.2 | 5 | 7 | 677 | 3.3 | 1.3 | 1.4 | <200 | 5.7 | 90 | 0.2 | 0.21 | 0.05 | 72 | 2.13 |
| Area 2 | L+M Zone | KM 1966-29,30 MINTO L+M ZONE | 1.1 | 1330 | 43 | 229 | 2.4 | 7 | 10 | 953 | 8.1 | 2.6 | 0.35 | 200 | 3 | 33 | 0.7 | 30.2 | 0.27 | 105 | 0.97 |
| Area 2 | N Zone | KM 1966-28 MINTO N ZONE | 1.4 | 672 | 25 | 145 | 1.1 | 5 | 12 | 591 | 3.8 | 4 | 1.04 | 200 | 4.7 | 75 | 0.5 | 0.8 | 0.08 | 97 | 1.24 |
| Area 2 | M Zone | KM 1966-29 MINTO M ZONE | 0.7 | 1005 | 9 | 150 | 0.9 | 5 | 10 | 869 | 7.1 | 1.3 | 0.35 | 200 | 2.9 | 30 | 0.4 | 0.13 | 0.24 | 81 | 0.91 |
| Area 2 | L Zone | KM 1966-30 MINTO L ZONE | 0.9 | 1580 | 42 | 253 | 1.3 | 6 | 13 | 1030 | 8.3 | 3.2 | 0.21 | 200 | 2.8 | 31 | 0.7 | 1.34 | 0.28 | 130 | 0.81 |
| Area 2 | O Zone | KM 1966-40 MINTO O ZONE | 1.1 | 1570 | 5 | 107 | 1.1 | 4 | 9 | 885 | 6.1 | 1.5 | 0.46 | 200 | 2.7 | 47 | 0.2 | 0.14 | 0.31 | 72 | 1.36 |
| Area 2 | P Zone | KM 1966-41 MINTO P ZONE | 0.9 | 1940 | 5 | 151 | 1.0 | 7 | 10 | 749 | 4.7 | 1.7 | 0.27 | 200 | 4.4 | 272 | 0.5 | 0.1 | 0.15 | 87 | 1.43 |
| Area 2 | Q Zone | KM 1966-42 MINTO Q ZONE | 0.8 | 1990 | 4 | 138 | 1.1 | 5 | 10 | 771 | 5.9 | 1.8 | 0.35 | 200 | 3.5 | 196 | 0.4 | 0.16 | 0.53 | 87 | 0.99 |

| Mine Area | Notes | Sample ID | P<br>% | La<br>ppm | Cr<br>ppm | Mg<br>% | Ba<br>ppm | Ti<br>% | B<br>ppm | Al<br>% | Na<br>% | K<br>% | W<br>ppm | Hg<br>ppm | Sc<br>ppm | Tl<br>ppm | S<br>% | Ga<br>ppm | Se<br>ppm |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | LOD | 0.001 | 1 | 1 | 0.01 | 1 | 0.001 | 20 | 0.01 | 0.001 | 0.01 | 0.1 | 0.01 | 0.1 | 0.1 | 0.05 | 1 | 0.5 |
| | | Method Code | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX | 1DX |
| Minto North | Locked cycle tails, G&T report KM2420 | 2420-10 Cu (Rougher + Cleaner Tails) Composite | 0.047 | 4 | 53 | 0.53 | 128 | 0.108 | <20 | 0.92 | 0.024 | 0.5 | <0.1 | <0.01 | 2.2 | 0.2 | <0.05 | 8 | 0.6 |
| Area 118- Lower | Locked cycle tails, G&T report KM2351 | Composite 2351-33 | 0.067 | 5 | 9 | 0.68 | 238 | 0.114 | <20 | 0.92 | 0.01 | 0.68 | <0.1 | 0.02 | 2.6 | 0.3 | <0.05 | 8 | 0.6 |
| Area 118- Upper | Locked cycle tails, G&T report KM2351 | Composite 2351-34 | 0.114 | 12 | 17 | 0.91 | 322 | 0.161 | <20 | 1.08 | 0.014 | 0.99 | <0.1 | <0.01 | 4.6 | 0.3 | 0.06 | 6 | 0.6 |
| Ridgetop East- Lower | Locked cycle tails, G&T report KM2351 | Composite 2351-35 | 0.089 | 8 | 10 | 0.82 | 246 | 0.143 | <20 | 0.96 | 0.012 | 0.84 | <0.1 | <0.01 | 3.7 | 0.3 | <0.05 | 7 | 0.7 |
| Ridgetop East- Upper | Locked cycle tails, G&T report KM2351 | Composite 2351-36 | 0.065 | 7 | 5 | 0.69 | 227 | 0.105 | <20 | 0.75 | 0.016 | 0.62 | 0.2 | <0.01 | 3.5 | 0.2 | <0.05 | 5 | 1.2 |
| Ridgetop East- Upper | Locked cycle tails, G&T report KM2351. | Composite 2351-37 | 0.067 | 6 | 6 | 0.68 | 227 | 0.106 | <20 | 0.73 | 0.014 | 0.62 | 0.2 | <0.01 | 3.7 | 0.2 | <0.05 | 5 | 0.9 |
| Area 2 | K1 Zone | KM 1966-13,22 MINTO K1 | 0.101 | 30 | 6 | 0.82 | 260 | 0.112 | <10 | 1.27 | 0.02 | 0.65 | 0.1 | <0.01 | 5.9 | 0.31 | 0.13 | 7.07 | 0.9 |
| Area 2 | L+M Zone | KM 1966-29,30 MINTO L+M ZONE | 0.061 | 4.2 | 12 | 0.76 | 260 | 0.154 | <10 | 1.24 | 0.02 | 0.71 | 0.08 | 0.84 | 3.5 | 0.3 | 0.1 | 12.4 | 1.7 |
| Area 2 | N Zone | KM 1966-28 MINTO N ZONE | 0.118 | 14.7 | 9 | 1.02 | 160 | 0.223 | <10 | 1.62 | 0.04 | 0.33 | 0.08 | 0.01 | 3.9 | 0.1 | 0.08 | 9.27 | 1.2 |
| Area 2 | M Zone | KM 1966-29 MINTO M ZONE | 0.051 | 4.6 | 9 | 0.66 | 170 | 0.116 | <10 | 1.02 | 0.02 | 0.53 | 0.06 | 0.01 | 3 | 0.22 | 0.07 | 11.85 | 1.3 |
| Area 2 | L Zone | KM 1966-30 MINTO L ZONE | 0.072 | 3.8 | 9 | 0.92 | 390 | 0.215 | <10 | 1.52 | 0.02 | 1.02 | 0.07 | 0.02 | 5.1 | 0.41 | 0.11 | 14.75 | 2.2 |
| Area 2 | O Zone | KM 1966-40 MINTO O ZONE | 0.047 | 4.7 | 6 | 0.54 | 200 | 0.017 | <10 | 0.71 | 0.01 | 0.12 | 0.05 | 0.06 | 1.9 | 0.06 | 0.11 | 9.45 | 1.7 |
| Area 2 | P Zone | KM 1966-41 MINTO P ZONE | 0.101 | 8.6 | 15 | 0.99 | 190 | 0.231 | <10 | 1.45 | 0.04 | 0.47 | 0.08 | 0.02 | 3.3 | 0.13 | 0.63 | 8.98 | 1.7 |
| Area 2 | Q Zone | KM 1966-42 MINTO Q ZONE | 0.06 | 7.1 | 9 | 0.7 | 130 | 0.142 | <10 | 1.12 | 0.02 | 0.33 | 0.06 | 0.01 | 2.7 | 0.11 | 0.29 | 10.5 | 2.3 |

# Appendix D

# Processing/Metallurgy

# MINTOEX.

# MINTO PHASE IV PRE-FEASIBILITY STUDY
## PROCESS DESIGN CRITERIA

**1921**

AUSENCO

Doc No.: 1921-DC-0001

| REV | DATE | DESCRIPTION | AUTHOR | CHECKED | APPROVED |
|---|---|---|---|---|---|
| C | 22-Oct-09 | Updated post client review and for 3750 tpd PFS | CAD | DJB | DJB |
| B | 31-Jul-09 | Issued for client review | CAD | PS | PS |
| A | 31-Jul-09 | Issued for internal review | CAD | PS | PS |

Treatment plant includes the following process areas:

**Description**
Mill Building
Crushing
Grinding
Flotation
Concentrate Handling
Tailings Disposal
Reagents
Plant Services

Sources of information used for the criteria are coded as follows:

| Code | Source |
|---|---|
| 1. | Client supplied data |
| 2. | Testwork - metallurgical, process |
| 3. | Consultant report, data |
| 4. | Operating practice, industry standard |
| 5. | Vendor data |
| 6. | Engineering handbook, Regulatory Standards, Codes |
| 7. | Environmental |
| 8. | Ausenco recommendation |
| 9. | Not available.  To be provided by Client, test work, others - as available. |

Metric units are used throughout, unless otherwise specified.

A period, not comma, is used as the decimal marker.
A comma is used to separate groups of three integers.
The reference conditions for gas volume are 0°C and 101.325 kPa, at molar (ideal) gas volume of 22.414 m3/(kg×mol).
Volume is shown as "$m^3$ (normal)" or abbreviated to "$Nm^3$".

Abbreviations for common terms:

| Unit | Abbreviation/Symbol |
|---|---|
| ampere per square meter | $A/m^2$ |
| average | ave |
| bed volume | BV |
| boiling point | bp |
| cubic meter | $m^3$ |
| day | d |
| decibel | dB |
| degree Celsius | °C |
| degrees | deg |
| diameter | dia |
| direct current | dc |
| hectare | ha |
| hour | h |
| inside diameter | ID |
| kilogram | kg |
| kilogram per cubic meter | $kg/m^3$ |
| kilowatthour | kWh |
| life of mine | LOM |
| litre | L |
| maximum | max |
| meter | m |
| meter per second | m/s |
| meter per second squared | $m/s^2$ |
| metric ton | t |
| micron | mic |
| minimum | min |
| minute | min |
| mole percent | mol % |
| molecular mass (weight) | mol wt |
| parts per billion | ppb |
| parts per million | ppm |
| power factor | PF |
| run of mine | ROM |
| second | s |
| specific gravity | SG |
| square meter | $m^2$ |
| temperature | T |
| tonnes per hour | t/h |
| volume | vol |
| volume by volume | v/v |
| week | w |
| weight (mass) | wt |
| weight (mass) percent | wt % |
| weight by mass | w/w |
| weight by volume | w/v |
| year | y |

These terms are used according to the definitions following.

Design Criteria Value:

The design criteria value provides the instantaneous process criterion value. This takes account of:
- variability in process parameters
- variability in operating conditions
- flow conditions that are not continuous

The design criteria value:
- is an individual rates used for sizing equipment.
- is intended as an attainable rate at the stated operating condition.
- does not include any additional design allowance(s), by engineer or vendor, to ensure attainment.

The combination of design values neither relate to the annual productions defined nor integrate to represent a metallurgical balance.

Where it is intended that the particular equipment will have an additional catch-up capacity or include additional allowances for sizing where these factors materially affect the equipment size, these are noted in the calculations.

Flowsheet balance:

Flowsheet balance values represent steady-state average rate during utilization time per operating period.
All flowsheet balance flow rates, together with the respective utilization factors, should be consistent with a single mass balance in which all ouputs are equivalent to all inputs.
The average flow includes the utilisation factor which allows for planned stoppage, sush as maintenance, and unplanned on-line disturbances.

Utilization:

Utilization reflects the combined effect of allowed (planned) availability for that facility and the utilization effect from on-line disturbances (unplanned) of upstream or downstream equipment, or from other factors.
A utilization factor of less than unity represents operation of equipment or a facility that is not continuously on-line.

Annual Rate:

Flowsheet balance hourly rate x 24 hour x equipment utilization factor x no. of operating days per year
Flowsheet Balance daily rate x no. of operating days per year
The number of operating days per year that the plant is available for operation is 365 days.

The process criteria listed are for flowsheet balance unless specified as for design.

Where criteria flow values are for time units of less than one hour, they are intended to represent the equivalent of continuous hourly rates in the relationship above.

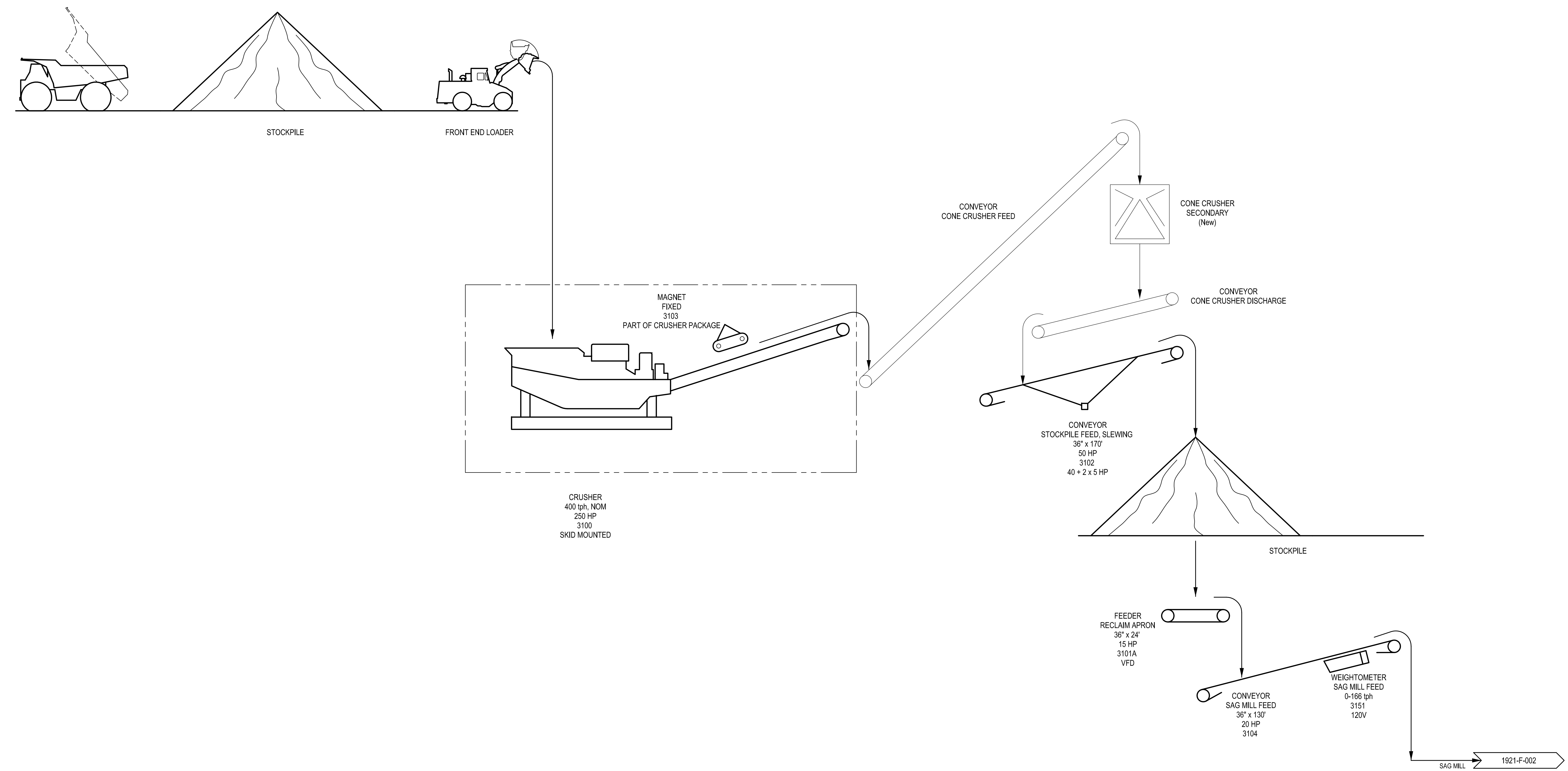
The criteria value in this column provide the instantaneous process criterion values that take account of flows that operate for less than 24 hours during one operating day, or where it is intended that the particular equipment will have an additional capacity to allow for maintenance, catch-up capability or for variability in process parameters. The Design values are intended as attainable continuous rates and do not include any additional design allowance(s), by engineer or vendor, to ensure attainment. The combination of Design values neither relate to the annual productions defined nor integrate to represent a metallurgical balance. The Design values are individual rates used for sizing equipment.

| Units | Measured Attribute | Unit | Symbol | Definition/Formula |
|---|---|---|---|---|
| Base | Length | metre | m | |
| | Mass | kilogram | kg | |
| | Time | second | s | |
| | Electric current | ampere | A | |
| | Thermodynamic temperature | Kelvin | K | |
| | Amount of substance | mole | mol | |
| Derived (basis) | Time | minute | min | 60 s |
| | | hour | h | 60 min |
| | | day | d | 24 h |
| | | (calendar) year | y | 365 d |
| | Mass | metric ton | t | 1,000 kg |
| Derived (other) | Acceleration | metre per second squared | $m/s^2$ | |
| | Angular acceleration | radian per second squared | $rad/s^2$ | |
| | Area | hectare | ha | 1 ha = 104 m2 |
| | Area | square metre | m2 | |
| | Celsius temperture | degree Celsius | °C | K-273.15 |
| | Concentration | mol per cubic metre | $mol/m^3$ | |
| | Current density | ampere per square metre | A/m2 | |
| | Density (mass) | kilogram per cubic metre | kg/m3 | |
| | Electric capacitance | farad | F | C/V |
| | Electric conductance | siemens | S | A/V |
| | Electric resistance | ohm | W | V/A |
| | Energy (electrical) | kilowatthour | kWh | 1 kWh = 3.6 MJ |
| | Energy, work, heat qty. | joule | J | N×m |
| | Force | newton | N | $kg \times m/s^2$ |
| | Frequency (periodic) | hertz | Hz | 1/s |
| | Molar energy | joule per mole | J/mol | |
| | Molar entropy | joule per mole Kelvin | J/(mol×K) | |
| | Molar heat capacity | joule per mole Kelvin | J/(mol×K) | |
| | Moment of force | newton metre | N×m | |
| | Pressure, stress | pascal | Pa | $N/m^2$ |
| | Qty. of electricity | coulomb | C | A×s |
| | Specific energy | joule per kilogram | J/kg | |
| | Specific entropy | joule per kilogram Kelvin | J/(kg×K) | |
| | Specific heat capacity | joule per kilogram Kelvin | J/(kg.K) | |
| | Specific volume | cubic metre per kilogram | m3/kg | |
| | Velocity | metre per second | m/s | |
| | Viscosity, dynamic | pascal second | Pa×s | |
| | Viscosity, kinematic | square metre per second | m2/s | |
| | Volume | litre | L | $1\ L = 10^{-3}\ m^3$ |
| | Volume | cubic metre | m3 | |

| | Units | Data | Source | Rev |
|---|---|---|---|---|
| **Site Conditions** | | | | |
| Site Location | | 240km NW of Whitehorse, Central Yukon | 1 | A |
| Altitiude | mAMSL | 787 | 1 | A |
| Ambient Temperatures | | | | |
| Maximum | ° C | 26.0 | 1 | C |
| Minimum | ° C | -43.3 | 1 | C |
| Wind Type | | | | |
| Prevailing Wind Direction | | TBA | 1 | A |
| Rainfall | | | | |
| Annual Total | mm | 378.5 | 1 | A |
| Maximum One Day Event | mm | 57.2 | 1 | A |
| **Mine** | | | | |
| Deposit | | **Minto North, Area 2/118, Ridgetop** | 1 | A |
| Ore type | | Oxide/Fresh | 1 | A |
| Main copper mineralisation assemblages | | Sulphides (Bornite, Chalcopyrite) | 2 | A |
| Ore reserves, (X.X g/t Au cut-off) | Mt | TBA | 9 | A |
| Annual ore treatment | t/y | 1,368,837 | 1 | A |
| **Feed Grade** | | | | |
| Minto North, Area 2/118, Ridgetop | | | | |
| Copper | % | TBA | 9 | A |
| gold | g/t | TBA | 9 | A |
| silver | g/t | TBA | 9 | A |
| sulphur, range | % | TBA | 9 | A |
| **Ore Properties - Crushing Circuit Design** | | | | |
| Unconfined compressive strength | MPa | - | | A |
| Crushing work index | kWh/t | - | | A |
| **Ore Properties - Grinding Circuit Design** | | | | |
| Design JK/SMC Parameters | | | | |
| A x b, range | | 65.0 | 2 | A |
| Drop Weight Index | | 4.2 | 2 | A |
| Bond rod mill work index, design | kWh/t | 10.0 | 2 | A |
| Bond ball mill work index, design | kWh/t | 13.0 | 2 | A |
| **Ore Properties - All Ore Types** | | | | |
| Bulk density of broken ROM mineralised ore, for design volume calculations | $t/m^3$ | 1.76 | 1 | A |
| Bulk density of broken ROM mineralised ore, for design mass calculations | $t/m^3$ | 1.85 | 8 | A |
| Bulk density of crushed mineralised ore, for design volume calculations | $t/m^3$ | 1.76 | 1 | A |
| Bulk density of crushed mineralised ore, for design mass calculations | $t/m^3$ | 1.85 | 8 | A |
| Specific gravity of mineralised ore, for design volumetric calculations | $t/m^3$ | 2.80 | 1 | A |
| Specific gravity of mineralised ore, for design pipe velocity calculations | $t/m^3$ | 2.90 | 8 | A |
| Bulk density of flotation concentrate at 8 % moisture, for volumetric design | $t/m^2$ | 2.20 | 8 | C |
| Specific gravity of flotation concentrate, for design volumetric calculations | $t/m^3$ | 3.00 | 4 | A |
| Specific gravity of flotation concentrate, for design pipe velocity calculations | $t/m^3$ | 3.20 | 8 | A |
| Fresh ore moisture content, for design | % $H_2O$ w/w | 3.0 | 1 | C |
| Abrasion index | | | | |
| design | | 0.6 | 1 | A |
| Angle of repose | | | | |
| ROM mineralised ore | degrees | TBA | | A |
| crushed mineralised ore | degrees | TBA | | A |
| Angle of drawdown | degrees | TBA | | A |
| **Production Schedules** | | | | |
| **Crushing** | | | | |
| Operating days per year | d/y | 365 | 4 | A |
| Shifts per day | | 2.0 | 4 | A |
| Hours per shift | h | 12.0 | 4 | A |
| Operational availability | % | 75.0 | 8 | A |
| Operating hours per year | h/y | 6,570 | | A |
| Ore throughput, nominal design | t/h | 228 | 8 | A |
| Plant throughput per day, average for mass balance | t/d | 3,750 | | A |
| Plant throughput per day, for design | t/d | 5,477 | | A |
| **Grinding** | | | | |
| Operating days per year | d/y | 365 | 4 | A |
| Shifts per day | | 2.0 | 4 | A |
| Hours per shift | h | 12.0 | 4 | A |
| Operational availability | % | 91.3 | 8 | A |
| Operating hours per year | h/y | 7,998 | | A |
| Ore throughput, nominal design | t/h | 171 | 8 | A |
| Plant throughput per day, average for mass balance | t/d | 3,750 | | A |
| Plant throughput per day, for design | t/d | 4,108 | | A |

| | Units | Data | Source | Rev |
|---|---|---|---|---|
| **ROM Ore** | | | | |
| ROM ore maximum lump size, $F_{100}$ | mm | 1,000 | 9 | A |
| ROM ore typical lump size, F80 | mm | TBA | 9 | A |
| Wheel loader to dump hopper, type | | TBA | 9 | A |
| ROM dump pocket live capacity | t | 100 | 8 | A |
| ROM bin grizzly aperture | mm x mm | 800 | 8 | A |
| **Existing Primary Crusher** | | | | |
| Product size $P_{99}$ | mm | 220 | 8 | A |
| Product size $P_{80}$ | mm | 115 | 8 | A |
| Nominal feed rate to crusher | t/h | 228 | | A |
| Maximum capacity at closed side setting | t/h | 370 | 5 | A |
| **New Secondary Crusher** | | | | |
| Product size $P_{99}$ | mm | 50.0 | 4 | C |
| Product size $P_{80}$ | mm | 26.5 | 4 | C |
| Crusher closed side setting | mm | 25.0 | 1 | C |
| Nominal feed rate to crusher | t/h | 228 | 1 | C |
| Maximum capacity at closed side setting | t/h | 205 | 5 | C |
| **Existing Crushed Ore Stockpile** | | | | |
| Maximum live storage capacity | t | TBC | 9 | C |
| Total stockpile capacity | t | TBC | 9 | C |
| **Existing Reclaim Feeder** | | | | |
| Number of feeders | | 1.0 | 1 | C |
| Type | | Apron | 1 | A |
| Design capacity per feeder, % total SAG feed | % | 110.0 | 8 | A |
| Design capacity per feeder | t/h | 188 | | A |
| **Existing grinding Circuit** | | | | |
| Circuit type | | No new mills. Upgrade and optimisation of pumps, cyclones and screens | 8 | A |
| SAG mill specific energy (Esp) | kWh/t | 3.5 | 8 | A |
| SAG power draw, at pinion | kW | 606 | 8 | A |
| SAG ball charge, % total filling | % | 18 | 8 | A |
| Ball mill specific energy (Esp) | kWh/t | 7.0 | 8 | C |
| Ball Mill power draw, at pinion (per mill) | kW | 602 | 8 | C |
| Ball Mill ball charge, % total filling | % | 33 | 8 | A |
| Cyclone overflow P80 | µm | 250.0 | 1 | A |
| **Flotation** | | | | |
| **Float Scale-up Factors** | | | | |
| Residence time scale up from laboratory batch test | | | | |
| rougher/scavenger | | 2.5 | 8 | C |
| Concentrate froth factor | | 2.0 | 8 | A |
| Aeration hold-up factor in float cell | | | | |
| rougher | % | 10.0 | 4 | A |
| scavenger | % | 10.0 | 4 | A |

| | Units | Data | Source | Rev |
|---|---|---|---|---|
| **Increased Rougher/Scavenger Flotation** | | | | |
| Type | | | | |
| Design | | New rougher flotation cells to provide sufficient residence time to allow existing scavenger cells to be converted to cleaner 1 cells | 8 | A |
| Description | | 2 x 1500 cubic foot Outotec cells | 8 | A |
| Residence time, design | | | | |
| rougher/scavenger | mins | 32.5 | 8 | A |
| Bulk rougher/scavenger concentrate weight recovery, nominal | % | 12.3 | 1 | A |
| Bulk rougher/scavenger concentrate weight recovery, for design | % | 15.0 | 8 | C |
| Specific gravity bulk concentrate | $t/m^3$ | 3.2 | 4 | A |
| **New Regrind Mill Circuit** | | | | |
| Type | | | | |
| Design | | Rougher and scavenger concentrate pumped to the regrind circuit. | | |
| Description | | VERTIMILL | 1 | C |
| Regrind Cyclone Cluster | | | | |
| Rougher/Scavenger Slurry Density | % w/w | 22 | 9 | C |
| Regrind cyclone cluster feed slurry density | % w/w | 40 | 8 | C |
| Regrind cyclone cluster underflow slurry density | % w/w | 60 | 8 | C |
| Regrind cyclone cluster overflow slurry density | % w/w | 22 | 8 | C |
| Regrind cyclone cluster feed F80 | µm | 210 | 8 | C |
| Regrind cyclone cluster overflow P80 | µm | 60 | 1 | C |
| Regrind mill circulating load - nominal | % | 250 | 8 | C |
| Regrind mill circulating load - maximum for design | % | 400 | 8 | C |
| Regrind cyclone operating pressure | kPa | 100-150 | 8 | C |
| Regrind Mill | | | | |
| Operating work index of concentrate, design | kWh/t | 13 | 8 | C |
| Regrind total specific energy - design | kWh/t | 8.16 | 8 | C |
| Regrind circuit fresh feedrate - design | t/h | 21 | 8 | C |
| Regrind mill installed power | kW | 225 | 8 | C |
| **Increased Cleaner Capacity** | | | | |
| Type | | | | |
| Design | | Existing 4 x 500 cubic foot rougher scavenger cells to be converted to Cleaner 1. Existing cleaner 1 and 2 cells will be converted to cleaner 2 and 3. | 8 | C |
| Residence time, design | | | | |
| Cleaner 1 | mins | 32.5 | 8 | C |
| Cleaner 2 | mins | 13.0 | 8 | C |
| Cleaner 3 | mins | 10.0 | 8 | C |
| **Existing Flotation Concentrate Thickening** | | | | |
| Description | | Installation of an upgraded feedwell to the thickener including auto-dilution | | A |
| **New Flotation Concentrate Thickener Overflow Clarifier** | | | | |
| Feed to clarifier | | Flot Conc. Thickener O/F | 1 | A |
| Solids suspended in concentrate thickener overflow, design | mg/L | 4,200 | 1 | C |
| Clarifier drainage sludge density | % w/w | 30.0 | 8 | A |
| Solution clarity, suspended solids, design | mg/L | ≤ 200 | 1 | C |
| **Existing Flotation Tails Thickening** | | | | |
| Description | | Installation of an upgraded feedwell to the thickener including auto-dilution | | A |
| Thickener underflow solids density | % w/w | 50.0 | 8 | C |
| **Exisiting Reagents/Consumables** | | TBC during feasibility | | A |
| **Existing Water and Air Services** | | TBC during feasibility | | A |




NOTES:

FADED ITEMS ARE NEW EQUIPMENT

STD DETAIL DRAWINGS APPLY



| No | BY | DATE | REVISION DETAILS | CHKD | APPR |
|---|---|---|---|---|---|
| D | MA | 11/24/09 | ISSUED FOR PHASE IV STUDY | CD | CD |
| C | MA | 10/29/09 | ISSUED FOR CLIENT APPROVAL | CD | CD |
| B | MA | 10/26/09 | ISSUED FOR CLIENT REVIEW | CD | CD |
| A | MA | 10/22/09 | PRELIMINARY ISSUE | CD | CD |

| DRAWING No. | REFERENCE DRAWING TITLE | No | BY | DATE | REVISION DETAILS | CHKD | APPR |
|---|---|---|---|---|---|---|---|

| | | |
|---|---|---|
| DRAWN | M. AGANON | 11/24/09 |
| CHECKED | C. DONKIN | 11/24/09 |
| DESIGNED | | |
| DES. APPR | | |
| PROJ APPR | | |

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CLIENT: MINTO EX

TITLE: MINTO PHASE IV STUDY
ORE STOCKPILE AND RECLAIM
PROCESS FLOWSHEET

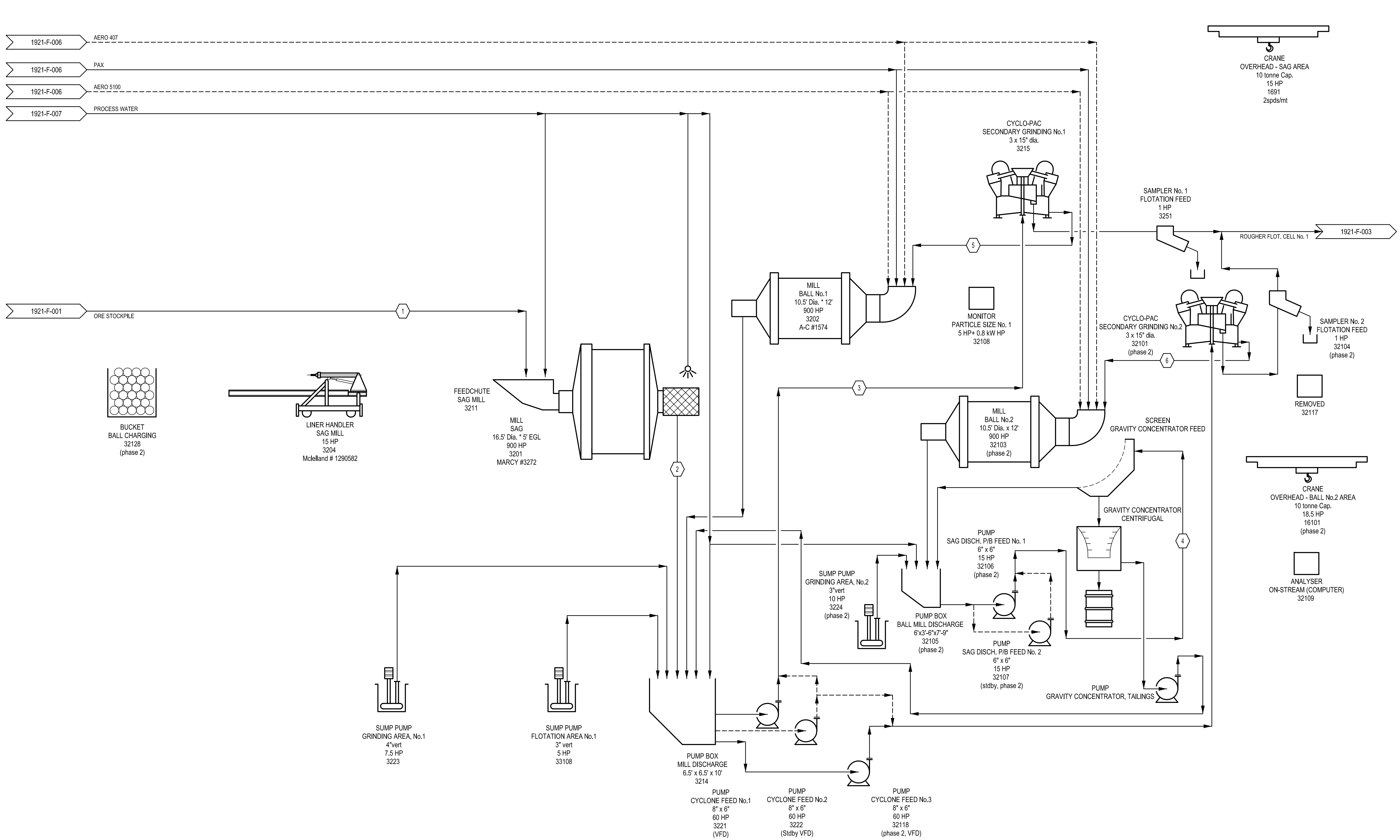
SCALE: NTS

SIZE: A1

ALL DIMENSIONS IN MILLIMETRES

REV No: A B C D

DRAWING No: 1921-F-001




STD DETAIL DRAWINGS APPLY



| DRAWING No. | REFERENCE DRAWING TITLE | No | BY | DATE | REVISION DETAILS | CHKD | APPR |
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| | | C | MA | 10/29/09 | ISSUED FOR CLIENT APPROVAL | CD | CD |
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| | | A | MA | 10/22/09 | PRELIMINARY ISSUE | CD | CD |

| | | |
|---|---|---|
| DRAWN | M. AGANON | 11/24/09 |
| CHECKED | C. DONKIN | 11/24/09 |
| DESIGNED | | |
| DES. APPR | | |
| PROJ APPR | | |

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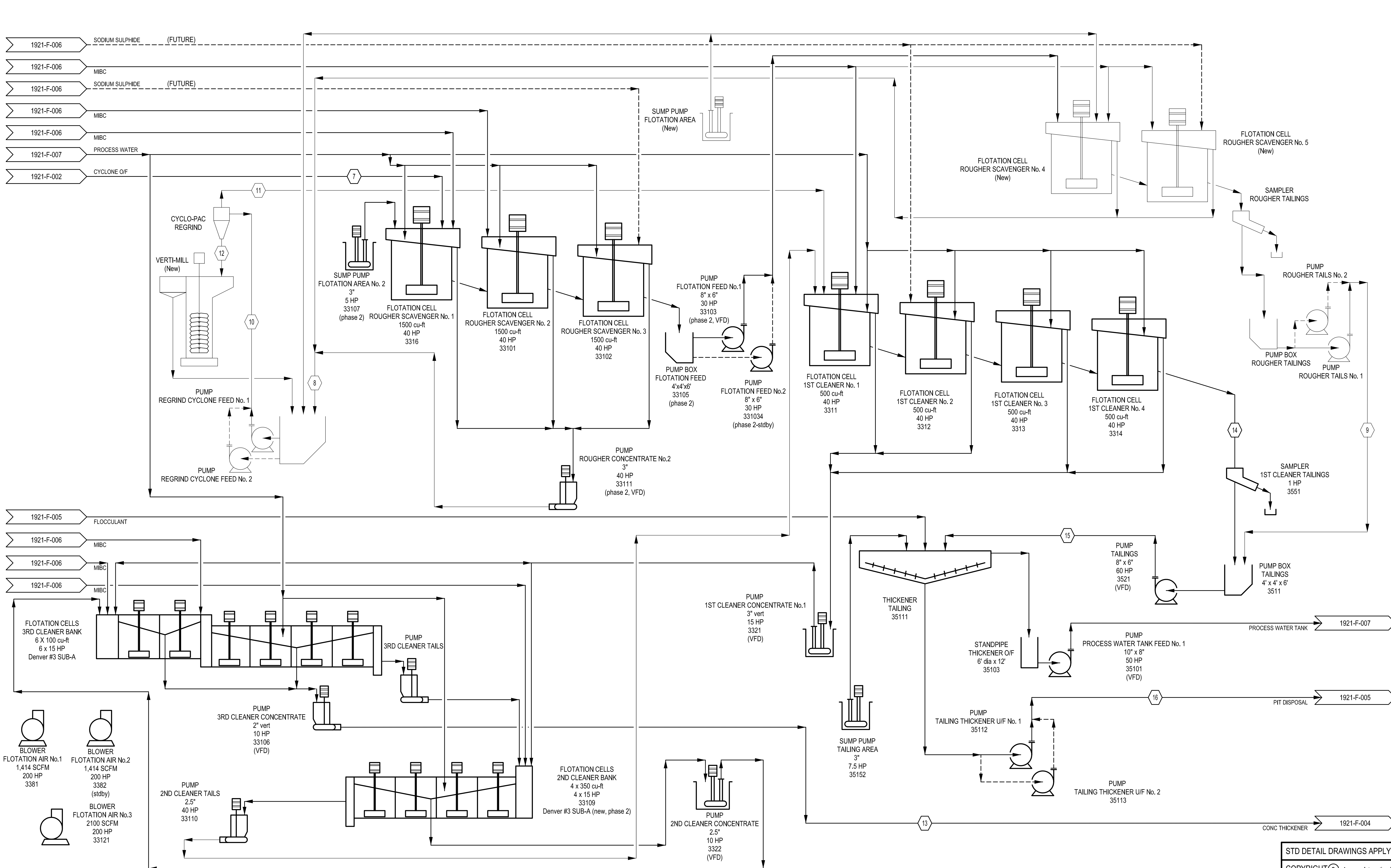
CLIENT
MINTO EX

TITLE
MINTO PHASE IV STUDY
GRINDING
PROCESS FLOWSHEET

SCALE NTS
SIZE A1
ALL DIMENSIONS IN MILLIMETRES
REV No A B C D
DRAWING No 1921-F-002




STD DETAIL DRAWINGS APPLY



| No | BY | DATE | REVISION DETAILS | CHKD | APPR |
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| B | MA | 10/26/09 | ISSUED FOR CLIENT REVIEW | CD | CD |
| A | MA | 10/22/09 | PRELIMINARY ISSUE | CD | CD |

| DRAWING No. | REFERENCE DRAWING TITLE | No | BY | DATE | REVISION DETAILS | CHKD | APPR |
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| | | |
|---|---|---|
| DRAWN | M. AGANON | 11/24/09 |
| CHECKED | C. DONKIN | 11/24/09 |
| DESIGNED | | |
| DES. APPR | | |
| PROJ APPR | | |

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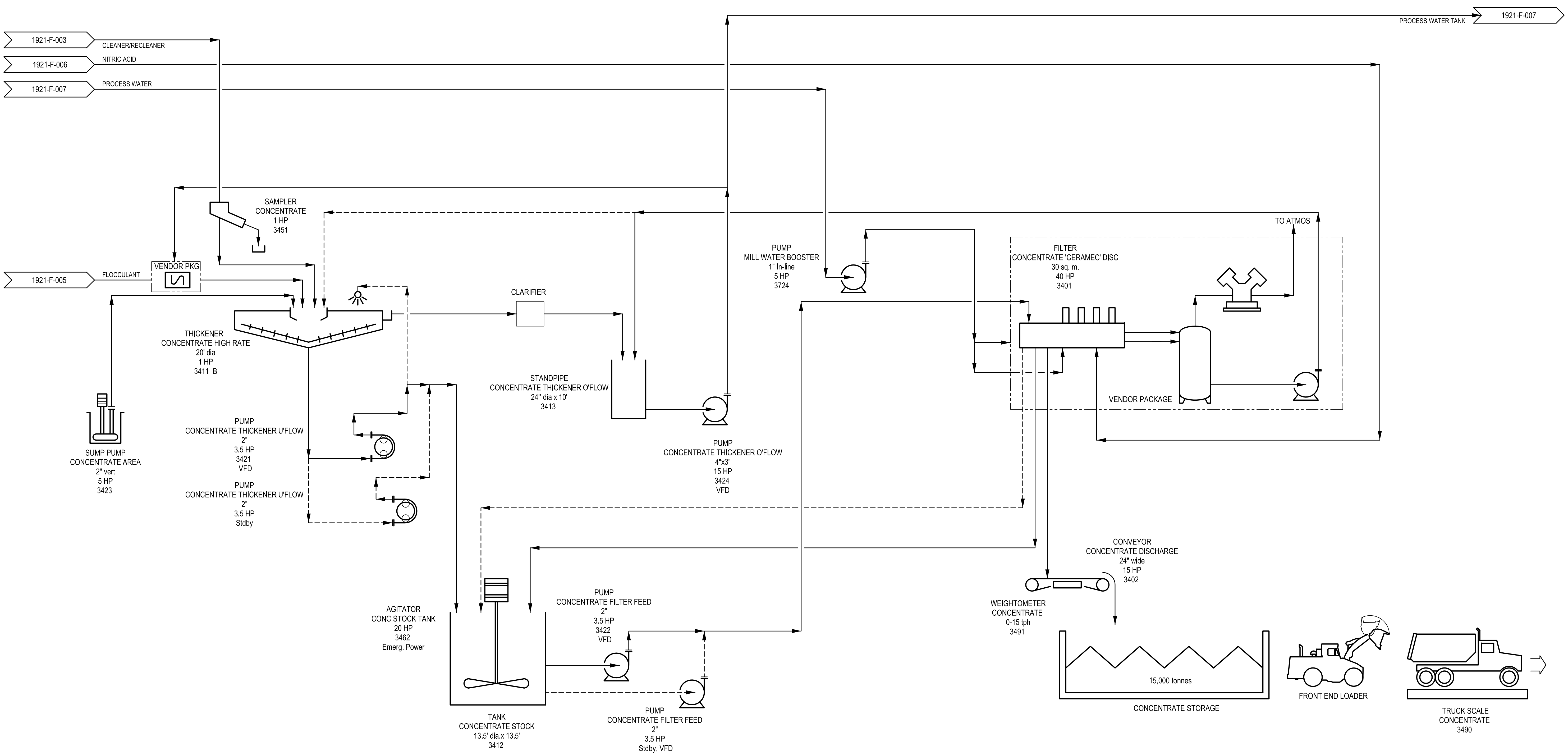
CLIENT
MINTO EX

TITLE
MINTO PHASE IV STUDY
FLOTATION
PROCESS FLOWSHEET

SCALE NTS
SIZE A1
ALL DIMENSIONS IN MILLIMETRES
REV No A B C D
DRAWING No
1921-F-003




STD DETAIL DRAWINGS APPLY



| DRAWING No. | REFERENCE DRAWING TITLE | No | BY | DATE | REVISION DETAILS | CHKD | APPR | No | BY | DATE | REVISION DETAILS | CHKD | APPR |
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| | | | | | | | | C | MA | 10/29/09 | ISSUED FOR CLIENT APPROVAL | CD | CD |
| | | | | | | | | B | MA | 10/26/09 | ISSUED FOR CLIENT REVIEW | CD | CD |
| | | | | | | | | A | MA | 10/22/09 | PRELIMINARY ISSUE | CD | CD |

| | | |
|---|---|---|
| DRAWN | M. AGANON | 11/24/09 |
| CHECKED | C. DONKIN | 11/24/09 |
| DESIGNED | | |
| DES. APPR | | |
| PROJ APPR | | |

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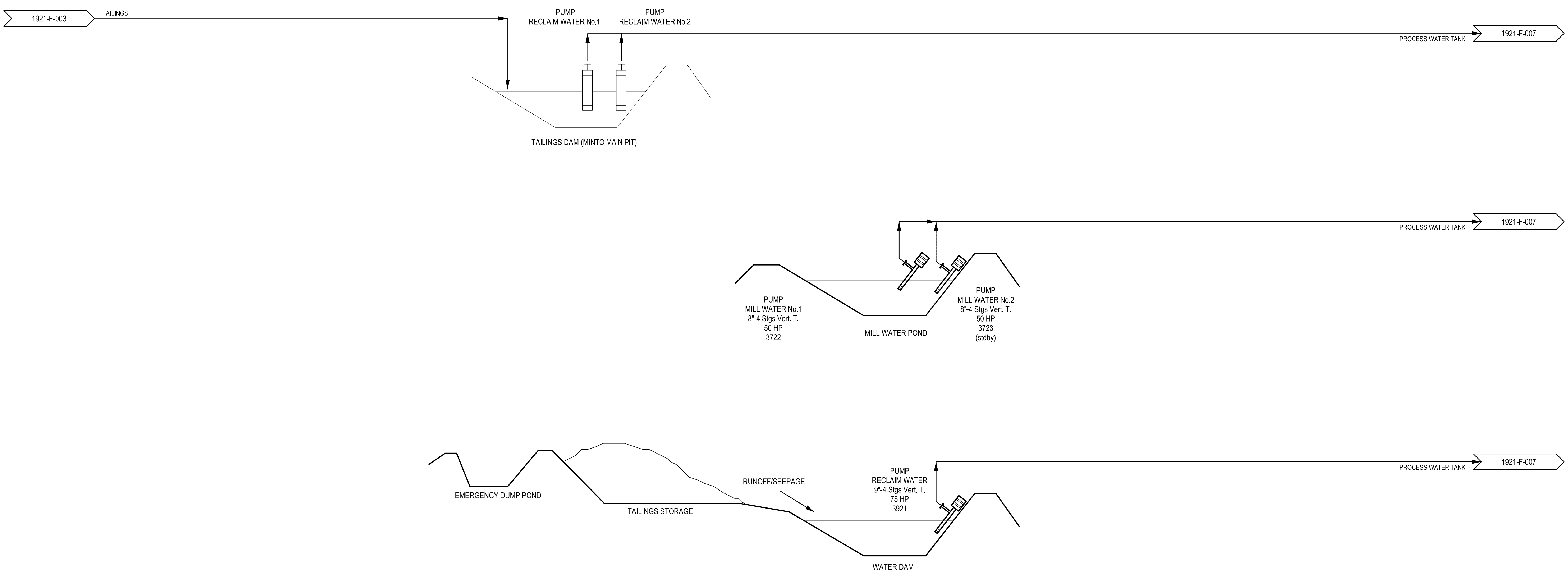
CLIENT
MINTO EX

TITLE
MINTO PHASE IV STUDY
DEWATERING, FILTRATION AND CONC. STORAGE
PROCESS FLOWSHEET

SCALE NTS
SIZE A1
ALL DIMENSIONS IN MILLIMETRES
REV No A B C D
DRAWING No
1921-F-004




STD DETAIL DRAWINGS APPLY



| DRAWING No. | REFERENCE DRAWING TITLE | No | BY | DATE | REVISION DETAILS | CHKD | APPR |
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| No | BY | DATE | REVISION DETAILS | CHKD | APPR |
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| C | MA | 10/29/09 | ISSUED FOR CLIENT APPROVAL | CD | CD |
| B | MA | 10/26/09 | ISSUED FOR CLIENT REVIEW | CD | CD |
| A | MA | 10/22/09 | PRELIMINARY ISSUE | CD | CD |

| | | |
|---|---|---|
| DRAWN | M. AGANON | 11/24/09 |
| CHECKED | C. DONKIN | 11/24/09 |
| DESIGNED | | |
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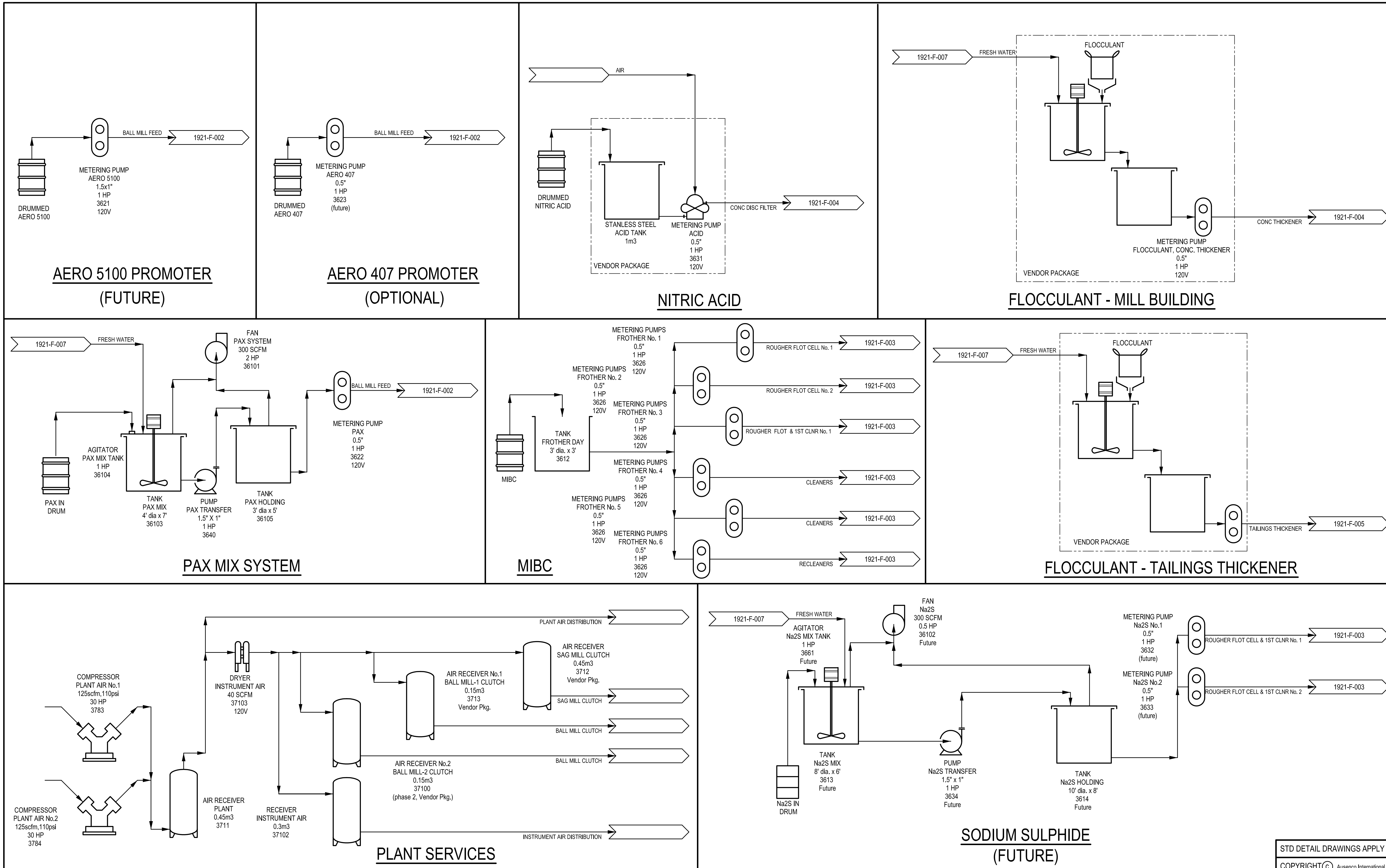
CLIENT
MINTO EX

TITLE
MINTO PHASE IV STUDY
TAILINGS, RECLAIM WATER AND PROCESS WATER
PROCESS FLOWSHEET

SCALE NTS
SIZE A1
ALL DIMENSIONS IN MILLIMETRES
REV No A B C D
DRAWING No
1921-F-005




STD DETAIL DRAWINGS APPLY



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| B | MA | 10/26/09 | ISSUED FOR CLIENT REVIEW | CD | CD |
| A | MA | 10/22/09 | PRELIMINARY ISSUE | CD | CD |

| DRAWING No. | REFERENCE DRAWING TITLE | No | BY | DATE | REVISION DETAILS | CHKD | APPR |
|---|---|---|---|---|---|---|---|

| | | |
|---|---|---|
| DRAWN | M. AGANON | 11/24/09 |
| CHECKED | C. DONKIN | 11/24/09 |
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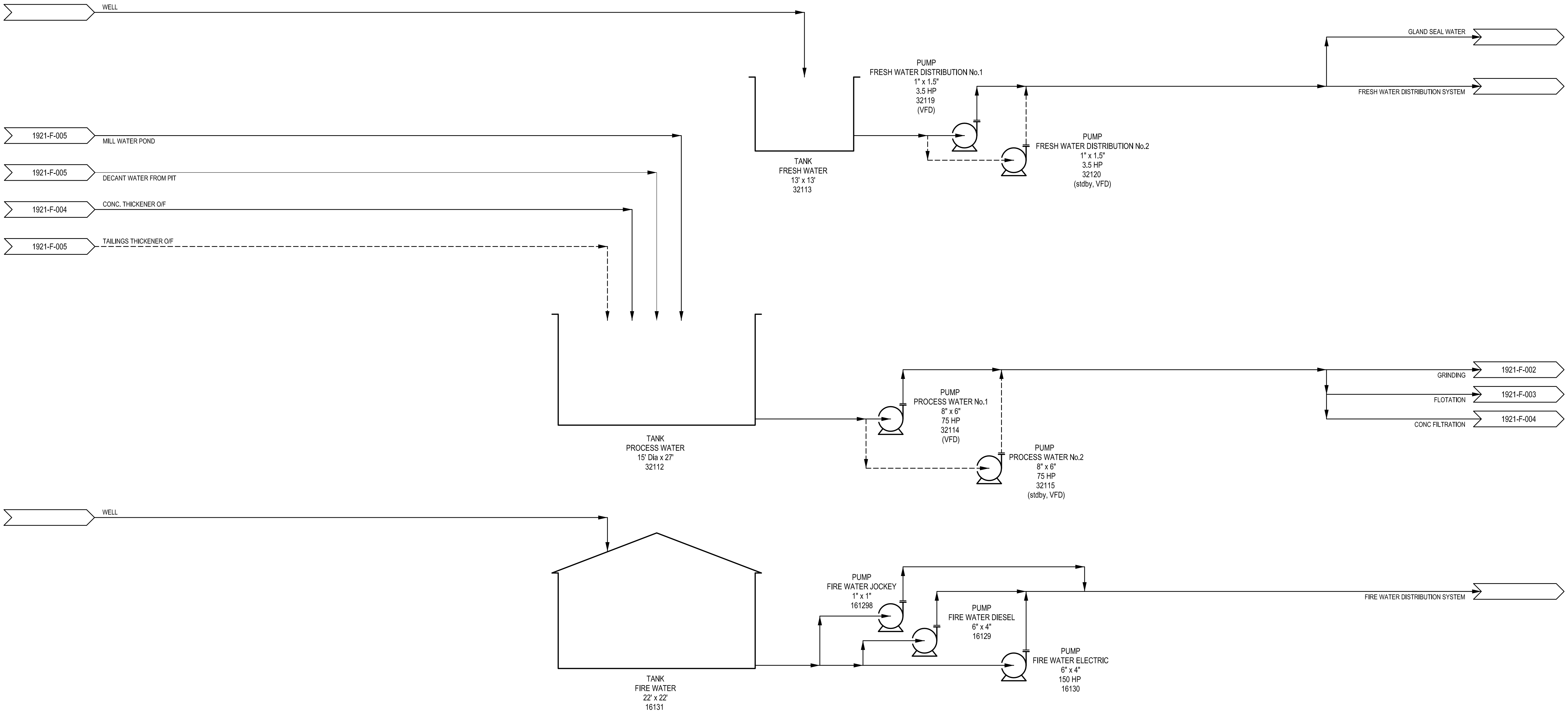
CLIENT: MINTO EX

TITLE: MINTO PHASE IV STUDY
REAGENTS AND UTILITIES
PROCESS FLOWSHEET

SCALE: NTS
SIZE: A1
ALL DIMENSIONS IN MILLIMETRES
REV No: A B C D
DRAWING No: 1921-F-006

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| | | | | | | | | C | MA | 10/29/09 | ISSUED FOR CLIENT APPROVAL | CD | CD |
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| | | | | | | | | A | MA | 10/22/09 | PRELIMINARY ISSUE | CD | CD |

| | | |
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| DRAWN | M. AGANON | 11/24/09 |
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CLIENT
MINTO EX

TITLE
MINTO PHASE IV STUDY
GENERAL WATER SYSTEM
PROCESS FLOWSHEET

STD DETAIL DRAWINGS APPLY



SCALE NTS

SIZE A1

ALL DIMENSIONS IN MILLIMETRES

REV No A B C D

DRAWING No
1921-F-007

AUSENCO Minerals

# MINTO PHASE IV PFS - MASS BALANCE

## MINTOEX.

| | |
|---|---|
| Job No | 1921 |
| Title | Minto Phase IV PFS - Mass Balance |
| Date | 23rd November 2009 |
| Rev | C |

| Stream Number (Phase IV Flowsheets) | | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Description | Unit | Total Griding Circuit Feed | Existing SAG Discharge | Cyclone Circuit 1 Feed | Cyclone Circuit 2 Feed | BM Circuit 1 Feed | BM Circuit 2 Feed | Flotation Feed | Rougher Concentrate | Rougher Tails | New Regrind Cyclone Feed | New Regrind Cyclone Overflow | New Regrind Cyclone Underflow | Final Concentrate | Cleaner 1 Tails | Flotation Final Tails | Tails Thickener Underflow |
| Solids | t/h | 171 | 171 | 342 | 342 | 257 | 257 | 171 | 21 | 150 | 74 | 21 | 53 | 10 | 11 | 161 | 161 |
| Water | t/h | 5 | 67 | 228 | 228 | 100 | 100 | 257 | 63 | 194 | 106 | 75 | 31 | 30 | 45 | 238 | 161 |
| Slurry | t/h | 176 | 238 | 570 | 570 | 357 | 357 | 428 | 84 | 344 | 180 | 96 | 84 | 40 | 56 | 400 | 322 |
| Solids | $m^3/h$ | 63 | 63 | 127 | 127 | 95 | 95 | 63 | 7 | 57 | 23 | 7 | 16 | 3 | 3 | 60 | 60 |
| Water | $m^3/h$ | 5 | 67 | 228 | 228 | 100 | 100 | 257 | 63 | 194 | 106 | 75 | 31 | 30 | 45 | 238 | 161 |
| Slurry | $m^3/h$ | 69 | 130 | 355 | 355 | 195 | 195 | 320 | 70 | 250 | 129 | 81 | 48 | 33 | 48 | 299 | 221 |
| Solids SG | | 2.70 | 2.70 | 2.70 | 2.70 | 2.70 | 2.70 | 2.70 | 3.20 | 2.70 | 3.20 | 3.20 | 3.20 | 3.20 | 3.20 | 2.70 | 2.70 |
| Water SG | | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 |
| Slurry SG | | 2.57 | 1.83 | 1.61 | 1.61 | 1.83 | 1.83 | 1.34 | 1.21 | 1.37 | 1.39 | 1.18 | 1.76 | 1.21 | 1.16 | 1.34 | 1.46 |
| Slurry % Solids | %w/w | 97.0 | 72.0 | 60.0 | 60.0 | 72.0 | 72.0 | 40.0 | 25.0 | 43.7 | 41.0 | 22.0 | 62.6 | 25.0 | 19.9 | 40.3 | 50.0 |

AUSENCO Minerals

# NEW EQUIPMENT LIST - DEBOTTLE NECKING OPTION

## MINTOEX.

## MINTO PHASE IV PFS

| REV | DATE | DESCRIPTION | BY |
|---|---|---|---|
| A | 07/10/09 | BASE CASE | JEE |
| B | 28/11/09 | General Revision | JEE |

**EQUIPMENT INFORMATION**

| Lead Engineer / Pkg Engineer | Facility No | Area | Equip-Type | Equip-No | Rev | Disc | Client Equip No | Stage | Stat | Equipment Description | Model and Specification | Duty / Standby | VSD | Consumed kW | Installed kW |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 02 | 20 | PU | | | M | | | | PUMP - CLARIFIER UNDERFLOW No. 1 | SPX 40 BREDEL | DUTY | YES | | 7.5 |
| | 02 | 20 | PU | | | M | | | | PUMP - CLARIFIER UNDERFLOW No. 2 | SPX 40 BREDEL | STANDBY | YES | | 7.5 |
| | 02 | 20 | CL | | | M | | | | HOPPER CLARIFIER | 3.5m DIAMETER, 16.1 $m^3$ | DUTY | NO | | 5.5 |
| | 02 | 30 | FC | | | M | | | | FLOTATION CELL - ROUGHER SCAVENGER No. 9 | TC40 | DUTY | NO | | 55 |
| | 02 | 30 | FC | | | M | | | | FLOTATION CELL - ROUGHER SCAVENGER No. 10 | TC40 | DUTY | NO | | 55 |
| | 02 | 30 | PU | | | M | | | | ROUGHER CONCENTRATE PUMP | SALA VT50 | DUTY | YES | | 5,5 |
| | 02 | 30 | HP | | | F | | | | PUMP BOX - ROUGHER TAILINGS | | | | | |
| | 02 | 30 | PU | | | M | | | | PUMP - ROUGHER TAILINGS No. 1 | WARMAN 8/6 E-AH | DUTY | YES | | 30 |
| | 02 | 30 | PU | | | M | | | | PUMP - ROUGHER TAILINGS No. 2 | WARMAN 8/6 E-AH | STANDBY | YES | | 30 |
| | 02 | 30 | PU | | | M | | | | ROUGHER FLOTATION AREA SUMP PUMP | 65 SP WARMAN | DUTY | NO | | 30 |
| | 02 | 30 | SA | | | M | | | | ROUGHER TAILINGS SAMPLER | 2 STAGE MULTOTEC SAMPLER | DUTY | NO | | 5.5 |
| | 02 | 30 | HP | | | F | | | | PUMP BOX - REGRIND CYCLONE FEED PUMP | | | | | |
| | 02 | 40 | PU | | | M | | | | PUMP - REGRIND CYCLONE FEED No. 1 | WARMAN 6/4 E-AH | DUTY | YES | | 55 |
| | 02 | 40 | PU | | | M | | | | PUMP - REGRIND CYCLONE FEED No. 2 | WARMAN 6/4 E-AH | STANDBY | YES | | 55 |
| | 02 | 40 | CY | | | M | | | | REGRIND CYCLONE CLUSTER | 6 gMAX10 CYCLONES | | | | |
| | 02 | 40 | ML | | | M | | | | REGRIND MILL | VTM-300 | DUTY | NO | | 225 |
| | 02 | 40 | CH | | | F | | | | REGRIND MILL DISCHARGE CHUTE | | | | | |
| | 02 | 40 | CH | | | F | | | | BALL CHARGING CHUTE | | | | | |
| | 02 | 40 | HT | | | M | | | | REGRIND AREA CRANE | 15t | | | | |
| | 02 | 40 | KB | | | F | | | | BALL CHARGING KIBBLE No. 1 | 10t CAPACITY | | | | |
| | 02 | 40 | KB | | | F | | | | BALL CHARGING KIBBLE No. 2 | 10t CAPACITY | | | | |
| | 02 | 40 | KB | | | F | | | | BALL CHARGING KIBBLE No. 3 | 10t CAPACITY | | | | |
| | 02 | 40 | KB | | | F | | | | BALL CHARGING KIBBLE No. 4 | 10t CAPACITY | | | | |
| | 02 | 40 | KB | | | F | | | | BALL CHARGING KIBBLE No. 5 | 10t CAPACITY | | | | |
| | 02 | 40 | FA | | | M | | | | FAN - WALL EXHAUST No. 1 | | DUTY | NO | | 1.5 |
| | 02 | 40 | FA | | | M | | | | FAN - WALL EXHAUST No. 2 | | DUTY | NO | | 1.5 |
| | 02 | 40 | FA | | | M | | | | FAN - WALL EXHAUST No. 3 | | DUTY | NO | | 1.5 |
| | 02 | 40 | HE | | | M | | | | HEATER No. 1 | | | | | 15 |
| | 02 | 40 | HE | | | M | | | | HEATER No. 2 | | | | | 15 |
| | 02 | 40 | HE | | | M | | | | HEATER No. 3 | | | | | 15 |