Preliminary Economic Assessment for the F2 Gold System, Phoenix Gold Project, Red Lake, Ontario

Report (amended and restated) Prepared for

Rubicon Minerals Corporation

RUBICON MINERALS

Report (amended and restated) prepared by

srk consulting

SRK Consulting (Canada) Inc.

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DATE AND SIGNATURES PAGE

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Cover: Phoenix Gold Project Area and Headframe, Red Lake, Ontario (Photo courtesy of Rubicon)

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 *Standards of Disclosure for Mineral Projects* Technical Report for Rubicon Minerals Corporation (Rubicon) by SRK Consulting (Canada) Inc. (SRK). The quality of information, conclusions, and estimates contained herein is consistent with the quality of effort involved in SRK's services. The information, conclusions, and estimates contained herein are based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Rubicon subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits Rubicon to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to National Instrument 43-101. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Rubicon. This report replaces and supersedes all prior Technical Reports for the F2 Gold System, Phoenix gold project. The user of this report should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

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CAUTIONARY STATEMENT

This preliminary economic assessment is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. The quantity and grade of reported Inferred mineral resources in this preliminary economic assessment are uncertain in nature and there has been insufficient exploration to define these Inferred mineral resources as an Indicated or Measured mineral resource and it is uncertain if further exploration will allow conversion to the Measured and Indicated categories or that the Measured and Indicated mineral resources will be converted to the Proven or Probable mineral reserves. There is no certainty that this preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability; the estimate of mineral resources in this preliminary economic assessment may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The projected mining method, potential production profile and plan and mine plan referred to in this preliminary economic assessment are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks which include, but are not limited to, the inclusion of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets and process plant designs may require additional detailed work and economic analysis and internal studies to ensure satisfactory operational conditions and decisions regarding future targeted production.

In the context of this report, the use of the word “will” should not be construed in its normal meaning, but rather in the context of “planned,” “potential,” or “possible.” In addition, the term “ore” is used in this report to differentiate between mineralized material (including dilution) above an economic cut-off grade and waste rock; there is no inference of mineral reserves.

Executive Summary

Introduction

The Phoenix gold project is a development mining project located in the gold mining district of Red Lake, Ontario, Canada. It is located approximately 265 kilometres (km) northeast of Winnipeg, Manitoba. Rubicon Minerals Corporation (Rubicon) wholly owns 100 percent (%) of the Phoenix gold project subject to a 2% net smelter return (NSR) royalty on the majority of the water portions of the property. Rubicon has the option to acquire a 0.5% interest in the NSR for US$675,000 at any time, in which case the NSR would be reduced to 1.5%.

This technical report documents a preliminary economic assessment prepared by SRK Consulting (Canada) Inc. (SRK) for the Phoenix gold project. The report also documents a Mineral Resource Statement prepared for the project. This technical report was prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1 and supersedes all prior technical reports prepared for the Phoenix gold project.

The preliminary economic assessment reported herein is a collaborative effort between Rubicon and SRK personnel. SRK was responsible for most of aspects of the study; Soutex Inc. (Soutex) was responsible for the mineral processing aspect of the study.

The projected mining method, potential production profile and plan, and mine plan referred to in this preliminary economic assessment are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks, which include, but are not limited to, the inclusion of Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets and process plant designs may require additional detailed work and economic analysis and internal studies to ensure satisfactory operational conditions and allow decisions regarding future targeted production.

Property Description and Location

The Phoenix gold project is located in the southwestern part of Bateman Township within the Red Lake mining district of northwestern Ontario, Canada. The total area of the mineral tenure is 510.4 hectares. The Phoenix gold project is centred on the historical McFinley shaft (now called the Phoenix shaft). The Phoenix gold project consists of 31 contiguous KRL or K numbered blocks.

The project site is accessible via an eight-kilometre gravel road accessed from the community of Cochenour, part of the Municipality of Red Lake. Located on East Bay of Red Lake, the project is also easily accessible by water.

History

The exploration and mining history of the Red Lake mining district dates back to 1925, when significant gold was first discovered by prospector L. B. Howey. The Phoenix gold property (previously known as the McFinley property) was initially staked and owned by McCallum Red Lake Mines Ltd. in 1922. After a series of ownership changes, Rubicon optioned the property from Dominion Goldfields Corporation in two agreements in 2002. The surface rights of the Patented Claims are now owned by 0691403 B.C. Ltd., a wholly owned subsidiary of Rubicon.

Geological Setting and Mineralization

The Phoenix gold project is located in the Uchi Subprovince of the Superior Province of the Canadian Precambrian Shield. Within the Uchi Subprovince, the Red Lake Greenstone Belt is host to one of Canada's preeminent gold producing districts with more than 26 million ounces of gold produced since the 1930s.

The tholeiitic and komatiitic metabasalts of the Balmer Assemblage, dated approximately between 3,000 and 2,988 million years before present (Ma), are the oldest volcanic rocks in the greenstone belt and host the major lode gold deposits in the Red Lake district. The lower and middle portions of the Balmer Assemblage are the host rocks for the major gold deposits of the Red Lake camp. The Phoenix gold project is underlain by the Balmer Assemblage, which is comprised of three sequences dominated by tholeiitic mafic volcanic rock, separated by distinct marker horizons of felsic and ultramafic volcanic rocks.

A strong north-northeast–trending structural fabric through the area is considered part of the East Bay Deformation Zone (EBDZ), which dominates the geology of the Phoenix gold project. The EBDZ is manifested by a well-developed, northeast-striking penetrative foliation (S1), which displays progressively steeper dips eastwards as the boundary with the adjacent F2 dominated domain is approached (eastern flank of the EBDZ). The property is interpreted to largely represent limb domains parallel to F1 structures. In the area of the existing mine shaft, the F1 foliation and the geological sequence dip approximately 50 degrees to the northwest whereas towards the southeast, in the area of the F2 gold system, which occupies the core of the EBDZ, the foliation dips are subvertical to steep northwest.

Three-dimensional lithological and structural modelling was completed by SRK to enable the recognition of faults and potential controlling structures on gold mineralization in the core zone of the F2 gold system.

Gold mineralization in the F2 gold system itself is characterized by vein and sulphide replacement styles that are preferentially hosted along the boundaries of two main rock types – HiTi basalts (high iron tholeiites) and felsic intrusive rocks (bounding units) – with additional mineralization associated with crosscutting structures. Gold, however, is distributed through all of the adjacent rock types, with the majority contained within the HiTi basalt.

In the F2 gold system, intense potassic alteration (biotite) is accompanied by variable amounts of carbonate, silica alteration, and quartz-carbonate veining. The amount of alteration throughout the F2 gold system is indicative of a robust hydrothermal system likely related to the observed gold mineralization. Gold mineralization domains were modelled in Leapfrog and GoCad software based on gold grade, structural trends, and lithological contacts. The main domains of the F2 gold system consist of 18 lower grade domains and 31 high-grade domains. The hanging wall domains consist of seven lower grade domains.

Exploration and Drilling

Since acquiring the Phoenix gold project in 2002, Rubicon has conducted an extensive exploration program, which includes geological mapping, re-logging of selected historic boreholes, digital compilation of available historical data, ground and airborne magnetic surveys, mechanical trenching, channel sampling, bathymetric survey, airborne high resolution resistivity and induced polarization (DCIP), Titan 24 geophysical survey, petrographic study, topographic survey, data modelling and processing, along with numerous drilling programs.

Since 2002 and up to November 1, 2012, Rubicon has completed 428,710 metres (m) of core drilling (229,164 m of surface drilling and 199,145 m of underground drilling) on the Phoenix gold project. During this period, 355,611 m were drilled on the F2 gold system.

In the opinion of SRK, the sampling procedures used by Rubicon are consistent with generally accepted industry best practices and the resultant drilling pattern is sufficiently dense to interpret the geometry and the boundaries of the gold mineralization with confidence. All drilling sampling was conducted by appropriately qualified personnel under the direct supervision of appropriately qualified geologists. Rubicon has postponed planned exploration to focus on the shaft construction.

Sample Preparation, Analyses and Security

All analytical or testing laboratories used are independent of Rubicon. Various analytical laboratories have been used by Rubicon over time. Samples collected before 2008 were sent to either the ALS Minerals (ALS) preparation lab in Thunder Bay, Ontario, or its analytical lab in Vancouver, British Columbia, or to Accurassay Laboratories (Accurassay), Thunder Bay, Ontario. Since January 2008, assays have been conducted by SGS Mineral Services (SGS) in Red Lake, Ontario. Umpire check assays have been completed on 5 % of these assays since January 2010 and were analyzed by ALS.

Prior to 2009, gold was analyzed using the fire assay process (with an atomic absorption or ICP finish) on a 30-gram subsample. If the sample contained greater than 10 grams per tonne (gpt) gold, it was sent for a gravimetric finish. Starting in October 2009, the assay subsample size was increased to 50 grams. Since 2009, core samples were also assayed for a suite of 50 trace elements using a multi-acid digestion followed by inductively coupled plasma atomic emission spectroscopy.

The analytical quality control program developed by Rubicon is appropriate for this exploration project and was overseen by appropriately qualified geologists. In the opinion of SRK, the exploration data from the Phoenix project were acquired using sample preparation, sample analyses, and security measures that are consistent with generally accepted industry best practices and are, therefore, adequate for resource delineation. After review, SRK considers that the sampling approach used by Rubicon did not introduce a sampling bias.

Data Verifications

Rubicon’s exploration work was conducted under a quality management system involving all stages of exploration, from drilling to data management. All field data were recorded digitally using standardized templates that ensure all relevant information was captured. Borehole data are reviewed by ioGlobal Pty Ltd. for quality assurance and quality control. Various levels of descriptive input were recorded, with appropriate validation procedures in place.

In accordance with National Instrument 43-101 guidelines, SRK visited the Phoenix gold project on various occasions between October 2011 and April 2013. At the time of the visits, underground drilling activities were ongoing on the project. The purpose of the site visits was to ascertain the geological setting of the project, witness the extent of exploration work carried out on the property, and assess logistical aspects, and other constraints relating to conducting mining activities in this area. SRK reviewed the exploration database and validation procedures, reviewed exploration procedures, defined geological modelling procedures, examined drill core, and interviewed project personnel.

Assay results for sample blanks and certified reference materials collected by Rubicon were summarized by SRK on time series plots to highlight the performance of the control samples. Paired data (field duplicates and umpire check assays) were analyzed using bias charts, quantile-quantile, and relative precision plots.

In the opinion of SRK, the results of the analytical quality control data received from SGS from 2008 to 2012 are sufficiently reliable for the purpose of resource estimation. Exploration data from before 2008 was not used for mineral resource estimation purposes.

Mineral Resource Estimates

The Mineral Resource Statement presented herein was prepared for the F2 gold system, Phoenix property in accordance with the Canadian Securities Administrators’ National Instrument 43-101.

The Mineral Resource Statement includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the Inferred mineral resources will be converted to the Measured and Indicated categories, or that the Measured and Indicated mineral resources will be converted to the Proven or Probable mineral reserves. Mineral resources that are not mineral reserves have not demonstrated economic viability; the estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The mineral resource model prepared by SRK considers 820 core boreholes drilled by Rubicon during the period of 2008 to 2012. The current drilling information is sufficiently reliable to interpret with confidence the boundaries of the gold mineralization and the assaying data is sufficiently reliable to support mineral resource estimation.

The gold mineralization wireframes were developed by SRK in collaboration with Rubicon based on sectional interpretations of geology provided by Rubicon, a three-dimensional lithological model prepared by SRK, and wireframes provided by Rubicon. Resource domains were defined using a 0.5 gpt gold threshold. Within the gold mineralization domains, narrower, high-grade subdomains were defined using a 3.0 gpt gold threshold. SRK defined 56 gold mineralization domains (31 high-grade and 25 lower grade domains) that were used to constrain mineral resource modelling. These 56 domains were combined into three groups based on their spatial orientation: Main, Main 45, and Hanging Wall (HW). The gold mineralization located outside the modelled domains was also evaluated unconstrained.

The mineral resources were modelled using a geostatistical block modelling approach constrained by the 56 gold mineralization domains. Four rotated subcell block models were generated with block sizes and orientation specific to the mineralization domain grouping. SRK chose a primary 2.5 by 5 by 10 m dimension for the Main and Main 45 domains, a 10 by 20 by 20 m dimension for the HW domain and a 5 by 10 by 20 m dimension for the External domain.

Gold and assay data were composited to a 1.0 m length and extracted for geostatistical analysis and variography. The impact of gold outliers was examined on composited data using log probability plots and cumulative statistics. SRK evaluated the spatial distributions of the gold mineralization using variograms and correlograms of original capped composited data as well as the normal score transform of the capped composited data. The block model was populated with a gold grade using ordinary kriging. Three estimation runs were used, each considering increasing search neighbourhoods and less restrictive search criteria. The first estimation pass considered search neighbourhoods adjusted to 80% of the modelled variogram ranges. A uniform specific gravity of 2.87 was applied to the lower grade domains and a value of 2.96 was assigned to the high-grade domains to convert volumes into tonnages.

SRK considers that blocks of the Main zone estimated during the first estimation pass can be classified in the Indicated category within the meaning of the CIM *Definition Standards for Mineral Resources and Mineral Reserves* (November 2010). SRK considers that for those blocks of the Main zone the level of confidence is sufficient to allow the appropriate application of technical and economic parameters to support mine planning and to allow the evaluation of the economic viability of the deposit. Conversely, all other modelled blocks were classified in the Inferred category as the confidence in the estimates is insufficient to allow for the meaningful application of technical and economic parameters or to enable an evaluation of economic viability.

SRK considers that the gold mineralization at the Phoenix gold project is amenable to underground extraction. SRK considers that it is appropriate to report the Phoenix gold project mineral resources at a cut-off grade of 4.0 gpt gold.

Mineral resources were estimated in conformity with generally accepted CIM *Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines*. The mineral resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent resource estimates. The Mineral Resource Statement for the Phoenix gold project is presented in Table i.

There is an opportunity to expand the currently reported mineral resources in areas adjacent to mineral resource blocks where, although boreholes are present with elevated gold grades, borehole density is insufficient to satisfy the applied mineral resource criteria. Targeting these areas for follow-up drilling may have a high probability of increasing the mineral resources.

Table i: Mineral Resource Statement*, Phoenix Gold Project, Ontario, SRK Consulting (Canada) Inc., June 24, 2013

Domain	Resource Category	Quantity (000't)	Grade Au (gpt)	Contained Gold (000'oz)
Main#	Measured	-	-	-
	Indicated	4,120	8.52	1,129
	Measured + Indicated	4,120	8.52	1,129
	Inferred	6,027	9.49	1,839
HW	Measured	-	-	-
	Indicated	-	-	-
	Measured + Indicated	-	-	-
	Inferred	151	5.21	25
External	Measured	-	-	-
	Indicated	-	-	-
	Measured + Indicated	-	-	-
	Inferred	1,274	8.66	355
Combined	Measured	-	-	-
	Indicated	4,120	8.52	1,129
	Measured + Indicated	4,120	8.52	1,129
	Inferred	7,452	9.26	2,219

* Mineral resources are not mineral reserves and do not have a demonstrated economic viability. All figures have been rounded to reflect the relative accuracy of the estimates. Reported at a cut-off grade of 4.0 gpt gold and assuming an underground extraction scenario, a gold price of US$1,500 per ounce, and metallurgical recovery of 92.5%.

\# The Main domain includes the Main 45 domain.

Mineral Reserve Estimates

There are no mineral reserves at the Phoenix gold project.

Conceptual Mining Methods

The projected mining method, potential production profile and plan, and mine plan are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks, which include, but are not limited to, the inclusion of Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets, and process plant designs may require additional detailed work and economic analysis and internal studies to ensure satisfactory operational conditions and decisions regarding future targeted production.

To the extent that the use of the term “ore” occurs in this technical report, its use is intended solely to differentiate between mineralized material (including dilution) above an economic cut-off grade and waste rock; there is no inference of mineral reserves.

Access for exploration to the F2 gold system was from the existing Phoenix shaft and levels. Levels are currently established at the 122 m, 183 m, 244 m, 305 m, 488 m, and 610 m horizons. The 244 level is currently advanced to within 50 m of the deposit. Mining on the 305 level includes drift access into the deposit where two approximately 1,000-tonne bulk samples were excavated. Additional access from the shaft to the deposit is required on all levels for mining.

A conceptual mining plan was developed by SRK to assess the feasibility of profitably extracting gold from the deposit. The plan envisages the development of an underground trackless longhole open stoping and cut and fill stoping operation with main track haulage on the 610 and 1285 levels. The maximum planned production rate is 2,250 tonnes per day (tpd) to be processed in a 2,500 tpd capacity mill.

A geotechnical assessment conducted by SRK included review of available data and previous studies, underground mapping on the 305 level, diamond drill core logging, and assessment of drill core photographs. It was estimated that 15% external dilution could be achieved with hanging wall cable bolting of the stopes.

Considering the steep dip of the deposit and the estimated rock mass strength, SRK determined that longhole (LH) mining with paste backfill was suitable for areas of the deposit with regular geometry while areas with irregular shape should be mined by cut and fill (CAF). The proportion of each method will be 90% LH and 10% CAF based on tonnage. The stoping outlines were based on mineral resources above a cut-off grade of 5 gpt gold and include an average inherent internal dilution of 26% from within the resource envelopes. Mining is estimated to economically extract 78% (by ounces) of the target mineral resources above a cut-off grade of 5 gpt gold. The potential mineable mineralization is based on an average external dilution of 15% grading 0.68 gpt gold, and a 95% mining recovery.

The interval between the main levels is nominally 61 m. This was based on analysis by SRK that considered the trade-off between minimizing lateral development and controlling external dilution. The main levels and sublevels will be accessed by an internal ramp system.

The variable shape of the mineralization requires a flexible mining plan that will provide access to four or five levels at any given time to achieve the peak production rate of 2,250 tpd. The average life-of-mine (LoM) production rate is 1,914 tpd. Four variations of the LH stoping method, as well as CAF mining, were adapted to the variable nature of the deposit to successfully achieve maximum recovery of the mineralization.

The main ramp will provide access to sublevels located between the main levels to facilitate mining of certain stopes located in the central mine area. Sublevels are set nominally at 30-metre intervals for the larger transverse LH stopes, with sublevel spacing reducing as the mining width becomes narrower. All stopes less than 5.0 m in width will have 15-metre spaced captive sublevels accessed by Alimak raises. The captive sublevels can be vertically adjusted closer or further apart to best match the stope geometry to the shape of the mineralization.

Five variations of mining methods were identified to manage the variation in stope widths and geometries:

- Irregular shapes – conventional or mechanized CAF; this method comprises 10% of all stope tonnes;
- 1.8 m to 3.5 m wide – Alimak longhole at 60 m high by 15 m panels on strike;
- 3.5 m to 5.0 m wide – Alimak longhole at 30 m high by 15 m panels on strike;
- 5.0 m to 10.0 m wide – longitudinal retreat longhole mining on 30-metre sublevels; and
- 10.0+ m wide – transverse longhole on 30-metre sublevels; this variation comprises 57% of all LH tonnes.

The resulting weighted average stope width for all mining method variations is 7.8 m. The width of the deposit is variable and exceeds 30 m in some areas.

For stopes less than 5.0 m wide, an Alimak raise will be driven between 60-metre spaced levels. At 15-metre vertical intervals, sublevels will be driven along strike to provide access for down-hole production drilling.

Drill borehole diameter will range from 100 - 115 millimetres (4.0 - 4.5″) for all stopes over 5.0 m wide and will be reduced to 54 millimetres (2-1/8″) for LH stopes less than 5.0 m wide. All blasted stope material will be removed by load-haul-dump (LHD) units using remote control as required. Blasted stope material will be hauled on the lower main level by LHD or truck to a central raise system designed to handle broken rock.

The stope boundaries will be determined by definition diamond drilling to decide the location of stope development drifts and raises. During development, drifts will be driven under survey control while the extent of mineralization will be further delineated by geological sampling. Stope outlines will be revised as required. The definition drilling will begin on a conservatively close pattern and be adjusted based on reconciliation of results from the in stope sampling.

Stopes will be backfilled with development rock where practical and cemented paste backfill.

Planned equipment sizes are limited by the shaft dimensions to 3.5-cubic-yards LHDs and 20-tonne trucks. This size equipment is available in battery powered models, which will be tested for short haulage distances on levels to reduce the ventilation requirements. Diesel equipment will be required for ramp development and longer haulage routes.

One central rock pass will handle development rock. Three muck passes for handling mineralized rock will be located to equalize LHD haulage distances on a tonnage weighted basis. The passes reporting to the 305, 610 and 1285 levels will have chutes for loading into trains. The existing 3-tonne manual loading pocket below the 305 level is available to support development above the 305 level. A 10-tonne loading pocket below the 610 level will be installed as part of the shaft deepening with ore and waste passes driven to support production between the 610 and 122 levels.

The existing hoist will be used to move employees and materials between the surface and underground levels as well as hoist mineralized production material and waste rock. Main levels access points from the shaft will be established on levels 122, 183, 244, 305, 488 and 610.

The shaft is currently being deepened to the 610 level (710 level shaft bottom). A loading pocket will be installed at the 680 level to service the planned 10-tonne skips. Other surface facilities have been installed for administration, camp facilities, diamond drill core logging, warehousing, and storage. A security gate and perimeter fence control access to the site.

The shaft will be extended later from the 710 level to the 1285 level (1400 m shaft bottom depth) to coincide with hoisting requirements for the development and production planned for proposed mining below the 610 level. Shaft deepening facilities, including a sinking winder on the 725 level below the existing shaft, will be accessed via a ramp. Levels will be established from the shaft on the 793, 976, 1098, and 1285-metre elevations. A 16-tonne loading pocket and crusher will be installed below the 1342 level, similar to the arrangement below the 610 level. The shaft extension will be directly in line with the existing shaft and the two shaft segments will be tied in when the deepening is completed. The hoisting facilities will be upgraded to enable skipping from the 1360 level loading pocket. Hoisting capacity from the 1360 level pocket with 16-tonne skips and hoisting upgrades is estimated at 3,000 tonnes in 10 hours. The estimated hoisting capacity from the 10-tonne loading pocket on the 680 level with the existing hoist is 3,000 tonnes in 12 hours.

Mine ventilation will be a push-pull system with supply and return fans located on surface. Fresh air will be supplied by a single raise and return will be via two raises, one at each end of the mineralization. Airflow on the levels will be modulated with ventilation regulators at the return air raise connections.

Fresh air will be supplied to the mine in two phases. The initial phase will provide 123 m^3/sec (260,000 CFM) to the development and proposed production activities above the 366 level. Fresh air will be introduced on the 305 level and a fresh air transfer drift will be driven parallel to the shaft crosscut to deliver air to the new mine workings. An auxiliary system will provide fresh air from the 305 level to the 366 level. A single return air raise will be driven from the 305 level to the 122 level. The fresh air and return air raises will connect with the intermediate 244 and 183 levels. Mine air will be heated to +5°C with a 28 Mbtuh propane-fired heater. Return air will be via a connecting drift on the 122 level from the new mine workings to the old workings and a return air raise from the 122 level to surface.

The final ventilation phase will provide 245 m^3/sec (520,000 CFM) by doubling up on the supply fan, mine air heater, and return air system. The fresh air raise will extend to the 1342 level and two return air raises, one at either end of the mineralization, will extend from the 1464 and 1342 levels to connect with the 305 level. The initial return air system from the 305 level to surface will be duplicated at the opposite end of the mineralization. The supply air installation will include two mine air heaters rated at 28 Mbtuh each.

Main dewatering stations will be established on the 610, 976 and 1285 levels to augment existing stations on the 122 and 305 levels. Water will be pumped to surface in stages from one station to the next. Level drainage will be handled by temporary sumps and pumps on the level as well as a system of drain holes directing the water to one of the main dewatering stations. Conditions are dry in areas developed to date and groundwater seepage is very low. However, in the event that water inflows occur in excess of pumping capacity, emergency water storage will be provided by installing a bulkhead, complete with valves, at the bottom of a mined out stope. Water discharged from the mine will report to the surface water handling system for recycling as process water or for treatment. The mine dewatering system will have a capacity of 3.0 m^3/min (800 USgpm).

Metallurgical Testing and Recovery Methods

Metallurgical testing was performed by Soutex in 2010 and 2011 as a basis for process design. The gold mineralization will be processed in a 1,250 tpd on-site mill. The design made allowance for upgrading the capacity to 2,500 tpd. The process consists of a single process line, starting with a semi-autogenous grinding (SAG) mill. The discharge from the SAG mill will be sent to the ball mill circuit arranged with hydrocyclones in closed circuit for classification. A gravity separation circuit is included in the closed circuit with to partially recover and concentrate any gravity recoverable gold. The remaining gold will be extracted in a conventional carbon-in-leach (CIL) circuit.

The loaded carbon will be washed with a hydrochloric acid solution to remove carbonates. Gold will then be removed from the loaded carbon by elution (stripping) followed by electrowinning. The electrowinning and the gravity circuit will both produce high-grade gold concentrates for smelting into doré in an electric induction furnace. The stripped carbon will be regenerated in a reactivation kiln before being reintroduced to the leaching process. Fine carbon will constantly be eliminated (and recovered) from the process to avoid gold loss, with fresh carbon being continuously added to the process.

The cyanide contained in the tailings from the CIL circuit will be eliminated in a cyanide destruction tank with the SO_2-air process. Once the cyanide is destroyed, the tailings will be sent to the tailings pond for disposal, or be directed to the paste backfill plant.

At the paste backfill plant, which is integral to the mill, tailings will be thickened and filtered, and cement will be added to meet the strength requirements needed in mined out stopes. Paste backfill will be pumped underground to the stopes through a dedicated distribution system of pipes and boreholes.

The remaining tailings will be thickened and pumped to the tailings management facility (TMF) near the mill and headframe. The tailings will be contained by a series of dams that will be raised periodically in stages as additional storage capacity is required.

Development waste rock will be placed into mined out stopes as backfill to the extent possible. Rock that is hoisted will be tested for acid generation and metal leaching potential. Clean rock will be used as construction material at the site and on the access road while the rock that does not pass the test criteria will be utilized for TMF construction and/or will be placed on the downstream TMF embankment to supplement buttressing and armoring. Testing to date indicates that rock between the shaft and the F2 gold system is expected to be clean and material in the mineralized material zone could be either clean, or will need to be contained.

Project Infrastructure

Project work is already in progress and as a result much of the requisite surface infrastructure is in place or under construction. The status of infrastructure currently on site includes:

- Offices, dry, worker accommodations, and sewage systems;
- 7.5 MVA electrical substation;
- A process and potable water supply;
- Diesel fuel storage and dispensing facility;
- Compressed air supply;
- Mine ventilation system;
- A 4.27-metre (14 ft) diameter double drum hoist and headframe;
- Shaft deepening to the 710 level is in progress;

- Underground services are being installed in the shaft to provide electrical power, communications, compressed air, and pipelines for mine water supply and mine dewatering;
- A mill and Stage 1 TMF are under construction;
- Water management and treatment systems; and
- VOIP telephone system with fibre optic cable connection.

Changes to the main infrastructure required to meet potential planned production levels are:

- The dams in the TMF will be raised periodically to provide additional capacity for ongoing tailings deposition;
- The site electrical substation has a continuous operating capacity of 7.5 MVA and the supply allotted to the Phoenix project by Hydro One is currently 5.3 MVA. Additional power will be required and Rubicon is currently in discussion with Hydro One in this regard;
- The mill will be upgraded to process 2,500 tpd to meet conceptual mining schedule of 2,250 tpd; and
- Compressed air supply will be increased to 15,920 m^3/hr (9,370 CFM).

Market Studies and Contracts

The gold produced is expected to be of a quality that can be sold directly to refiners at prevailing spot gold prices.

Rubicon has successfully entered into contracts with suppliers under current market conditions. Similar contracts could be readily entered into as the project moves forward.

Environmental Studies, Permitting, and Social Impacts

Permitting is in place for an average underground mine production of 1,250 tpd and the site has an approved closure plan. For the processing plant’s maximum designed rate of 2,500 tpd, it will be necessary for Rubicon to amend select permits, including the closure plan.

Rubicon commenced an advanced exploration phase at the Phoenix project in the first quarter of 2009. Development work at the site is currently ongoing and Rubicon is permitted for commercial production at an annual average rate of 1,250 tpd.

The project site is situated on the McFinley Peninsula that is adjacent to a valued recreational lake. As such, emphasis for physical environmental sensitivities has been placed on potential off-site discharges of water, fugitive dust, and noise.

The project’s social aspects include consultation with Aboriginal communities under the guidance of government agencies. To supplement the guidance from the government agencies, Rubicon commissioned an independent traditional land use study that concluded the project site is within the traditional territory of Lac Seul First Nation (LSFN) and Wabauskang First Nation (WFN) (Forbes 2011).

An archaeological study of the McFinley Peninsula was commissioned by Rubicon, comprising a desktop study as well as field work. The study did not identify any sites with a high potential to host a cultural heritage value site within the development footprint (Ross Associates 2010). Also, as the project involves the re-development of the existing footprint with only moderate expansion, the potential for impacts to cultural heritage values as a result of the re-development of the area is considered to be negligible. Accordingly, it has been deemed reasonable to solely engage LSFN, WFN, and the Métis Nation of Ontario (MNO) to further discuss and identify potential cultural heritage value sites within the development footprint that may warrant protection.

Annual public information sessions have been held in the Red Lake community since 2008. No negative comments have been received to date during these sessions.

Rubicon maintains an issues tracking matrix as part of its environmental management system (EMS) to effectively track and manage potential concerns as they arise.

Rubicon has planned and intends to execute the project in a manner that is consistent with industry best practices and conducive to a walk-away closure. Details of decommissioning requirements during potential production and upon closure have been determined and form part of the closure plan. Chemical and physical stability requirements have been identified and will be satisfied and monitored in accordance with regulatory requirements and the Phoenix Project Closure Plan, which was filed by Ministry of Northern Development and Mines on December 2, 2011 in accordance with Section 141 of the *Mining Act*.

Capital and Operating Costs

The estimated total capital cost for the project is C$650 million (M). C$224 M is to be expended on a go-forward basis during the pre-production period from Q3-2013 to Q2-2014 and C$426 M is sustaining capital in the potential production period from Q3-2014 to Q3-2027.

The productive mine life is 13.25 years and the average operating cost, which includes operating development, is estimated at C$151 per tonne of resource material milled.

The changing potential production profile presented a challenge for calculating representative operating costs. To compensate, operating costs were split into fixed and variable components. The fixed component remained unchanged with production rate where the variable component was prorated according to production rate.

Financial Analysis

The Phoenix gold project's base case economics were evaluated on a post-tax cash flow basis. The indicative post-tax economic results include:

- Cumulative cash flow of C$897 M;
- Internal rate of return (IRR) of 27% calculated on a mid-period basis; and
- Net present value (NPV) of C$531 M at a discount rate of 5% calculated on a mid-period basis.

The project has a payback period of approximately 3.7 years for the base case from the start of commercial production in Q3-2014. The average operating cost per ounce of gold recovered is C$629 per ounce (/oz). Sustaining capital cost per tonne over the life of the mine is C$46.62 per tonne milled or C$194 per ounce of gold recovered.

The metal price for the base case scenario was US$1,385 per ounce for gold as advised by Rubicon. The exchange rate (C$/US$) of 1.05 is consistent with both the consensus forecast and historical rates.

Sensitivity of the base case post-tax NPV of C$531 M was calculated for gold price, gold grade, capital cost, operating cost and process recovery. Sensitivity analysis indicates that NPV is most sensitive, and almost equally sensitive, to changes in gold price and gold grade, which are both related to revenue.

A 20% increase in gold price from C$1,385/oz to C$1,662/oz results in a 59% increase in post-tax NPV to C$845 M. This compares to a 20% decrease in gold price to C$1,108/oz which yields a 61% decrease in post-tax NPV to C$205 M.

With respect to gold grade, a 20% increase to 9.68 gpt gold results in a 59% higher post-tax NPV of C$845 M while a 20% decrease to 6.45 gpt gold yields a 61% lower post-tax NPV of C$206 M.

This preliminary economic assessment is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the Inferred mineral resources will be converted to the Measured and Indicated categories, or that the Measured and Indicated mineral resources will be converted to the Proven or Probable mineral reserves and there is no certainty that this preliminary economic assessment will be realized. Mineral resources that are not mineral reserves have not demonstrated economic viability; the estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The projected mining method, potential production profile and plan, and mine plan are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks, which include, but are not limited to, the inclusion of Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets, and process plant designs may require additional detailed work and economic analysis and internal studies to ensure satisfactory operational conditions and decisions regarding the future targeted production.

Adjacent Properties and Other Relevant Data and Information

There are no adjacent properties that are considered relevant to this technical report. There is no other relevant data available about the Phoenix gold project.

Conclusions and Recommendations

SRK reviewed and audited the exploration data available for the Phoenix gold project as well as the exploration methodologies adopted to generate this data. SRK is of the opinion that the exploration data are sufficiently reliable to interpret with confidence the boundaries of the gold mineralization and support evaluation and classification of mineral resources in accordance with generally accepted CIM *Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines* and CIM *Definition Standards for Mineral Resources and Mineral Reserves*.

This is a preliminary economic assessment and does not constitute, nor is it intended to be, a preliminary feasibility or feasibility study. This preliminary economic assessment is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the Inferred mineral resources will be converted to the Measured and Indicated categories, or that the Measured and Indicated mineral resources will be converted to the Proven or Probable mineral reserves and there is no certainty that this preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability; the estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The projected mining method, potential production profile and plan and mine plan are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks which include, but are not limited to, the inclusion of Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets and process plant designs may require additional detailed work and economic analysis and internal studies to ensure satisfactory operational conditions and decisions regarding future targeted production.

The Mineral Resource Statement documented herein represents a significant change relative to the previous Mineral Resource Statement released in June 2011. It is significant to note the material increase in tonnages reported in 2013 at reduced grades compared to the tonnages reported in 2011. Reported tonnages and grades are the product of contrasting mineral resource estimation methodologies applied during the two studies. The additional infill drilling since 2011 contributed to the significant increase in the reported Indicated mineral resources in 2013.

The results of this preliminary economic assessment support the continued advancement of the Phoenix gold project and work related to further technical studies. No production decision has been made at this time. Such a decision, if reached, will require such additional technical studies and ongoing evaluation by Rubicon of the construction and advancement of the Phoenix gold project and would not be based solely on the results of this report. Future work on the property should address key risks and opportunities that have been identified in this technical report. SRK recommends a work program that includes further exploration drilling, geotechnical studies, metallurgical testing, and other various studies aimed at completing the characterization of the project in preparation for further engineering studies.

Geology and Exploration

Additional core drilling is required to achieve three objectives:

- Infill the remnant gaps in the drilling data with the potential to increase the mineral resources and to potentially improve resource classification;
- Step-out drilling to test to the lateral and depth extensions of the gold mineralization; and
- In preparation for production, delineation drilling, in particularly the shallow levels, will be required for short term mine planning.

Mine Geotechnical and Hydrogeology

The following testing and studies will provide information required for optimization of the potential mining plans and opportunities:

- Additional geotechnical data is required to improve the mine design. Stopes may be more or less stable than currently estimated and external dilution may be highly variable as well as dilution grade;
- In the event of water inflow from an ungrouted diamond drill borehole, provisions have been made for emergency water storage in a bulkhead mined out stopes until the borehole can be properly grouted. Mining near surface could weaken the crown pillar to the extent that mining induced cracks become water bearing. Currently, a 45 m crown pillar has been planned and other measures such as conventional grout curtains will mitigate this risk; and
- The risk of uncontrolled stope failures due to unknown structural weaknesses in the rockmass cannot be predicted.

Mine Design and Planning

A key to efficient mining is determining the outline of the gold mineralization prior to and during stope development and mining. This is necessary due to the variable shape of the deposit and the mineral concentration. The following recommendations are made to adapt to these conditions:

- Develop procedures to obtain information about the shape and extent of mineralization as a basis for mine design and stoping;
- Reconcile the block model with the data obtained as an aid to future mine planning;
- Continue to refine mining methods and equipment selection to adapt to specific conditions of deposit shape and extent of mineralization; and
- Revise the mining plan based on the revised mineral resource model(s) resulting from the proposed exploration drilling and resource modelling program.

On a larger scale, it is recommended that investigations of supplements or alternatives to the Phoenix shaft be conducted. Trade-off studies could include a new shaft or ramp to alleviate congestion in the Phoenix shaft. Possible risks to successful operation include:

- Attracting and retaining key technical staff who have the ability to work effectively with the complex nature of deposit;
- Attracting, training, and retaining the required company workforce; and
- Securing additional required project financing.

Mineral Processing

Two recommendations relating to mineral processing risks have been identified:

- Perform additional metallurgical testwork on drill core samples throughout the deposit to decrease the processing uncertainties and have a more accurate estimation of the range of possible gold recovery; and
- Further testwork aimed at reducing cyanide concentration is recommended to keep ammonia within the regulated limit for discharge.

Infrastructure

Recommendations for the surface facilities include:

- Confirm the quantity requirements for the increased design processing rate of 2,500 tpd and ensure the site is permitted for a sufficient withdrawal from Red Lake; and
- Maintain negotiations with Hydro One for an increase in electric power allotment that is required for the future operation of the mill and mine ventilation fans.

Environmental

Environmental risk is low except in the area of Aboriginal relations where a lawsuit was brought against the project by the Wabauskang First Nation in 2012. Recommendations for environmental affairs are to:

- Continue negotiations of a benefits agreement with Wabauskang First Nation;
- Continue organizing and documenting Aboriginal and public consultations;
- Continue to maintain the site environment, health and safety system in good order;
- Continue to manage the project approvals and permits with special attention to amendments required for future operations; and
- Maintain monitoring and mitigation plans for the recognized environmental sensitivities relating to water discharge, fugitive dust, and noise.

Costs and Economic Analysis

The main results of this preliminary economic assessment are:

- Mineral resource at 4 gpt gold cut-off grade: Indicated tonnage of 4.1 million (M) tonnes at 8.52 gpt gold containing 1.1 M ounces gold and Inferred tonnage of 7.5 M tonnes at 9.26 gpt gold containing 2.2 M ounces gold;
- Mining method*: five longhole and cut and fill mining methods were successfully adapted to the variable shape and thickness of the deposit to achieve an average resource extraction of 78%;
- External dilution: weighted average external dilution from all mining methods was estimated to be 15% grading 0.68 gpt gold;
- Potential mineable mineralization at 5.0 gpt gold cut-off grade: 9.13 M tonnes at 8.06 gpt gold containing 2.37 M oz;
- Recovered ounces: 2.19 M oz at 92.5 percent (%) processing recovery*;
- Production: production period is 13.25 years from Q3-2014 to end of Q3-2027; production rate reaches to a maximum of 2,250 tpd during the years 2022 to 2025;
- Recovered ounces of gold per year: average 165,300 oz/year;
- Revenue after refining cost and royalty: C$3,127 M;
- Operating cost: C$1,378 M total; average of C$151/t or C$629/oz gold;
- Capital costs:
  - Total C$650 M; average of C$71.14/t or C$297/oz gold;
  - Pre-production C$224 M (including a 20% contingency)**; and
  - Post-production C$426 M; average of C$46.62/t or C$194/oz gold;
- The indicative post-tax financial results for the Phoenix gold project are:
  - Cumulative cash flow of C$897 M;
  - NPV(5%)*** of C$531 M on a mid-period calculation basis;
  - IRR*** of 27%; and
  - 3.7 years payback from start of commercial production starting Q3-2014.

* The projected mining method, potential production profile and plan and mine plan are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks, which include, but are not limited to, the inclusion of Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets, and process plant designs may require additional detailed work and economic analysis and internal studies to ensure satisfactory operational conditions and decisions regarding future targeted production.

** As of May 31, 2013, Rubicon had working capital totalling C$118 million.

***Based on a 30-day trailing spot gold price assumption of US$1,385 per ounce and a C$/US$ consensus exchange rate of 1.05:1.00 (Source: Bloomberg C$/US$ FX Forecast 2013 through 2017, as of June 18, 2013).

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1 Introduction and Terms of Reference

The Phoenix gold project is a development mining project located in the district of Red Lake, Ontario, Canada. It is located approximately 265 kilometres (km) northeast of Winnipeg, Manitoba. Rubicon Minerals Corporation (Rubicon) wholly owns 100 percent (%) of the Phoenix gold project.

In December 2012, Rubicon commissioned SRK Consulting (Canada) Inc. (SRK) to prepare an updated geological model, a mineral resource model, and to complete a preliminary economic assessment for the Phoenix gold project. The services were rendered between December 2012 and June 2013 leading to the preparation of the Mineral Resource Statement, a proposed mining study and a preliminary economic assessment reported herein that was disclosed publically by Rubicon in a news release on June 25, 2013.

This technical report documents a preliminary economic assessment prepared by SRK for Rubicon's Phoenix gold project. This technical report was prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1. The Mineral Resource Statement reported herein was prepared in conformity with generally accepted CIM *Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines*.

A previous mineral resource model dated June 15, 2011 was prepared by AMC Mining Consultants (Canada) Pty Ltd. (AMC). That mineral resource model was considered by AMC for the preparation of an initial preliminary economic assessment for the Phoenix gold project and documented in a technical report dated August 8, 2011.

1.1 Scope of Work

The scope of work, as defined in a letter of engagement executed on December 12, 2012 between Rubicon and SRK, includes the construction of a mineral resource model for the gold mineralization delineated by drilling on the Phoenix gold project, the design at a conceptual level of an underground mine and ancillary facilities, and the preparation of a preliminary economic assessment of the resultant conceptual project. This work involves the assessment of the following aspects of this project:

- Topography, landscape, access;
- Regional and local geology;
- Exploration history;
- Audit of exploration work carried out on the project;
- Geological modelling;
- Mineral resource estimation and validation;
- Preparation of a Mineral Resource Statement;
- Mining shapes design following geotechnical guidelines;
- Applying external dilution and mining loss factors to estimate potential mineable mineralization;
- Three-dimensional underground mine modelling;
- Mine production rate estimation;
- Life-of-mine development and production scheduling;
- Equipment and manpower determination;
- Infrastructure requirement estimations;
- Mine operating and capital costs estimation;

- Economic modelling and financial analysis; and
- Recommendations for additional work.

1.2 Work Program

The preliminary economic assessment reported herein is a collaborative effort between Rubicon and SRK personnel.

The exploration database was compiled and maintained by Rubicon and it was audited by SRK. The geological model and outlines for the gold mineralization were constructed by SRK with input from Rubicon staff. In the opinion of SRK, the geological model is a reasonable representation of the distribution of the targeted mineralization at the current level of sampling. Geostatistical analysis, variography, and grade models were completed by SRK during the months of March and April 2013.

A new conceptual underground proposed mining plan was generated by SRK from the revised mineral resource model. The plan includes the stoping areas, development, fleet selection, ventilation system, ore and waste handling systems, underground infrastructure, pastefill distribution, dewatering facilities, and mine services. Schedules were generated up for production, development and underground infrastructure.

The production schedule was used to prepare an economic model that was used as a basis for financial analysis of the project.

Several other aspects of the project were reviewed by SRK to confirm existing completed work. These include geotechnical studies, metallurgical studies, environmental issues, and estimates for initial capital, sustaining capital, and operating expenses. Soutex Inc. (Soutex) was responsible for the mineral processing aspect of the conceptual mining project. Most of the tasks contributing towards the preliminary economic assessment documented herein were completed during the period April to June 2013.

A Mineral Resource Statement and the results of the preliminary economic assessment reported herein were presented to Rubicon in memorandum reports on June 24, 2013 and were disclosed publically in a news release dated June 25, 2013.

The technical report was assembled in Toronto between the months of January to July 2013.

1.3 Basis of Technical Report

This report is based on information collected by SRK on various site visits performed between October 2011 and April 2013 and additional information provided by Rubicon throughout the course of SRK's investigations. SRK has no reason to doubt the reliability of the information provided by Rubicon. Other information was obtained from the public domain. This technical report is based on the following sources of information:

- Technical discussions with Rubicon personnel;
- Inspection of the Phoenix gold project area, including underground exposure and drill core;
- Review of exploration data collected by Rubicon;
- Information extracted from the AMC August 2011 technical report; and
- Additional information from public domain sources.

1.4 Qualifications of SRK and SRK Team

The SRK Group comprises of more than 1,600 professionals, offering expertise in a wide range of resource engineering disciplines. The independence of the SRK Group is ensured by the fact that it holds no equity in any project it investigates and that its ownership rests solely with its staff. These facts permit SRK to provide its clients with conflict-free and objective recommendations. SRK has a proven track record in undertaking independent assessments of mineral resources and mineral reserves, project evaluations and audits, technical reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies, and financial institutions worldwide. Through its work with a large number of major international mining companies, the SRK Group has established a reputation for providing valuable consultancy services to the global mining industry.

A tabulation of the qualified persons responsible for each section of this report is provided in Table 1. The mineral resource evaluation work was completed by Sébastien Bernier, PGeo (APGO#1847), with the assistance of Chris MacInnis, PGeo (APGO# 2059). Additional geological modelling contributions were provided by Dr. Jean-François Ravenelle, PGeo (APGO # 2159), Dominic Chartier, PGeo (OGQ #874), Dr. Iris Lenauer, Dr. James Siddorn, PGeo, and Dr. Julia Kramer Bernhard. Geological and mineral resource modelling benefited from the senior review of Glen Cole, PGeo (APGO#1416).

The mining study work was completed by Stephen Taylor, PEng (PEO # 90365834) and Daniel Hewitt, PEng (PEO # 19465012). Tim Coleman, BEng, Brad Klassen (APEGS#21370), and Nico Viljoen, PEng (ECSA#20120488) contributed to the geotechnical evaluation. Mr. Hewitt was also responsible for infrastructure, environment, costs and financial evaluation.

Table 1: Qualified Persons Accepting Professional Liability for this Technical Report

Author	Company	Report Section (s)
Sebastien Bernier, PGeo	SRK	1 to 11, and 13
Glen Cole, PGeo	SRK	1 to 11, 13, 24, 25
Daniel Hewitt, PEng	SRK	15, and 17 to 25
Stephen Taylor, PEng	SRK	14, 15, 24, and 25
Pierre Roy, PEng	Soutex	10.3, 11.3, 12, 13.12.2, 16, 24.7 and 25

Mr. Gary Poxleitner, PEng (PEO#100059860), contributed to the costs and financial evaluation.

The processing and recovery aspects of the report were completed by Mr. Sylvain Caron, Eng (OIQ #38767) and Mr. Pierre Roy, Eng (OIQ #045201) of Soutex.

The compilation of Sections 1 to 13 of the technical report benefited from the contributions of Dominic Chartier, PGeo (OGQ #874) and Dr. Iris Lenauer. Ms. Zoe Demidjuk (MAusIMM) assisted with the compilation and review of the analytical quality control data.

By virtue of their education, membership to a recognized professional association, and relevant work experience, Mr. Bernier, Mr. Cole, Mr. Hewitt, Mr. Taylor, and Mr. Roy are independent Qualified Persons as this term is defined by National Instrument 43-101.

Drafts of this technical report were reviewed by Dr. Jean-Francois Couture, PGeo (APGO#0197), Ken Reipas, PEng (PEO#100015286), Brian Connolly, PEng (PEO#90545203), Ms. Sophia Karadov and Ms. Alison Harrington prior to their delivery to Rubicon as per SRK's internal quality management procedures.

1.5 Site Visit

In accordance with National Instrument 43-101 guidelines, Dr. Ravenelle, Mr. Hewitt, and Mr. Taylor of SRK as well as Mr. Caron and Mr. Roy of Soutex visited the Phoenix gold project separately on multiple occasions accompanied by Rubicon personnel.

Dr. Ravenelle visited the property between October 1 and 7, 2011 and between December 3 and 8, 2011. The site visits aimed at investigating the geological and structural controls on the distribution of the gold mineralization in order to aid the construction of three-dimensional gold mineralization domains.

Mr. Hewitt visited the property on April 15, 2013. The purpose of the visit was to review the status of the project and possible data gaps with respect to geotechnical conditions in the proposed mining areas, existing underground openings, mining plans and schedule, processing and backfill facilities, environmental programs and compliance, property boundary, and closure planning.

Mr. Taylor visited the property on January 22, 2013 and again on April 17, 2013. The purpose of the January site visit was to collect data on ground conditions encountered previously, review the results of the bulk sampling program, and observe shaft sinking development. In April, the purpose of the site visit was to review existing infrastructure and equipment as well as discuss gaps in infrastructure to be addressed in the ongoing mine design work.

Mr. Roy and Mr. Caron visited the property on February 16 to 18, 2011 and on April 18 to 20, 2011. Mr. Roy also visited the property on March 10, 2011. The purpose of their visits was related to the subsampling of the two bulk samples that were extracted from underground and to review the surface infrastructure for confirmation of the planned mineral processing facility implementation. An underground visit was also made to the location on the 305 level where the bulk samples were taken.

SRK was given full access to relevant data and conducted interviews with Rubicon personnel to obtain information on the past exploration work, to understand procedures used to collect, record, store, and analyze historical and current exploration data.

1.6 Acknowledgement

SRK would like to acknowledge the support and collaboration provided by Rubicon personnel for this assignment. Their collaboration was greatly appreciated and instrumental to the success of this project.

SRK acknowledge the contributions of Mr. Michael Lalonde, PEng, President and Chief Executive Officer, Mr. Matthew Wunder, PGeo (APGO #1316), Vice President Exploration, Mr. Daniel Labine, PEng (PEO # 25165556), Vice President Operations, Mr. Bob Lewis, CFO, Mr. Ian Russell, Manager Special Projects, Darryl Boyd, Manager of Regulatory Affairs, and Mr. Jerrett Landry, MSc, Project Controller, who provided invaluable technical insight and direction to the project.

1.7 Declaration

SRK's opinion contained herein and effective **June 25, 2013** is based on information collected by SRK throughout the course of SRK's investigations, which in turn reflects various technical and economic conditions at the time of writing this report. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate, or an affiliate of Rubicon, and neither SRK nor any affiliate has acted as advisor to Rubicon, its subsidiaries, or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

SRK was informed by Rubicon that there is no known litigation potentially affecting the Phoenix gold project, other than the application for judicial review brought by Wabauskang First Nation described below.

2 Reliance on Other Experts

SRK has not performed an independent verification of the land title and tenure as summarized in Section 3 of this report. SRK did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties, but has relied upon Rubicon's reliance on the legal opinion of a specialist in corporate and commercial law with Weiler, Maloney, Nelson of Thunder Bay, Ontario as expressed in a letter provided to Rubicon on May 31, 2013 with an effective date of May 27, 2013**.** The reliance applies solely to the legal status of the rights disclosed in Section 3.1 below.

3 Property Description and Location

The Phoenix gold project is located in the south western part of Bateman Township within the Red Lake mining district of northwestern Ontario, Canada (Figure 1). The town of Red Lake is approximately 150 kilometres (km) northwest of Dryden, Ontario and 265 km northeast of Winnipeg, Manitoba.

The total area of mineral tenure is 510.4 hectares. The Phoenix gold project is centred on the historical McFinley Shaft (now called the Phoenix Shaft), located at latitude 51.13 degrees north and longitude 93.74 degrees west. Rubicon has a 100 percent (%) interest in the Phoenix gold project subject to a 2% net smelter return (NSR) royalty on the majority of the water portions of the property to Franco-Nevada Corporation. Rubicon has the option to acquire a 0.5% interest in the NSR for US$675,000 at any time, in which case the NSR would be reduced to 1.5%

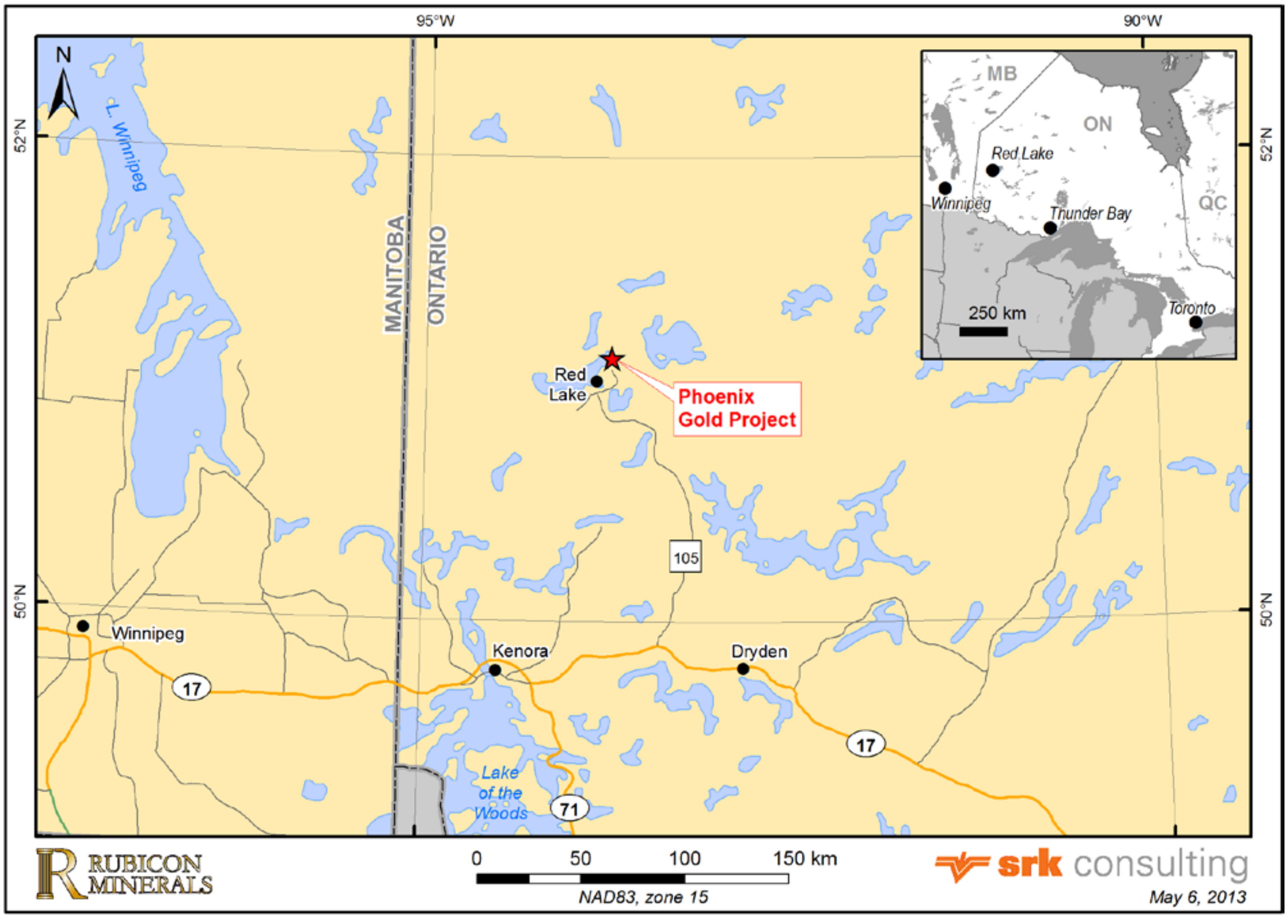


Figure 1: Phoenix Gold Project and Its Surrounding Infrastructure (SRK 2013)

3.1 Mineral Tenure

The Phoenix gold project consists of 31 contiguous KRL or K numbered blocks (Table 2 and Figure 2) comprised of:

- One Mining Lease covering four KRL blocks;
- Sixteen Patented Claims covering land portions of the property;
- Twenty-five Mining Licences of Occupation covering water portions of the property; and
- One Staked Claim.

A single KRL or K numbered block can consist of a land portion (Patented Claim) and associated water portion (Mining Licences of Occupation containing a separate number) when it covers land and water within its boundaries. A single KRL or K numbered block can also consist of solely land portions or solely water portions of the property.

The mineral resources reported herein are contained within the Patented Claims and associated Mining Licences of Occupation in blocks K1499, KRL18376, KRL18735, KRL246, KRL18375, KRL247, KRL18374, and K1493.

The perimeter of the Phoenix property was surveyed legally by certified Ontario land surveyor Jim Bowman on February 7, 1985. This legal survey defined the Phoenix gold property at the time of the original mining lease application on October 20, 1986. This land survey was verified by Rubicon via professional land surveying services of Geomatics Inc. on August 3, 2012.

The mineral and surface rights of the Mining Lease and the mineral rights of the Patented Claims are registered under Rubicon with Ontario's Electronic Land Registration System. The surface rights of the Patented Claims are registered under 0691403 B.C. Ltd, a wholly owned subsidiary of Rubicon, with Ontario's Electronic Land Registration System. The mineral and surface rights of the Mining Licences of Occupation (MLOs) and the Staked Claim are registered under Rubicon with the Mining and Minerals Division of the Ministry of Northern Development and Mines (MNDM).

The MLOs are subject to a payment of rents shown on the face of each license. They do not have a stated term but exist during the pleasure of the Crown. No application for renewal is required.

The Mining Lease is for a standard fixed term. The current term has been extended to October 31, 2028. Prior to expiry of the extended term, an application must be made under the Ontario *Mining Act* for the Minister's consent to extend the leasehold for a further fixed term.

On June 22, 2009, mineral rights for one Staked Claim were recorded with the MNDM. To maintain the claim in good standing it is required by Rubicon to carry out eligible assessment work of C$400 prior to June 22, 2016.

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

Table 2: Mineral Tenure Information for Phoenix Gold Project

KRL or K Numbered Block(s)	Number	Start Date	Expiry Date	Hectares	Current Resource
Mining Lease					
KRL503297, KRL503298, KRL503299, KRL526262	108126	November, 1986	October 31, 2028	56.0	
Patented Mining Claims (Land Portion)					
K1498	992	October 1, 1945	Not Applicable	3.0	
K1499	993	October 1, 1945	Not Applicable	11.5	Yes
K1493	994	March 1, 1946	Not Applicable	5.1	Yes
K1494	995	March 1, 1946	Not Applicable	8.4	
K1495	996	March 1, 1946	Not Applicable	10.4	
KRL246	997	March 1, 1946	Not Applicable	15.0	Yes
KRL247	998	March 1, 1946	Not Applicable	17.9	Yes
K1497	999	March 1, 1946	Not Applicable	13.5	
KRL11481	1446	November 1, 1941	Not Applicable	4.2	
KRL11482	1447	November 1, 1948	Not Applicable	6.9	
KRL11483	1448	November 1, 1941	Not Applicable	12.2	
KRL11487	1452	November 1, 1941	Not Applicable	15.3	
K954 (recorded as KRL 18152)	1977	January 1, 1947	Not Applicable	6.9	
K955 (recorded as KRL 18515)	1978	January 1, 1947	Not Applicable	4.3	
KRL18457	2449	January 1, 1950	Not Applicable	7.9	
KRL18735	2450	January 1, 1950	Not Applicable	20.9	Yes
Licenses of Occupation (Water Portion)					
KRL2155	3186	August 1, 1945	Not Applicable	9.9	
KRL2156	3187	August 1, 1945	Not Applicable	13.7	
K1498	3289	October 1, 1945	Not Applicable	11.0	
K1499	3290	October 1, 1945	Not Applicable	2.4	Yes
K1493	3370	March 1, 1946	Not Applicable	5.0	Yes
K1494	3371	March 1, 1946	Not Applicable	18.7	
K1495	3372	March 1, 1946	Not Applicable	10.1	
K1497	3380	March 1, 1946	Not Applicable	6.1	
KRL246	3381	March 1, 1946	Not Applicable	4.3	Yes
KRL247	3382	March 1, 1946	Not Applicable	4.5	Yes
KRL11483	10495	November 1, 1941	Not Applicable	6.7	
KRL11482	10496	November 1, 1948	Not Applicable	5.6	
KRL11481	10497	November 1, 1941	Not Applicable	14.1	
KRL11487	10499	November 1, 1941	Not Applicable	5.7	
KRL11038, KRL11039	10830	January 1, 1947	Not Applicable	28.7	
KRL11031	10834	January 1, 1947	Not Applicable	17.9	
K954 (recorded as KRL18152)	10835	January 1, 1947	Not Applicable	9.3	
K955 (recorded as KRL18515)	10836	January 1, 1947	Not Applicable	10.0	
KRL18514	10952	October 1, 1947	Not Applicable	17.5	
KRL18735	11111	January 1, 1950	Not Applicable	12.2	Yes
KRL18457	11112	January 1, 1950	Not Applicable	11.0	
KRL18373	11114	January 1, 1950	Not Applicable	7.7	
KRL18374	11115	January 1, 1950	Not Applicable	19.7	Yes
KRL18375	11116	January 1, 1950	Not Applicable	22.9	Yes
KRL18376	11117	January 1, 1950	Not Applicable	15.0	Yes
Staked Claim					
KRL4229741	N/A	June 22, 2009	June 22, 2016	1.0	
Total Area				**510.4**	

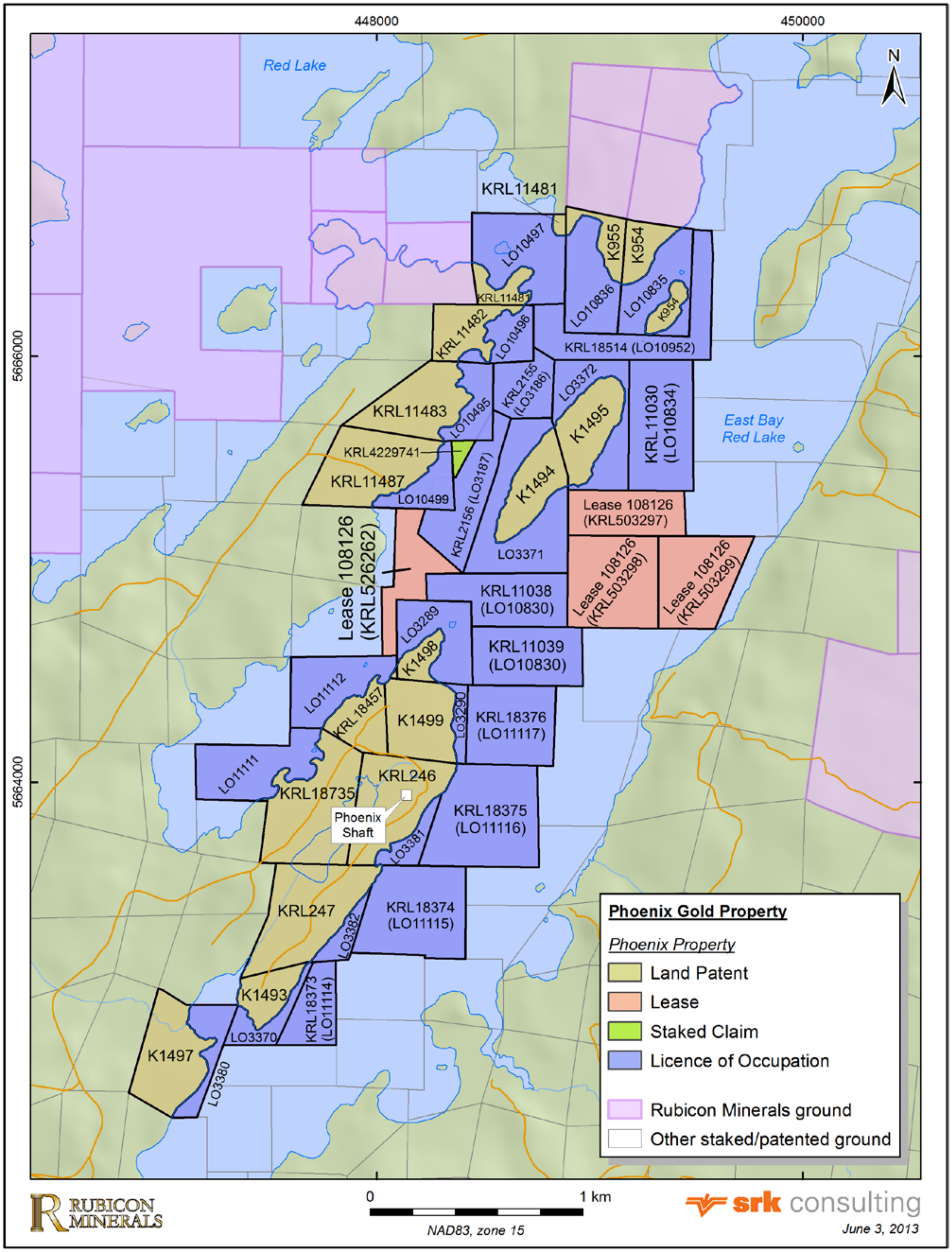


Figure 2: Land Tenure Map of the Phoenix Gold Project (SRK 2013)

3.2 Underlying Agreements

Rubicon’s 100% interest in the property was acquired in two separate agreements entered into with Dominion Goldfields Corporation (DGC) in 2002. The 25 MLOs and the one Mining Lease were optioned from DGC in January 2002 by agreeing to pay C$800,000 in cash, issue 260,000 shares to DGC, and complete US$1,300,000 of exploration work prior to March 31, 2006. During 2004, Rubicon acquired the MLOs and Mining Lease from DGC after meeting all the required payments and expenditures. The MLOs and the Mining Lease were subsequently transferred to Rubicon.

The water portions of the property, except the Staked Claim, are subject to a NSR royalty to Franco-Nevada Corporation of 2%. Franco-Nevada Corporation purchased the NSR from DGC in August 2011. Advance royalties of US$50,000 are due annually to a maximum of US$1,000,000 prior to commercial production of which US$500,000 was paid by Rubicon to 15 May 2013. Rubicon has the option to acquire a 0.5% NSR royalty for US$675,000 at any time, in which case the NSR royalty to Franco-Nevada Corporation would be reduced to 1.5%. Upon a positive production decision, Rubicon would be required to make an additional advance royalty payment of US$675,000. Rubicon has confirmed that the annual payments are up to date.

The mineral rights of the 16 Patented Claims were optioned from DGC in June 2002. The surface rights of the Patented Claims are owned by 0691403 B.C. Ltd, a wholly owned subsidiary of Rubicon. On October 25, 2011, Rubicon announced that by execution of its right of first refusal under its agreement with DGC, it had acquired and thereby extinguished all royalties on the blocks covering the land portions of the property. On closing the agreement, Rubicon issued a total of 1,216,071 of its common shares to DGC, at a deemed price per share of C$3.50, for total consideration of C$4,256,248.50.

3.3 Permits and Authorization

Rubicon currently holds all material permits required for it to carry out its drilling, underground exploration, development, and potential future production on the Phoenix gold project at an annual average rate of 1,250 tpd. Amendments to some of these permits will be obtained for increases to the potential future production rate.

A full list of permits, applications and their status, as advised by Rubicon, is given in Section 19.

3.4 Environmental Considerations

The current and potential production phase environmental liabilities associated with the project site are described in the *Phoenix Project Closure Plan (December 2, 2011)*, filed with the Ontario provincial government pursuant to Part VII of the Mining Act. SRK understands that there are no significant chemical or physical stability liabilities associated with the project site and financial assurance is being provided to the Government of Ontario by Rubicon to rehabilitate all identified features of the project site in accordance with the Mining Act.

3.5 Mining Rights in Ontario

The Phoenix gold project is located in Ontario, a province that has a well understood permitting process in place and one that is coordinated with the federal regulatory agencies. As is the case for similar mine developments in Canada, the project is subject to a federal and provincial environmental assessment process. Due to the complexity and size of such projects, various federal and provincial agencies have jurisdiction to either provide authorizations or permits that enable project construction to proceed.

Federal agencies that have significant regulatory involvement at the pre-production phase include the Canadian Environmental Assessment Agency, Environment Canada, Natural Resources Canada, and Fisheries and Oceans Canada.

On the provincial agency side, the Ministry of Northern Development and Mines, Ministry of Environment, Ministry of Transportation, and the Ministry of Natural Resources each have key project development permit responsibilities.

4 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

4.1 Accessibility

The Phoenix gold project is centred within the Red Lake area of northwestern Ontario, approximately 565 kilometres (km) by road (430 km direct) northwest of Thunder Bay and approximately 475 km by road (265 km direct) east-northeast of Winnipeg, Manitoba. Red Lake can be reached via Highway 105, which branches off the Trans-Canada Highway 17 some 170 km south of Red Lake. Red Lake is also serviced with daily flights from Thunder Bay and Winnipeg by Bearskin Airlines. Bus service is also available from Kenora, Ontario.

The project site is accessible via a 8 kilometre gravel road accessed from the community of Cochenour, part of the Municipality of Red Lake. Located on East Bay of Red Lake the project is also easily accessible by water.

4.2 Local Resources and Infrastructure

The Red Lake Municipality comprises six communities: Red Lake, Balmertown, Cochenour, Madsen, McKenzie Island, and Starratt Olsen. The latest Canada Census of 2011 measured the population of the Municipality at 4,366. Mining is the primary industry and employer in the area. Other industries include small scale logging and tourism focused on hunting and fishing.

The Phoenix gold project site is currently supplied by a 10.4-kilometre power transmission line connected to Hydro One’s 44 kilo Volts (kV) grid in the Municipality of Red Lake. Currently, the site is authorized for a load of 5.3 Mega Volt ampere (MVA) utilizing the 10 MVA substation installed on site to step down distribution voltages to 4,160 volts (V) for surface and underground. Further voltage step downs are utilized locally as required for specific equipment installations.

Mine water is pumped to a holding tank on site from the nearby East Bay of Red Lake. The water is piped underground via a 100 millimetre water line for drilling use, muckpile watering, etc. A potable water plant is now fully commissioned and operating at the processing plant. Rubicon is currently commissioning a separate portable water treatment and distribution system for the camp area. Rubicon has all the surface rights required to conduct its potential operations. Rubicon has also access to local workers and fly in, fly out workers.

4.3 Climate

The climate in this portion of northwestern Ontario is considered subarctic with temperature extremes generally ranging from winter lows of approximately -45ºC to summer highs of roughly 30ºC. Average winter temperatures are in the range of -15ºC to -20ºC and average summer temperatures are in the range of 15ºC to 20ºC. Between 1971 and 2000, annual average precipitation was measured at 64 centimetres (cm) with 47 cm of rain and 193 cm of snow. Average winter snow depths in the region range from 40 to 50 cm. Weather conditions have minimal impact on underground operations and the operating season.

4.4 Physiography

The topography within much of the project is mildly rugged. The elevation is commonly less than 15 metres (m) above the level of Red Lake. The topography is dominated by glacially scoured southwest trending ridges, typically covered with jack pine and mature poplar trees. Swamps, marshes, small streams, and small- to moderate-sized lakes are common. Rock exposure varies locally, but rarely exceeds 15 percent and is mostly restricted to shoreline exposures. Glacial overburden depth is generally shallow, rarely exceeding 10 m, and primarily consists of ablation till, minor basal till, minor outwash sand and gravel, and silty-clay glaciolacustrine sediments.

Vegetation consists of thick boreal forest composed of black spruce, jack pine, trembling aspen, and white birch. Figure 3 illustrates the typical landscape around the Phoenix gold project and the associated vegetation.

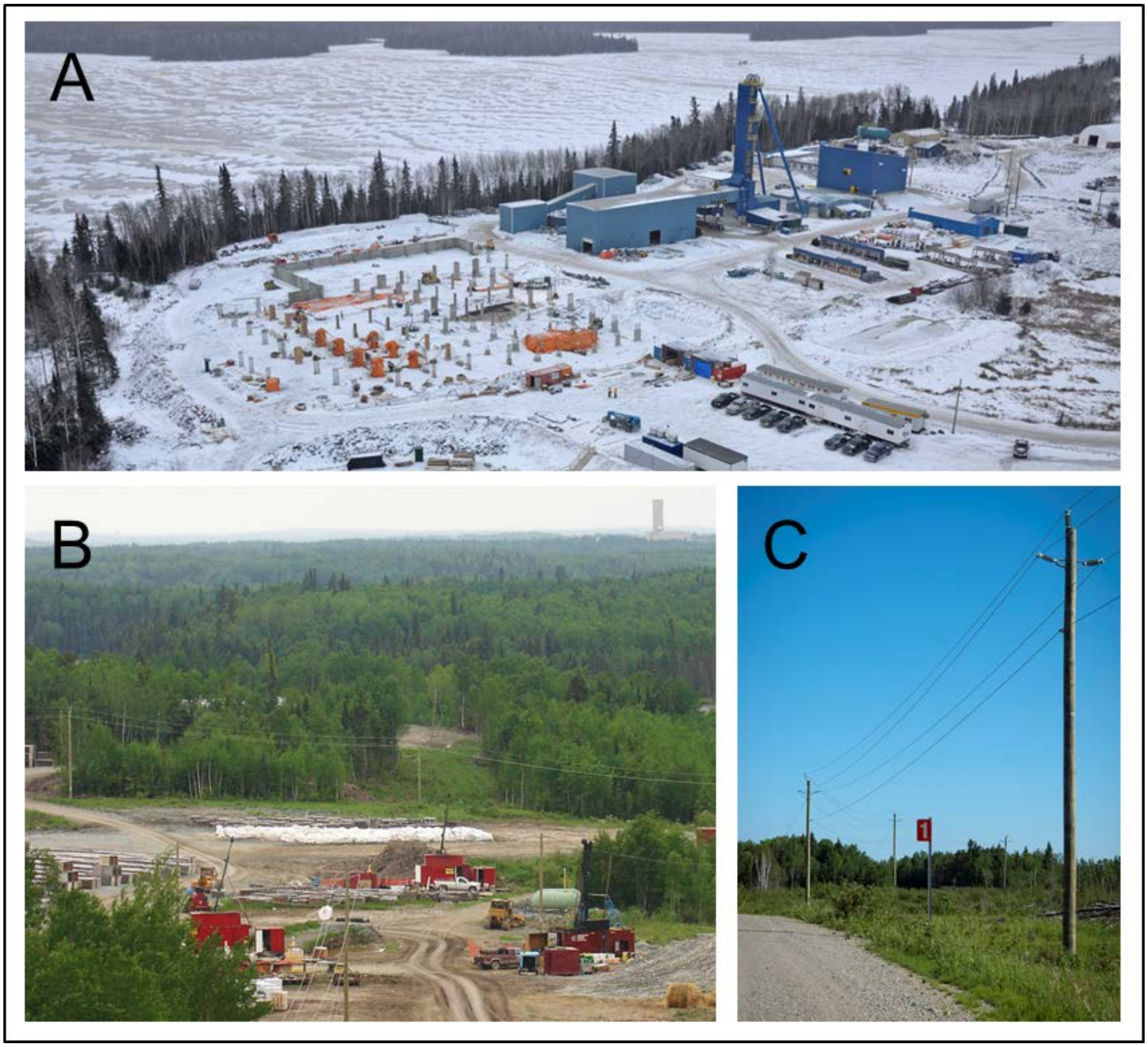


Figure 3: Typical Landscape in the Phoenix Gold Project Area (Photos courtesy of Rubicon)

A. Aerial view of project area looking south. East Bay in background.
B. Typical landscape with drill rigs in foreground and Goldcorp's Cochenour Headframe in distance.
C. Gravel road and power line leading to project area.

A portion of the project is covered by the East Bay of Red Lake with McFinley Island, directly to the north of McFinley Peninsula, representing the largest island on the property. Recent seismic surveys indicate average accumulations of 10 to 20 m of lake sediments and overburden at the lake bottom, with the water depth less than 8.5 m within the property boundary. The location of the tailings storage area and other site infrastructure are covered in Section 17.

5 History

Information in this section is summarized from a previous technical report on the Phoenix gold project prepared by AMC Mining Consultants (2011b).

Gold was originally reported in the Red Lake area in 1897 by R. J. Gilbert of the North Western Ontario Development Company (Parrot 1995). The exploration and mining history of the Red Lake mining district dates back to 1925, when significant gold was first discovered by prospector L. B. Howey. The gold bearing veins he discovered were developed into Red Lake's first producing mine – the Howey Mine.

The Phoenix gold property (previously known as the McFinley property) was initially staked and owned by McCallum Red Lake Mines Ltd. in 1922. Between 1944 and 1974, the property was owned by McFinley Red Lake Gold Mines Ltd. (McFinley Red Lake Gold Mines). In 1974, Sabina Industries Ltd. (Sabina) earned a 60 percent (%) interest in the property. McFinley Red Lake Gold Mines changed its name to McFinley Red Lake Mines Ltd. (McFinley Red Lake Mines) in 1975 and in 1983 by a plan of arrangement Sabina transferred its 60% in the project to McFinley Red Lake Mines.

In 1984, McFinley Red Lake Mines joint ventured the project with Phoenix Gold Mines Ltd. (42.9%) and Coniagas Mines Ltd. (7.1%). This 50% joint venture interest was subsequently repurchased by McFinley Red Lake Mines in 1986 with financial backing from Alexandra Mining Company (Bermuda) Ltd.

Financial difficulties experienced by McFinley Red Lake Mines in 1989 subsequently led to a period of inactivity between 1990 and 2002 with the eventual acquisition of the property by creditors in lieu of unpaid debts. Dominion Goldfields Corporation (DGC) was awarded title to the MLOs and Mining Lease of the project in 1999 and 2002 through vesting orders from the Superior Court of Ontario. DGC and its wholly-owned subsidiary, 1519369 Ontario Ltd., were subsequently granted ownership of the mining rights and surface rights respectively by a vesting order of the Superior Court of Ontario in 2002.

Rubicon optioned the property from DGC in two agreements in 2002. The surface rights of the Patented Claims are now owned by 0691403 B.C. Ltd, a wholly-owned subsidiary of Rubicon.

5.1 Historical Exploration

The extensive history of exploration activities on the project have been described in detail in two previous reports prepared by G. M. Hogg (2002a; 2002b). One report covered the Patented Claims, with the second document discussing historical work completed on the MLOs and Mining Lease, which comprise the project.

All historical information regarding property ownership, previous exploration work, and mineral resources prepared prior to 2002 is summarized below in Table 3.

Table 3: Exploration History of the Phoenix Gold Project

Year	Description of Work
1922	Original staking in 1922 undertaken to cover a high-grade silver occurrence on the McFinley Peninsula, the first mineral prospect on record in the area. Trenching, sampling, and shallow drilling was undertaken by McCallum Red Lake Mines Ltd. Wide-spread but erratic gold mineralization was noted in cherty metasedimentary rock on both McFinley Peninsula and McFinley Island.
1941 – 1942	Mineral occurrences were drilled as part of the Wartime Minerals Evaluation program.
1944 – 1946	McFinley Red Lake Gold Mines Ltd. carried out ground magnetic surveys, a 48 borehole drilling program consisting of 167 m (548 feet [ft]) of drilling over the McFinley Peninsula, and a 1,487-metre (4,877 ft) drilling program from the ice of Red Lake.
1946 – 1955	Fourteen boreholes (M Series) were completed for approximately 1,585 m (5,200 ft) of diamond drilling.
1955 – 1956	Little Long Lac Gold Mines sank a 130-metre (428 ft) vertical shaft on claim KRL 246 and completed 414 m (1,358 ft) of exploratory underground development on two levels. Work terminated in 1956.
1974 – 1975	Sabina completed 25 diamond boreholes for approximately 3,048 m (10,000 ft) of drilling on the project; ground magnetic and electromagnetic surveys; and 10 boreholes for approximately 735 m (2,410 ft) of diamond drilling over a portion of the lake properties.
1981 – 1983	Sabina and McFinley Red Lake Mines completed a magnetic/electromagnetic geophysical survey over the McFinley Peninsula area, surface bulk sampling, and 3,672 m (12,046 ft) of surface diamond drilling in 33 boreholes.
1983 – 1984	McFinley Red Lake Mines and Sabina completed seven boreholes for approximately 646 m (2,120 ft) of diamond drilling.
1984 – 1985	An agreement with Phoenix Gold Mines Ltd. allowed the reopening of the McFinley Shaft (now called the Phoenix Shaft) and completion of a total of 479 m (1,570 ft) of drifting and crosscutting on the 150 ft (46 m) and 400 ft (122 m) levels. Metallurgical work and mineral processing were carried out. Eighty underground boreholes totalling 1,829 m (6,000 ft) and 69 surface boreholes totalling 10,628 m (34,870 ft) of diamond drilling were completed. Funding difficulties resulted in the project being placed on temporary standby in February 1985.
1985 – 1987	A total of 1,151 m (3,775 ft) of drifting and crosscutting was carried out on the 150 ft (46 m) and 400 ft (122 m) levels. A total of 7,111 m (23,333 ft) of underground drilling, 9.14 m (30 ft) of raising, and an extensive chip-sampling program were completed. A program of 12,763 m (41,874 ft) of diamond drilling was also completed in 61 surface boreholes.
1987 – 1989	In recognition of a nugget effect in sampling results, a decision was made to proceed with a minimum 15,000 ton bulk sample. A 150-tpd mill and TMF was constructed. Underground development (2,890 m/9,482 ft) continued on the 150 ft (46 m) and 400 ft (122 m) levels, a new 275 level (at 84 m) and on a ventilation raise from the 400 ft (122 m) level to surface. Additional sampling, diamond drilling (8,730 m/28,642 ft), and metallurgical testing were completed. Bulk sampling operations commenced in July 1988 with sampling indicating head grades in the range of 0.25 ounces per ton (oz/t) gold (8.23 grams per tonne [gpt] gold) from prepared stope areas. Mill design problems, lack of income from bulk sampling, and lack of exploration funding forced the closure of the operation after an estimated 2,500 tons of material were milled. Total historical development in drifting, crosscutting and raising is estimated to be more than 5,791 m (19,000 ft). Total historical diamond drilling focused on the McFinley Peninsula area is estimated to be 45,110 m (148,000 ft) from surface and 35,814 m (117,500 ft) from underground. An estimated 54,864 m (180,000 ft) of core is stored on the property.
1999 – 2002	DGC foreclosed on the MLOs and Mining Lease and was awarded title to the lake portion of the Phoenix gold project in 1999 and 2002, respectively. DGC and its subsidiary were subsequently awarded title to the Patented Claims of the project in 2002.

5.2 Previous Mineral Resource Estimates

5.2.1 McFinley Red Lake Mines – 1986

An historical mineral resource estimate was prepared by McFinley Red Lake Mines staff in 1986 (Hogg 2002a; Hogg 2002b). The McFinley Red Lake Mines historical mineral resource is located approximately 450 m northwest of the F2 gold system. The estimate refers to the shaft area located on the McFinley Peninsula where historic underground exploration and development, and extensive sampling were carried out. The shaft area is in stratigraphic units separate to the current F2 gold system. The 1986 historical mineral resource estimate was developed using underground sampling results augmented with closely spaced borehole data.

5.2.2 GeoEx Limited – 2010 and 2011

GeoEx Limited (GeoEx) prepared two mineral resource estimates for the F2 gold system in 2010 and 2011 (GeoEx 2011a: GeoEx2011b).

5.2.3 AMC Mining Consultants (Canada) Ltd. – 2011

AMC prepared a Mineral Resource Statement for the F2 gold system using a block modelling approach based on drilling information available to February 28, 2011. The model was not constrained vertically by a potential crown pillar and was extended to incorporate all drilling data. The Mineral Resource Statement was reported at a cut-off grade of 5.0 gpt gold (Table 4).

Table 4: Mineral Resource Statement, F2 Gold Project, AMC Mining Consultant (Canada) Ltd., June 15, 2011

Classification	Million Tonnes	Grade (gpt gold)	Million Ounces of Gold
Indicated	1.028	14.5	0.477
Inferred	4.230	17.0	2.317

1. CIM definitions used for mineral resources
2. Cut-off grade of 5.0 gpt gold applied
3. Capping value of 270 gpt gold applied to composites
4. Based on drilling results to February 28, 2011

A total of 511 boreholes were used in the modelling. Rubicon's interpretations of lithologies, mineralization controls and geology domains were reviewed and accepted by AMC. Twelve mineralized domains were interpreted by AMC using a low gold threshold (0.1 gpt gold), and were further expanded to incorporate all significant mineralized zones.

A composite length of 1.0 metre (m) was chosen and gold composites were capped at 270 gpt gold. The parent block size was 2 by 8 by 12 m, with sub blocking was utilized. Search parameters were 8 by 24 by 36 m for the first pass, two times these parameters for the second pass and three times these parameters for the third pass. The search ellipse was orientated to take cognizance of the predominant orientation of the mineralization as known. The model blocks were assigned a gold grade using and inverse distance (power of three) estimator. An average bulk density value of 2.90 tonnes per cubic metres (t/m^3) was used for all rock types.

Blocks were classified considering data support as a main criterion with a manual review creating volumes based on borehole density and number of samples to inform a block.

5.3 Past Production

There is no past production on the property. Mining exploration activities on the property were terminated in 1989 after test-milling of an estimated 2,250 tonnes of material unrelated to the F2 gold system.

6 Geological Setting and Mineralization

6.1 Regional Geology

The following description of the geology of the Red Lake Greenstone Belt was modified from Sanborn-Barrie et al. (2004) and the references therein.

The Phoenix gold project is located in the Uchi Subprovince of the Superior Province of the Canadian Precambrian Shield. Within the Uchi Subprovince, the Red Lake Greenstone Belt is host to one of Canada's preeminent gold producing districts with over 26 million ounces of gold produced since the 1930s.

The belt is interpreted to have evolved on the south side of the North Caribou Terrane, an ancient continental block originating approximately 3 billion years before present (Ga) (Figure 4). The terrane evolved from extensive magmatic and sedimentary activity that occurred from 3.0 to 2.7 Ga with multiple events of intense deformation, metamorphism, hydrothermal alteration, and gold mineralization. Regional metamorphic assemblages range from greenschist to amphibolite.

The tholeiitic and komatiitic metabasalts of the Balmer Assemblage, dated approximately between 3,000 million years and 2,988 million years before present (Ma), are the oldest volcanic rocks in the greenstone belt and host the major lode gold deposits in the Red Lake district. The assemblage consists of lower, middle, and upper massive to pillowed tholeiitic metabasalt sequences separated by distinctive felsic and ultramafic metavolcanic rock.

Metasedimentary rocks also occur within the assemblage, mainly as thinly bedded magnetite-chert ironstone. There is an angular unconformity between the Balmer Assemblage and all other younger assemblages in the district. The lower and middle portions of the Balmer Assemblage are the host rocks for the major gold deposits of the Red Lake camp.

Underlying the northwestern portion of the Red Lake Greenstone Belt is the Ball Assemblage (approximately 2,940 to 2,925 Ma) consisting of a thick sequence of metamorphosed intermediate to felsic calc-alkaline flows and pyroclastic rocks.

The Slate Bay Assemblage (approximately 2,903 to 2,850 Ma) extends the length of the belt and consists of clastic rocks of three main lithological facies varying from conglomerates, quartzose arenites, wackes, and mudstones. The contact of the Slate Bay Assemblage with the Ball and Balmer assemblages represents an unconformity (Figure 5).

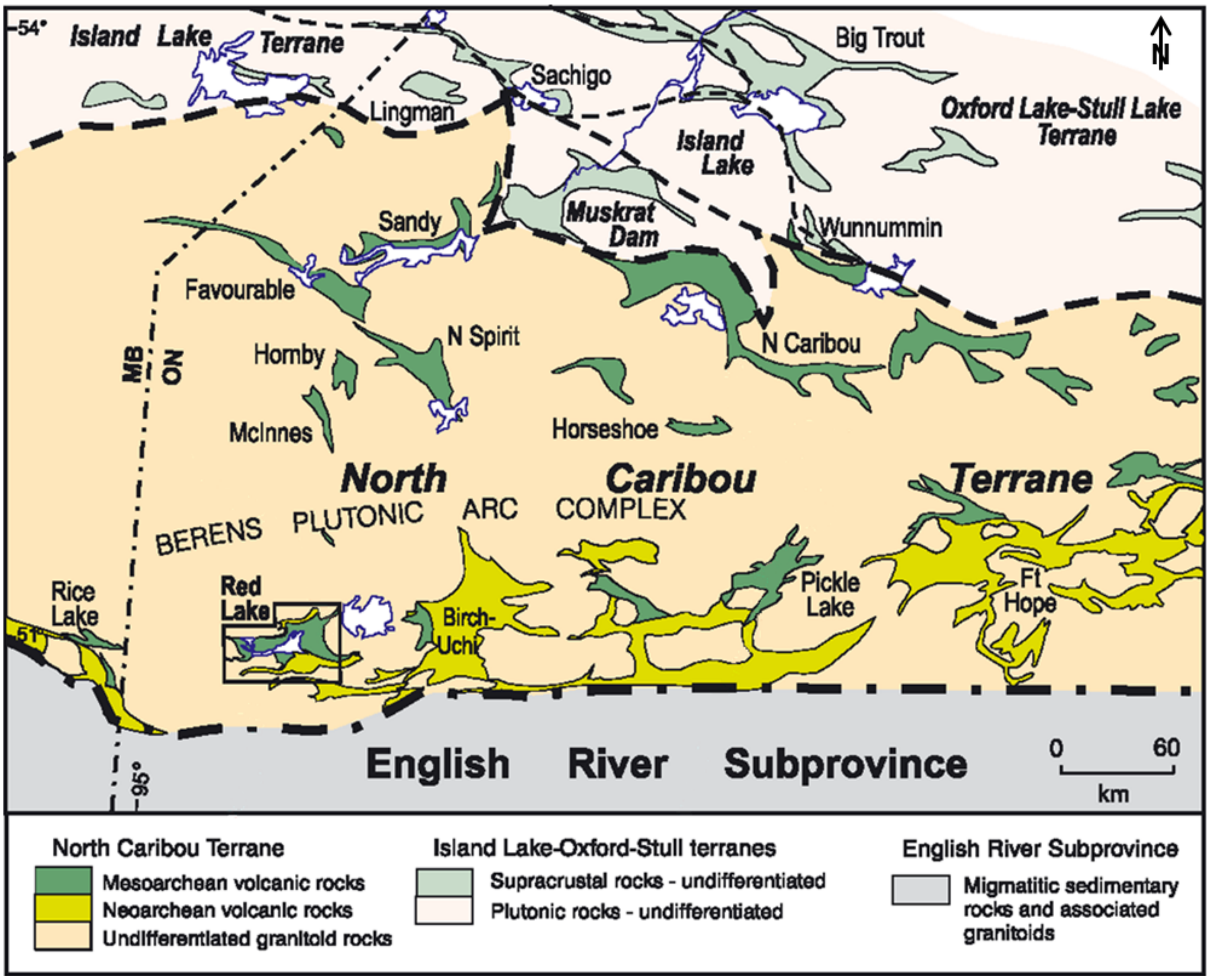


Figure 4: Geology of the North Caribou Terrane of the Superior Province
(Source: Sanborn-Barrie et al. 2004)

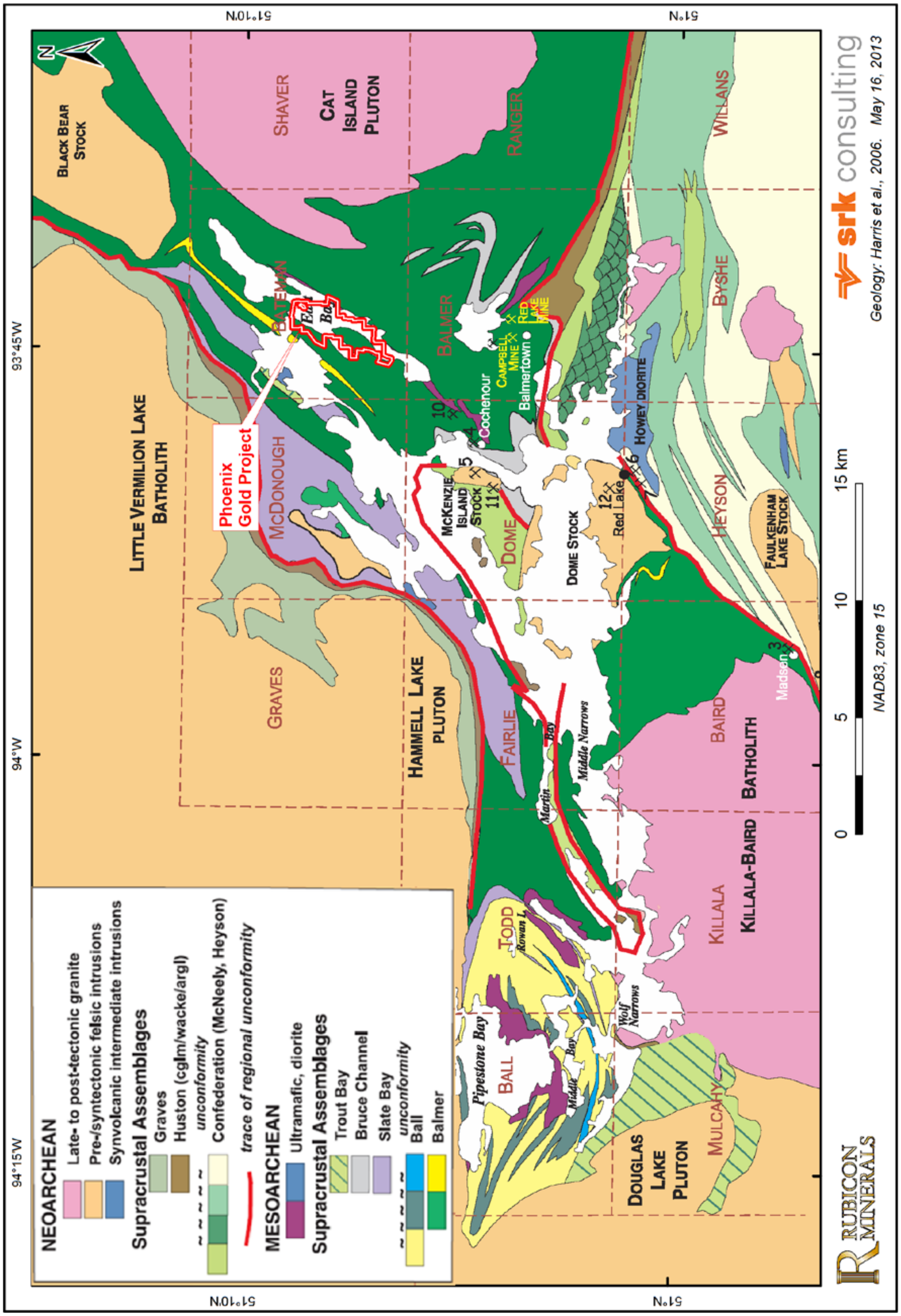


Figure 5: Simplified Geology of the Red Lake Greenstone Belt

(Source: Harris et al. 2006)

A thin sequence of calc-alkaline dacitic to rhyodacitic pyroclastic rocks of the Bruce Channel Assemblage (approximately 2,894 Ma) were deposited and overlain with clastic sediments and a chert-magnetite iron formation. Enriched LREE trace element profiles relative to the Balmer Assemblage are interpreted to indicate crustal growth at a juvenile continental margin.

The Trout Bay Assemblage (approximately 2,853 Ma) is exposed in the southwest portion of the Red Lake Greenstone Belt. It is a volcano-sedimentary sequence consisting of a lower tholeiitic basalt unit overlain by clastic rocks and interbedded with an intermediate tuff and a chert-magnetite-iron formation.

Following a lull in volcanic activity for approximately 100 million years, the Confederation Assemblage represents a time of widespread calc-alkaline volcanism (approximately 2,748 to 2,739 Ma).

Overlying the McNeely sequence in the Confederation Assemblage is the Heyson sequence of tholeiitic basalts and felsic volcanics. Isotopic and geochemical data suggests the McNeely rocks were formed during a shallow marine to subaerial arc on the existing continental margin with later intra-arc extension and eruption forming the Heyson sequence. In the Madsen area, the strata of the Confederation and Balmer assemblages depict an angular unconformity with opposing facing directions. The Balmer Assemblage was, thus, overturned prior to the deposition of the Confederation Assemblage.

Following the Confederation Assemblage, the Huston Assemblage (approximately between 2,742 and 2,733 Ma) records a time of clastic sedimentary deposition varying from immature conglomerates and wackes. The Huston Assemblage has been compared to the Timiskaming conglomerates commonly associated with gold mineralization in the Timmins camp of the Abitibi Greenstone Belt (Dubé et al. 2003). The Huston was followed by the Graves Assemblage (approximately 2,733 Ma) of calc-alkaline volcanism dominated by andesitic to dacitic pyroclastic tuff, and synvolcanic diorite and tonalite.

Plutonic rocks found in the Red Lake Greenstone Belt correlate with various stages of volcanism. These include mafic to ultramafic intrusions during Balmer and Ball time periods, gabbroic sills related to Trout Bay volcanism, felsic dikes and diorite intrusions during the Confederation Assemblage, and intermediate to felsic plutons, batholiths and stocks of Graves assemblage age. Post-volcanism plutonic activity is also evident from granitoid rocks such as the McKenzie Island stock, Dome stock and Abino granodiorite (2,720 and 2,718 Ma) that were host to past producing gold mines. The last magmatic event recorded in the belt is from about 2.7 Ga with a series of potassium-feldspar megacrystic granodiorite batholiths, plutons, and dikes, including the Killala-Baird Batholith.

Structurally, the belt displays evidence of several deformational events with associated hydrothermal activity and gold mineralization. The main episode of penetrative deformation occurred after the Confederation volcanism, which took place at 2.74 Ga. This D1 deformation event resulted in the formation of north-trending south-plunging F1 folds and associated fabrics. The likely cause of deformation is a change in plate dynamics such as the shallowing of a subducted slab creating compression in the upper plate and the displacement of magmatic activity.

A second important deformational event superimposes D1 structures. East- to northeast-trending D2 structures occur in western and central Red Lake, and southeast-trending folds and fabric are present in eastern Red Lake such as at the Campbell and Red Lake mines. The onset of penetrative D2 strain across the belt from 2.72 Ga is interpreted to document the collision of the North Caribou Terrane and the Winnipeg River Subprovince to the south.

6.2 Property Geology

The Phoenix gold project is underlain by the Balmer Assemblage, which is comprised of three sequences dominated by tholeiitic mafic volcanic rocks, separated by distinct marker horizons of felsic and ultramafic volcanic rocks. The lower Balmer sequence is comprised of mafic to pillowed tholeiitic basalts, with local pillowed and massive komatiitic volcanics. The middle Balmer sequence is comprised of a lower andesite unit, which is overlain by pillowed, variolitic tholeiitic basalts, with thin bedded chert-magnetite metasediments and intermediate to felsic flows and pyroclastics, as well as komatiitic flows near the top of the middle Balmer sequence. The upper Balmer sequence is comprised of tholeiitic mafic volcanic rocks.

A strong north-northeast–trending structural fabric through the area is considered part of the East Bay Deformation Zone (EBDZ), which dominates the geology of the Phoenix gold project. Mine Grid lies at an orientation of +45 degrees to the UTM grid roughly parallel to the EBDZ. The EBDZ is in sharp structural contact with a later F2 domain to the southeast, where northwest trending (F2) fold axes are perpendicular to the EBDZ. The EBDZ represents a very large structural zone or “break” separating two major geological domains. In addition, the amphibolite/greenschist isograd is developed in association with a pluton emplacement commonly associated with mineralization in the Red Lake district.

6.2.1 Lithology

The F2 gold system lies within the Phoenix gold project boundaries and comprises a northeast-trending, west-dipping sequence of ultramafic to mafic volcanics ± intrusives, felsic intrusives and minor sedimentary rock types. Extensive mapping, trenching, diamond drilling, and geophysical surveys have defined a very consistent geological sequence that can be correlated along the length of the property for over 4 kilometres (km). A summary of the stratigraphic units found within the project area is shown in Table 5 and Figure 6.

Table 5: Summary of Project Stratigraphy

Sequence	Stratigraphy
West Peninsula Sequence	Pillowed to massive basalts with banded iron formation (BIF), graphitic BIF and chert, banded silty to arenaceous sediment/epi-sediments and significant pyrite/pyrrhotite.
Central Basalt Sequence	Pillowed and massive tholeiitic basalts with flow top breccias occasional BIF and (graphitic) argillite.
Intrusive Komatiite Sequence	Massive, spinifex, and columnar jointed basaltic komatiite bounded by Hanging Wall BIF to the east and by Main BIF to the west. BIF possible in central part of sequence.
McFinley Sequence	Bounded to the west by Hanging Wall BIF and to the east by the Footwall BIF. At least five horizons of silica/oxide (carb.) facies BIF within pillowed and amygdaloidal basalt.
Hanging Wall Basalt Sequence	Pillowed to massive, amygdaloidal basalts. Variably carbonate altered, variable foliation.
East Bay Serpentinite	Extrusive and intrusive ultramafics. Variable talcose alteration.
High Titanium Basalt (HiTi basalt)	Variable biotite alteration, sulphides (pyrite, pyrrhotite). Silica flooding, quartz breccia, and quartz veining throughout. The HiTi basalt is the main host to gold mineralization in the F2 gold system.

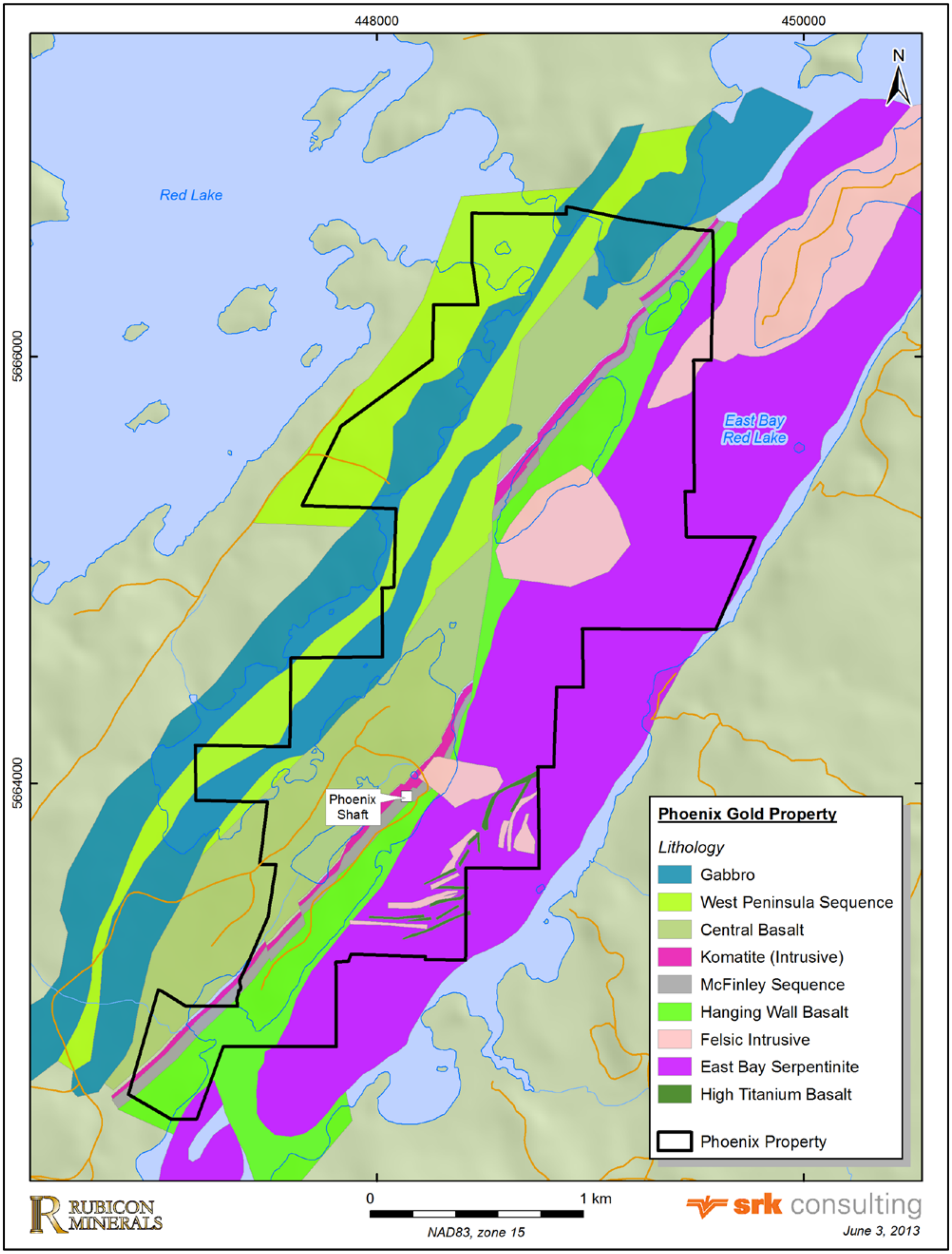


Figure 6: Lithological Map of Phoenix Gold Project (Rubicon 2013)

6.2.2 Structural Geology

At the Phoenix gold project, the EBDZ is manifested by a well-developed, northeast-striking penetrative foliation (S1), which displays progressively steeper dips eastwards as the boundary with the adjacent F2 dominated domain is approached (eastern flank of the EBDZ). Foliation is parallel to lithological boundaries, except rarely where F1 closures are mapped.

The property is interpreted to largely represent limb domains parallel to F1 structures. In the area of the existing mine shaft, the F1 foliation and the geological sequence dip approximately 50 degrees to the northwest whereas towards the southeast, in the area of the F2 gold system which occupies the core of the EBDZ, the foliation dips are subvertical to steep northwest. Within the F2 gold system, the F1 stratigraphy is folded by later F2 folding. This F2 deformation is observed as broad, open folding or warping of the F1 stratigraphy (Figure 7).

Three-dimensional (3D) lithological and structural modelling was completed to enable the recognition of faults and potential controlling structures on gold mineralization in the core zone of the F2 gold system. The 3D lithological modelling focused on building wireframes for the High Titanium (HiTi) basalt and felsic intrusive rocks, but also included banded iron formation (BIF) of the McFinley Sequence and ‘regular’ basalt. A total of 87 lithological domains and nine potential faults were modelled (Figure 8). It should be noted that the term HiTi basalts is a local, property terminology.

Four of the nine faults modelled represent brittle faults modelled based on borehole intervals logged as Fault Zone in the project database. The 3D modelling work appears to show that the steeply-dipping faults (Faults 1 and 2) terminate against the moderate, southwest-dipping faults (Faults 3 and 4). The brittle faults within the F2 core zone appear to partially control the distribution of the various lithological domains. For example, toward the northern extent of the F2 core zone, a large felsic intrusive unit is juxtaposed against HiTi basalt along Fault 1.

Five other faults termed “uncharacterized faults,” were modelled based on apparent offsets observed in the distribution of HiTi basalt and felsic dikes in the F2 core zone. These faults are west-trending (mine grid), have a subvertical dip, and show only limited strike-separation. The nature of these uncharacterized faults and their timing relative to gold mineralization is not understood. They may represent west-trending (mine grid) D2 shear zones documented underground in the F2 core zone.

The lithological modelling also defined two attitudes of folding: moderately- to steep-plunging folds with east-trending axial planes (mine grid) in the northern part of the F2 core zone and shallow-plunging folds with north-trending axial planes (mine grid) that can be seen in cross-section.

The east- and north-trending phases of folding may be attributed to D2 and D3 deformation phases, respectively. This is based on SRK’s structural analysis of underground workings that determined the presence of an east-striking, steep-dipping S2 foliation locally axial-planar to moderate-plunging F2 folds in the S1 foliation, and a south-striking, steep-dipping S3 crenulation cleavage axial planar to shallow-plunging open F3 folds in the S1 foliation.

East-trending F2 folds in the northern part of the F2 core zone may have been generated through fault drag of lithological units along the northern uncharacterized fault.

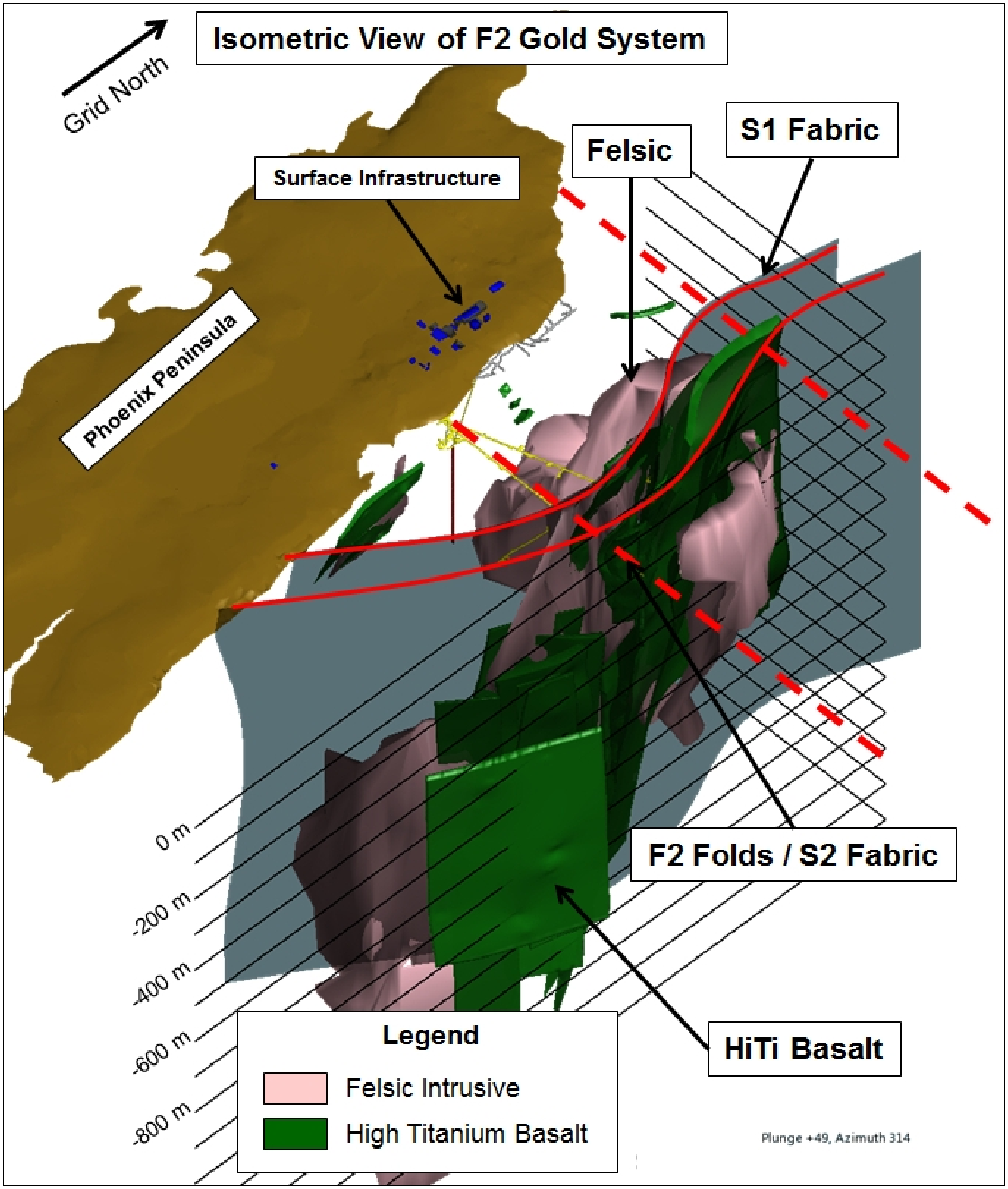


Figure 7: Isometric View of Structure and Geology of the F2 Gold System
(Source: Rubicon, 2013)

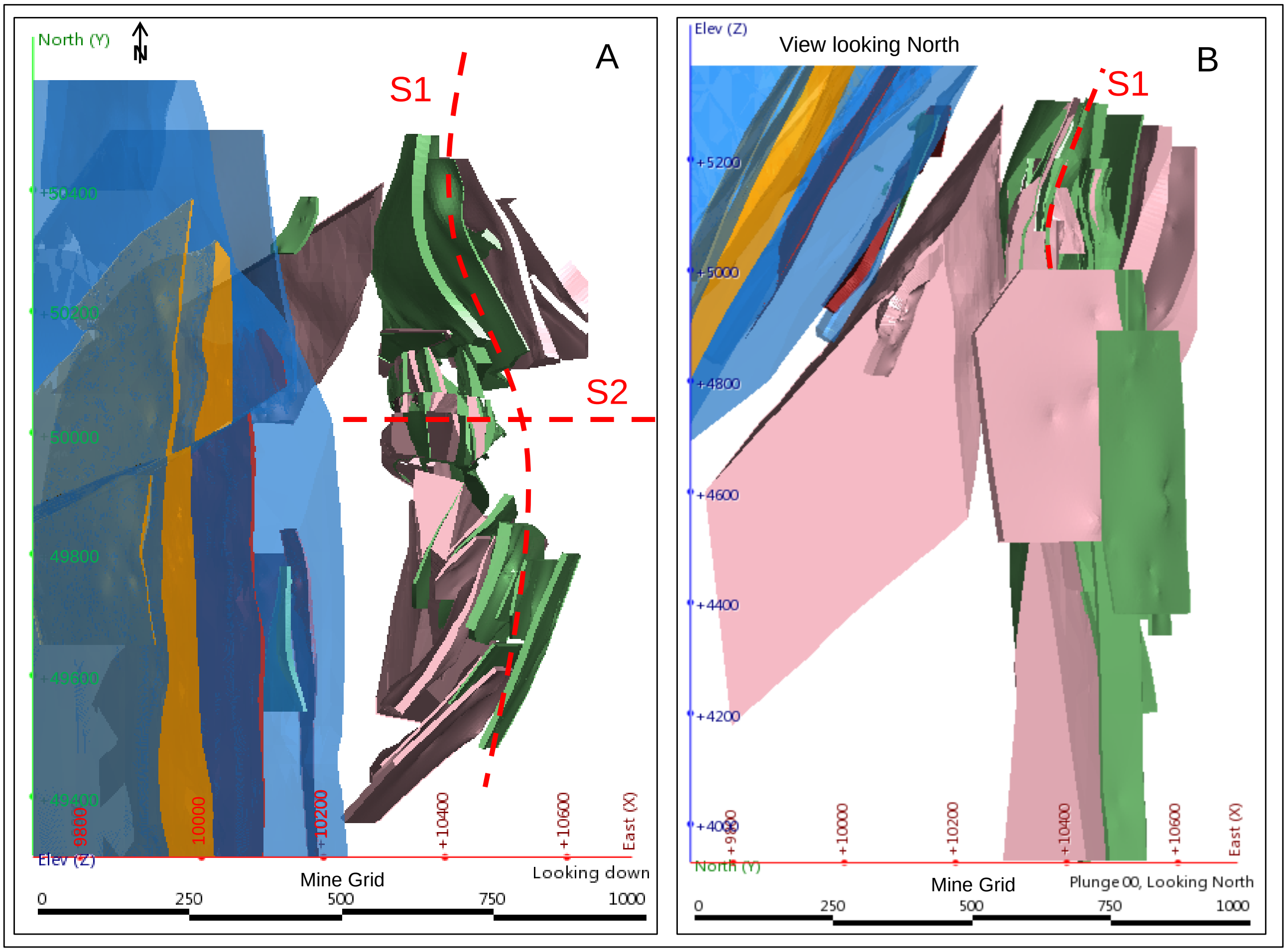


Figure 8: Plan View (A) and Cross-Section (B) Showing 3D Lithological Modelling of the F2 Core Zone

(Source: SRK 2013)

Legend: Felsic intrusive rock (pink) and HiTi basalt (green) show at least two phases of folding. Quartz-feldspar porphyry dike (red), banded iron formation (orange) and basalt (blue).

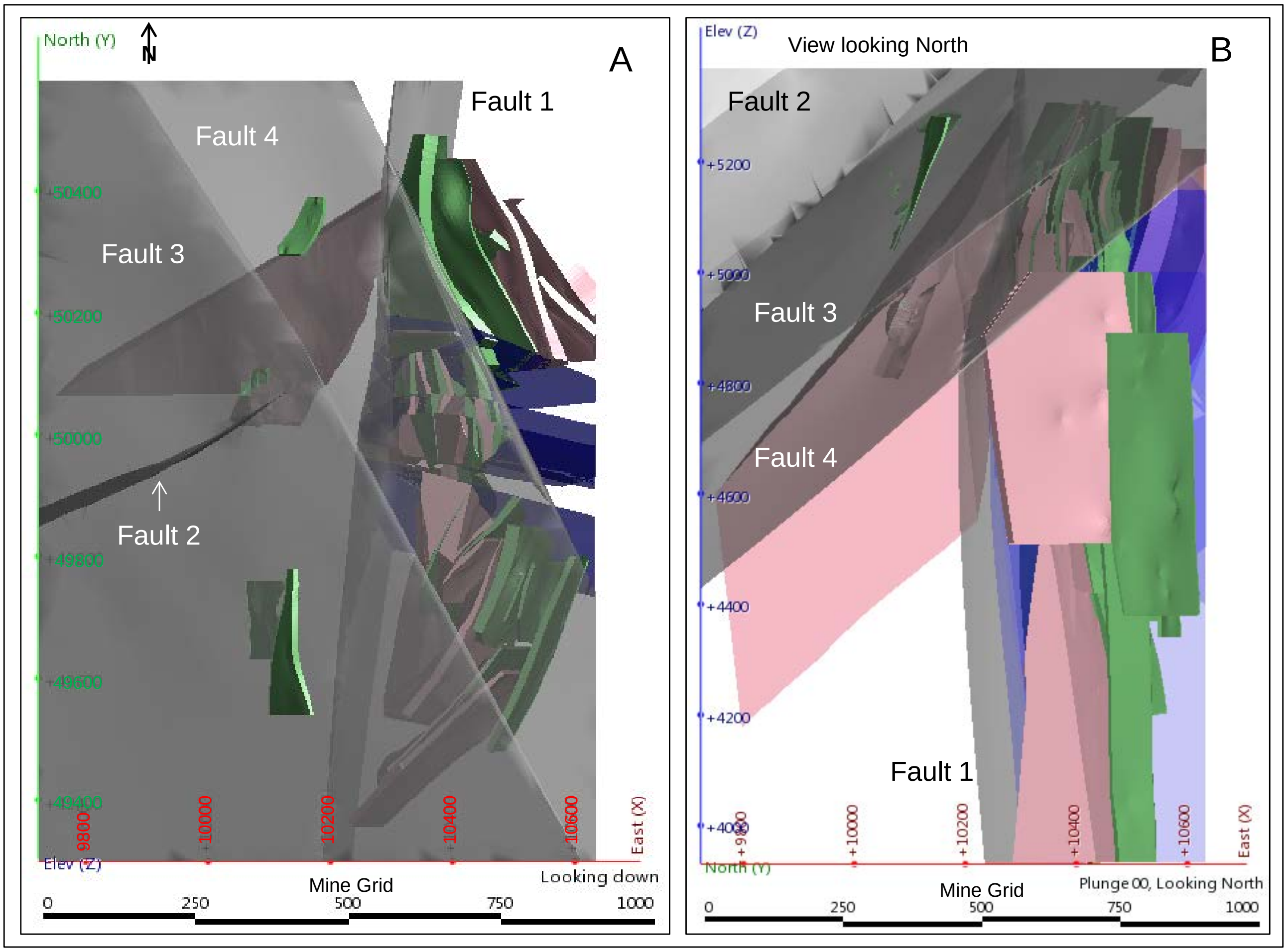


Figure 9: Plan View (A) and Cross-Section (B) Showing Lithological Modelling of the F2 Core Zone with Brittle Faults (Grey)

(Source: SRK 2013)
Legend: Felsic intrusive dike (pink), HiTi basalt (green)

6.2.3 Mineralization

Gold deposits in the Red Lake district have been classified into three main categories (Pirie 1981; Andrews et al. 1986): mafic volcanic-hosted; felsic intrusive-hosted; and strata-bound. The majority of the productive zones in the Red Lake camp, including the Campbell and Red Lake mines, are of the mafic volcanic-hosted type and occur as vein systems within a lower mafic to komatiitic and ultramafic volcanic sequence.

Gold mineralization in the F2 gold system itself is characterized by vein and sulphide replacement styles which are preferentially hosted along the boundaries of two main rock types – HiTi basalts (high iron tholeiites) and felsic intrusive rocks (bounding units) – with additional mineralization associated with crosscutting structures. Gold, however, is distributed through all of the adjacent rock types, with the majority contained within the HiTi basalt (Figure 10).

In the F2 gold system, the gold mineralization is generally conformable with lithological boundaries and is locally modified by observed and inferred crosscutting high angle F2 structures. The gold mineralization is characterized by vein and sulphide replacement styles. The HiTi basalts are fine grained and, where fresh examples exist, comprise amphibole ± plagioclase. Geochemically, these rocks are iron tholeiites and have a very similar geochemical character to gold bearing iron tholeiites in the Red Lake Greenstone Belt as at the Campbell-Red Lake Complex and Cochenour Mine. The felsic dikes, where less altered, are fine- to medium-grained albite, quartz ± biotite bearing, sill-like bodies. Both HiTi basalts and felsic intrusives are heavily altered by potassium, (biotite), iron carbonate (ankerite) ± silica associated with gold mineralization. Both rock types can be readily identified chemically on aluminium-titanium (Al-Ti) plots. Such plots are used to confirm the identity of rock types in areas of intense alteration. Extensive ultramafic rocks comprise the majority of the remainder of the F2 gold system. Host rock types have been correlated over vertical distances of approximately 1,500 metres (m) and horizontal distances of approximately 1,200 metres. Mineralized zones are associated with the contacts of these major rock types.

The main zones identified to date within the F2 gold system generally display a northeast strike, although there appears to be a distinction between zones identified to the western and eastern parts of the property. To the east, the dip on the mineralization is generally subvertical to steep west dipping and the dips shallow slightly to a 70 to 80 degree west dip near surface. To the west in the hanging wall domains, mineralized zones typically dip at about 55 to 65 degrees. The overall plunge of the mineralization identified to date is 55 to 65 degrees to the south-southwest in all areas.

In the F2 gold system, intense potassic alteration (biotite) is accompanied by variable amounts of carbonate, silica alteration, and quartz-carbonate veining. The amount of alteration throughout the F2 gold system is indicative of a robust hydrothermal system likely related to the observed gold mineralization.

The northeast striking hanging wall domains are on the western margin of the currently defined F2 gold system. The mineralization is typically hosted by variably biotite and amphibole altered basalts (hanging wall basalt sequence), with minor pyrite / pyrrhotite and associated quartz / quartz-carbonate veins. Mineralization is also observed along the footwall contact of the hanging wall basalt where it is in (structural) contact with variably talc altered ultramafic rocks. The three-dimensional modelling of the gold mineralized domains was completed based on a low cut-off grade of 0.5 gpt gold (Figure 11) and a high cut-off grade of 3.0 gpt gold (Figure 12). The domains were modelled in Leapfrog and GoCad software based on gold grade, structural trends, and lithological contacts. The main domains of the F2 gold system consist of 18 lower grade domains and 31 high grade domains. The hanging wall domains consist of seven lower grade domains.

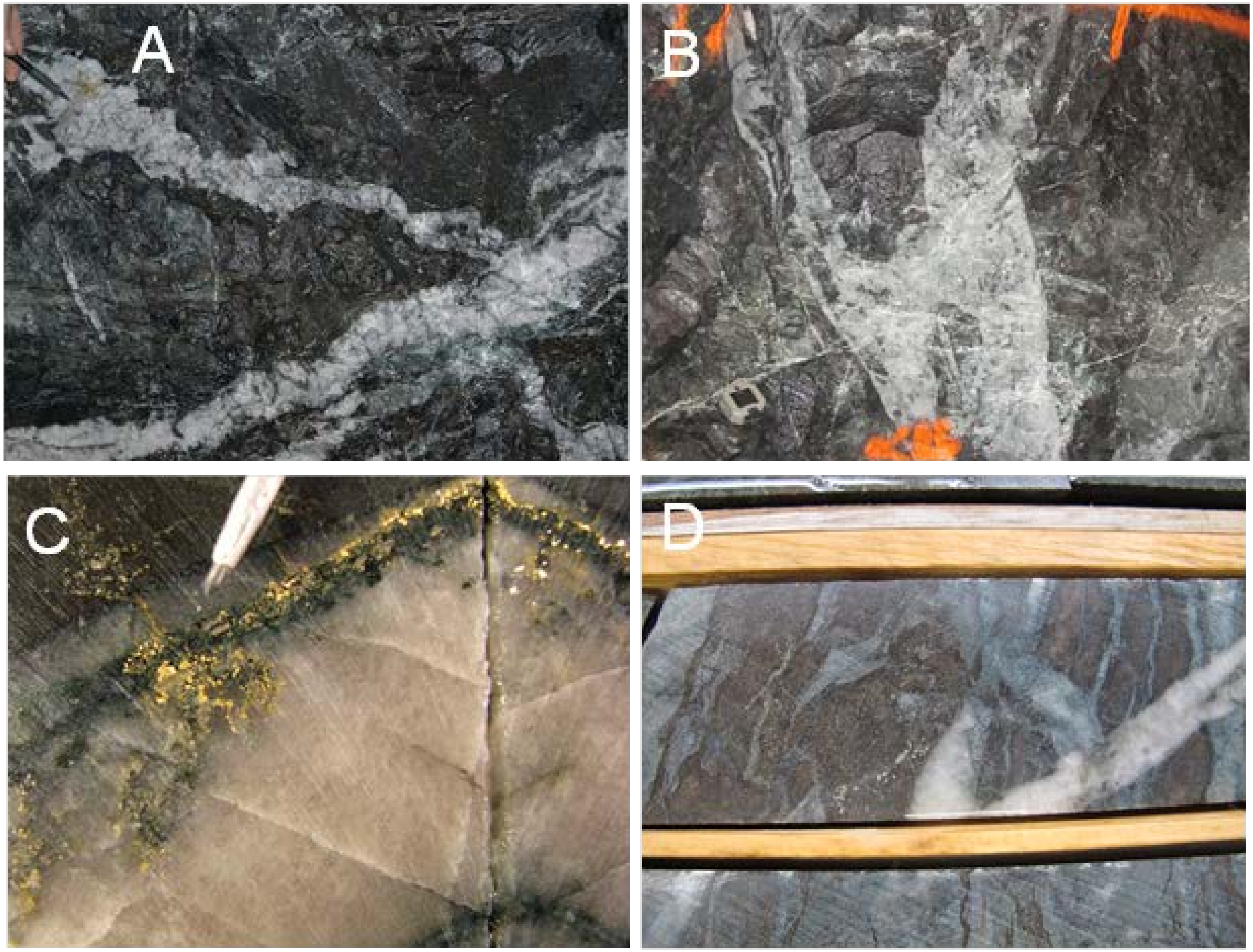


Figure 10: Gold Mineralization Examples from Underground Exposure and Core

A: Visible gold in quartz amphibolite vein in F2 core zone. Pen and fingers for scale.
B: Steep white breccia quartz vein in F2 core zone. Compass for scale.
C: Visible gold in quartz amphibolite vein. Borehole F2-29 at 143.2 metres depth.
D: Altered HiTi hosted actinolite and white quartz veins. Borehole D305-04-014 at 80 metres depth.

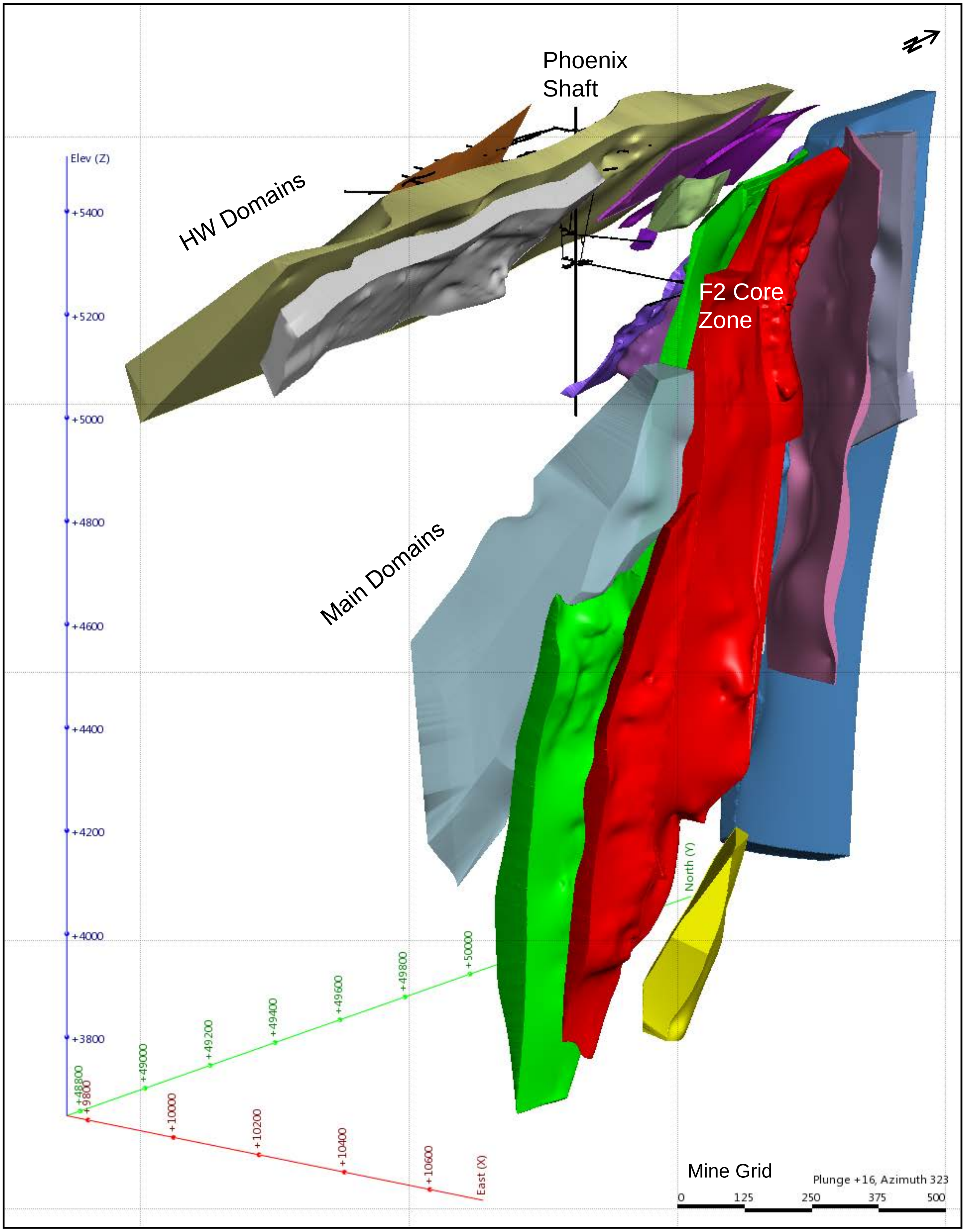


Figure 11: Oblique View Looking North (Mine Grid NW) of Lower Grade Gold Mineralization Wireframes (SRK 2013)

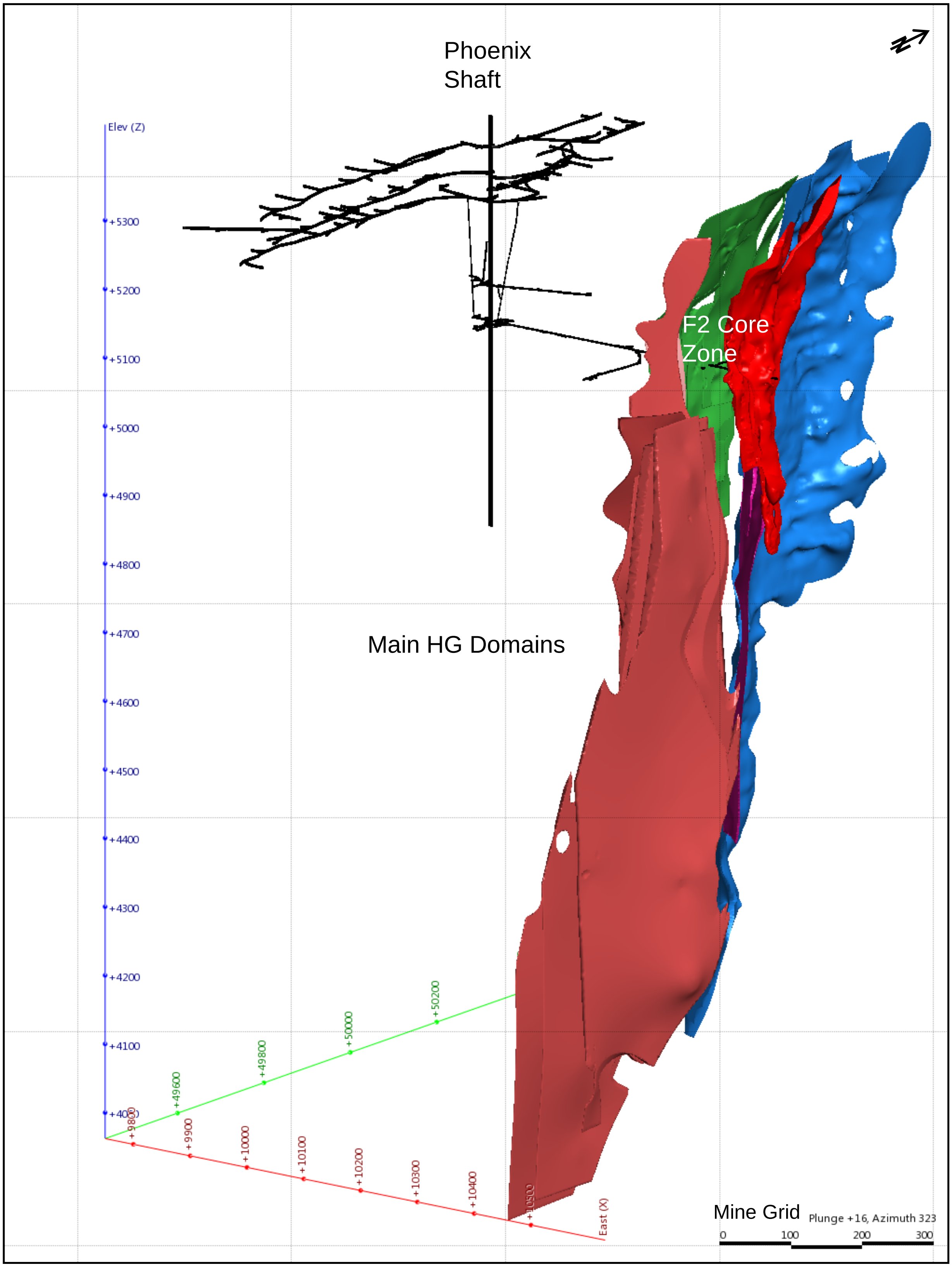


Figure 12: Oblique View Looking North (Mine Grid NW) of High Grade Gold Mineralization Wireframes (SRK 2013)

7 Deposit Types

The Red Lake district is a world-class gold mining district located in the Red Lake Greenstone Belt, which is host to various styles of gold deposits. Twenty-eight mines have operated in the district since 1930 producing over 26 million ounces of gold from three main producing mines: Campbell, Dickenson/Red Lake and Madsen. The gold deposits of the Red Lake Greenstone Belt have been classified into three groups (after Pirie 1981), according to the stratigraphic or lithological associations described below.

Most of the gold production from Red Lake is derived from high-grade quartz-carbonate veins associated with deformation and folding in Balmer Assemblage metamorphosed volcanic, sedimentary and granitoid rocks (Sanborn-Barrie et al. 2004). At the Campbell-Red Lake mine, the main source of gold is found within quartz-carbonate veins associated with the Campbell and Dickenson fault zones and locally controlled by F2 folding (Dubé et al. 2001). Gold in the No. 8 Zone at Madsen mine and at Starratt-Olsen mine are from similar quartz-carbonate veins. A spatial relationship exists between the ultramafic rocks and gold mineralization, with the majority of gold mineralization at Cochenour-Willans and Red Lake gold mines occurring within a few hundred metres of ultramafic bodies. Dubé et al. (2001) suggest that a competency contrast between basalt and ultramafic units is important in the formation of extensional carbonate veins in fold hinge zones during deformation, which are then later replaced by gold-rich siliceous fluids.

The majority of the F2 gold system on the Phoenix gold project is interpreted as being this first type of gold mineralization supported by its location along the East Bay Deformation Zone within favourable Balmer Assemblage mafic and ultramafic rocks as well as the alteration assemblage present. Also of note is the close proximity to the amphibolite-greenschist isograd. Current underground mapping and sampling has demonstrated the existence of most of the associations within this group.

A second type of gold deposit in the Red Lake district is replacement-style, disseminated gold which corresponds to the main source of historical production at the Madsen mine (Dubé et al. 2000). The hydrothermal alteration consists of a changing distribution of andalusite, staurolite, garnet, chloritoid, biotite, and quartz (Andrews et al. 1986). The replacement-style disseminated gold mineralization at Madsen is located at the deformed unconformity between Balmer and Confederation assemblages. Gold mineralization is hosted by mafic volcaniclastic rocks and basalt flows, and consists of heavily disseminated sulphides within a potassic alteration zone, grading outward into an aluminous, sodium depleted zone. Strata-bound replacement style mineralization is present locally throughout the F2 gold system.

A third deposit type in the Red Lake mining district is polymetallic stockwork-style mineralization. The largest of this type of deposit, the McKenzie mine, produced over 650,000 ounces of gold (Andrews et al. 1986). These are typically sulphide-poor quartz-carbonate veins hosted in sedimentary or intermediate to felsic volcanic rocks. They are frequently associated with dikes following the same structural weaknesses. In general these systems develop as narrow, steep-dipping, tabular or splayed veins that are in parallel and offset geometries. Veins can grade into large zones of stockwork or even breccia. Sulphides such as galena, sphalerite, argentite, molybdenite, arsenopyrite, and sulphosalt minerals are often coarse-grained in pods or patches with finer-grained dissemination throughout the vein system. Along the western portion of the F2 gold system, mineralization is hosted locally at the contacts of the felsic dikes and is contained primarily within quartz veins representing this type of gold mineralization.

8 Exploration

8.1 Historical Exploration Work

The history of exploration activities from 1922 to 2002 is discussed in Section 5. Exploration conducted by previous owners is summarized in Table 3.

8.2 Exploration by Rubicon

Since acquiring the Phoenix gold project in 2002, Rubicon has conducted an extensive exploration program which includes geological mapping, re-logging of selected historic boreholes, digital compilation of available historical data, ground and airborne magnetic surveys, mechanical trenching, channel sampling, bathymetric survey, and induced polarization Titan 24 survey, petrographic study, topographic survey, data modelling and processing along with numerous drilling programs. A summary of the exploration activities undertaken at the Phoenix gold project between 2002 and 2012 by Rubicon is shown in Table 6.

Table 6: Summary of Exploration Activities by Rubicon from 2002 to 2012

Period	Exploration Activity
2002	Geological mapping Cataloguing, numbering and re-boxing of historical core cross-piled on property (over 60,000 m) Digital compilation of historical data High resolution airborne magnetic survey 22,000 m^2 of mechanical trenching and power washing (in 2002 and 2004) Channel sampling (876 samples between 2002 and 2004) Overwater bathymetric survey of Red Lake within property boundary 1,900 m of drilling on the Phoenix Peninsula
2003	Re-logging of selected historical boreholes (approximately 23,000 m from 161 boreholes) Digital compilation of historical data Phase 1 drilling program with 9,600 m of winter drilling including ice drilling Phase 2 drilling program consisting of 3,000 m drilled on the Phoenix Peninsula
2004	Continued mechanical trenching and power washing Continued channel sampling Winter drilling program with 7,300 m drilled
2005	11,800 m of surface drilling
2006	1,614 m of surface drilling
2007	13,444 m of surface drilling
2008	First phase of Titan 24 DCIP and MT survey 43,800 m of surface drilling
2009	Second and final phase of airborne Titan 24 survey completed Preliminary petrographic study Surface (44,675 m) and underground (25,512 m) core drilling
2010	Topographic survey utilizing airborne LiDAR technology (light detection and ranging) Surface (37,823 m) and underground (82,068 m) core drilling
2011	Surface (5,462 m) and underground (74,337 m) core drilling
2012	Surface (40,900 m) and underground (17,627 m) core drilling (to cut-off date of November 1, 2012)

A core re-logging program initiated in 2002 formed a solid basis for understanding the nature of mineralization hosted within the hanging wall volcanic units of the EBDZ.

The airborne magnetometer survey flown by Fugro Airborne Surveys in 2002 provided the data necessary to allow re-interpretation of the local geology within the Phoenix property boundary including the extrapolation of known geological contacts, the identification of local structural offsets, and the identification of large target areas such as magnetic lows, which potentially represent magnetic destruction through hydrothermal alteration processes.

The 2008 Titan 24 DCIP survey by Quantec Geoscience was completed after the discovery of the F2 gold system and successfully detected several known near surface gold zones and appears to have detected the alteration related to the F2 gold system. The extensive chargeability anomaly is over 1,500 m long and appears to correlate with strongly altered hosts rocks and sulphide bearing gold mineralization, extending from the southern extents of the F2 gold system to the North Peninsula Zone. The F2 Titan anomaly is one of a number of similar anomalies developed along 3.0 kilometres (km) of prospective stratigraphy extending to the northeast on the property. The anomalies range from vertical depths of 200 to over 800 m and constitute high priority regional targets.

Preliminary petrographic analysis performed by Vancouver Petrographics in 2009 on select representative core samples from the F2 gold system indicated that 90 to 95 percent of the native gold occurs in quartz as equant grains, mainly from 20-100 microns in size. Petrography identified that such fragments should be liberated relatively easily. Finer grains of native gold (mainly 5-20 microns), both in fragments of meta-andesite and less commonly in quartz, will be more difficult to liberate. Most likely the recovery of gold would not increase greatly with grinding below 15 microns.

The procedures and parameters applied for down-hole surveys are discussed in Section 9.2.1, whereas the drillhole sampling methodology and approach are discussed in Section 9.2.2 and Section 9.3.

In the opinion of SRK, the sampling procedures used by Rubicon are consistent with generally accepted industry best practice and the resultant drilling pattern is sufficiently dense to interpret the geometry and the boundaries of the gold mineralization with confidence. All drilling sampling was conducted by appropriately qualified personnel under the direct supervision of appropriately qualified geologists. Accordingly there are no known factors that could materially impact the accuracy and reliability of the results.

The results of the drilling sampling are used to model the geology of the F2 gold system and evaluate mineral resources as described in Section 13.

9 Drilling

9.1 Historical Drilling

The history of exploration activities including drilling activities from 1922 to 2002 is discussed in Section 5. Drilling conducted by previous owners is summarized in Table 3. Historical drilling results were not considered for geology and mineral resource modelling.

9.2 Drilling by Rubicon

Since 2002 and up to November 1, 2012, Rubicon has completed 428,710 metres (m) of core drilling (229,164 m of surface drilling and 199,545 m of underground drilling) on the Phoenix gold project (Table 7). During this period, 355,611 m were drilled on the F2 gold system.

Table 7: Core Drilling Programs

Year	Surface Boreholes		Underground Boreholes		Total
	Number	Metres	Number	Metres	Metres
2002 - 2005	188	41,480			41,480
2006	11	1,614			1,614
2007	24	13,444			13,444
2008	62	43,766			43,766
2009	69	44,675	42	25,512	70,187
2010	49	37,823	199	82,068	119,891
2011	6	5,462	296	74,337	79,799
2012	90	40,900	36	17,627	58,582
Total	**499**	**229,164**	**573**	**199,545**	**428,710**

The property has been evaluated within the context of the current knowledge about the controls on the distribution of the gold mineralization for the producing mines in the Red Lake region and on information gained from extensive exploration on the project. The majority of core drilling by Rubicon has targeted areas outside of the historical McFinley Red Lake Mines areas that were historically perceived to have exploration potential. Key target areas on the Phoenix gold project are presented in Figure 13. At the F2 gold system, infill drilling from surface continues to improve the definition of the gold mineralization at borehole spacing to a 50-metre grid spacing or better. Additionally, underground fans have been drilled approximately 25-35 m apart horizontally, with boreholes collared in 10^{o} vertical increments on section and crossing the F2 core mineralization. The confidence in the continuity of the gold mineralization is robust.

In 2011, a total of 302 core boreholes were drilled (79,799 m), including 5,462 m from surface and 74,337 m from underground (Figure 14). Underground core drilling was conducted on the 305 level, from seven separate drill stations, 305-02 through 305-08. The majority of the drilling was focused on the F2 core zone with a number of boreholes testing the extension of the zone along strike.

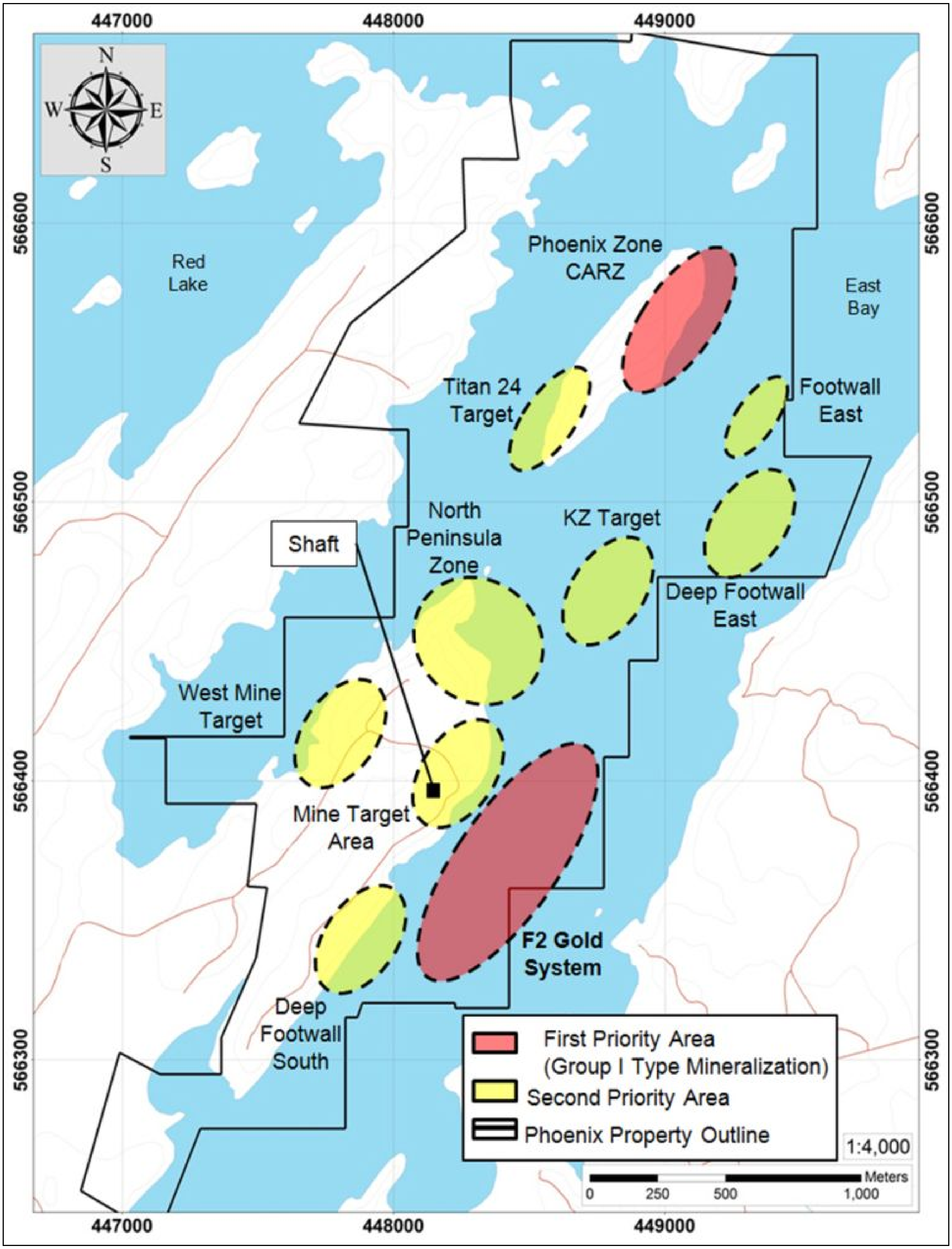


Figure 13: Key Target Areas on the Phoenix Gold Project
(Source: Rubicon 2013)

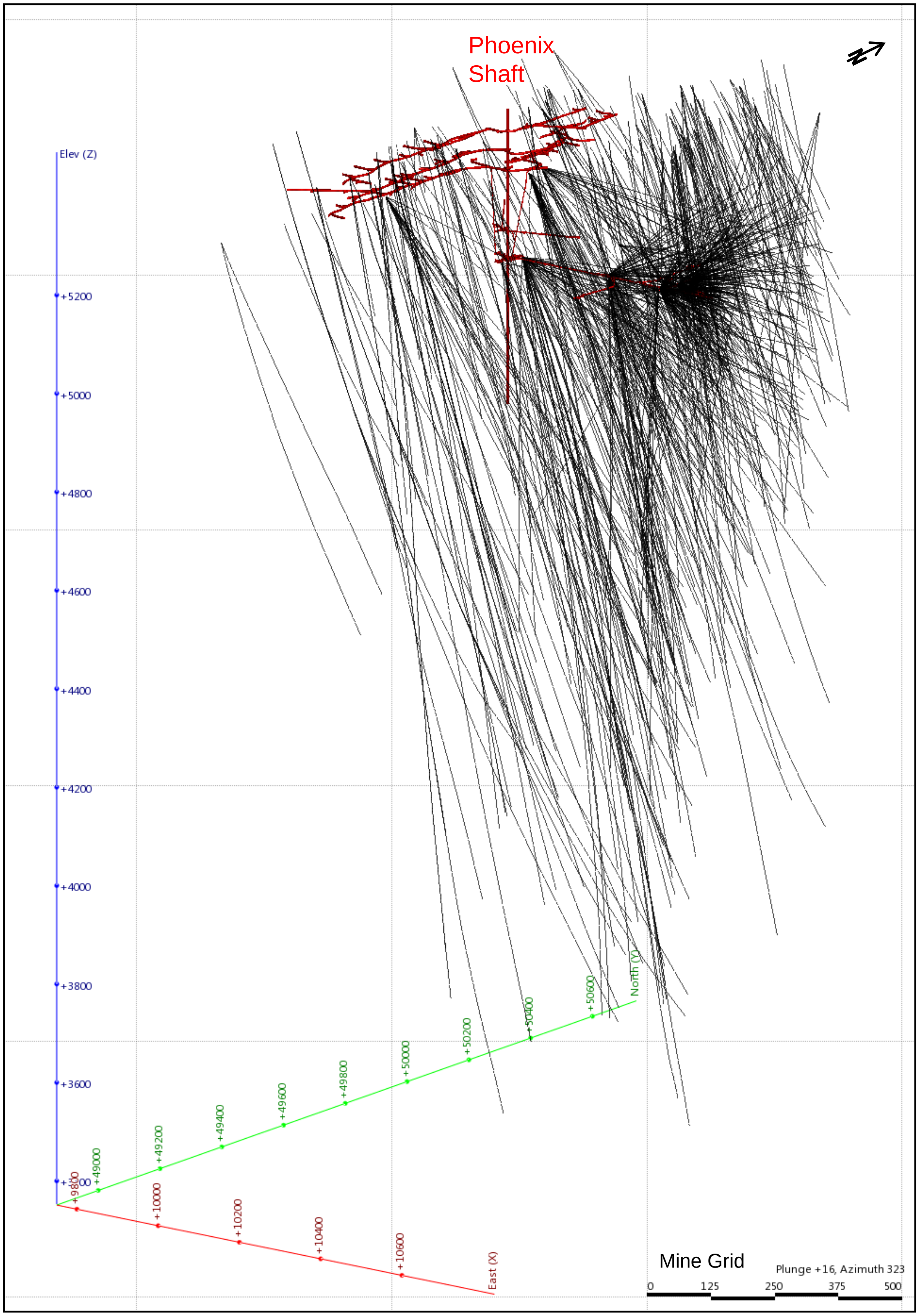


Figure 14: Distribution of Underground and Surface Drilling in the F2 Gold System of the Phoenix Gold Project (Source: SRK 2013)

Legend: Mine shaft and drifts in red, borehole trace in black

The 2011 drilling campaign has continued to define the northeast-trending (F1) gold mineralization associated with silicification, quartz veining and strong alteration within, and adjacent to, favourable host rock types. Gold mineralization also occurs in northwest-trending structures that are generally confined within, or immediately adjacent to, northeast-trending bounding geological units and parallel to the regional F2 fold trend direction. Typically, this mineralization occurs as local quartz veining and brecciation.

In 2012, a total of 126 boreholes (58,528 m) were drilled up to November 1, 2012, the cut-off date for drilling considered for geology and mineral resource modelling. Underground core drilling was conducted from the 305, 244, and 122 levels, from four separate drill stations (305-02, 305-03, 244-09 and 122-03). Surface drilling was carried out on the ice during the winter months, as well as from land. The drilling was focused on the up-plunge of the F2 core zone as well as a series of deep targets.

The 2012 drilling program was successful at demonstrating continuity of the gold mineralization and in extending gold zones within the overall F2 gold system. Drilling continues to define the trend of high grade intercepts and broad lower grade gold zones. Although the main focus of the 2012 drilling campaign was infill, it also expanded the known strike length of the system by 71 m and the depth by 105 m.

A detailed geological and structural interpretation of the stratigraphy suggests that the predominant S1 fabric, parallel to most of the geological contacts, trends northeast and dips to the northwest at 65 to 85 degrees. This northeast fabric is warped by a later F2 folding event, trending northwest.

Faults are oriented in both a northeast (S1) and northwest direction (S2).The gold mineralization is predominantly associated with the contacts between geological units (S1) where there is a strong rheological contrast and can be observed in both the footwall and hanging wall contacts. Gold mineralization is also observed along the crosscutting S2 fabric and faults related to the later folding event (F2). Based on the distribution of the drilling to date an apparent plunge of the gold mineralization to the southwest is observed. The system remains open along strike and at depth.

9.2.1 Drilling Procedures

All proposed land and ice borehole collars were surveyed with a handheld global positioning system (GPS) instrument with an accuracy of ±3 m. Two foresight pickets were also surveyed and drills were set up under the direct supervision of a Rubicon geologist or geological technician. Collars for barge boreholes were also surveyed with a handheld GPS instrument and then marked with a buoy; the same foresight procedure was carried out. Changes in actual borehole location from planned locations, due to local ice conditions or other technical reasons were noted with the true easting and northing coordinates. Final collar locations are surveyed with a differential GPS unit (sub-metre accuracy) and recorded in the database. All surveys currently use the mine grid, which lies at an orientation of +45^{o} to the UTM grid.

The majority of the core drilling performed on the F2 gold system has been carried out by Hy-Tech Drilling of Smithers, British Columbia using Tech-4000 diamond core drills both from surface (on land, ice or barge) having a depth capacity of 2,500 m and from underground having a depth capacity of 1,500 m. Layne Christensen Canada Limited of Sudbury, Ontario was also contracted to complete deep boreholes using their skid-mounted CS 4002, which has a depth capacity of 2,500 m. Orbit Garant Drilling of Val-d'Or, Quebec was contracted to complete underground drilling using either a B-20 or Orbit 1500, which have a depth capacity of 1,500 m. Each drilling program was supervised by a Rubicon geologist. Generally, NQ2 (50.8 millimetres [mm] diameter) or NQ (47.6 mm diameter) core was drilled.

Casing for boreholes collared on land were left in place, plugged, cemented, and covered with aluminum caps with the borehole number etched or stamped into the cap. Boreholes that were drilled from the ice or barge were plugged with a Van Ruth plug at 30 m down the borehole from the base of the casing, and then cemented to the top of the borehole. All casing was removed from these boreholes. Since January 2012, all boreholes drilled from the ice or barges are cemented from the bottom of the hole to the base of the casing.

A Reflex or Ranger electronic single shot survey instrument was used to take down-hole surveys recording azimuth, inclination, magnetic tool face angle, gravity roll angle, magnetic field strength, and temperature at 60-metre intervals.

9.2.2 Sampling Method and Approach

Core was laid in wooden core boxes at the drilling site, with depth markers at every 3 m, and it was sealed with a lid and strapped with plastic bindings. Boxes were delivered once a day by the drilling contractor or Rubicon personnel to the on-site core logging facility.

Rock quality designation (RQD) and total core recovery were routinely measured after each drilling run. Core recovery was measured as actual recovered core length against drilled run length and recorded as a percentage. Core recovery was generally very good (greater than 98 percent).

Upon delivery of the core boxes to the core shack, the core boxes were placed in sequential order for description by an appropriately qualified geologist. The description procedure involved collecting elaborate information about colour, lithology, alteration, weathering, structure, and mineralization. Data was captured directly into a standardized computerized database.

Core sampling intervals were marked by considering geology by an appropriately qualified geologist. Core assay samples were collected from half core sawed lengthwise with a diamond saw. Sampling intervals of mineralized zones were set at a standard 1-metre length or less considering geological contacts.

9.3 SRK Comments

In the opinion of SRK, the sampling procedures used by Rubicon are consistent with generally accepted industry best practice and the resultant drilling pattern is sufficiently dense to interpret the geometry and the boundaries of the gold mineralization with confidence. All drilling sampling was conducted by appropriately qualified personnel under the direct supervision of appropriately qualified geologists. Accordingly there are no known factors that could materially impact the accuracy and reliability of the results.

The results of the drilling sampling are used to model the geology of the F2 gold system and evaluate mineral resources as described in Section 13.

10 Sample Preparation, Analyses, and Security

10.1 Sample Preparation and Security

Upon arrival at the core storage facility, the core was washed, logged, and split using a diamond blade saw under the on-site supervision of a Rubicon geologist. Samples were moved directly from the core shack to the cutting shack and then they were cut and shipped in individual zip tied sample bags. Approximately 10 individual bagged samples were placed in a large rice bag that was sealed with a security zip tie containing a unique numbered tamper-proof security seal. Since 2008, samples were delivered directly from the mine site to the SGS Canada Inc. (SGS) laboratory in Red Lake by Rubicon staff. Each sample number and security seal was recorded and then verified by SGS with a written acknowledgment upon receipt.

Individual samples received at the laboratory typically ranged from 0.5 to 2 kilograms in weight. The samples were dried prior to any sample preparation at the laboratory. The entire sample was crushed to 2 millimetres in an oscillating steel jaw crusher and either an approximate 250 gram split, or, in the case of metallics fire assay, the whole sample was pulverized in a chrome steel ring mill. The coarse reject was bagged and stored. The samples were then crushed to 90% -8 mesh, split into 250- to 450-gram subsamples using a Jones Riffle Splitter and subsequently pulverized to 90% -150 mesh in a shatter box using a steel puck. Prior to analysis, the samples were homogenized. Silica cleaning between each sample was also performed to prevent any cross-contamination. All samples were sent for fire assay and the pulps remained on-site.

The logged and sampled core is stored at the project site in a secured area (locked building) near the core shack. There is only one road into the mine site, which has a gate with 24-hour security and restricted access. The pulps and rejects were returned from SGS and stored on the project site for long-term storage.

10.2 Sample Analyses

All analytical or testing laboratories used are independent of Rubicon. Various analytical laboratories have been used by Rubicon over time and these are discussed below. Samples collected before 2008 were sent to either the ALS Minerals (ALS) preparation lab in Thunder Bay, Ontario, or its analytical lab in Vancouver, British Columbia, or to Accurassay Laboratories (Accurassay), Thunder Bay, Ontario. Since January 2008, assays have been conducted by SGS in Red Lake, Ontario. Umpire check assays have been completed on 5 percent (%) of these assays since January 2010 and were analyzed by ALS.

All three laboratories are accredited to ISO/IEC Guideline 17025 by the Standards Council of Canada for conducting certain testing procedures, including the procedures used for assaying gold.

Dr. Barry Smee, PGeo, Consultant Geochemist, audited the sample preparation facilities of SGS in Red Lake, Ontario on behalf of Rubicon. Recommendations from his audit were provided to SGS and corrective measures were implemented (Smee and Associates Consulting Ltd., 2009 and 2011).

Assay results from the historical core, when sampled, are taken as indicative since the drilling of these boreholes was not conducted under Rubicon supervision. Data deemed historic in nature have not been included in the resource estimate.

10.2.1 ALS Minerals (From 2002 – 2007)

Gold concentrations were determined by fire assay fusion of a 50-gram subsample with an atomic absorption spectroscopy (AAS) finish. This is the standard procedure used in ongoing umpire check analyses. The gold-metallics assay, also known as screen fire assaying, required 100% pulverization of the sample and screening of the sample through a 150 mesh (100 micron). Material remaining on the screen was retained and analyzed in its entirety by fire assay fusion followed by cupellation and a gravimetric finish. The -150 mesh (pass) fraction was homogenized and two 50-gram subsamples were analyzed by standard fire assay procedures. In this way, the magnitude of the coarse gold effect can be evaluated via the levels of the +150 mesh material.

Representative samples for each geological rock unit and, generally, at least one sample every 20 metres (m), were selected for multi-element assaying using inductively-coupled plasma atomic emission spectroscopy (ICP-AES), following four-acid digestion. Copper, lead, and zinc values exceeding ICP-AES limits were re-assayed using wet chemistry. Only a few samples were assayed for whole rock major elements using X-ray fluorescence spectrometry (XRF).

Results were reported electronically to the project site in Red Lake and to the head office in Vancouver to multiple recipients with assay certificates filed and catalogued at Rubicon's head office in Vancouver.

10.2.2 Accurassay Laboratories (From 2002 – 2007)

Gold was determined by fire assay using a 30-gram fire assay charge. This procedure used lead collection with a silver inquart. The beads were then digested and an atomic absorption or ICP finish was used. All gold assays greater than 10 gpt were automatically re-assayed by fire assay with a gravimetric finish. A Sartorius micro-balance was used with a sensitivity of 1 microgram (six decimal places) giving a 5 parts per billion (ppb) detection limit.

Screen metallics analysis included the crushing of the entire sample to 90% -10 mesh and using a Jones Riffle Splitter to split the sample to a 1 kilogram subsample. The entire subsample was then pulverized and subsequently sieved through a series of meshes (80, 150, 200, 230, 400 mesh). Each fraction was then assayed for gold (maximum 50 gram). Results were reported as a calculated weighted average of gold in the entire sample.

Core samples were also assayed for a suite of 32 trace elements using a multi-acid digestion followed by ICP-AES.

As with ALS, results were reported electronically to the project site in Red Lake with assay certificates filed and catalogued at Rubicon's head office in Vancouver.

10.2.3 SGS Mineral Services (Since January 2008)

Prior to 2009, gold was analyzed using the fire assay process on a 30-gram subsample. If the sample contained greater than 10 grams per tonne (gpt) gold, it was sent for a gravimetric finish. Starting in October 2009, the assay subsample size was increased to 50 grams on the recommendations of Smee (2009). All gold assays greater than 10 gpt were automatically re-assayed by fire assay with gravimetric finish.

A select suite of sample pulps were also assayed for a suite of 50 trace elements by the SGS Laboratory in Toronto, Ontario, using a multi-acid digestion and ICP-AES.

Results were reported electronically to the project site in Red Lake and to the head office in Vancouver to multiple recipients with assay certificates filed and catalogued at Rubicon's head office in Vancouver and added to the master Microsoft Access database stored on the Vancouver and Red Lake servers.

10.2.4 Handling of Multiple Assay Values for One Sample

In cases where multiple assays were completed on an individual sample, gold values produced by the metallic fire assay are deemed to supersede fire assay gold values owing to the larger size of the sample analyzed and/or the better reproducibility in samples with coarse gold.

10.3 Sample Analyses of Metallurgical Testwork

10.3.1 G&T Metallurgical Services

Metallurgical testwork was completed at the G&T Metallurgical Services Ltd. (G&T) facility in Kamloops, British Columbia. Gold was measured by fire assay method using a 30 gram assay charge. When requested, metallic sieve preparation method was also used. The lab has a complete written procedure and participates in a Proficiency Testing Program accredited by the Standards Council of Canada. This facility also performed assays for iron and arsenic content using a multi-acid digestion and ICP-AES method, and assays for sulfur and carbon by combustion furnace.

G&T also performed different metallurgical testing for the characterisation of the mineralized material. All the tests performed were done using industry recognized methods for the testwork. The facility was visited by Soutex personnel. The facility has a well-documented controlled procedure for all types of testing and the procedures are available at the site. The quality management includes ISO-9001 accreditation.

10.3.2 ALS Minerals

All the assays related to the treatment of the bulk samples at SMC (Canada) Ltd. (SMC)'s McAlpine mill in Cobalt, Ontario during the summer and fall of 2011 were sent to ALS Minerals' accredited laboratories. Gold assays were done with fire assay on a 30 gram assay charge. All samples that were head grade samples and tailings samples were prepared with screen metallic sieve preparation done on the whole received sample. All samples of gold concentrate were assayed without screen metallic sieve preparation. The samples were expedited and received at the Val d'Or facility and the assays were performed in ALS Minerals' laboratory in North Vancouver. A series of blank, duplicate and certified samples were also sent to the laboratory for quality control.

10.4 Specific Gravity Data

The specific gravity database includes 6,666 records generated by Rubicon from measurements on core from 470 boreholes (Table 8). Of these records, 6,562 measurements are available within the mineralization envelopes modelled by SRK. Specific gravity measurements were taken from selected core samples intervals 1-metre in length that is representative of the major rock types. Specific gravity was measured using a water dispersion method. The samples were weighed in air, and then placed in a basket suspended in water and weighted again.

Table 8: Specific Gravity Data by Lithology Type

Rock Code	Description	Count	Specific Gravity			
			Average	STDV	Min	Max
E1H	High titanium basalt	1,396	2.96	0.10	2.20	3.72
EOT	Talc rich unit	1,600	2.90	0.05	2.61	3.15
I3	Felsic intrusives	847	2.67	0.07	2.36	3.08
E0	Ultramafic flow	1,264	2.92	0.08	2.50	3.76
E0B	Komatiitic basalt	370	2.98	0.07	2.61	3.24
E1A	Basalt	198	2.89	0.09	2.67	3.54
AGZ	Altered green zone	97	2.93	0.09	2.69	3.20
Other	Other	894	2.88	0.12	1.85	3.45
Total		**6,666**				

10.5 Quality Assurance and Quality Control Programs

Quality control measures are typically set in place to ensure the reliability and trustworthiness of exploration data. These measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management and database integrity. Appropriate documentation of quality control measures and regular analysis of quality control data are important as a safeguard for project data and form the basis for the quality assurance program implemented during exploration.

Analytical control measures typically involve internal and external laboratory control measures implemented to monitor the precision and accuracy of the sampling, preparation and assaying. They are also important to prevent sample mix-up and to monitor the voluntary or inadvertent contamination of samples.

Assaying protocols typically involve regularly duplicating and replicating assays and inserting quality control samples to monitor the reliability of the assaying results throughout the sampling and assaying process. Check assaying is normally performed as an additional test of the reliability of the assaying results; it generally involves re-assaying a set number of sample rejects and pulps at a secondary umpire laboratory.

Rubicon relied partly on the internal analytical quality control measures implemented by the primary laboratories used. In addition, Rubicon implemented external analytical control measures starting in 2008 on all sampling conducted at the Phoenix gold project. Analytical control measures by Rubicon consist of inserting control samples (blank, certified reference material, and field duplicates) in all sample batches submitted for assaying.

The blank consisted of store-bought white garden stone (quartz or quartzite). In 2010, Rubicon used material sourced from a granite boulder located near Red Lake, just off a northern road in the bush. From February 2011, Rubicon has been using granite slab purchased from Nelson Granite in Vermillion Bay, Ontario.

Field duplicates consist of half core and have been taken since June 2009. Twenty-seven gold commercial certified reference materials sourced from CDN Resource Laboratories Ltd. (CDN) were used in sampling between 2008 and 2012. Control samples used range from 0.121 to 29.21 gpt gold (Table 9).

Table 9: Specifications of CDN Certified Control Samples Used by Rubicon on the Phoenix Gold Project between 2008 and 2012

Gold Reference Material	Recommended Value (gpt Au)	Standard Deviation (gpt)	Number of Samples
CDN-GS-P1	0.121	0.011	58
CDN-GS-P5B	0.44	0.02	90
CDN-GS-P7A	0.77	0.03	93
CDN-GS-P8	0.78	0.03	178
CDN-GS-10	0.82	0.05	3
CDN-GS-1J	0.946	0.051	170
CDN-GS-1H	0.972	0.054	297
CDN-GS-1G	1.14	0.05	91
CDN-GS-1E	1.16	0.03	1,649
CDN-GS-1P5A	1.37	0.06	16
CDN-GS-1P5B	1.46	0.06	83
CDN-GS-9	1.75	0.07	123
CDN-GS-2B	2.03	0.06	77
CDN-GS-2A	2.04	0.095	5
CDN-GS-2C	2.06	0.075	243
CDN-GS-3E	2.97	0.135	107
CDN-GS-3D	3.41	0.125	180
CDN-GS-5C	4.74	0.14	1
CDN-GS-5E	4.83	0.185	1,244
CDN-GS-5J	4.96	0.21	162
CDN-GS-5A	5.1	0.135	10
CDN-GS-5F	5.3	0.18	431
CDN-GS-6A	5.69	0.24	306
CDN-GS-7A	7.2	0.3	121
CDN-GS-6A	9.99	0.25	8
CDN-GS-11A	11.21	0.435	17
CDN-GS-30B	29.21	0.615	170

Control samples (including blanks, gold mineralized reference material, and field duplicates) were inserted every 25 samples. In addition, umpire laboratory testing was performed on approximately 5% of samples since the beginning of 2010.

10.6 SRK Comments

In the opinion of SRK, the sampling preparation, security, and analytical procedures used by Rubicon are consistent with generally accepted industry best practices and are, therefore, adequate for the purpose of mineral resource estimation.

11 Data Verification

11.1 Verifications by Rubicon

The core drilling discussed in this report was undertaken by experienced and competent Rubicon geologists under the supervision of Matthew Wunder, PGeo (APGO #1316), Rubicon Vice President, Exploration, Terry Bursey, PGeo (APGO#1375), Manager, Community Relations, and Ian Russell, Manager, Special Projects.

Rubicon performs logging, surveying, sample selection, and inserts analytical quality control samples. Data are verified and double checked by senior geologists at site (for data entry verification, error analysis, plus assay pass/fail against standards and blanks, etc.). Borehole data are reviewed by ioGlobal Pty Ltd. (ioGlobal) for quality assurance and quality control.

Blank and standards assay protocols were developed in 2003 and revisited in 2009 and 2011 with input from Dr. Barry Smee, PhD, PGeo, independent geochemist, in consultation with Rubicon personnel.

Sample batches are re-analyzed when anomalies are observed. Rubicon initiated an assay check sampling program in 2010 where 5 percent (%) of the sample pulps are reassayed by ALS. Controls samples were also inserted to provide quality control on the re-assays samples. Results from this check assay program are reviewed for accuracy and tracked in an action log as part of the standard quality assurance and quality control procedures. Failures are addressed and re-assayed as required.

11.2 Verifications by SRK

11.2.1 Site Visit

In accordance with National Instrument 43-101 guidelines, SRK visited the Phoenix gold project on various occasions between October 2011 and April 2013. At the time of the visits, underground drilling activities were ongoing on the project. The purpose of the site visits was to ascertain the geological setting of the project, witness the extent of exploration work carried out on the property, and assess logistical aspects, and other constraints relating to conducting mining activities in this area. SRK reviewed the exploration database and validation procedures, reviewed exploration procedures, defined geological modelling procedures, examined core, and interviewed project personnel.

11.2.2 Verification of Analytical Quality Control Data

Rubicon provided SRK with internal and external analytical control data containing the assay results for the quality control samples for the F2 gold system between 2008 and 2012. All data was provided in a Microsoft Access database.

SRK aggregated the assay results of the external analytical control samples for further analysis. Blanks and gold certified reference material data were summarized on time series plots to highlight the performance of the control samples. Paired data (field duplicates and umpire check assays) were analyzed using bias charts, quantile-quantile, and relative precision plots.

The external analytical quality control data produced for the F2 gold system between 2008 and 2012 are summarized in Table 10 and presented in graphical format in Appendix A. The external quality control data produced on this project represents 12.5% of the total number of samples assayed.

Table 10: Summary of Analytical Quality Control Data Produced By Rubicon on the F2 Zone, Phoenix Gold Project between 2008 and July 2012

	Total	**(%)**	**Comment**
Sample Count	130,126		
Blanks	5,808	4.46%	Store-bought white garden stone; granite boulder; granite slab from Nelson Granite
Certified Reference Material	5,963	4.58%	
CDN-GS-P1	58		CDN (0.121 ppm Au)
CDN-GS-P5B	90		CDN (0.44 ppm Au)
CDN-GS-P7A	93		CDN (0.77 ppm Au)
CDN-GS-P8	178		CDN (0.78 ppm Au)
CDN-GS-10	3		CDN (0.82 ppm Au)
CDN-GS-1J	170		CDN (0.946 ppm Au)
CDN-GS-1H	297		CDN (0.972 ppm Au)
CDN-GS-1G	91		CDN (1.14 ppm Au)
CDN-GS-1E	1,649		CDN (1.16 ppm Au)
CDN-GS-1P5A	16		CDN (1.37 ppm Au)
CDN-GS-1P5B	83		CDN (1.46 ppm Au)
CDN-GS-9	123		CDN (1.75 ppm Au)
CDN-GS-2B	77		CDN (2.03 ppm Au)
CDN-GS-2A	5		CDN (2.04 ppm Au)
CDN-GS-2C	243		CDN (2.06 ppm Au)
CDN-GS-3E	107		CDN (2.97 ppm Au)
CDN-GS-3D	180		CDN (3.41 ppm Au)
CDN-GS-5C	1		CDN (4.74 ppm Au)
CDN-GS-5E	1,244		CDN (4.83 ppm Au)
CDN-GS-5J	162		CDN (4.96 ppm Au)
CDN-GS-5A	10		CDN (5.10 ppm Au)
CDN-GS-5F	461		CDN (5.30 ppm Au)
CDN-GS-6A	306		CDN (5.69 ppm Au)
CDN-GS-7A	121		CDN (7.20 ppm Au)
CDN-GS-6A	8		CDN (9.99 ppm Au)
CDN-GS-11A	17		CDN (11.21 ppm Au)
CDN-GS-30B	170		CDN (29.21 ppm Au)
Field Duplicates	4,426	3.40%	Half Core
Total QC Samples	**16,197**	**12.45%**	
Check Assays			
ALS, Thunder Bay	**4,406**	**3.39%**	Pulp Duplicates

In general, the performance of the control samples (blank, certified reference material, and field duplicates) inserted with samples submitted for assaying is acceptable. Less than 1% of blank samples returned assay values above 0.055 gpt gold, the batch assessment criteria threshold (failure limit) determined by Rubicon. However, a number of blanks (above 0.088 gpt gold) have gold values similar to certified reference material, indicating possible sample misidentification. The blank material used prior to October 2009 was suspected by Rubicon of containing lower levels of gold and new blank material was sourced and inserted in sample batches submitted for assaying. However, the new blank material may have low levels of gold as well.

Rubicon uses three standard deviations as a batch assessment criteria threshold (failure limit). A number of individual certified reference materials are outside three standard deviations. However, the certified reference materials have similar means to the recommended value and/or less than 5% of the samples are outside three standard deviations. A number of certified reference materials have gold values similar to the blanks or other certified reference materials, indicating possible sample misidentification. SGS had difficulty with the precision and accuracy of certified reference material CDN-GS-2B and CDN-GS-3D. These control samples were used between 2008 and 2009.

Paired assay data for field duplicates produced by SGS and examined by SRK suggest that that gold grades are difficult to reproduce from the pulverized sample. Ranked half absolute relative difference (HARD) plots suggest that only 33% to 41% of the field duplicate sample pairs have HARD below 10%. The poor reproducibility of field duplicate results is to be expected in gold deposits with a strong nugget effect. The reproducibility does, however, improve between 2008 and 2012.

Rubicon also submitted approximately 3.39% of assay pulp duplicates originally assayed at SGS to ALS for umpire laboratory testing. Between 25.9% and 49.6% of the umpire check assay pairs tested have a HARD below 10%. The low reproducibility between the two laboratories could be related to the poor reproducibility of field duplicate results at SGS. There is, however, no apparent bias between the two laboratories.

Overall, SRK considers that the analytical quality control data reviewed by SRK delivered by the primary laboratory used by Rubicon are sufficiently reliable for the purpose of mineral resource estimation. The data sets examined by SRK do not present obvious evidence of analytical bias.

11.3 Verifications by Soutex

In accordance with National Instrument 43-101 guidelines, Soutex visited the Phoenix gold project on two occasions in February and April 2011.

The G&T facility in Kamloops, British Columbia was also visited by Soutex personnel. Quality control procedures are well documented for all types of testing and the procedures are available at the site. The quality management of the laboratory includes ISO-9001 accreditation.

In the opinion of Soutex, the sampling preparation, security, analytical procedures and specific testing used in all the metallurgical testwork are consistent with generally accepted industry best practices and are, therefore, adequate for the purpose of predicting the metallurgical performance that can be obtained on the material extracted from the Phoenix gold project.

12 Mineral Processing and Metallurgical Testing

12.1 Background

In September 2008, Vancouver Petrographics Ltd. (Vancouver Petrographics, September 2008) performed a petrographic analysis on 10 thin sections derived from representative mineralized core samples from the F2 Zone.

In October 2010, Rubicon completed a metallurgical testwork program (the “2010 study”) performed by Soutex (Soutex, October 2010). The study was done on small samples from different underground zones. The testwork program was conducted at G&T under the supervision of Soutex (G&T, July 2010). This study included running a metallurgical testwork program, developing a preliminary milling process, and designing a preliminary concentrator. The design addressed the gold recovery process from a material delivered by the mine skip to the cyanide-free tailings going to the tailings management facilities (TMF), and the production of gold doré. Paste plant considerations and TMF were not included in the study.

In September 2011, Rubicon completed a further metallurgical testwork program (the “2011 study”) performed by Soutex. The study was done on representative subsamples (composites) extracted from two approximately 1,000-tonne bulk samples representing two underground areas on the 305 level. The metallurgical testwork program was conducted at G&T under the supervision of Soutex. (G&T, October 2011).

Characterization of mineralized material competency for semi-autogenous grinding (SAG) milling was performed by G&T under the supervision of JKTech Pty Ltd. (JKTech, March 2011). Grinding circuit design was validated by simulation with SGS Minerals Services (SGS, June 2011). In July 2012, the processing of the two approximately 1,000-tonne bulk samples was completed at SMC McAlpine mill (SMC) under the supervision of Soutex in order to reconcile the bulk sample grades against the resource estimate (Soutex, July 2013).

12.2 The Nature and Extent of the Testing Materials

Three sets of samples were available for characterization and metallurgical testwork programs:

- 2010 study samples: seven composite samples composed of 155 core samples. Five samples were quarter core and two samples were assay rejects. Testwork was done at G&T;
- 2011 study samples: two 10-tonne subsamples (composites) from the two approximately 1,000-tonne bulk samples. Testwork was done at G&T; and
- 2011 bulk samples: two approximately 1,000-tonne bulk samples. Testwork was done at SMC under the continuous on-site supervision of Soutex personnel.

12.2.1 Elemental Characteristics

2010 Study Samples - Core and Assay Reject Composite Samples

In the 2010 study, of the seven representative composite samples treated, five samples (RL-01-01 to RL-01-05) originated from core samples and two samples (RL-02-01 and RL-02-02) originating from core assay rejects. Composite drill core samples consisted of ¼ sawn NQ2 core samples.

The iron and arsenic were assayed using a multi-acid digestion and ICP-AES method, while sulfur and carbon were assayed using combustion furnace. The main chemical elements were assayed using specific methods. Gold assays were conducted by fire assay with a 30 gram assay charge using a metallic sieve preparation on 100 gram samples to reduce assay variability. All assays were done at the G&T Metallurgical Service facility. Table 11 presents the head grade results for all samples.

Table 11: 2010 Study Sample Head Grades

Sample	Grade				
	Au (gpt)	Fe (%)	S (%)	C (%)	As (%)
RL-01-01	8.85	5.56	2.11	1.18	0.03
RL-01-02	6.04	9.15	2.82	1.04	0.08
RL-01-03	4.12	8.85	2.19	0.51	0.04
RL-01-04	9.14	9.95	2.81	1.10	0.01
RL-01-05	4.89	5.18	1.57	1.00	0.01
RL-02-01	12.80	5.47	2.16	1.18	0.02
RL-02-02	8.96	8.80	2.66	1.11	0.06
Average	**7.82**	**7.42**	**2.33**	**1.01**	**0.04**

Highlights are the following:

- On average, the samples contained 7.82 gpt gold (ranging from 4.12 to 12.8 gpt);
- On average, the samples contained 2.33% sulphur (ranging from 1.57% to 2.82%). These results are consistent with the presence of widespread sulphide minerals (pyrite and pyrrhotite) in the mineralized zones. The sulphide minerals contribution comes from pyrrhotite and lesser amounts of pyrite. Generally, only trace amounts of arsenopyrite are present in the samples, although the RL-01-02 and RL-02-02 samples contained somewhat increased levels of arsenopyrite;
- On average, the samples contained 0.04% arsenic (ranging from 0.01% to 0.08%), which is relatively low; and
- On average, the samples had a specific gravity of 2.78 (ranging from 2.67 to 2.84).

2011 Study Samples - 10 Tonne Subsamples (G&T)

In 2011, two representative 1,000-tonne bulk samples were extracted from the 305 level and coarse crushed to 90% minus ½ inch. Two 10-tonne subsamples were extracted as splits from the 1,000-tonne bulk samples and shipped to G&T Labs in Kamloops, BC for analysis. Ten separate 15 kg composite samples were then collected as splits taken from the 10-tonne subsamples. These composite samples were analysed and the iron and arsenic were assayed using a multi-acid digestion and ICP-AES method while sulfur and carbon were assayed using combustion furnace. Gold assays were conducted by fire assay using a 30 gram assay charge. All the assays were done at the G&T Metallurgical Service facility. Table 12 presents the head grade results for both subsamples.

Highlights are the following:

- On average, the composite samples from bulk sample #1 were assayed at 7.6 gpt gold (ranging from 4.94 to 20.3 gpt gold) and the composite samples from bulk sample #2 contained 8.6 gpt gold (ranging from 5.47 to 23.3 gpt gold);

- On average, the samples from bulk sample #1 were assayed at 2.6% sulphur (ranging from 2.49% to 2.66%) and the samples from bulk sample #2 contained 1.7% sulphur (ranging from 1.64% to 1.80%);
- On average, the samples from bulk sample #1 were assayed at 0.043% arsenic (ranging from 0.038% to 0.048%) and the samples from bulk sample #2 contained 0.052% arsenic (ranging from 0.047% to 0.069%), which is relatively low; and
- As part of the drop weight testing (DWT) (JKTech), specific gravities were determined from 30 randomly selected particles ranging in the size from 26.5 to 31.5 millimetres. The samples had an average specific gravity of 3.05 (ranging from 2.87 to 3.37).

Table 12: 2011 Study Sample Head Grades

Composite	Sample	Au Assay gpt		Assay Percent		
		A	B	Fe	S	As
Composite 1	Head 1	10.00	4.95	11.00	2.60	0.046
	Head 2	5.88	5.34	10.90	2.54	0.039
	Head 3	5.49	5.82	10.60	2.49	0.045
	Head 4	7.49	20.30	10.70	2.64	0.045
	Head 5	5.64	12.40	10.90	2.62	0.042
	Head 6	9.13	7.38	10.70	2.57	0.042
	Head 7	7.93	5.64	10.80	2.65	0.040
	Head 8	5.14	6.69	10.10	2.51	0.040
	Head 9	8.85	8.43	10.00	2.66	0.045
	Head 10	4.94	5.78	9.90	2.63	0.038
	Laboratory Head 1	4.98	5.27	10.30	2.55	0.048
	Laboratory Head 2	12.90	5.40	10.40	2.60	0.047
	Average	**7.60**		**10.50**	**2.60**	**0.043**
Composite 2	Head 1	8.10	5.47	6.40	1.71	0.053
	Head 2	8.09	7.17	6.30	1.64	0.055
	Head 3	6.45	7.40	6.20	1.74	0.047
	Head 4	5.92	6.33	6.60	1.80	0.069
	Head 5	10.10	16.20	6.60	1.78	0.049
	Head 6	8.47	7.12	6.10	1.73	0.047
	Head 7	23.30	12.60	6.40	1.73	0.053
	Head 8	6.96	6.42	6.70	1.75	0.049
	Head 9	10.40	7.82	6.40	1.67	0.052
	Laboratory Head 1	6.79	5.80	6.50	1.77	0.052
	Laboratory Head 2	6.07	7.16	6.40	1.70	0.049
	Average	**8.60**		**6.40**	**1.70**	**0.052**

During the processing of the two 10-tonne subsamples, a series of bench scale gravity floatation and cyanide leach test were completed to estimate the final gold grades from the production samples collected. Gold assays were conducted by fire assay with a 30 gram assay charge. Metallurgical balances for the concentrates and the tailings grades were summed to compile the gold content for each bulk sample and are presented in Table 13 .

These results from the metallurgical balance represent the most accurate method available for estimating the gold grades because the head grade was calculated from the concentrate and the tailings grades.

All the assays were done at the G&T facility.

Table 13: 2011 Study Sample Gold Head Grades

Description	Bulk Sample #1 Composite (gpt)	Bulk Sample #2 Composite (gpt)
Gold Head Grade	6.79	8.41

2011 Bulk Samples – 1,000 Tonne Bulk Samples (SMC McAlpine Mill)

In 2011, the residual bulk sample material (after removing the 10-tonne subsamples) representing two approximately 1,000-tonne bulk samples were trucked from the project site to SMC's McAlpine mill for processing. Table 14 represents the resulting head grades returned for each of the bulk samples processed at the mill.

Table 14: 2011 Bulk Sample Gold Head Grades

Description	Bulk Sample #1 Composite (gpt)	Bulk Sample #2 Composite (gpt)
Gold Head Grade, 2011 Calculated	7.10	8.20
Gold Head Grade, 2012 Umpire Assays	5.87	7.28

All assays were performed under ALS Minerals' Val d'Or facility management which sends the samples to the North Vancouver laboratory of the same company where the assays were realized. All the head samples were prepared by screen metallic sieve on 2,000 gram samples with a 30 gram assay charge. All the tailings samples were prepared by screen metallic sieve on 200 gram samples with a 30 gram assay charge. All concentrate samples were assayed with a 30 gram assay charge. Gold assays were conducted by fire assay using a metallic sieve preparation in order to reduce assay variability.

Since the G&T facility is a better controlled environment than the SMC facility to recover all the gold present at different locations in the processing circuit, it was decided that the final bulk sample gold grades would be those obtained at G&T.

12.2.2 Grindability

Grindability testing based on the Bond work indexes was done first on the 2010 study core samples. Results are presented in Table 15.

Table 15: Grindability Results on Core Samples (2010 Study)

Sample	Bond Rod Mill Work Index (kWh/t)	Bond Ball Mill Work Index (kWh/t)
RL-01-01	14.4	11.6
RL-01-02	16.2	12.5
Average	**15.3**	**12.1**

Table 16: Grindability Results on Composite Samples (2011 Study)

Sample	Bond Rod Mill Work Index (kWh/t)	Bond Ball Mill Work Index (kWh/t)
Composite 1	17.7	13.1
Composite 2	15.3	10.3
Average	**16.5**	**11.7**

Additional grindability testing based on Bond work indexes was done on the 2011 study samples (composite samples). Results are presented in Table 16.

Bond grindability results from both sets of samples show similar behaviour, facilitating design of a grinding circuit that would be appropriate for all the tested samples.

To complete the grindability testing, DWT was performed on the 2011 study samples (composite samples) in order to allow the sizing of the SAG mill and the design of the grinding circuit using JKSimMet software (JKTech, March 2011) (SGS, June 2011). The results are presented in Table 17.

Table 17: DWT Results on Composite Samples (2011 Study)

Sample	A	b	A * b	ta
Composite 1	61.6	0.48	29.6	0.29
Composite 2	75.7	0.40	30.3	0.27

Highlights of the DWT testing on the tested samples are the following:

- Parameter “A*b” is utilized to characterize the competency of a material to SAG milling. The smaller parameter “A*b” is, the greater is the resistance to impact breakage (or as commonly said, the harder is the material). Figure 15 shows the frequency distribution of the “A*b” parameter based on the JKTech database (JKTech is the provider of JKSimMet software). The database shows that the value of 30 obtained from the tested samples is at the 12th percentile of all values. This means that only 12% of the materials tested by JKTech are more difficult to grind with SAG milling than the samples tested; and
- Parameter “ta” is utilized to characterize the abrasiveness of a material. The smaller parameter “ta” is, the more resistant the material is to abrasion. Figure 16 shows the frequency distribution of the “ta” parameter based on the JKTech database. The values obtained with the tested samples classified the mineralized material as hard, which is the second hardness class.

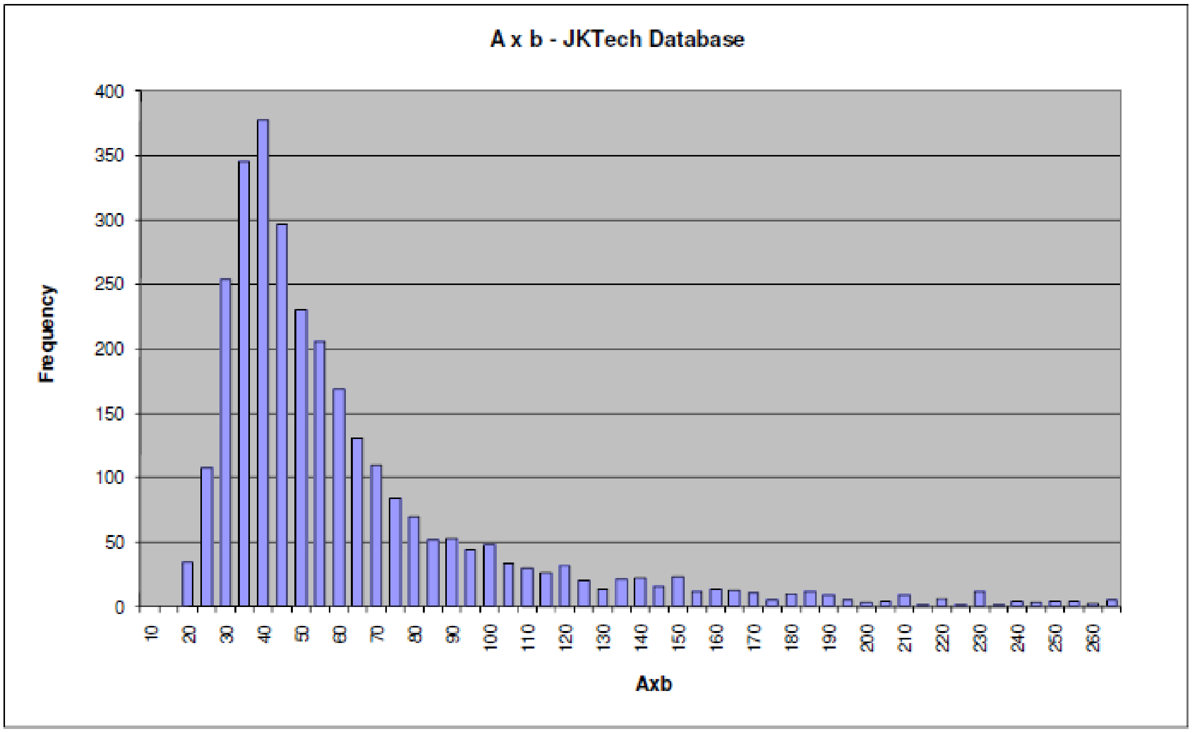


Figure 15: Frequency Distribution of A*b (JK Tech Database)

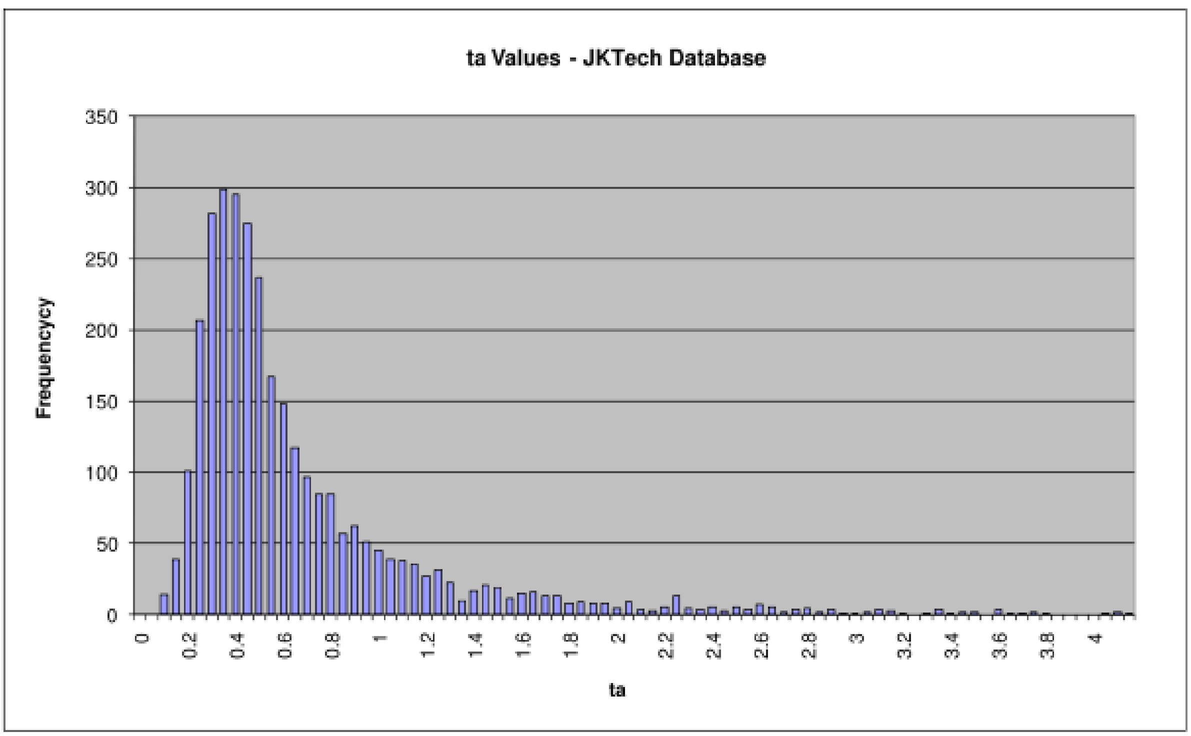


Figure 16: Frequency Distribution of ta (JK Tech Database)

12.2.3 Implications for SAG Mill Design

Based on the DWT parameters presented in Table 17, SAG mill sizing can be completed, including the main physical characteristics and operating conditions. This work was done by SGS using the JKSimMet software.

The purpose of the work was to determine a SAG mill sizing capable of handling 1,250 tpd, but also of being able to double capacity after a few years of operation to 2,500 tpd. Thus, simulations on different scenarios were carried out and a pebble crusher circuit was considered at 2,500 tpd.

The main guidelines for the simulations were the following:

- Optimize energy efficiency of the overall grinding circuit;
- Use only one SAG mill in the grinding circuit, even at the 2,500 tpd throughput;
- Evaluate the effect of adding a pebble crusher to achieve the 2,500 tpd throughput by crushing the circulating load (the coarser SAG mill product size fraction obtained by use of larger mill discharge grate openings and a mill discharge screen); and
- Evaluate the effect of mill dimensions, grinding media charge, and mill rotation.

Table 18 presents a summary of the results obtained from the JKSimMet simulations.

Results of the JKSimMet simulations allow the following conclusions:

- Option 2, consisting of one 6.1 m by 3.4 m (20 ft by 11.25 ft) flange to flange (F/F), 3.0 m (10 ft) effective grinding length (EGL) SAG mill supplied with a minimum 1,491 kW (2,000 HP) motor and one 3.4 m by 4.9 m (11 ft by 16 ft) F/F ball mill supplied with a minimum 447 kW (600 HP) motor is the preferred option;
- In order to avoid any shortage that may arise with harder mineralized material at 2,500 tpd, a 1,790 kW (2,400 HP) motor with variable speed drive will be installed on the SAG mill and a 597 kW (800 HP) motor will be installed on both ball mills; and
- For the 1,250 tpd throughput, the SAG mill is expected to be operated at a lower rotation speed with a reduced ball charge. This will result in reduced power consumption.

Table 18: JKSimMet Simulation Results

Option Number	Mill Size (Ins. Liner Dia. x EGL) (ft)	Ball Charge (% Vol)	Mill Speed (% Crit.)	Grate Size (mm)	Class. Slots Size (mm)	SAG Recycle (%)	Pebble Crusher (Y/N)	Total SAG Mill Power Requirement (HP)	Total Ball Mill Power Requirement (HP)
				1 250 tpd (57 t/h)					
Option 0	22 by 12	-	-	-	-	-	-	-	-
Option 1	24 by 9	5	75	15	3.4	-	No	2,312	-
Option 2	**20 by 10**	**10**	**71**	**12**	**-**	**0**	**No**	**1,669**	**453**
Option 3	20 by 10	10	71	12	-	0	No	1,669	453
Option 4	20 by 12	10	71	12	-	0	No	1,642	346
Option 5	-	-	-	-	-	-	-	-	-
Option 6	-	-	-	-	-	-	-	-	-
				2 500 tpd (113 t/h)					
Option 0	22 by 12	10	75	25	9.5	10	No	2,756	1,101
Option 1	24 by 9	10	75	25	9.5	10	No	2,643	1,190
Option 2	**20 by 10**	**12**	**77**	**63**	**9.5**	**20**	**Yes**	**1,971**	**1,221**
Option 3	20 by 10	12	77	63	6.4	24	Yes	1,976	1,150
Option 4	20 by 12	12	77	63	6.4	24	Yes	1,954	892
Option 5	20 by 12	12	77	25	6.4	18	No	2,315	1,181
Option 6	20 by 12	12	77	25	6.4	18	No	2,287	918

12.2.4 Gold Particle Size Analysis

After the processing of the two 10-tonne subsamples at G&T, the gravity concentrate obtained was sampled to carry out a gold size particle analysis (G&T, March 2012). The purpose of the analysis was to obtain reliable information about the dimension of the biggest gold particles present in the mineralization in order to refine the process design and determine the sampling protocols.

12.2.5 Gold Recovery

Gold recovery testing was done on the 2010 study core samples. For that testing, two of the five core samples were tested using four different flowsheet arrangements. The results on selected flowsheet (Flowsheet 2) are presented in Table 19. The tested flowsheet arrangements were:

- Flowsheet 1: gravity followed by rougher flotation;
- Flowsheet 2: gravity followed by cyanide leaching for 48 hours;
- Flowsheet 3: rougher flotation only; and
- Flowsheet 4: cyanide leaching only.

Highlights of the gold recovery testing on the tested samples are:

- The samples responded well to gravity;
- The samples responded reasonably well to flotation, but the results were generally lower than the cyanide leaching results;
- Cyanide leaching without gravity recovered, on average, 93% of the feed gold after 48 hours. Based on the experimental leaching curve, it is expected that 36 hours will be appropriate for complete gold dissolution;
- Gravity ahead of cyanide leaching did not appear to significantly improve overall gold recovery but it is recommended for security reasons; and
- The cyanide consumptions measured during the testing program were relatively low but were not optimized.

Because of the nature of the mineralized zones, a carbon-in-leach (CIL) process is preferred. This process is commonly used in the mining industry.

Table 19: Gold Recovery Results on Core Samples for Flowsheet 2 (2010 Study)

Sample	Gravity Recovery (%)	Leach Feed Au (gpt)	Leach Recovery (%)	Tailings Au (gpt)	Total Au Recovery (%)
RL-01-01	35.3	5.83	89.9	0.59	93.5
RL-01-02	24.1	4.70	89.9	0.48	92.3

12.3 Basis for Assumptions Regarding Recovery Estimates

12.3.1 Range of Gold Recovery

The gold recovery results obtained from only two core samples (RL-01-01 and RL-01-02) were used to evaluate the average gold recovery using gravity and cyanide leaching. Unfortunately, there was not enough material from the three other core samples to test the selected flowsheet (Flowsheet 2). However, the selected flowsheet was also tested on the two assay reject samples (RL 02-01 and RL-

02-02). Even though the size distribution of these samples was too fine to compare to what can be obtained with an industrial grinding circuit but as they were higher gold grade samples, they were of interest to estimate the effect of the gold grade on the recovery. After the testing, four test tailings samples (one from each core and assay reject sample) were chosen to perform a gold size by size analysis.

In order to compare all tested samples together on a grade-recovery basis, the testing results from the assay reject samples were corrected by matching the size distribution of the assay reject tailings to the core tailings size distribution. This allowed correcting the assay reject tailings gold grades and finally, the gold recoveries. This correction has lowered the recoveries of the assay reject samples by about 1%.

The average corrected recovery from the tests performed on the assay reject samples and the average recovery of the tests performed on the core samples are presented in Figure 17.

The figure shows that an increase in gold recovery is obtained when the gold grade increases. This is a relationship commonly observed in the gold industry. But as this relationship was obtained with only one size by size analysis for each sample, there is a significant uncertainty in the results.

Considering the level of accuracy required for a preliminary economic assessment, it is acceptable to apply this grade-recovery relationship. This relationship can be applied for head grades in the range of the obtained results while the minimum obtained recovery will be applied to material with lower grades and the maximum to the material with higher grades.

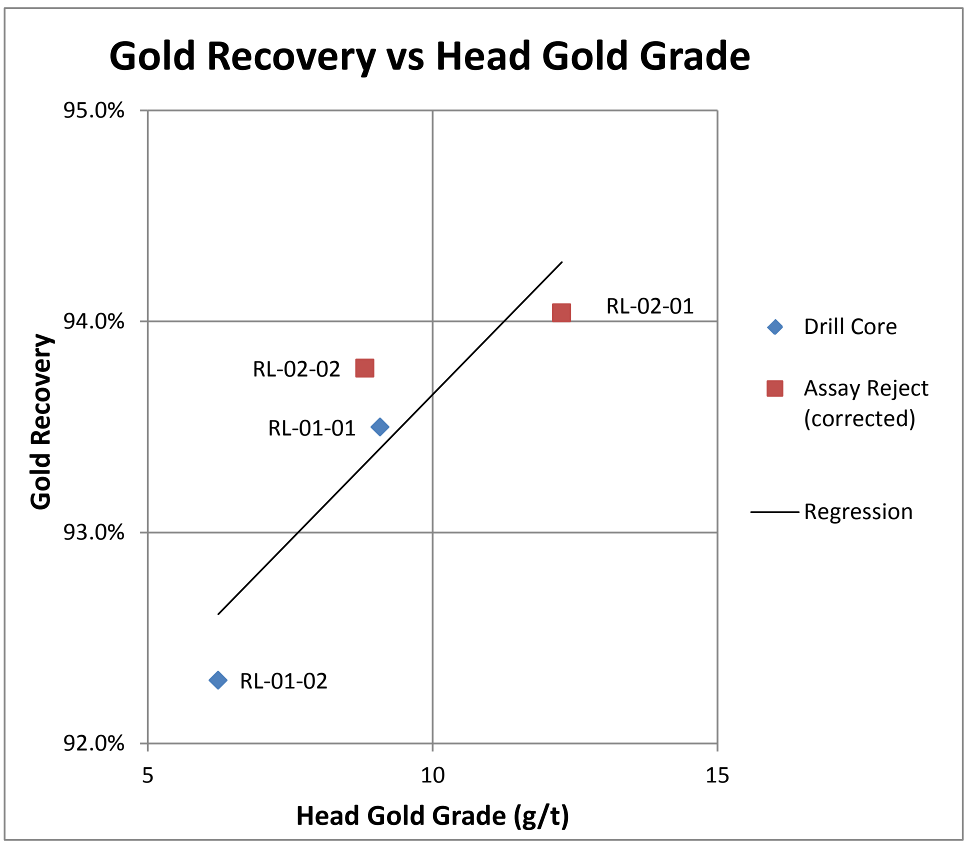


Figure 17: Effect of Head Gold Grade on Gold Recovery

Considering that an estimated soluble gold loss of 0.4% should be subtracted from the recovery values presented in the Figure 17, gold recoveries will be applied as follows:

- Material with a gold grade of less than 6 gpt will have a recovery of 92.0%;
- Material with a gold grade between 6 and 13 gpt will have a recovery that increase from 92.0% up to 94.0% in relation with the grade; and
- Material with a gold grade of more than 13 gpt will have a recovery of 94.0%.

In addition, for the economic sensitivity analysis usually performed with a range of possible gold recoveries, the estimated range is based on previous experience rather than results obtained during the metallurgical testwork program during which the number of tested core samples was limited. It is estimated that a realistic gold recovery will be in the range of -1.5 to +2.5% around the gold recovery average of 92.5% obtained during the metallurgical testwork program. The range is therefore 91% to 95% and covers the uncertainties related to the mineralogy of the mineralized material and scale-up methodology.

12.3.2 Improvement in Gold Recovery

It is anticipated that a better knowledge of the gold mineralization mineralogy together with continuous improvement efforts may increase gold recovery to greater than 92.5% over the years of operation.

Should gold recovery be less than initial expectations, then additional project work may be implemented to improve gold recovery. The scopes and potential returns on investment for such projects would be defined as required.

12.4 Sample Representativeness

12.4.1 2010 Study Samples – Core Samples

A protocol for selecting the cores needed for the 2010 study samples was designed by Soutex in collaboration with the Geology department of Rubicon.

Five composite drill core samples were used in the metallurgical testwork program. The first four samples (RL-01-01 to RL-01-04) were taken from predetermined locations according to the initially envisioned short- to medium-term mining scenario. The last sample (RL-01-05) was a blend of material (envisioned to be typical run of mine) taken from predetermined locations according also to the initially envisioned short- to medium-term mining scenario.

Core samples were prepared by complying with the following elements:

- The quantity of material should be obtained by selecting the number of material quarters from one core borehole distributed equally all along the core sections of interest representing the zone;
- The samples should not be crushed before the shipping;
- All samples should be put in separate, well-identified bags;
- Also, the Geology department of Rubicon had to prepare a sampling report presenting the following information:
  - Identification of all core sections (borehole number and depth) for material going to the metallurgical testwork program;

- Expected average gold grade;
- Method used to select sections from cores (when applicable); and
- Spatial distribution of all sampled boreholes.

12.4.2 2011 Study Samples – Subsamples (Composites)

Two bulk samples were collected to assess the head grades for two different underground zones on the 305 level and to complete the metallurgical testwork (G&T, October 2011). The guidelines for extracting and sampling the bulk samples were identified in order to ensure that the high variability of gold found in typical gold-bearing deposits is properly addressed.

A protocol for extracting and sampling the bulk samples was designed by Soutex in collaboration with Rubicon's geology department. From the bulk samples, several subsamples were generated and these were referenced as the "2011 study samples". It is important to note that the protocol was based on the Gy theory.

The highlights of the extraction and sampling protocol for each of the bulk samples are:

- Extraction from underground of 1,000-tonne of minus 230 mm material from a specific zone.
- Crushing all material in a jaw crusher to minus 75 mm;
- Hand sampling of one 170 l (45-gallon) drum;
- Crushing all material in a cone crusher to minus 12 mm;
- Tower sampling of one 10-tonne sample for confirming the head grade at the G&T pilot plant facility (see Section 16 for details);
- Tower sampling of two 1.0-tonne samples for future testwork; and
- Storage of the remaining approximately 988 tonnes for confirming the head grade at SMC Canada Ltd. McAlpine concentrator (custom milling facility) (see Section 16 for details).

The targeted head grade variability of each of the 10-tonne samples (expressed as the relative standard deviation of 1 σ) was about 7%, meaning that there is 95% probability that the head grade will be within a range of +/-14% (or +/- 2 σ) around the head grade of the material extracted from a specific zone. After the data analysis of all gold assays of the samples collected during the processing of the two 10-tonne samples at G&T, the head grade variability was evaluated to 1.3%. This means that in the future, the size of the bulk sample collected for each mining zone of interest can be reduced significantly.

12.5 Factors with Possible Effect on Potential Economic Extraction

12.5.1 2010 Study Samples

The following factors were identified during the treatment of the 2010 study samples.

Tested Samples

The metallurgical testwork was done on core samples originating from two representative zones and the current average metallurgical performances (especially grindability and the gold recovery), however, the samples selected might not show the variations that can be encountered throughout the short- to medium-term of the mine life.

To decrease the uncertainties and have a better estimation of the range of possible gold recovery to allow meeting the production target, a larger number of core samples statistically representing the different zones to be exploited should be collected.

As for the current mitigation plan, the concentrator design has been adjusted. Concerning grindability, more power has been selected for the SAG mill and the ball mills. Concerning the gold recovery, the risk of not recovering the dissolved gold is low because the selected carbon adsorption process was designed to recover more gold than the tested sample grades.

Main Process Equipment

At the grinding circuit for the operation at 1,250 tpd, it is expected that the SAG mill would be operated at a lower speed with a reduced ball charge. For the envisaged future expansion at 2,500 tpd, the ball charge and mill speed would be increased and a pebble crusher might be needed to crush the SAG mill recirculating load in order to reach the production target. One additional ball mill is expected at 2,500 tpd as well as a second hydrocyclone cluster and a gravity concentrator. Provision has also been made for a pre-crushing unit to support the grinding circuit if required and a second stripping column.

At the paste plant, for operation at 1,250 tpd two disc filters should meet the production target for the filter cake at 80% solids. For the envisaged future expansion at 2,500 tpd, a third disc filter is planned and provision has been made for a thickener, if required.

Plant Tailings Toxicity

The characteristics of the tested samples suggest the use of a conventional carbon-in-leach (CIL) process. Once the plant tailings are properly treated, there are no particular environmental issues that can be expected.

However, some sulphides were identified in the feed samples and this needs some attention to ensure proper treatment during the design of the TMF. Also, there is no significant arsenic or other deleterious elements present in the tested samples.

TMF Effluent

The cyanide concentration was not optimized during the CIL testwork. The cyanide is destroyed with the SO_2-air process while producing cyanate ions that will be degraded, thus producing ammonia. As ammonia is a regulated discharge parameter, it is necessary to keep it within the allowable limit. Testwork is currently being carried out for the cyanide destruction to ensure proper tailings treatment especially in the case of high cyanide consumption but further testwork is needed aimed at reducing the cyanide concentration.

12.5.2 2011 Study Samples

One factor was identified during the treatment of the 2011 study samples.

Gold Assaying

A series of gold grade assays was made in duplicate at G&T on about 10 samples pulverised at 95% minus 100 µm. Each sample weight was 250 grams and two fire assays were initially performed on 30-gram aliquots. Table 20 shows the results for both bulk samples. The results indicate that the respective difference relative standard deviations are at 45.9% and 33.4%.

These relative standard deviation values are higher than 16%, thus indicating, based on the Gy sampling theory, that the variability in the results follows a Poisson distribution. Such behaviour indicates that the 30-gram weight used for the analysis is too low to obtain the required precision. Thus, in the future, it will be necessary to perform a metallic sieve preparation adapted to the tested material to lower the gold assay variability.

Table 20: Gold Grades for Bulk Samples #1 and #2 (30-gram Aliquots)

Sample	Bulk Sample #1			Bulk Sample #2		
	Assay #1 (gpt)	Assay #2 (gpt)	Difference (gpt)	Assay #1 (gpt)	Assay #2 (gpt)	Difference (gpt)
Head 1	10.00	4.95	5.05	8.10	5.47	2.63
Head 2	5.88	5.34	0.54	8.09	7.17	0.92
Head 3	5.49	5.82	-0.33	6.45	7.40	-0.95
Head 4	7.49	20.30	-12.81	5.92	6.33	-0.41
Head 5	5.64	12.40	-6.76	10.10	16.20	-6.10
Head 6	9.13	7.38	1.75	8.47	7.12	1.35
Head 7	7.93	5.64	2.29	23.30	12.60	10.70
Head 8	5.14	6.69	-1.55	6.96	6.42	0.54
Head 9	8.85	8.43	0.42	10.40	7.82	2.58
Head 10	4.94	5.78	-0.84	-	-	-
Head σ_{REL} (%)	**26.38**	**57.52**	**-**	**54.32**	**41.51**	**-**
Difference σ_{REL} (%)	**-**	**-**	**45.90**	**-**	**-**	**33.42**

13 Mineral Resource Estimates

13.1 Introduction

This section describes the methodology and summarizes the key assumptions considered by SRK to prepare a Mineral Resource Statement for the Phoenix gold project. In the opinion of SRK, the resource evaluation reported herein is a reasonable representation of the global gold mineral resources found in the F2 gold system at the current level of sampling. The mineral resources have been estimated in conformity with generally accepted CIM *Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines* and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101.

The database used to estimate the Phoenix gold project mineral resources was audited by SRK. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries of the gold mineralization and that the assay data are sufficiently reliable to support mineral resource estimation.

This mineral resource estimate undertaken by SRK includes new drilling information acquired since February 28, 2011, the cut-off date for the previous resource estimate by AMC (see Section 5.2). The mineral resources reported herein consider drilling information available to October 31, 2012 and was evaluated using a geostatistical block modelling approach constrained by 56 gold mineralization wireframes. These 56 domains were subsequently combined by SRK into three groups based on their spatial orientation: Main, Main 45, and Hanging Wall (HW).

The construction of the mineral resource model was a collaborative effort between Rubicon and SRK personnel. The construction of the three-dimensional gold mineralization domains was completed under the direction of Dr. Jean-François Ravenelle, PGeo, whereas most of the resource evaluation work was completed by Sébastien Bernier, PGeo (OGQ#1034, APGO#1847), with the assistance of Chris MacInnis, PGeo (APGO# 2059). The assignment benefited from the senior review of Glen Cole, PGeo (APGO#1416). The full SRK team that contributed to various portions of the gold mineralization domain modelling and mineral resource model is presented in Table 21.

Table 21: SRK Mineral Resource Modelling Team

Professional	Position	Site Visit	Responsibility
Mineralization Domain Modelling			
Dr. Jean-François Ravenelle, PGeo	Senior Consultant	X	3D domain modelling
Dr. Iris Lenauer	Consultant		3D domain modelling
Dr. Julia Kramer-Bernhard	Senior Consultant		3D domain modelling
Dominic Chartier, PGeo	Senior Consultant	X	3D domain modelling
Dr. James Siddorn, PGeo	Principal Consultant	X	Domain model review
Mineral Resource Estimation			
Sébastien Bernier, PGeo	Principal Consultant		Geostatistics and resource estimation
Chris MacInnis, PGeo	Consultant		Resource estimation
Dr. Oy Leuangthong, PEng	Principal Consultant		Geostatistical analysis review
Senior Review			
Glen Cole, PGeo	Principal Consultant		Domain and estimation review

13.2 Resource Estimation Procedures

The mineral resources reported herein have been estimated using a geostatistical block modelling approach informed from core borehole data. Resource domains were defined using a traditional wireframe interpretation constructed from a sectional interpretation of the drilling data that took into consideration underground mapping and previous structural, lithological, and alteration modelling undertaken by SRK.

The evaluation of the mineral resources involved the following procedures:

- Database compilation and verification;
- Generation of three-dimensional wireframe models and verification;
- Data extraction and processing (compositing and capping), statistical analysis, and variography;
- Selection of estimation strategy and estimation parameters;
- Block modelling and grade estimation;
- Validation, classification, and tabulation;
- Assessment of "reasonable prospects for economic extraction" and selection of reporting cut-off grades; and
- Preparation of the Mineral Resource Statement.

13.3 Resource Database

The Phoenix exploration database up to October 31, 2012 comprises of 820 core boreholes (355,611 metres [m]), all drilled by Rubicon since 2008. This database comprises 618 core boreholes drilled from underground platforms and 202 core boreholes drilled from the surface. In the resource area, a total of 31 historical core boreholes were drilled by Rubicon before 2007 (7,400 m). The 31 historical boreholes were not used for the mineral resource estimation due to uncertainties associated with the analytical quality control at the time. They were used to guide the geological modelling. SRK received the borehole sampling data in a Microsoft Access file and subsequently converted the data into a series of CSV files for import into CAE Studio 3. SRK performed the following validation steps:

- Checked minimum and maximum values for each quality value field and confirmed/edited those outside of expected ranges; and
- Checked for gaps, overlaps, and out of sequence intervals in the assays tables.

A significant proportion of the core intervals were not sampled (Table 22). Unsampled intervals are considered unlikely to contain significant gold mineralization based on the rock type and alteration as supported by check samples through these intervals. For the mineral resource estimation process, SRK assigned a value of 0.00 grams per tonne (gpt) gold to all unsampled intervals prior to compositing the data.

Dr. Ravenelle and Mr. Dominic Chartier visited the Phoenix gold project between October 1 and 7, 2011 as well as between December 3 and 8, 2011 to examine exposures in the existing underground workings and core. SRK is satisfied that the exploration work carried out by Rubicon has been conducted in a manner consistent with generally recognized industry best practices and, therefore, the exploration drilling data are sufficiently reliable for the purpose of supporting a mineral resource evaluation.

Table 22: Sampling Statistics – Original Data

Domain	Element	Sample Count	Sampled Intervals	Unsampled Intervals	Percentage Unsampled
Main	Au (gpt)	74,309	18,976	55,359	26%
Main 45	Au (gpt)	20,586	5,216	15,370	25%
HW	Au (gpt)	11,070	5,716	5,354	52%
External	Au (gpt)	210,470	169,509	40,961	81%

13.4 Solid Body Modelling

Gold mineralization in the F2 gold system is characterized by vein and sulphide replacement styles which are preferentially hosted within and along the boundaries of two main rock types: high-TiO_2 (HiTi) basalts (high iron tholeiites) and felsic intrusions (bounding units); with additional mineralization associated with crosscutting structures. Gold, however, is distributed through all of the adjacent rock types, with the majority contained within the HiTi basalt. Several structural events affected the gold mineralization:

- The earlier stage of gold mineralization is overprinted by the northeast-trending S1 foliation while the later stage of gold mineralization cuts the S1 foliation; and
- Two generations of structures postdate the S1 foliation:
  - A northwest-trending, steeply dipping S2 foliation documented in the northern part of the underground workings; and
  - A northeast-trending, steeply dipping S3 foliation documented in the western part of the underground workings.

Gold mineralization wireframes were developed by SRK in collaboration with Rubicon based on sectional interpretations of geology provided by Rubicon, a three-dimensional lithological model prepared by SRK, and wireframes provided by Rubicon representing a 0.5 gpt gold threshold used to define domains hosting gold mineralization that were subsequently modified by SRK (Figure 11).

Within the gold mineralization domains, narrower, high-grade domains characterized by continuous grades above 3.0 gpt gold were also created by SRK (Figure 12). The high-grade domains essentially represent high-TiO_2 basalt units, whereas the lower grade domains generally occur in more felsic rock, surrounding the high-TiO_2 basalt units.

SRK defined 56 gold mineralization domains (31 high-grade and 25 lower grade domains) that were used to constrain mineral resource modelling. These domains represent a refinement of the gold mineralized geometries and an overall improvement of continuity of the modelled gold mineralization, which constrains the gold interpolation in the mineral resource model. These refined wireframe geometries closely reflect the current interpretation of the underlying geology framework. These 56 domains were subsequently combined into three groups based on their spatial orientation: Main, Main 45, and Hanging Wall (HW). The Main 45 domain is a domain proximal to the Main domain and is characterized by a unique structural orientation.

Locally, significant gold mineralization exists outside the modelled domains. To account for this mineralization, the areas located outside the 56 modelled domains were also evaluated as a separate “unconstrained” domain (External domain).

13.5 Compositing and Outlier Analyses

Borehole gold assay data were extracted and examined for determining an appropriate composite length (Figure 18). A modal composite length of 1.0 metre was applied to all data. The impact of gold outliers was examined on composited data using log probability plots and cumulative statistics. Basic statistics for gold assays, composites, and capped composites are summarized in Table 23. Data located external to the modelled gold mineralization wireframes were grouped into an unconstrained domain termed External. Examples of the basic statistics, histograms, and cumulative probability plots examined for the Main domain are provided in Figure 19. The plots for all other domains are available in Appendix B.

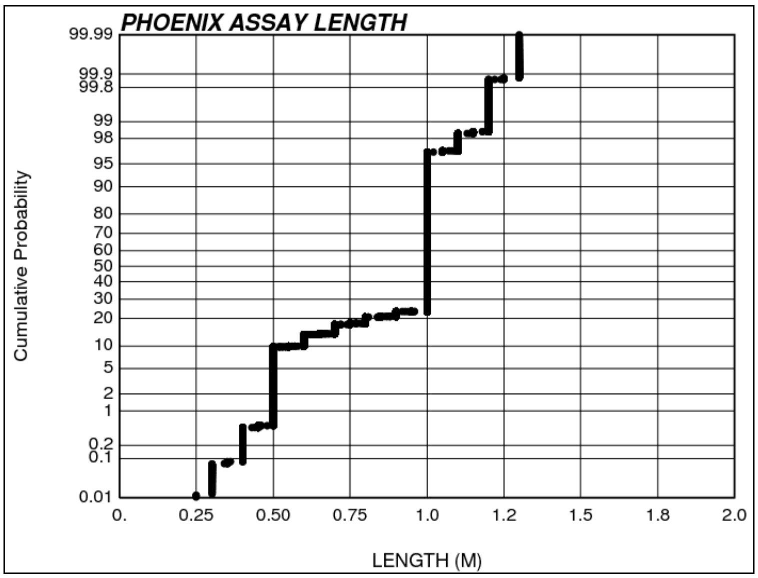


Figure 18: Distribution of the Sample Length Intervals

Table 23: Basic Statistics – Original, Composite and Capped Composite Data

Domain	Element	Sample Count	Minimum	Maximum	Mean	Standard Deviation	Coefficient of Variation	Capped Count
				Original Assay Data				
Main	Au (gpt)	74,309	0.00	2,620.70	1.04	16.10	15.54	
Main 45	Au (gpt)	20,586	0.00	1,219.64	0.55	6.53	11.82	
HW	Au (gpt)	11,070	0.00	3,151.10	0.38	22.21	58.96	
External	Au (gpt)	210,470	0.00	278.41	0.04	0.87	23.85	
				Composites				
Main	Au (gpt)	68,974	0.00	1,406.84	1.05	12.52	11.87	
Main 45	Au (gpt)	19,453	0.00	611.95	0.55	4.74	8.56	
HW	Au (gpt)	10,300	0.00	1,591.56	0.38	15.92	41.98	
External	Au (gpt)	205,896	0.00	275.46	0.04	0.78	21.49	
				Capped Composites				
Main	Au (gpt)	68,974	0.00	200.00	0.95	6.00	6.33	24
Main 45	Au (gpt)	19,453	0.00	200.00	0.53	2.30	4.32	1
HW	Au (gpt)	10,300	0.00	30.00	0.19	1.14	5.93	6
External	Au (gpt)	205,896	0.00	150.00	0.04	0.59	16.53	1

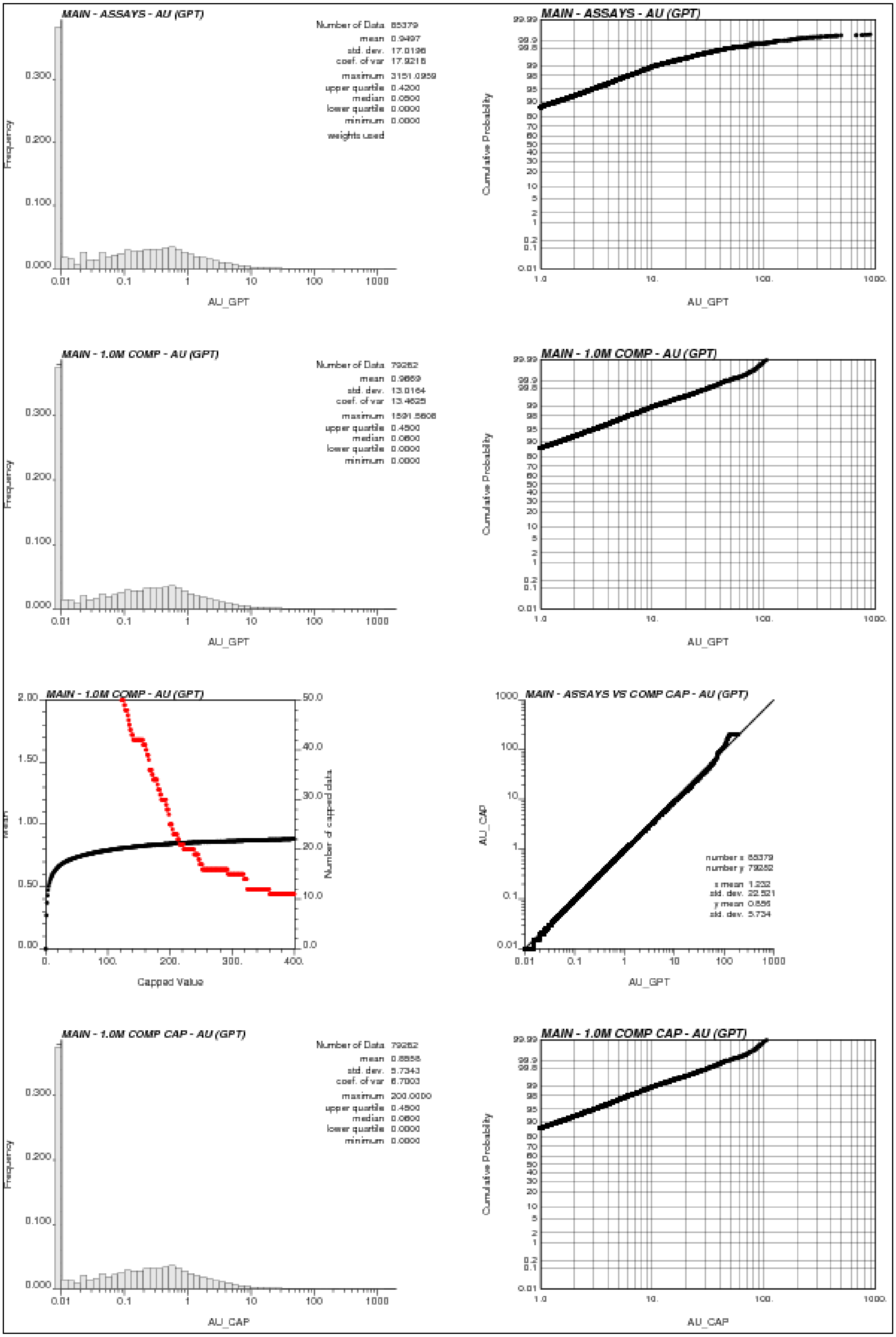


Figure 19: Basic Statistics of the Gold Assay Data for the Main Domain

13.6 Variography

SRK evaluated the spatial distributions of the gold mineralization using variograms and correlograms of original capped composited data as well as the normal score transform of the capped composited data. A total of eight spatial metrics were considered to infer the correlation structure of each element for use in the grade estimation.

Continuity directions were assessed based on the orientation of the resource domains, composites, and their spatial distribution. Further, variogram calculation considered sensitivities on orientation angles prior to finalizing the correlation orientation. All variogram analysis and modelling was performed using CAE Studio 3 and the Geostatistical Software Library. Variogram modelling is based on the combination of the four metrics of the uncapped composites, and in all cases, the traditional variogram using normal score transform data yields reasonably clear continuity long-range structures that are amenable to variogram fitting.

For the Main domain, the high-grade subdomains were used to model the variogram. The significant amount of data at or below the detection limit (approximately 40 percent [%]) in the lower grade subdomains introduced an artificially highly variable data set in normal score and prevented the modelling of a reliable variogram. The high-grade domain variogram was applied to the lower grade domains. The same approach was used in the Main 45 domain. The HW domain variogram was modelled using all the data present in this domain. No variogram modelling was attempted for the External domain due to the large data spacing and the amount of data at or below the detection limit. The variogram model of the Main domain was applied to the External domain.

The modelled variograms used for the estimation of gold for each domain are presented in Appendix C. Figure 20 shows an example of the traditional variogram using normal score transformed data calculated and modelled for gold in the Main domain.

Table 24 summarizes the modelled gold variograms. Gold values were estimated using ordinary kriging informed by capped composite data. Three estimation passes were used to inform model blocks, using increasing search neighbourhoods sized from variography results. The search parameters used for the estimation are summarized in Table 25 for the Main, 45 Trend, and HW domains; and in Table 26 for the External Domain. Each domain was estimated separately using composites from that domain only.

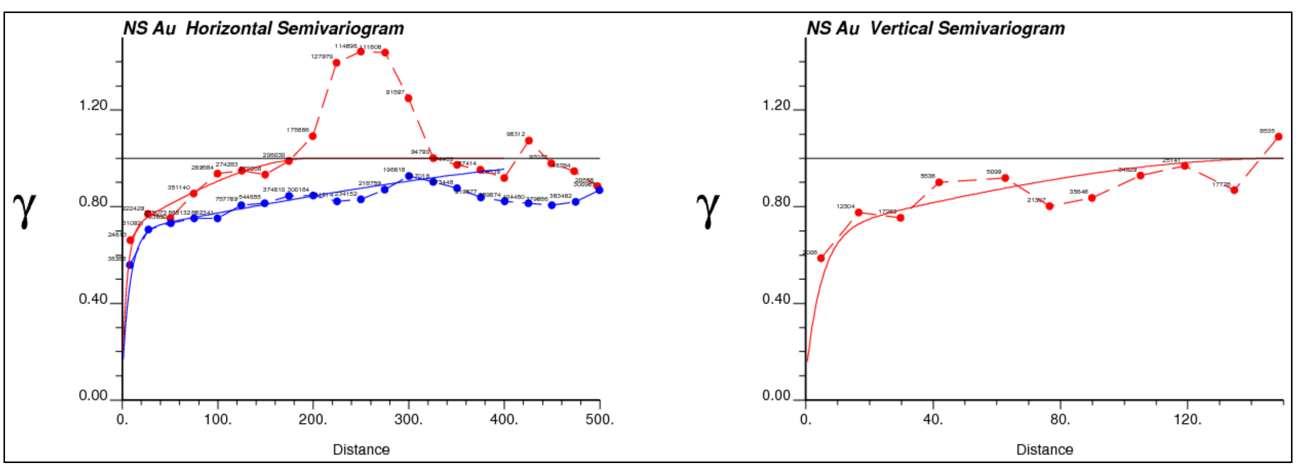


Figure 20: Traditional Variogram Using Normal Score Transforms Data for the Main Domain that Forms the Basis for Variogram Fitting

Table 24: Gold Variogram Parameters for the Phoenix Gold Project

Domain	Structure	Contribution	Model	R1x (m)	R1y (m)	R1z (m)	Angle* 1	Angle* 2	Angle* 3	Axis 1	Axis 2	Axis 3
Main	C0	0.10	Nugget	-	-	-	165	65	95	3	1	2
	C1	0.60	Exp	25	15	15	165	65	95	3	1	2
	C2	0.30	Sph	600	200	150	165	65	95	3	1	2
Main 45	C0	0.10	Nugget	-	-	-	210	75	90	3	1	2
	C1	0.55	Exp	75	10	80	210	75	90	3	1	2
	C2	0.35	Sph	80	150	85	210	75	90	3	1	2
HW	C0	0.10	Nugget	-	-	-	260	30	0	3	1	2
	C1	0.70	Exp	35	10	35	260	30	0	3	1	2
	C2	0.20	Sph	150	120	60	260	30	0	3	1	2
External	C0	0.10	Nugget	-	-	-	165	65	95	3	1	2
	C1	0.60	Exp	15	25	15	165	65	95	3	1	2
	C2	0.30	Sph	200	600	150	165	65	95	3	1	2

* The rotation angles are shown in CAE Studio 3 convention.

Table 25: Summary of Gold Estimation Search Parameters for the Main, 45 Trend, and HW Domains

Parameter	1st Pass	2nd Pass	3rd Pass
Element estimated	Au	Au	Au
Interpolation method	Ordinary kriging	Ordinary kriging	Ordinary kriging
Search range X	80% Var range	90% Var range	90% Var range
Search range Y	80% Var range	90% Var range	90% Var range
Search range Z	80% Var range	90% Var range	90% Var range
Minimum number of composites	8	8	6
Maximum number of composites	16	24	24
Octant search	Yes	Yes	No
Minimum number of octant	3	3	-
Minimum number of composites per octant	3	3	-
Maximum number of composites per octant	12	12	-
Maximum number of composites per borehole	5	5	5

Table 26: Summary of Estimation Search Parameters for Gold in the External Domain

Parameter	1st Pass
Element estimated	Au
Interpolation method	Ordinary kriging
Search range X	95 m
Search range Y	290 m
Search range Z	75 m
Minimum number of composites	6
Maximum number of composites	24
Octant search	No
Minimum number of octant	-
Minimum number of composites per octant	-
Maximum number of composites per octant	-
Maximum number of composites per borehole	5

13.7 Block Model and Grade Estimation

The criteria used in the selection of a block size included the borehole spacing, composite assay length, the geometry of the modelled zones, and the anticipated mining technique. In collaboration with Rubicon, SRK chose a block size of 2.5 by 5 by 10 m for the Main and Main 45 domains. The HW domain cell dimension was increased to 10 by 20 by 20 m to account for less dense borehole spacing and the locally disseminated nature of the gold mineralization in this domain. The External domain was estimated using 5 by 10 by 20 m cells and represents an area for future exploration drilling.

Subcells were used (20 splits in all directions for all models), allowing a variable resolution in all directions to honour the geometry of the modelled mineralization. Subcells were assigned the same grade as the parent cell. The four block models were rotated to honour the orientation of the geological wireframe and direction of the mineralization trend. The block model coordinates are based on a local mine grid developed by Rubicon for Phoenix. The characteristics of the block model are summarized in Table 27.

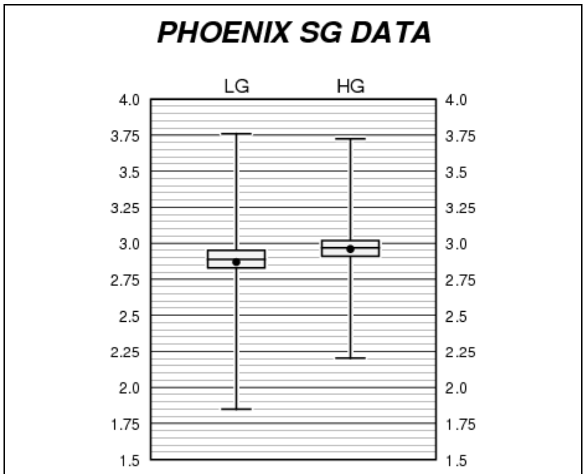
Table 27: Phoenix Gold Project Block Models Specification

Domain	Axis	Block Size (m)		Origin*	Number of Cells	Rotation Angles	Rotation Axis
		Parent	Sub cell				
Main	X	5.0	0.125	9,000	460	165	3
	Y	10.0	0.250	50,700	250	65	1
	Z	2.5	0.062	5,100	500	95	2
45 Trend	X	10.0	0.250	10,000	196	210	3
	Y	5.0	0.125	49,800	132	165	1
	Z	2.5	0.125	5,400	280	90	2
HW	X	20.0	0.500	9,984	92	260	3
	Y	10.0	0.250	48,698	37	-60	1
	Z	20.0	0.500	4,872	39	0	-
External	X	10.0	1.000	8,905	286	165	3
	Y	20.0	1.000	51,361	151	65	1
	Z	5.0	1.000	5,075	336	95	2

* Expressed as mine grid coordinates converted from the local UTM grid (Nad83 datum).

13.8 Specific Gravity

Rubicon measured specific gravity on small representative samples of selected assay intervals using a water displacement technique. A total of 6,562 specific gravity measurements are located within the geological domains modelled by SRK. Considering the different host rocks of the high-grade material (generally titanium-rich basalt) compared to the more felsic lower grade domain, the specific gravity values were divided between these two grade domains (Figure 21). After review of distributions, a uniform specific gravity of 2.87 was applied to the lower grade domains and a value of 2.96 was assigned to the high-grade domains to covert volumes into tonnages.



	LG	HG	
Number of data	5223	1339	Number of data
Mean	2.8701	2.9604	Mean
Std. Dev.	0.1206	0.0978	Std. Dev.
Coef. of Var.	0.042	0.033	Coef. of Var.
Maximum	3.7612	3.7246	Maximum
Upper quartile	2.9518	3.0184	Upper quartile
Median	2.8882	2.968	Median
Lower quartile	2.8307	2.9103	Lower quartile
Minimum	1.849	2.2033	Minimum

Figure 21: Summary of the Specific Gravity Database

13.9 Model Validation and Sensitivity

The block model estimated gold grades were validated through:

- Visual comparison of original borehole data with mineral resource block data on plans, sections, and down the plunge of the mineralization (illustrative sections through the Main Domain block model is presented in Appendix D);
- Validation of the estimation parameters and associated parameter sensitivity by varying the parameters and comparing the outcome statistically and visually against the informing borehole data and against the original block model for the Main domain;
- Comparison of the basic statistics of ordinary kriging estimates for gold with de-clustered mean informing capped composite data and with the original source data for the Main domain. The graphical outputs of this comparison is provided in Figure 22; and
- Change-of-support comparison of the informing capped composites against the block model estimated grade for the Main domain.

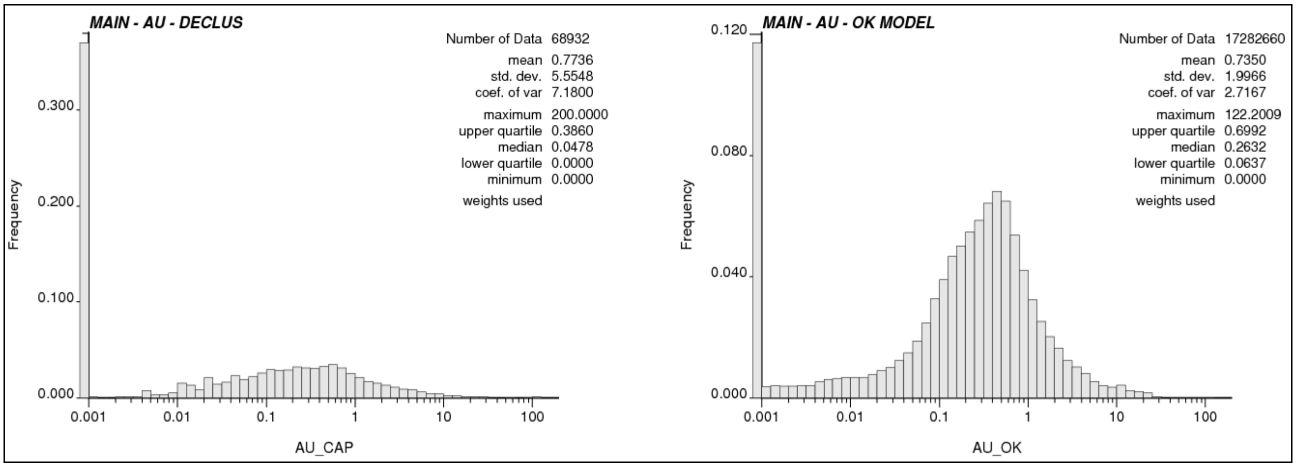


Figure 22: Block Model Validation of the Gold Estimated Value for the Main Domain

13.10 Mineral Resource Classification

Block model quantities and grade estimates for Phoenix were classified according to the CIM *Definition Standards for Mineral Resources and Mineral Reserves* (November 2010) by Sébastien Bernier, PGeo (APGO#1847) and Chris MacInnis, PGeo (APGO#2059).

Mineral resource classification is typically a subjective concept, and industry best practices suggest that resource classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification.

SRK is satisfied that the geological and gold mineralization model for Phoenix honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation and do not present a risk that should be taken into consideration for resource classification. The mineral resource model is informed from core boreholes drilled with pierce points generally spaced approximately 25 to 50 m apart in the core of the deposit. The geological information is sufficiently dense to demonstrate the continuity of the stockwork gold mineralization associated with a northeast-trending sequence of interbedded ultramafic, high-TiO_2 basaltic rocks and felsic sills and dikes. The confidence in the geological model is good.

SRK considers that it is reasonable to classify blocks of the Main zone satisfying the following criteria in the Indicated category within the meaning of the CIM *Definition Standards for Mineral Resources and Mineral Reserves*. SRK considers that for those blocks of the Main zone the level of confidence is sufficient to allow appropriate application of technical and economic parameters to support mine planning and to allow the evaluation of the economic viability of the deposit. Criteria applied to define Indicated resources include:

- Blocks located within a resource domain, informed by data that is supported by analytical quality control measures;
- Blocks estimated during the first or second pass within the 90% of the variogram range;
- Blocks estimates informed by four or more boreholes; and

- Blocks located within areas exhibiting good grade and geological continuity at a cut-off grade of 4.0 gpt gold.

To assist with classification, a wireframe was constructed manually within CAE Studio 3 to delineate the areas encompassing all of the above parameters. The wireframe was defined by moving through the deposit in 25-metre sections (dropping to 12.5 m where required) in plan view from surface to the base of the deposit. All blocks within that classification wireframe were classified as Indicated.

Conversely, all other modelled blocks were classified in the Inferred category as the confidence in the estimates is insufficient to allow for the meaningful application of technical and economic parameters or to enable an evaluation of economic viability.

13.11 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves define a mineral resource as:

"[A] concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial 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 that takes into account extraction scenarios and processing recoveries.

SRK considers that the gold mineralization at the Phoenix gold project is amenable to underground extraction. To select an appropriate reporting cut-off grade, SRK considered the assumptions listed in Table 28. SRK considers that it is appropriate to report the Phoenix mineral resources at a cut-off grade of 4 gpt gold.

Table 28: Assumptions Considered for Reporting Cut-Off Grade Determination

Parameter	Value
Production rate (tonnes per day)	1,800
Mining cost (C$/tonne)	$89.60
General and administration (C$/tonne)	$32.22
Process cost (C$/tonne milled)	$28.12
Gold recovery (%)	92.50%
Mining recovery/Mining dilution (%)	95/15
Gold price (US$/ounce)	$1,500
Revenue factor	1

The mineral resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent resource estimates. The mineral resources may also be affected by subsequent assessments of mining, environmental, processing, permitting, taxation, socio-economic, and other factors.

The Mineral Resource Statement for the Phoenix gold project is presented in Table 29.

Table 29: Mineral Resource Statement*, Phoenix Gold Project, Ontario, SRK Consulting (Canada) Inc., June 24, 2013

Domain	Resource Category	Quantity (000't)	Grade Au (gpt)	Contained Gold (000'oz)
Main#	Measured	-	-	-
	Indicated	4,120	8.52	1,129
	Measured + Indicated	4,120	8.52	1,129
	Inferred	6,027	9.49	1,839
HW	Measured	-	-	-
	Indicated	-	-	-
	Measured + Indicated	-	-	-
	Inferred	151	5.21	25
External	Measured	-	-	-
	Indicated	-	-	-
	Measured + Indicated	-	-	-
	Inferred	1,274	8.66	355
Combined	Measured	-	-	-
	Indicated	4,120	8.52	1,129
	Measured + Indicated	4,120	8.52	1,129
	Inferred	7,452	9.26	2,219

* Mineral resources are not mineral reserves and do not have a demonstrated economic viability. All figures have been rounded to reflect the relative accuracy of the estimates. Reported at a cut-off grade of 4.0 gpt gold and assuming an underground extraction scenario, a gold price of US$1,500 per ounce, and metallurgical recovery of 92.5%.

The Main domain includes the Main 45 domain.

SRK has worked with Rubicon to design a conceptual infill drilling program to upgrade the classification of the mineral resources in the Main domain above the 610 level from conceptual exploration drifts on the 244 and 425 levels.

In addition to upgrading a significant portion of Inferred mineral resources to an Indicated classification, the proposed drilling program has the potential to expand the mineral resources within these shallower levels that are planned to be extracted at an early stage of the proposed conceptual mining schedule.

SRK has applied specific gold grade interpolation criteria deemed to be appropriate for the style of gold mineralization. These criteria are potentially conservative incorporating criteria such as the requirement of a minimum of three informing boreholes, and a minimum/maximum of 8/24 samples for the first and second interpolation pass and 6/24 for the third interpolation pass.

SRK investigated the variance in potential gold mineralization thickness within the Main domain by interrogating the mineral resource model at a cut-off grade of 4 gpt gold for each mining level (61 m vertical). Understanding that the confidence in estimated gold mineralization thickness deteriorates with depth (lower drilling density), SRK estimate the range of gold mineralization thickness to generally be between 2.0 and 16.0 m with a weighted average of 7.8 m at a cut-off grade of 4 gpt gold. The width of the deposit is variable and exceeds 30 m in some areas.

There is an opportunity to expand the currently reported mineral resource in areas adjacent to mineral resource blocks where although boreholes are present with elevated gold grades, borehole density is insufficient to satisfy the applied mineral resource criteria. Targeting these areas for follow-up drilling has a high probability to increase the mineral resource.

13.12 Reconciliations and Comparisons

13.12.1 Comparison between the AMC 2011 and SRK 2013 Mineral Resource Statements

The Mineral Resource Statement documented herein represents a significant change relative to the previous Mineral Resource Statement released in June 2011 (Table 30). It is significant to note the material increase in tonnages reported in 2013 at reduced gold grades compared to the tonnages reported in 2011. Reported tonnages and grades are the product of contrasting mineral resource estimation methodologies applied during the two studies. The additional infill drilling since 2011 contributed to the significant increase in the reported Indicated mineral resources in 2013.

Table 30: Mineral Resource Statement*, Phoenix Gold Project, Ontario, AMC Mining Consultants (Canada) Ltd., June 15, 2011

Resource Category	Quantity (000't)	Grade Au (gpt)	Contained Gold (000'oz)
Measured	-	-	-
Indicated	1,028	14.50	477
Measured + Indicated	**1,028**	**14.50**	**477**
Inferred	4,230	17.00	2,317

The June 2011 Mineral Resource Statement (Table 30) was reported using a 5 gpt gold cut-off grade, whereas the Mineral Resource Statement documented herein is reported to a 4 gpt gold cut-off grade. For a more equitable comparison to the June 2011 Mineral Resource Statement, quantities and grades reported to 4 gpt gold are tabulated in Table 31, whereas comparative quantities and grades reported to 5 gpt gold are tabulated in Table 32.

Table 31: Comparison between 2011 and 2013 Quantities and Grades Reported at 4 gpt Gold

Classification	Quantity (000' tonnes)			Grade Au (gpt)			Contained Gold (000' ounces)		
	2011	2013	Change	2011	2013	Change	2011	2013	Change
Indicated	1,430	4,120	188%	11.63	8.52	-27%	535	1,129	111%
Inferred	5,674	7,452	31%	13.83	9.26	-33%	2,523	2,219	-12%

Table 32: Comparison between 2011 and 2013 Quantities and Grades Reported at 5 gpt Gold

Classification	Quantity (000' tonnes)			Grade Au (gpt)			Contained Gold (000' ounces)		
	2011	2013	Change	2011	2013	Change	2011	2013	Change
Indicated	1,028	3,116	203%	14.50	9.82	-32%	477	984	106%
Inferred	4,230	5,604	32%	17.00	10.84	-36%	2,317	1,954	-16%

13.12.2 Reconciliation of Bulk Sample

During 2011, Rubicon collected two approximately 1,000-tonne gold mineralized bulk samples from underground stoping on 305 level. A protocol for extracting and sampling the bulk samples was designed by Soutex in collaboration with the Geology department of Rubicon. The bulk samples were coarse crushed to approximately 90 % -½ inch and a 10-tonne subsample split from each. The 10-tonne subsamples were shipped to G&T in Kamloops, British Columbia for metallurgical test work. G&T reported the average gold contents of the head samples collected from the two bulk samples to be 7.6 and 8.6 gpt gold, respectively (composite samples) and the metallurgical balance was reported as 6.79 gpt and 8.41 gpt gold based on a pilot treatment of the entire 10 tonne subsamples.

The mineral resource model tonnes and grade within the stope volumes from which the two bulk samples were taken (wireframes of the stopes received from Rubicon) were extracted by SRK. Rubicon independently estimated the weighted average of the gold grade from sampled drilling data that intersected the two volumes extracted for bulk sampling.

A comparison of bulk sample gold grades as estimated by SRK (mineral resource model), G&T (average head grades) and Rubicon (weighted average of drilling data after a top-cut was applied) is provided in Table 33. Although SRK is unable to comment on the bulk sample gold grade estimation methodologies applied by G&T and Rubicon, it is noteworthy that the gold grade estimates from these exceed that estimated by SRK from the mineral resource model.

Table 33: Comparison of Four Estimates for the Bulk Sample Gold Grade

Bulk Sample	Source	Domain	Volume (m^3)	Tonnage	Density	Grade (Au gpt)
A	SRK	Main	189	560	2.96	17.91
		Main 45	462	1,327	2.87	0.76
		External	7	20	2.87	0.02
		Combined	**658**	**1,906**	**2.90**	**5.79**
	G&T (met balance)			976		6.79
	Rubicon					5.80
B	SRK	Main	540	1,574	2.91	3.47
		Main 45	-	-	-	-
		External	71	203	2.87	0.02
		Combined	**611**	**1,777**	**2.91**	**3.08**
	G&T (met balance)			1,107		8.41
	Rubicon					9.14
A and B combined	SRK	Main	729	2,134	2.93	7.26
		Main 45	462	1,327	2.87	0.76
		External	78	223	2.87	0.02
		Combined	**1,269**	**3,683**	**2.90**	**4.48**
	G&T			2,083		7.60
	Rubicon					9.10

13.13 Grade Sensitivity Analysis

Mineral resources are sensitive to the selection of a reporting cut-off grade. To illustrate this sensitivity, classified resource model quantities and grade estimates are presented in Table 34 at different cut-off grades.

The reader is cautioned that the figures presented in the tables should not be misconstrued with Mineral Resource Statements. The figures are only presented to show the sensitivity of the block model estimates to the selection of a cut-off grade. Figure 23 presents this sensitivity as a grade tonnage curve.

Table 34: Global Quantities and Grade Estimates* at Various Cut-Off Grades

	Indicated Mineral Resources			Inferred Mineral Resources		
Cut-Off (Au gpt)	**Quantity (000't)**	**Grade Au (gpt)**	**Contained Au (000'oz)**	**Quantity (000't)**	**Grade Au (gpt)**	**Contained Au (000'oz)**
0.50	16,318	3.29	1,726	146,116	1.44	6,745
1.00	11,439	4.38	1,611	51,776	2.81	4,683
1.50	8,904	5.28	1,512	29,176	4.06	3,810
1.75	8,036	5.68	1,467	23,660	4.63	3,524
2.00	7,360	6.03	1,427	19,680	5.19	3,285
2.25	6,779	6.36	1,386	16,552	5.78	3,074
2.50	6,271	6.69	1,349	14,286	6.31	2,899
3.00	5,396	7.33	1,272	11,119	7.34	2,623
3.50	4,686	7.95	1,198	8,968	8.32	2,399
4.00	**4,120**	**8.52**	**1,129**	**7,452**	**9.26**	**2,219**
4.50	3,620	9.11	1,060	6,372	10.11	2,070
5.00	3,116	9.82	984	5,604	10.84	1,954
5.50	2,721	10.48	917	5,056	11.45	1,861
6.00	2,425	11.06	862	4,617	11.99	1,780
6.50	2,168	11.63	811	4,293	12.42	1,715
7.00	1,959	12.16	766	3,962	12.90	1,643
7.50	1,785	12.63	725	3,589	13.48	1,556
8.00	1,627	13.11	686	3,274	14.03	1,477

* The reader is cautioned that the figures presented in this table should not be misconstrued as a Mineral Resource Statement. The reported quantities and grades are only presented as a sensitivity of the deposit model to the selection of a cut-off grade.

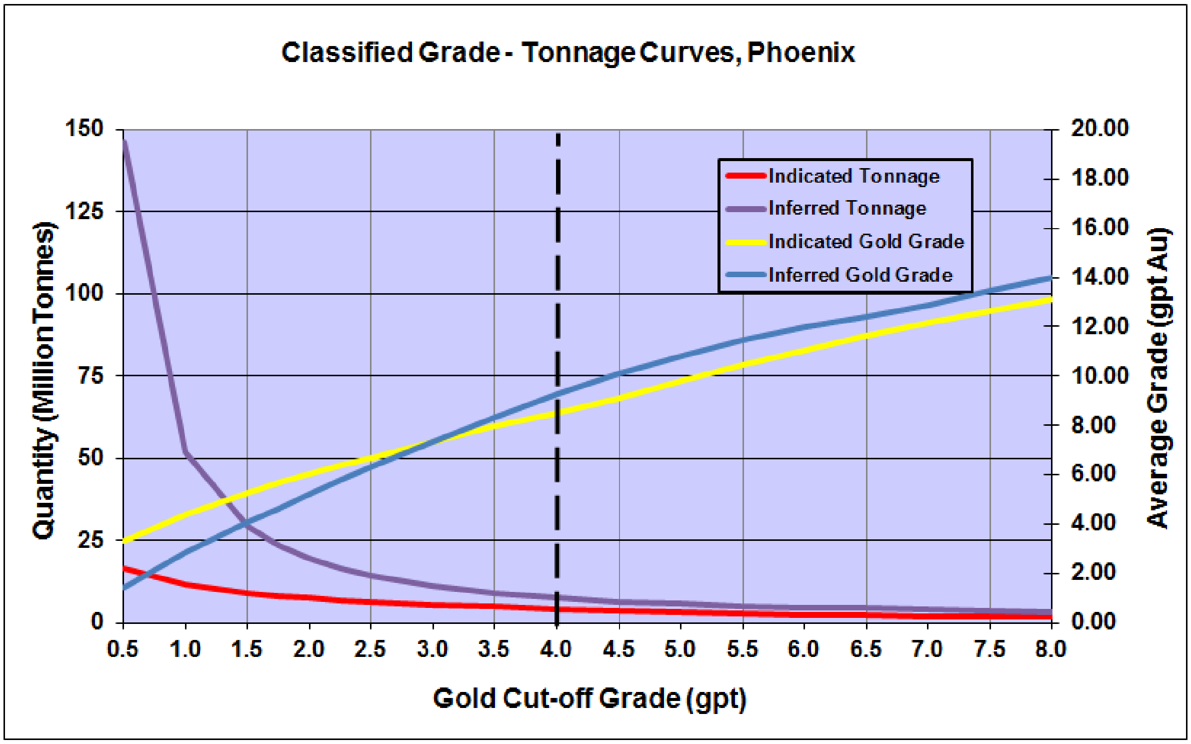


Figure 23: Phoenix Gold Project Classified Grade Tonnage Curves

13.14 Crown Pillar

Similarly to the June 2011 Mineral Resource Statement, the statement presented herein does not account for a crown pillar below the lake bottom. Detailed geotechnical modelling of the crown pillar has not been completed to date to adequately define the appropriate dimensions of a crown pillar.

A conceptual 45-metre crown pillar will be considered by SRK for the preliminary economic assessment study. A significant amount of geotechnical and structural investigation will be required to determine adequate dimensions for the crown pillar. Until these investigations are undertaken, SRK considers that some of the material within the potential crown pillar has “reasonable prospects for economic extraction.”

SRK estimate the quantity and grade of material within the conceptual 45-metre crown pillar to be 807,000 tonnes at 10.29 gpt gold (reported to a 4 gpt gold cut-off grade), which represents about 8% of the total reported mineral resource (in terms of gold ounces). All the material within the conceptual 45-metre crown pillar is classified as Inferred.

SRK strongly recommend that geotechnical and structural investigations are undertaken as soon as possible to determine adequate dimensions for the crown pillar. Once these are determined, the mineral resource within the crown pillar should be excluded from the reported mineral resources as material within the crown pillar does not have “reasonable prospects for economic extraction.”

14 Mineral Reserve Estimates

There are no mineral reserves to report at the Phoenix gold project.

15 Mining Methods

The projected mining method, potential production profile and plan, and mine plan are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks which include, but are not limited to, the inclusion of Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets and process plant designs may require additional detailed work, economic analysis, and internal studies to ensure satisfactory operational conditions and decisions regarding future targeted production.

To the extent that the use of the terms “ore,” “mineable,” “production,” and “mining” occurs in the following discussion, its use is intended solely to differentiate between mineralized material (including dilution) above an economic cut-off grade and waste rock; there is no inference of mineral reserves.

15.1 Overview

The Red Lake district is a world-class gold mining district. Twenty-eight mines have operated in the district since 1930 producing more than 26 million ounces of gold from three main producing mines: Campbell, Dickenson/Red Lake, and Madsen. The gold deposits of the Red Lake Greenstone Belt have been classified into three groups (after Pirie 1981), according to the stratigraphic or lithological associations described in Section 7.

The majority of the F2 gold system is interpreted as being the first type with similarities to the mineralization found at the Campbell-Red Lake, Starratt-Olsen, the Cochenour-Willans mines, and the No.8 Zone of the Madsen mine.

15.2 Previous Mining

The property has never been in commercial production to date, though a number of bulk samples have been taken in the past on both the F2 gold system and the unrelated mineralization that the McFinley mine was exploring.

In 1956, a 129-metre deep exploration shaft was sunk by McFinley Red Lake Gold Mines Ltd. and followed up with 414 m of lateral workings on two levels before work was suspended in mid-1957 (G.M. Hogg & Associates Ltd. 1983).

In 1984, the shaft was re-opened as the Phoenix Shaft and an additional 479 metres (m) of lateral development was completed on the 46 m (150 ft) and 122 m (400 ft) levels. After a temporary shutdown starting in February 1985, a further 1,151 m of lateral development and 10 m of raise development was completed prior to the decision to take a bulk sample in 1987. The bulk sample program started in July 1988 from prepared stoping areas (OMNDM 2013).

The level naming convention for the mine was originally measured in feet below the shaft collar. The 400 foot level was the original bottom level of the McFinley mine and is now referred to by its metric equivalent, the 122 level. The project uses the metric system and all measurements are metric.

Mining exploration activities on the property were terminated in 1989 after test-milling of an estimated 2,250 t of material unrelated to the F2 gold system.

Rubicon acquired the property in June 2002 and resumed exploration work.

In 2009, the existing shaft was dewatered and reconditioned to support an advanced exploration program. In June 2009, shaft sinking started to deepen the existing shaft to 350 m and a 3-tonne loading pocket was installed to support development at the 305 level, followed by lateral and vertical development on the 244 m and 305 levels. This lead to two approximately 1,000-tonne bulk samples being excavated on the 305 level in 2011 using development methods.

Shaft sinking resumed again in July 2012 after upgrading the headframe and hoisting plant, but it has been slowed significantly due to a zone of squeezing ground encountered during this phase of the shaft sinking through ultramafic units. The installation of concrete reinforcing rings and other measures were taken to ensure these issues will not cause potential future planned production delays. At the time of writing, the shaft was at 670 m depth with the intention of stopping at 710 m depth and installing a 10-tonne loading pocket to support production.

Surface infrastructure to support future production that is either installed or is under construction includes:

- Administration and camp facilities;
- Security shack and gate;
- Core logging and storage buildings;
- Warehousing and yard storage laydown areas;
- 7.5 MVA substation with a supply allotment of 5.3 MVA from Hydro One (additional capacity and allotment required for planned expansion);
- Construction of a 1,250 tpd mill, upgraded to 1800 tpd, is in progress; and
- Tailings management and water treatment facilities construction is in progress.

15.3 Geotechnical Evaluation

15.3.1 Introduction

SRK completed a scoping level geotechnical evaluation. The analysis pertaining to excavation stability is presented below.

15.3.2 Data Sources

Data used for the geotechnical assessment of the Phoenix deposit consisted of geotechnical data collected from boreholes and underground mapping during a site visit conducted by Brad Klassen and Nico Viljoen of SRK in April 2013. During the site visit, four exploration boreholes were geotechnically logged and five areas were geotechnically mapped on the 305 level. The locations of the boreholes that were logged are shown in Figure 24, the locations of underground mapping areas are shown in Figure 25. The Laubscher (1990) RMR method was used for both core logging and underground mapping.

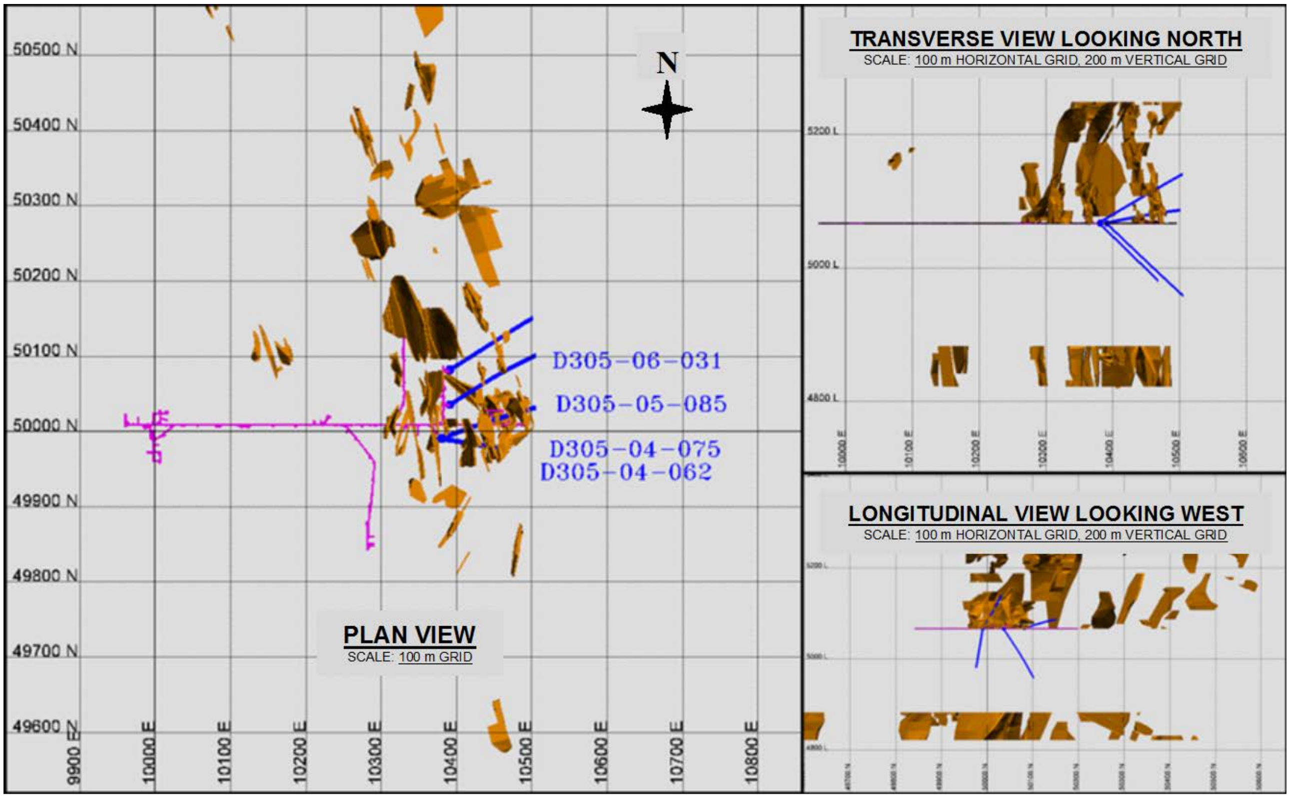


Figure 24: Geotechnical Borehole Locations in Relation to Mining Shapes (SRK 2013)

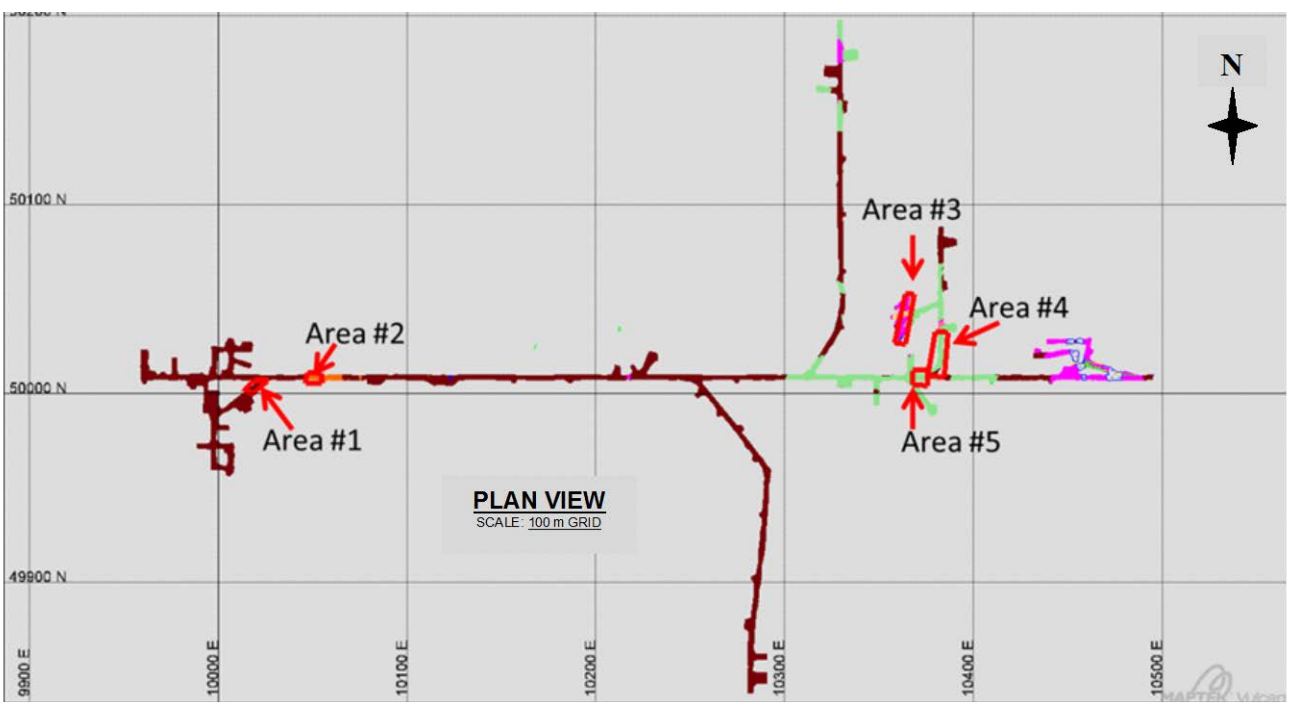


Figure 25: Geotechnical Mapping Locations on 305 Level (SRK 2013)

SRK was provided with a borehole database that included geological data and core photos for the Phoenix gold project. The provided borehole database contained rock quality designation (RQD) data for exploration boreholes that was collected by Rubicon Minerals (Rubicon).

SRK was provided with a crown pillar assessment conducted by AMC Consultants (AMC). No new geotechnical data has been collected in the crown pillar area since the study was conducted by AMC. Boreholes containing geotechnical data from the AMC crown pillar assessment are shown in Figure 26.

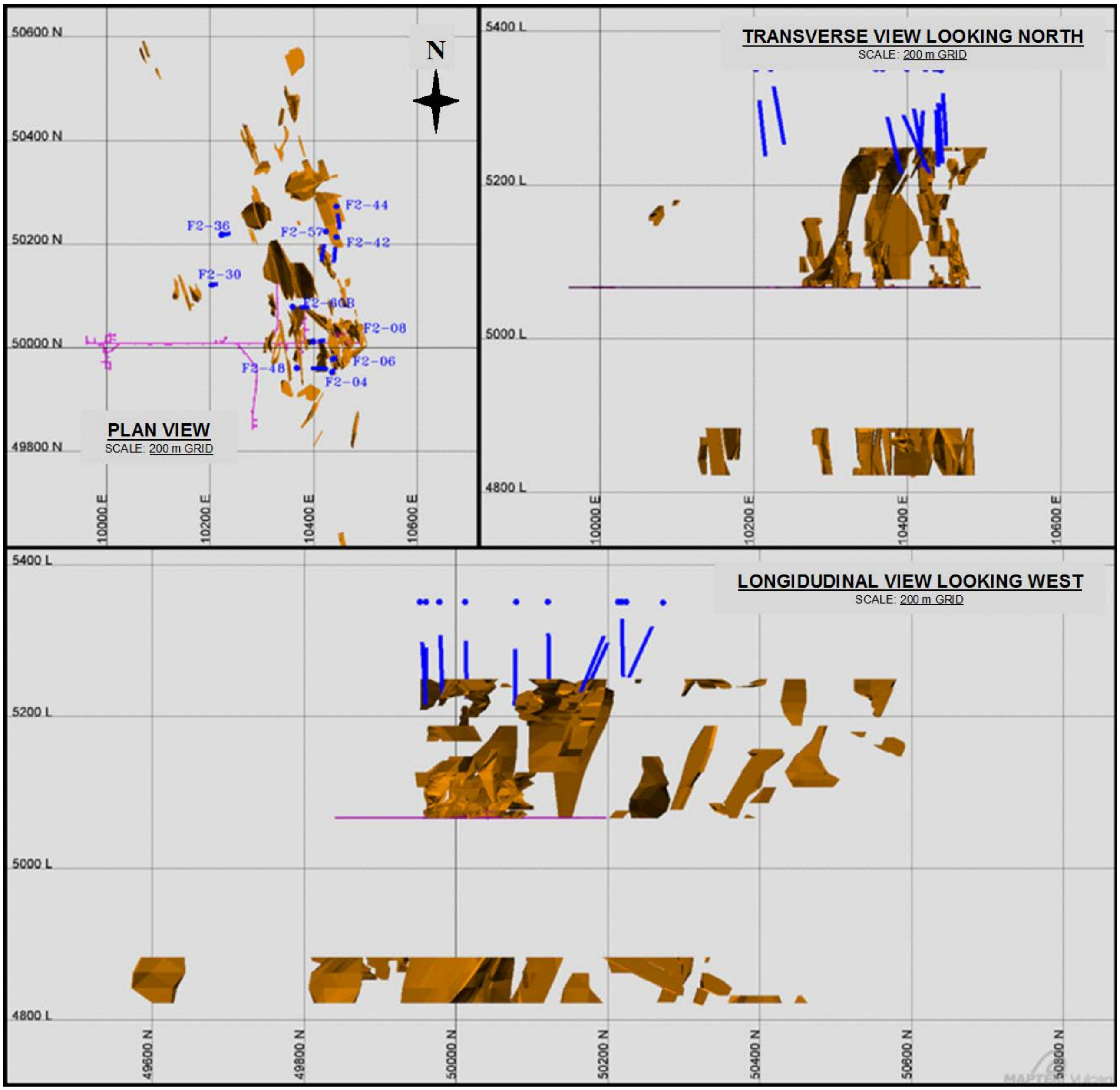


Figure 26: Location of Boreholes Logged by AMC for Crown Pillar Study (SRK 2013)

SRK was provided with wireframes for the following geological units, which are shown in Figure 27. SRK was not provided with a wireframe for the ultramafic rock unit.

- AGZ group (altered green zone);
- Basalt group;
- BIF group (banded iron formation);
- Brittle faults;
- Felsic group;
- HT group (high titanium basalt);
- QFP group (quartz feldspar porphyry); and
- Uncharacterized faults.

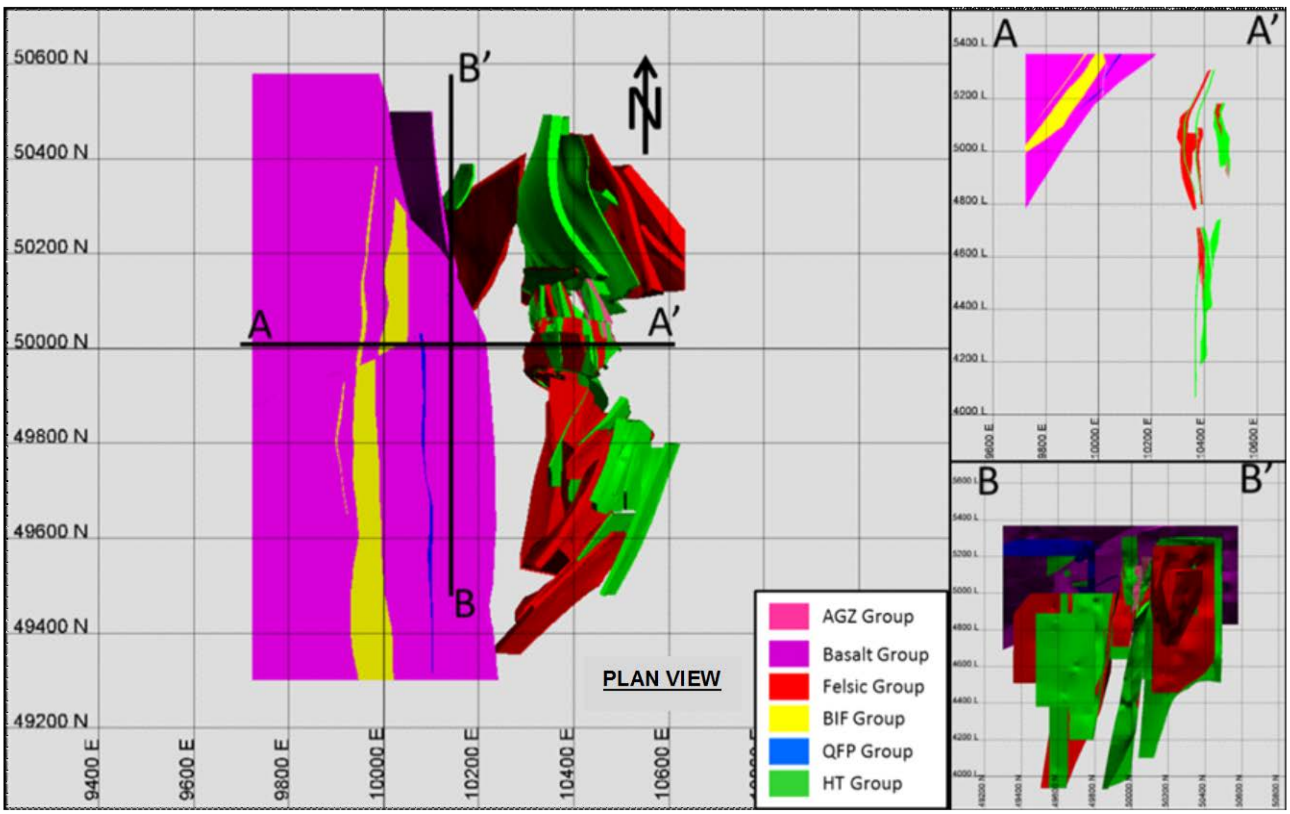


Figure 27: Geological Unit Wireframes (SRK 2013)

15.3.3 Geotechnical Assessment

This section outlines the major factors controlling the underground engineering design. Based on the discussions presented in this section, design recommendations are discussed below in Section 15.3.4.

Structural Geology

Fault interpretations provided to SRK include brittle faults that have been observed in core as well as uncharacterized faults that were interpreted by offsets in the geological model. The uncharacterized faults are of low confidence as they are not observed in core. Going forward to advanced study stages, it will be important to understand if the uncharacterized faults exist and the properties of these faults as they are located within the proposed mining area.

Rock Fabric Assessment

The ultramafic domain has varying levels of foliation. The general trend of the foliation is subparallel to mineralization with localized areas that have been folded. Additional oriented core will assist with understanding the localized foliation behaviour.

Going forward, it will be important to understand the orientation of joint sets and foliation as the intersection of these features can implement the risk of kinematic failures in mine workings. It is recommended that oriented core and/or televiewer data be collected in order to better understand the orientations of joints and foliation.

Geotechnical Domains

Based upon the interpretation of the currently available data, four major geotechnical domains are proposed for the Phoenix gold project. Within these domains there are localized subdomains that could not be separated at this time due to a lack of geotechnical data within the units.

The following major geotechnical domains are proposed:

- Domain 1 – Altered green zone
- Domain 2 – Felsic group
- Domain 3 – HT group
- Domain 4 – Ultramafics

Rock Mass Characterization

The proposed geotechnical domains were analyzed using the geotechnical core logging data that intersected each domain. Rock mass parameter ranges were determined for each domain as presented in Table 35. An evaluation of the core photographs was completed with a summary of rock mass rating as well as lithology.

Table 35: Rock Parameter Ranges per Geotechnical Domain

Parameter		AGZ_Group	Felsic_Group	HT_Group	Ultramafic
Total Logged (m)		**12**	**48**	**75**	**339**
RQD	Low	99	97	99	97
	Likely	100	99	99	98
	High	100	100	100	100
FF/m	Low	0.63	0.70	0.30	0.30
	Likely	0.70	1.00	0.30	1.00
	High	0.73	1.85	0.70	1.30
RMR90	Low	63	60	63	63
	Likely	66	63	68	67
	High	67	65	68	69
Q	Low	9	6	9	8
	Likely	11	8	14	13
	High	13	10	14	16

Note: Q calculated from RMR90 using formula RMR=9ln(Q)+44

Assumption for JW=1 and SRF=1 used therefore Q'=Q

During the site visit in April 2013, five areas were geotechnically mapped underground on the 305 level. The areas that were mapped are outlined in Figure 25 with the RMR90 results shown in Table 36. The results of the underground mapping are within the range of RMR90 values determined from geotechnical core logging.

Table 36: Underground Geotechnical Mapping Results

Site	Mapping Area	Domain	Rock Type	RMR90
Phoenix	1	A	Ultramafic	53
Phoenix	1	B	Ultramafic	30
Phoenix	2	A	Komatiite Basalt	52
Phoenix	3	A	HiTi Basalt	58
Phoenix	4	A	Felsic	67
Phoenix	5	A	Felsic	61

15.3.4 Underground Excavation Design Recommendations

The design of the underground excavations benefits from a number of well-established empirical and semi-empirical rules. These enable estimates to be made of the expected mining conditions and support requirements on the basis of a detailed description of the rock mass. The design procedure involves two steps: the quality of the rock mass is rated using a pre-defined classification system, and then the expected performance of the underground openings is predicted using an empirically derived correlation with the rock quality. This section discusses the excavation design methodology and recommendations based upon the rock mass characterization, in situ stress regime, and proposed mining methods.

The stability of stope geometries was assessed using the empirical Matthew’s Method (Trueman 2000; Mawdesley 2001); the same input parameters were used to estimate mineral dilution due to overbreak using the ELOS approach (Clark & Pakalnis 1997; Clark 1998).

Geotechnical and operational constraints will limit strike and vertical extent of the stopes. The dimensions outlined in this section are assumed reasonable at this stage but will vary based on further data collection. The geotechnical parameters used for the following analyses are derived from an evaluation of detailed geotechnical core logging and core photographs discussed in the preceding sections. These parameters should be re-evaluated in the specific stoping areas as additional geotechnical data becomes available as the project progresses.

In Situ Stresses

The performance of an excavation is also influenced by the in situ stresses in the rock mass. In the absence of site specific in situ stress data, a brief literature review was conducted to identify the likely pre-mining state of stress data at the project location.

The general trend of the mineralization of the Phoenix deposit is northeast-southwest and dipping from near vertical to ~80 degrees to the northwest. In the Canadian Shield, the general direction of the maximum principal stress is typically northeast; this is approximately parallel to the strike of mineralization. The minor principal stress in the Canadian Shield is typically in the vertical direction. Pre-mining in situ stress measurements have not been conducted in the immediate project study area. Pre-mining stress levels required of the assessment of excavation stability were estimated using the method discussed in Arjang (2006) for use in projects located within the Canadian Shield.

Empirical Design

For the purpose of empirical design, a range of RMR90 values of 60-70 were considered suitable for each domain. Man-entry spans, which are based on the empirical recommendations after Ouchi et al. (2004), are shown in Figure 34. A span of 10 m approaches the boundary of the potentially unstable zone for the range of RMR90 values considered.

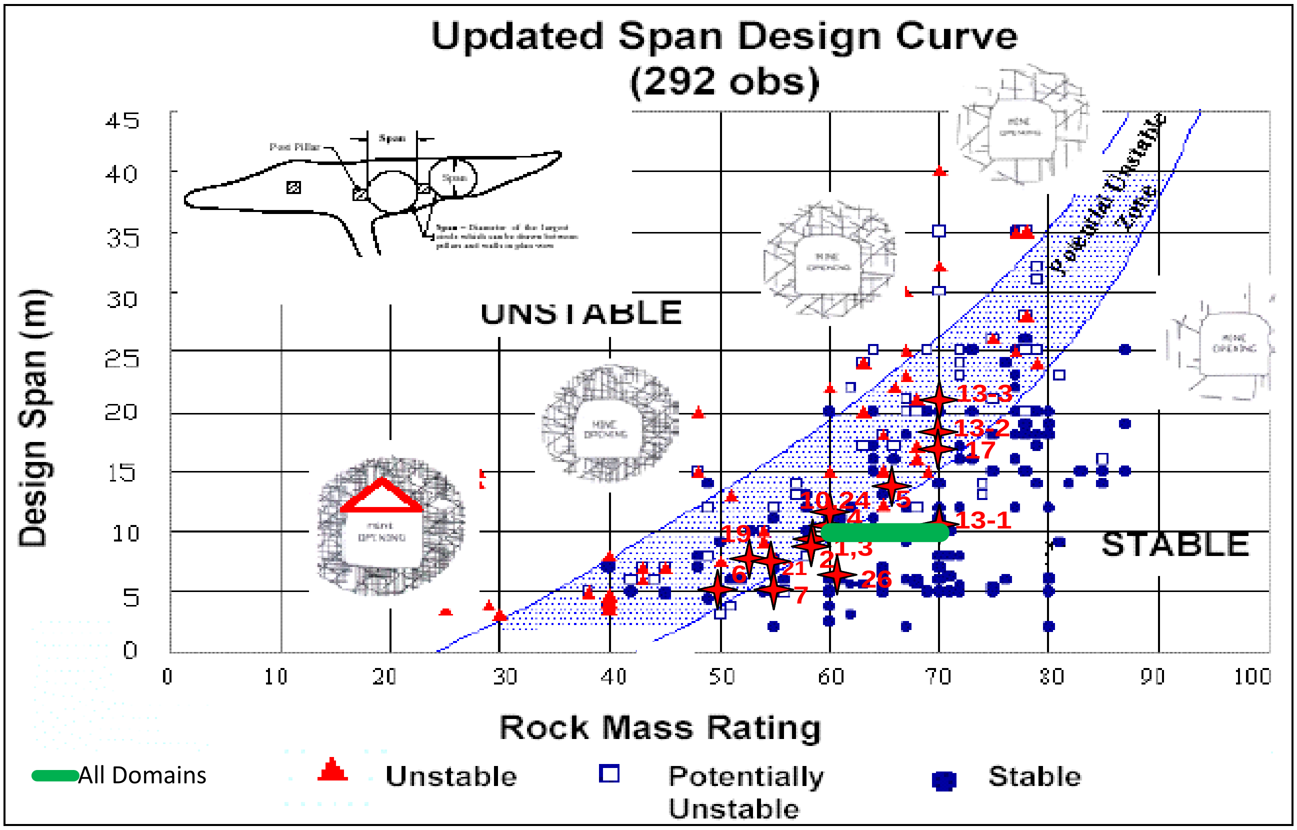


Figure 28: Man-entry Recommended Design Span (Ouchi et al. 2004)

Stope stability was analyzed for a range of sublevel spacing and depths for non-man entry stopes using the Mathews Method after Trueman (2000) and Mawdesley (2001) to determine stable stope dimensions appropriate for the mineralized wireframes. Example stope stability and probability of failure plots for the considered stope dimensions at a depth of 750 m are shown in Figure 29 and Figure 30.

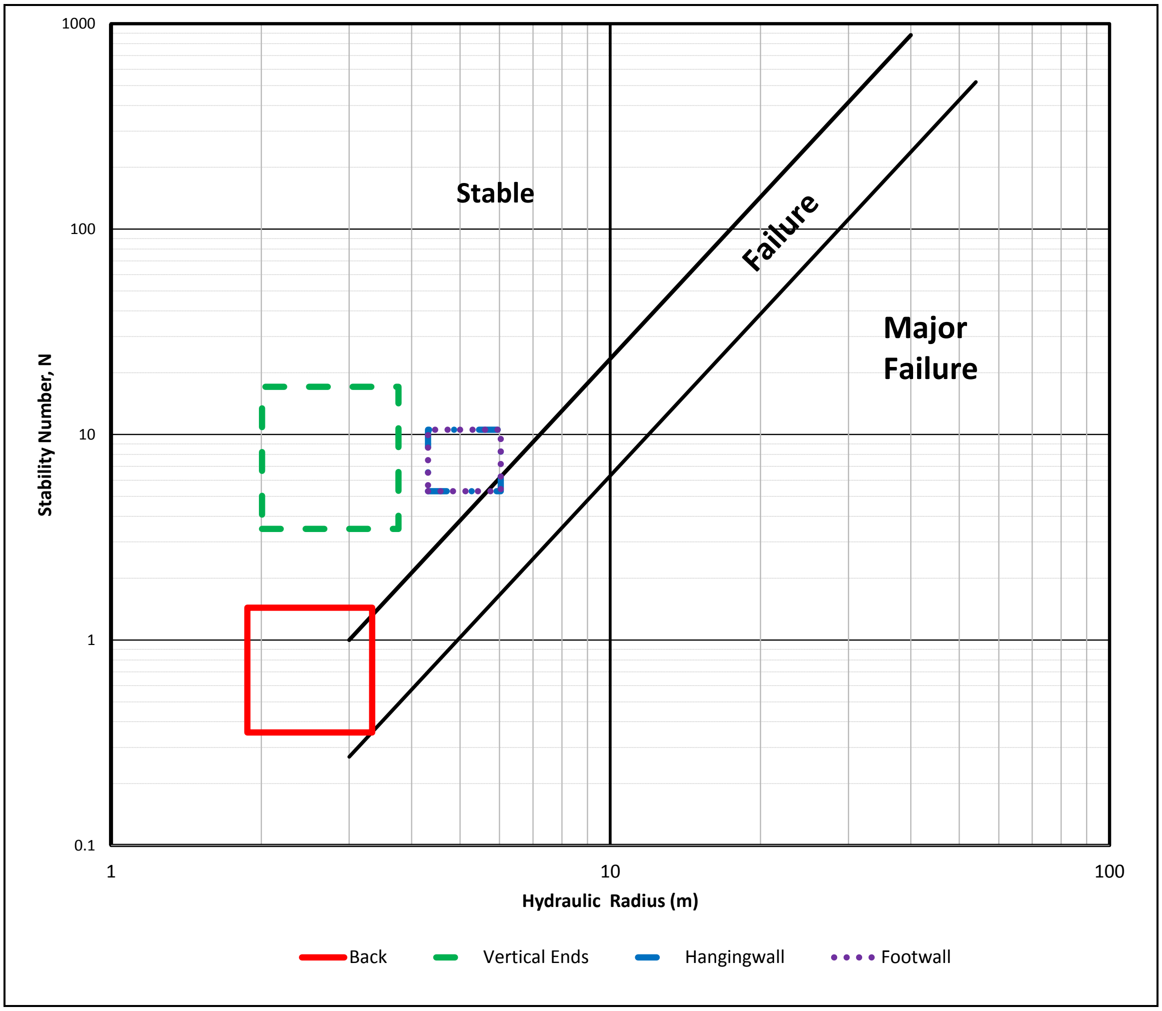


Figure 29: Non-Man Entry Stope Stability Plot for Considered Stope Dimensions at 750 m Depth (after Trueman 2000)

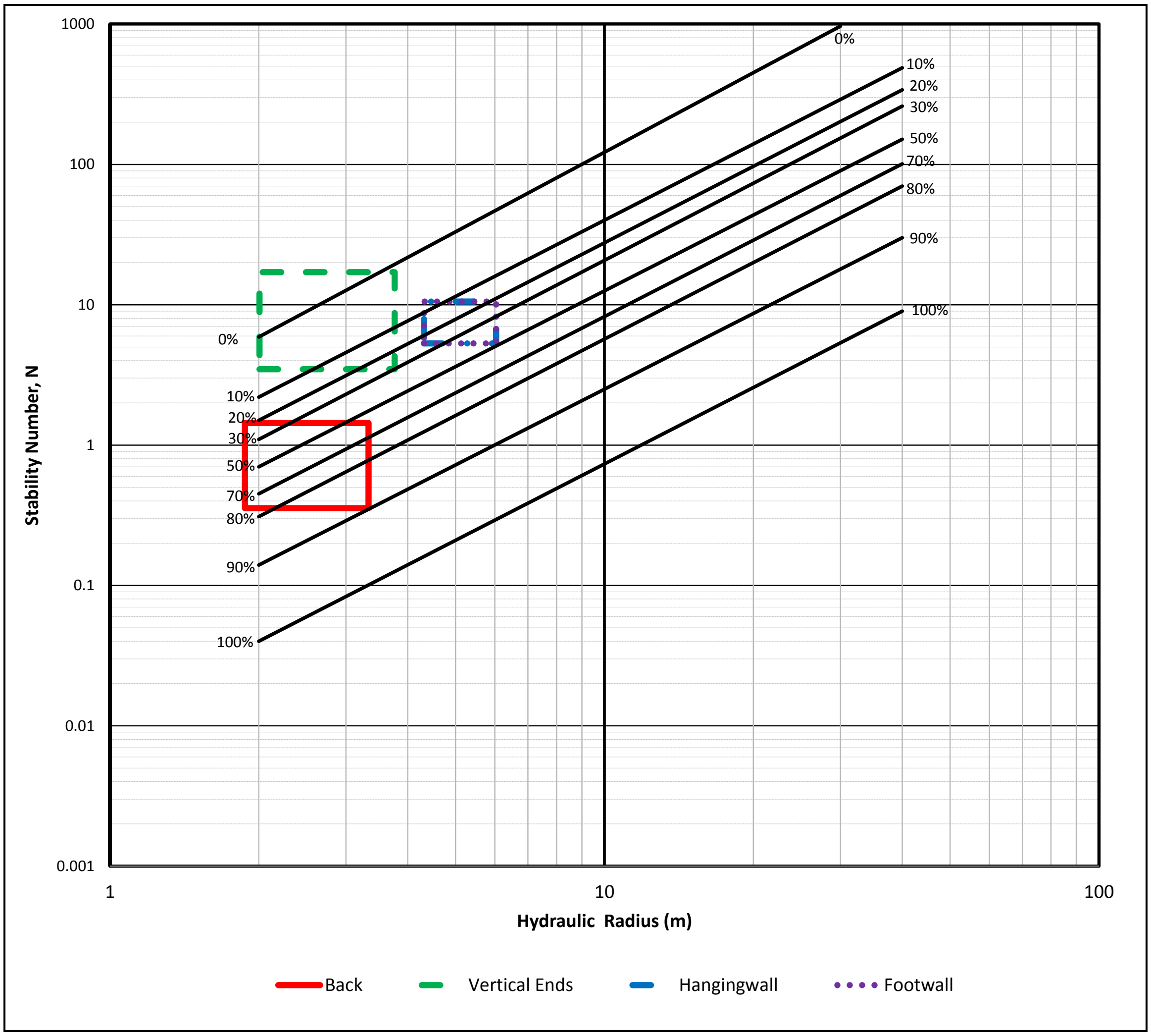


Figure 30: Non-Man Entry Probability of Failure Plot for Considered Stope Dimensions at 750 m Depth (after Mawdesley & Trueman 2001)

By using this approach and combined with engineering judgement, a range of stable excavation geometries were determined for a range of sublevel spacings at various mining depths. These are summarized in Table 37. It is important to note that for each stope size analyzed, the back falls within the potentially unstable zone and will require ground support for stability. Due to the lack of joint orientation data, excavation potential is restricted by conservative B and C parameters. It is anticipated that the collection of oriented core data may allow for the B factor to be adjusted to less conservative and potentially allow the mining of stopes with larger spans.

These stope dimensions are based upon a geotechnical data set limited to the proximity of the 305 level and the crown pillar assessment conducted by AMC. As the distance from these locations increases, the level of confidence of achieving these excavation geometries decreases. Review of similar properties in the Red Lake area and elsewhere in the Canadian Shield would suggest that the excavation geometries summarized in Table 37 would be achievable for 70 percent (%) of the deposit.

Table 37: Maximum Stable Span and Strike per Sublevel Spacing

Depth (m)	Sublevel Spacing (m)	Maximum Span (m)	Maximum Strike (m)
250	15	8	55
	20	8	35
	25	8	25
	30	8	20
500	15	8	50
	20	8	35
	25	8	20
	30	8	15
750	15	8	45
	20	8	20
	25	8	12
	30	8	10
1000	15	8	30
	20	8	10
	25	8	10
	30	8	8

Dilution

Sources of dilution of the mined mineralized material are influenced by two key factors: geotechnical, taking into consideration the quality of the rock mass and the excavation and ground support design; and operational, where the influence of blasting, mining, geological, and survey controls are included. Where stopes are mined in narrow vein type deposits, dilution of mineralized material due to geotechnical factors is often small in relation to the dilution realized from operational factors.

The potential dilution effects were considered using the method developed by Clark (1997), where he relates modified stability number, N′, and the hydraulic radius to the equivalent linear overbreak/slough (ELOS), which can then be used to calculate percentage dilution. The range of values for N′ and the stable stope dimensions outlined in Table 35 were used in the dilution assessment and are represented by the zones on the ELOS dilution design chart in Figure 31.

Using this method, dilution is estimated to be between 1 and 2 m for both the hanging wall and footwall of each stope at the lower end of the rock mass quality considered. Cable bolting stope walls is incorporated into the mine design to control dilution.

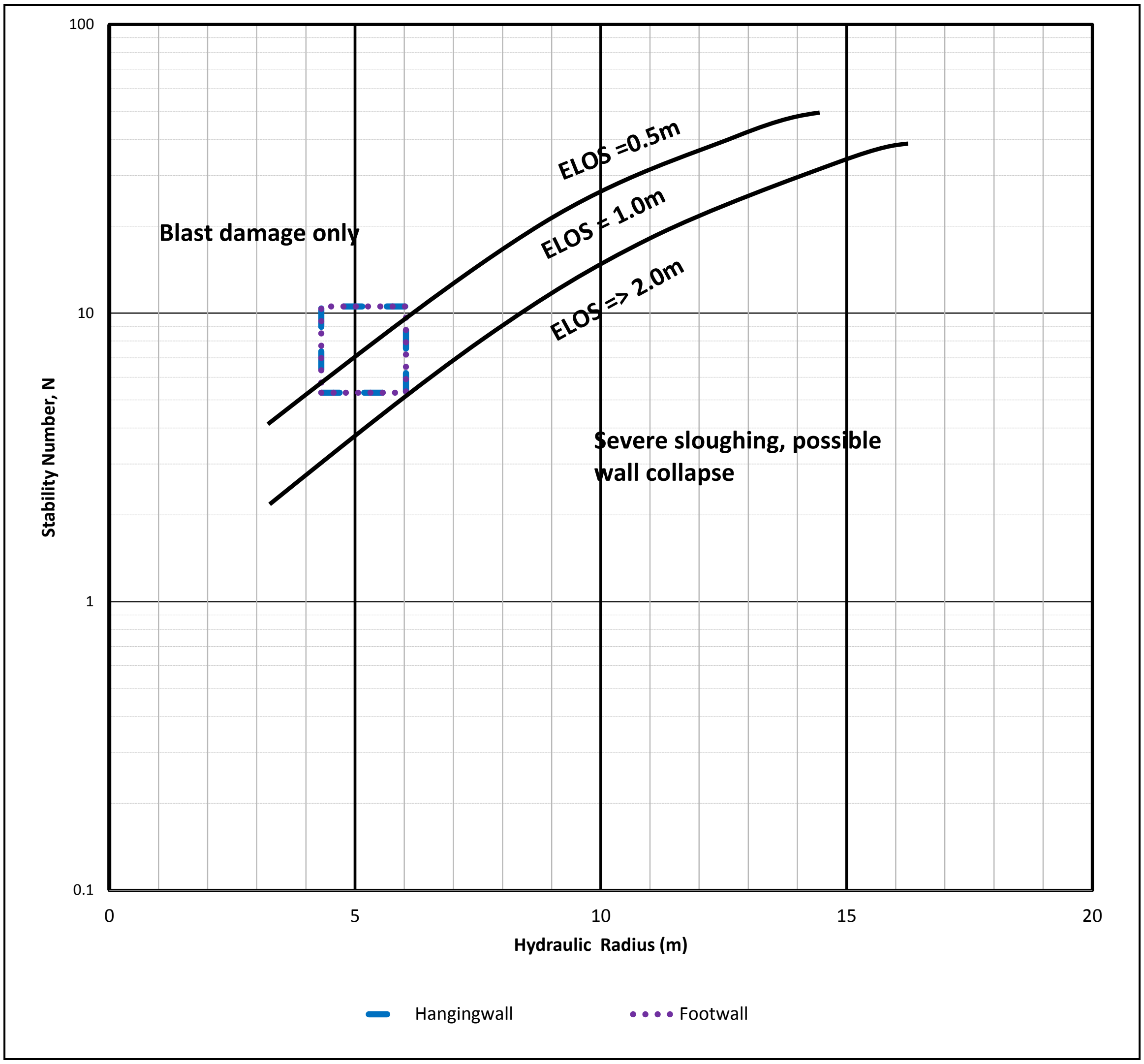


Figure 31: Dilution Estimation Using ELOS Method (after Clark 1998)

15.3.5 Ground Support Recommendations

The development support recommendations presented below are based on the empirical support design approaches of Laubscher (1990) and Grimstead and Barton (1993) as well as engineering judgement. The ranges of rock mass values are plotted on the ground support chart in Figure 32.

For the purpose of empirical design, a range of Q values of 6-17 were considered suitable for each domain as based on an RMR90 range of 60-70 converted to Q using the formula:

$$RMR = 9 \times \ln(Q) + 44$$

To account for differences in the excavation performance expectations, the excavation span is adjusted using the excavation support ratio (ESR). In permanent infrastructure excavations, an ESR of 1.6 is used, i.e., a 5 m true span is reduced to a 3.1 m adjusted span, and for stopes an ESR of 3.0 is used.

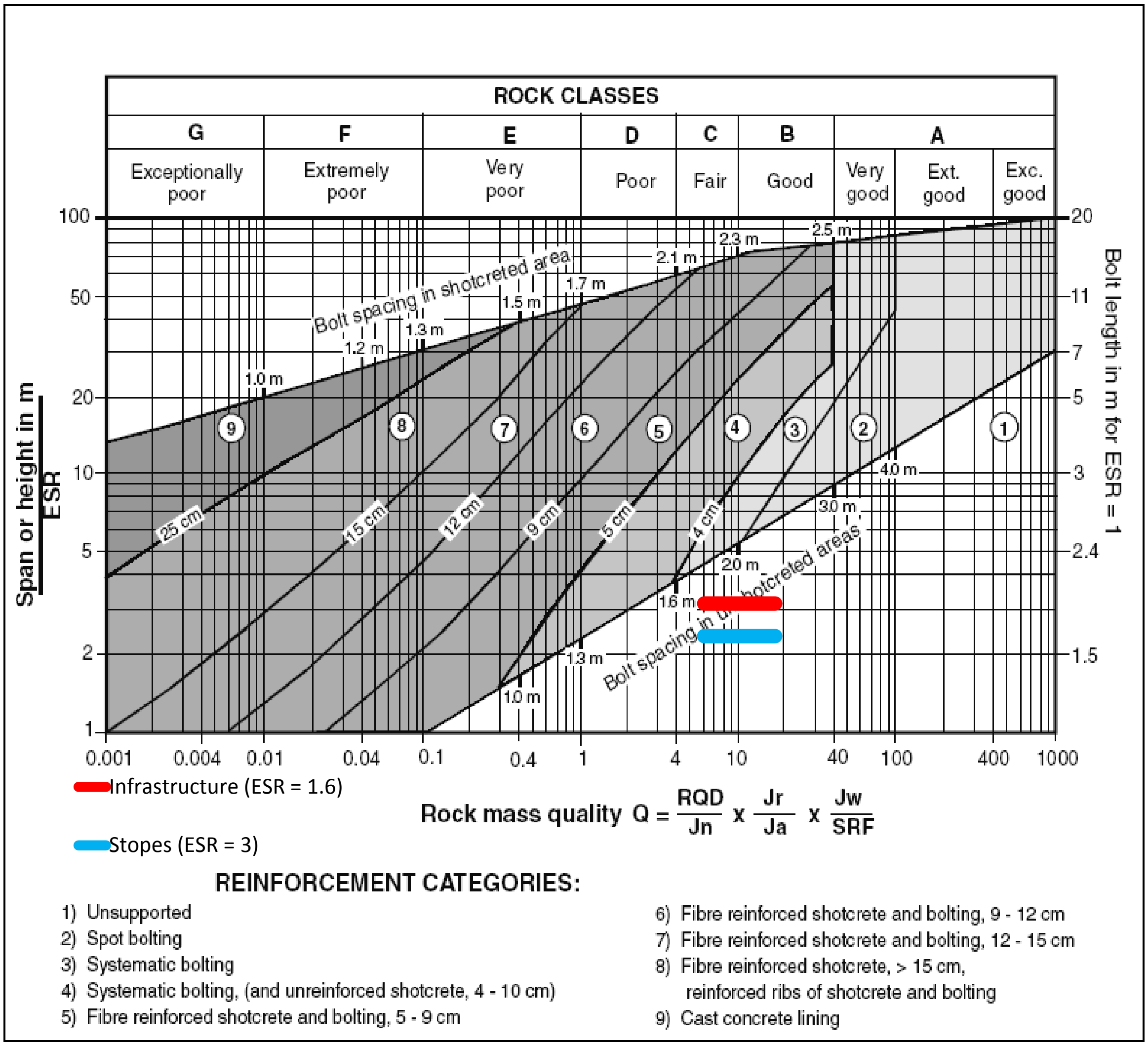


Figure 32: Ground Support Recommendations (after Grimstad and Barton 1993)

The ground support measures indicated by the geotechnical assessment are:

- For infrastructure excavations up to 5 m span, minimum 1.8 m long full column resin grouted #8 rebar on 1.5 m spacing extending down to 2 m from the floor;
- For stope excavation spans up to 5 m, minimum 1.8 m long resin grouted rebar on 1.5 m spacing extending down to 2 m from the floor;
- For stope excavation spans from 5 m up to 10 m, minimum 2.4 m long resin grouted rebar on 1.5 m spacing extending down to 2 m from the floor;
- Where ultramafic units are encountered, additional localised ground support in the form of welded wire mesh and shotcrete may be required, especially in areas of alteration. Extensive exposures of altered ultramafic units may require spiling and/or thicker applications of shotcrete;
- Altered portions of the ultramafic units are expected to result in localised instabilities as observed on the 305 level during the April 2013 site visit. In order to minimise the possible adverse impact on excavation stability, increased ground support requirements, and mining advance rate, it is strongly recommended that ultramafic units be avoided where possible.

Where it is not possible to avoid these units, it is strongly recommended that mining through ultramafic rock be oriented as close to perpendicular to the strike of the unit as possible;

- Ground support requirements for permanent infrastructure excavations with larger spans such as crusher chambers, pump chambers and workshops, will need to be assessed on a site by site basis; and
- Mesh should be used, as a minimum, in areas that are accessed on a proportionately higher basis, such as refuge stations, explosive magazines, and shops.

Kinematic assessment of excavation stability and ground support adequacy will be required once sufficient joint orientation data has been collected in later stages of the project.

15.3.6 Crown Pillar Assessment

SRK has reviewed the crown pillar assessment that was conducted by AMC Consultants. No new geotechnical data has been collected in the area of the crown pillar since the work conducted by AMC. SRK conducted a photo review of the crown pillar geotechnical core and calculated the RMR using the original data collected by AMC's programme.

The data indicates that the crown pillar is expected to be stable for the planned mining design for at least 50 years. Applying the methodology described in Carter (2008) and using a range of RMR values of 60 to 70 (Q' of 5.9 to 18), and maximum stope span and strike length of 10m and 40m respectively a 45m thick crown pillar plots in the Class F region of the crown pillar stability plot in Figure 33 below.

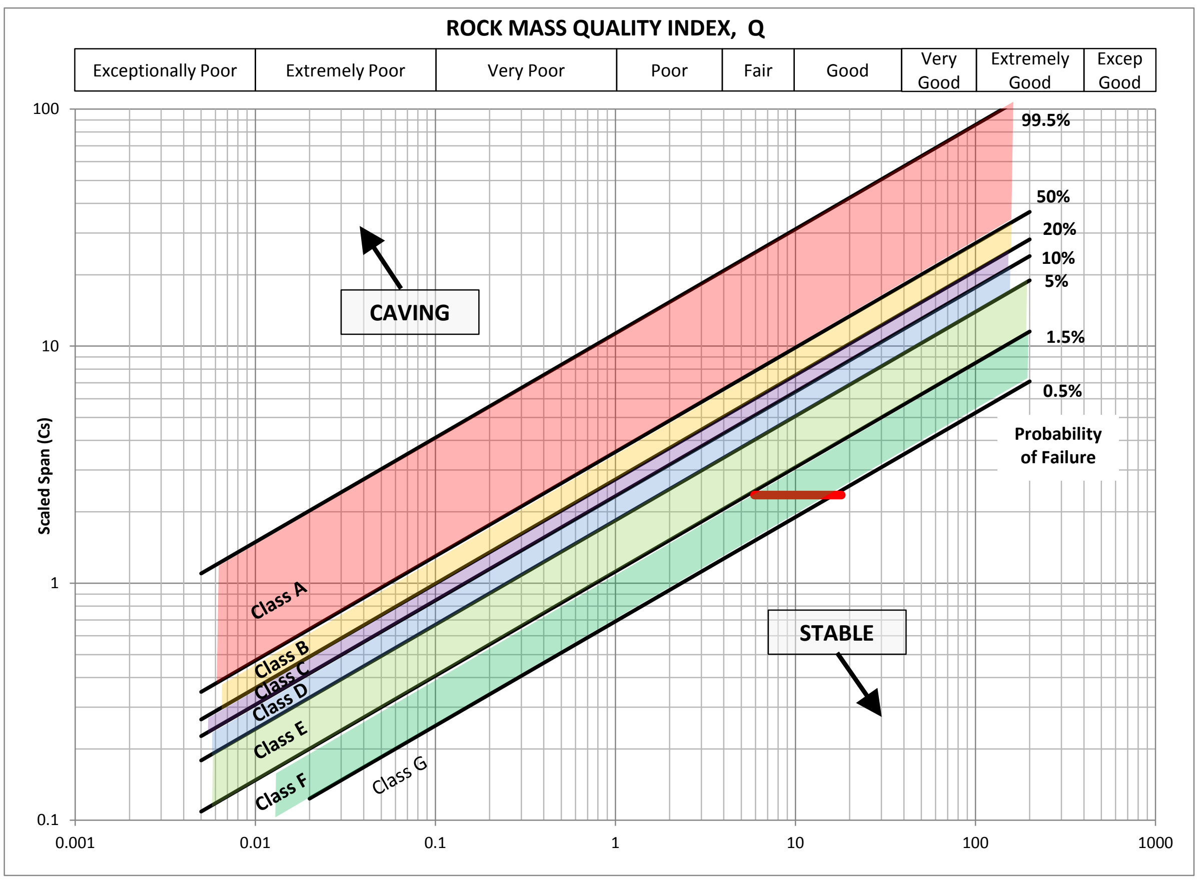


Figure 33: Crown Pillar Stability Plot (after Carter 2008)

The AMC crown pillar assessment was based on a mining span of approximately 3 m where the mining shapes in this report can have spans greater than 10 m below the crown pillar. Further assessment of the crown pillar is recommended to consider the influence the larger stope spans as well as of any major structures that may be present.

15.4 Planned Mining Methods

The previous preliminary economic assessment completed by AMC (2011b) envisioned a non-mechanized narrow vein gold mine using conventional captive cut and fill methods, track access, handheld drills, slushers, and mucking machines. This approach provides a highly selective, flexible mining strategy, but limits the practical production rate and requires high levels of skilled manpower resulting in a high operating cost.

Subsequent structural geology work has led to a revised interpretation of the structural controls on mineralization that has resulted in a different interpretation of the mining widths that are expected. The current mining plan envisions a complex deposit that is relatively discontinuous, somewhat disseminated in nature, has weak visual indicators, and a strong nugget factor.

In order to optimize the recovery of the mineral resource, it will be important to delineate the auriferous zones appropriately and to develop good quality local block models to represent the grade distributions within them. These block models may represent one or more levels or only one part of a level interval, depending on the local variations in shape and extent of mineralization.

The mine engineering group will use these block models to design stopes and associated development in 3D (three-dimensional) to support mineral reserve estimates and the long term production plan. Once the design of a stope or group of stopes has been finalized, the 3D shapes will be used by the short-term mine planners to prepare detailed layouts for stope development and longhole production drilling. Development layouts will be executed under survey control with adjustments made as additional geological data becomes available from mapping and sampling the exposed mineralization.

A few cycles of reconciling the actual mining results with the block model predictions will help refine this method to give reasonable estimates of future production, mitigating much of the uncertainty caused by the nugget effect and the disseminated nature of the deposit.

This process could take more than a year to complete from the start of definition drilling to the start of development.

15.4.1 Conceptual Mining Method Selection

The main deposit characteristics (context) relevant to the conceptual mining method selection are:

- The deposit is located approximately 400 m east of the existing shaft;
- The upper levels have already been established potentially as a track mine on 61 m level intervals from 122 level to 305 level;
- The small size of the existing shaft limits practical equipment size to 6.7-tonne class load-haul-dump (LHDs) and 20-tonne trucks;
- About half of the deposit is below the current shaft bottom;
- The deposit consists of four zones, each with a separate block model;
- No material from the fourth zone (HW zone) is included in the PEA LoM plan as it is distant from the other zones and lower grade;

- The mineralized zone is 150 to 200 m wide in section, up to 1,000 m along strike and discontinuous to the extent that much of this area is below the design cut-off grade;
- There are isolated mineralized zones outside the main corridor;
- The deposit dips at between 75 and 80 degrees with the shaft on the hanging wall side;
- Individual mineralized zones range in dip from 65 degrees to vertical;
- Mineralized zones above the conceptual mine design cut-off grade vary in true width from less than 1 m to greater than 30 m;
- Mineralized zones above the conceptual mine design cut-off grade can pinch and swell rapidly along strike and along dip;
- It is a high grade, high value deposit requiring good mining recovery;
- The deposit is located under a lake, therefore a stable crown pillar must be maintained;
- Any extraction from the crown pillar should wait until the end of the potential mine life;
- The conceptual mine plan ranges from 122 level (bottom of crown pillar) to 1586 level, a vertical distance of 1,464 m;
- The mineralized zone has contacts that are difficult to identify visually;
- Water inflows do not appear to be an issue as the known geological units have low permeability;
- Grade continuity in the mineralized zones above design cut-off grade is generally variable, which is indicative of a strong nugget effect; and
- The 122, 244 and 305 levels have already been started as track drifts.

To successfully mine a deposit with the above characteristics will require a high level of geological effort to understand the mineralization trends at the stope level, including closely spaced definition drilling and chip sampling programs. Development headings in mineralized material and cut and fill mining will be mined under geology control due to the general lack of visual indicators of mineralization.

This complex deposit will challenge the mine engineers to develop and employ a comprehensive "tool box" of mining solutions to optimize the extraction of the mineral resources. This will require the employment of multiple mining methods and variations on those mining methods to deal with situations where the mineralization is not continuous to the next level or has an irregular geometry.

SRK envisions the following conceptual mining methods as having a place in the deposit extraction plan:

- Sublevel longhole from an Alimak access raise for stopes between 1.8 to 5 m wide;
- Longitudinal retreat longhole with either a captive mucking sublevel or ramp/level access for stopes between 5 and 10 m wide;
- Transverse longhole with either a captive mucking sublevels or ramp/level access for stopes over 10 m wide;
- Uppers longhole stoping to extract small mineral resources, sill pillars, or other remnants;
- Captive conventional cut and fill (CAF) for isolated areas not amenable to longhole methods; and
- Mechanized cut and fill (MCF) for areas not amenable to longhole methods that are close to a ramp or level.

There are also a number of other mining methods that could be utilized but have not been considered as part of this PEA:

- Alimak longhole method (drill and blast horizontal longholes from an Alimak raise climber);
- Avoca method; and
- Shrinkage mining methods.

The mine design strategy was to design as many areas as practical using variations on the longhole mining method, and using overhand cut and fill mining methods for areas too irregular to employ longhole mining methods. Areas where an access ramp can be developed will employ mechanized cut and fill methods. Conventional captive cut and fill methods may be employed where no ramps or levels are nearby.

15.4.2 Alimak Access Sublevel Longhole Conceptual Mining Method (1.8 to 3.5 m Wide)

Approximately 13% of the total stoping tonnes (11% of life-of-mine [LoM] tonnes) are based on mineralized zones with a horizontal width between 1.8 and 3.5 m that are amenable to longhole mining methods.

Prior to SRK becoming involved, AMC and Rubicon had developed this approach as an alternative to the conventional captive cut and fill method proposed in the 2011 preliminary economic assessment, which was based on experience at Bell Creek Mine in Timmins. Rubicon requested SRK to consider the application of this method, but the deposit structural re-interpretation has reduced its role in the LoM plan compared to Rubicon’s original expectations.

SRK concurs that the method is workable, but has not conducted any trade-off studies to determine if it is optimal. Cable bolting of hanging walls has been incorporated in the mine design and operating costs to control potential hanging wall dilution. Test stopes are recommended in order to prove up and fine tune the method.

A typical stope will contain 20,000 tonnes of mineral resource, with dimensions of 61 m height by 45 m length by 2.5 m width, comprised of three 15-metre wide panels of 6,600 t each.

Advantages:

- Moderately selective and flexible method;
- Able to follow dip and strike changes fairly well;
- Requires less lateral waste development than other longhole methods as there are no intermediate sublevels in waste; and
- Higher productivity and lower operating cost than CAF which could also be utilized.

Disadvantages:

- Long lead time to develop and prepare the stope to start potential production, with development under survey and/or geology control;
- Lower productivity and higher operating cost per tonne than other longhole methods;
- Requires labour with specialized skills;
- Little mechanization, requiring increased manual labour;
- Possibility of high external dilution, which is controlled in the conceptual mine design by cable bolting of hanging walls in stopes.

The proposed conceptual mining sequence will begin with stope development by driving three or more crosscut drifts into the mineralized zone on 15 m centres and silling out the mineralized zone at the main level elevations (top and bottom of the stope). Once the full strike length is known at the main level elevations, an Alimak service raise will be driven within the mineralized zone from the bottom level to the top level. Sublevels will be collared every 15 m up the raise with the centre crosscut on the bottom level being used as the Alimak nest.

Once the service raise breaks through, the Alimak raise climber will be re-configured to enter from the top of the raise and becomes a means of moving men and materials to the sublevels. The access to the bottom of the raise becomes the drawpoint for removing the sublevel development muck slushed into the raise.

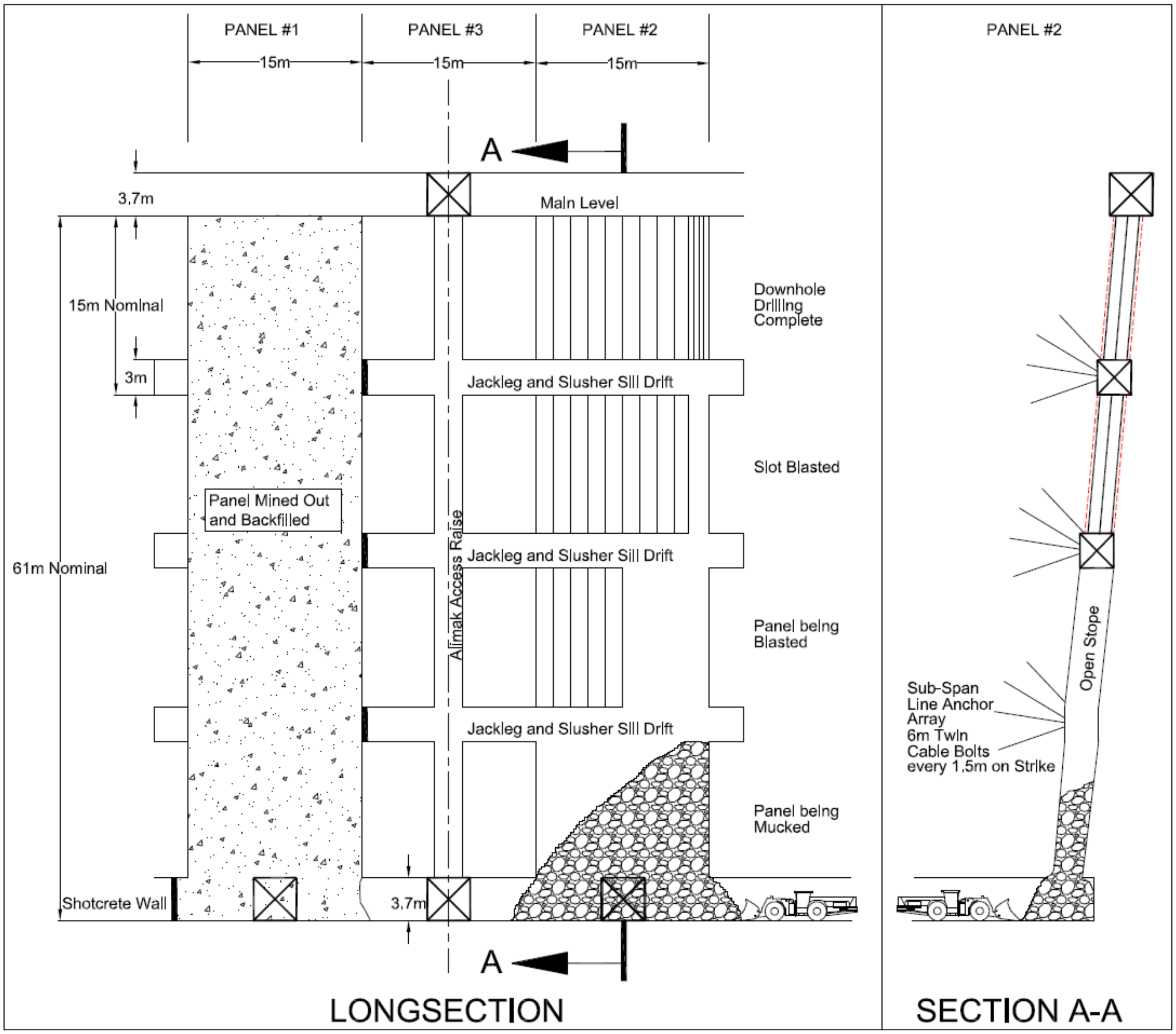


Figure 34: Alimak Access Sublevel Longhole 1.8 to 3.5 m Wide (SRK 2013)

The sublevels are to be driven to the extents of the mineralized zone by jacklegs and slusher using the Alimak raise as a mill hole. Once all three sublevels are developed, a small longhole drill can be moved in for hanging wall cable bolt holes and production drilling. Drilling will retreat from the extremities to the Alimak raise employing 54 to 64 mm (2.125″ to 2.50″) diameter down-holes. Stope blasting will not start until the drilling is complete to minimize the length of time that the hanging wall is exposed. It will begin with the blasting of slot drop raises at each end of the sublevels.

The stopes are organized into two to five panels, each no more than 15 m in strike length. Blasting will start on one end of the bottom sublevel, and progress up until the first panel has been blasted. The mined out panel will then be prepared for backfilling. While Panel #1 is being backfilled using paste fill, blasting will start in Panel #3 on the opposite end panel. Mining will progress until only Panel #3, the centre panel containing the service raise, remains and the rest of the stope has been filled. Blasting Panel #3, the last panel to be blasted, will use the Alimak raise as a slot and use the Alimak raise climber to retreat out of the stope as blasting progresses from the bottom up.

Where practical and economically viable, the stope will be divided into two stopes vertically by driving an additional mucking horizon at the midpoint elevation to reduce the geotechnical risk. Refer to report Section 15.4.3.

15.4.3 Alimak Access Sublevel Longhole Mining Method (3.5 to 5 m Wide)

Approximately 8% of the total stoping tonnes (6% of LoM tonnes) are based on mineralized zones with a horizontal width between 3.5 and 5.0 m wide that are amenable to longhole mining methods. All aspects of this method are the same as described in Section 15.4.2 with the exception that an additional mucking horizon will be developed at the midpoint elevation, thus breaking the stope into two 30 m high stopes with one sublevel each at 15 m. This reduces the geotechnical risk and controls external dilution by reducing the exposed area of hanging wall and footfall at any given time. Dilution is also mitigated by cable bolting of hanging walls which has been incorporated in the mine design. A typical stope will contain approximately 17,000 t of mineral resource with dimensions of 30 m height by 45 m length by 4.25 m width, comprised of three 15-metre-wide panels of 5,600 t each.

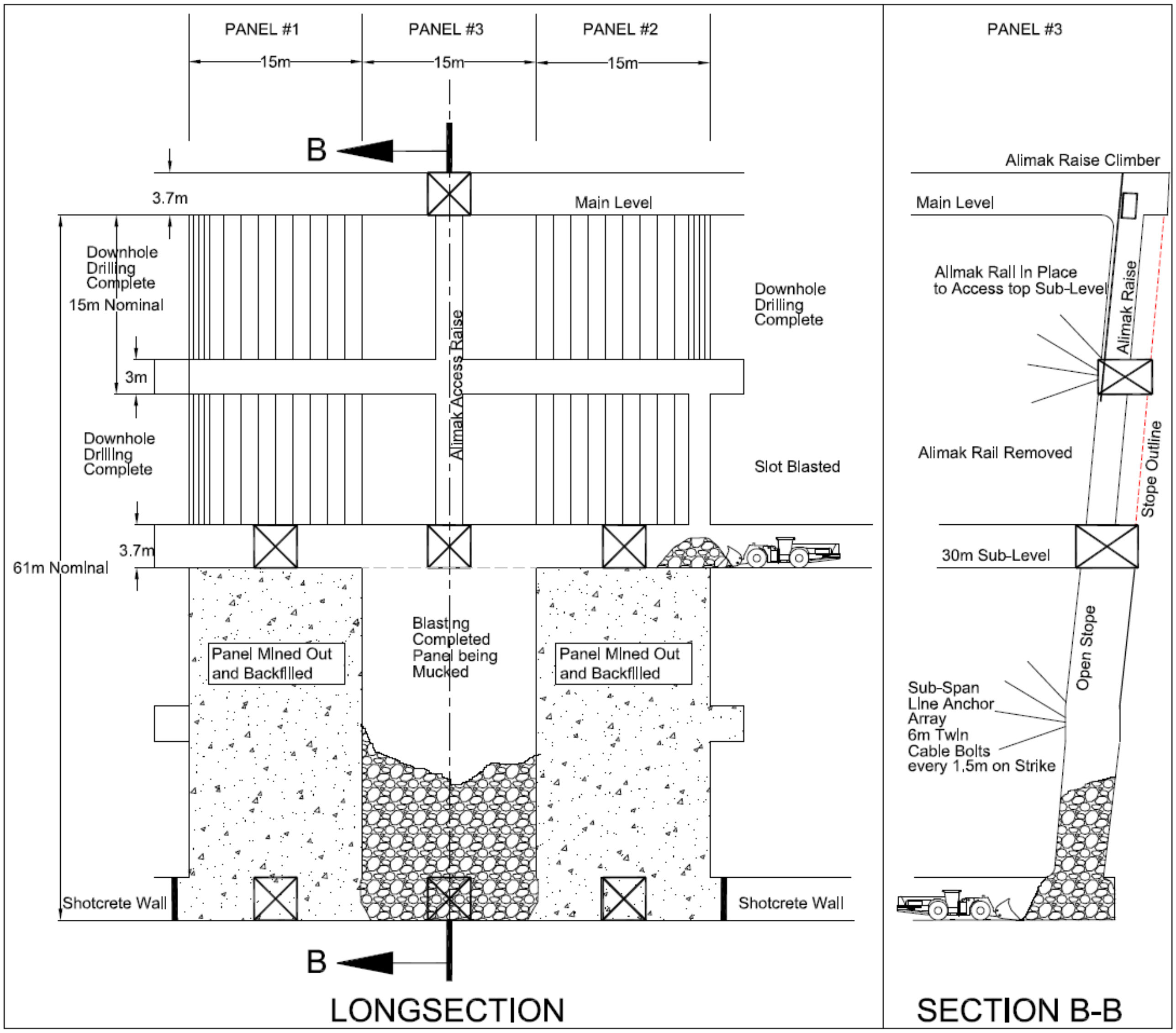


Figure 35: Modified Alimak Access Sublevel Longhole 3.5 to 5 m Wide (SRK 2013)

15.4.4 Longitudinal Retreat Longhole (5 to 10 m Wide)

Approximately 18% of the total stoping tonnes (14% of LoM tonnes) are based on mineralized zones with a horizontal width between 5 and 10 m wide that are amenable to longhole mining methods.

A typical stope will contain between 6,500 and 10,000 t of mineral resource with dimensions of 20 to 30.5 m height by 15 m length by 7.5 m width.

Advantages:

- More productive than cut and fill methods or Alimak access sublevel longhole methods;
- More common, well proven mining method;
- Lower operating costs per tonne; and
- Development proceeds faster as it will be under survey control.

Disadvantages:

- Less selective, but stope grade estimation is more reliable when extracting larger blocks of mineralization; and
- More lateral waste development than Alimak access sublevel longhole (but costs spread over larger tonnage).

The mining sequence will begin by crosscutting the mineralized zone and developing the top and bottom sill drifts at the main level elevations. Intermediate sublevels are also required on 20 to 30 m vertical spacing. These sublevels are best accessed from the ramp system, but can be developed as captive sublevels with dedicated manways and passes in isolated areas. It is envisioned that all development in the mineralized zone will be done under survey control for any stope over 5 m wide.

Depending on width, additional development for slot crosscuts may be required. For the purposes of the mine plan, it has been assumed that a separate slot raise will be required for each stope. The slot raises will typically be drilled and blasted as drop raises.

Once the slot raise has been opened, the remainder of the stope will be drilled off using contractor supplied drills. During production, stope extraction will be retreated to the drawpoint at one end of the stope.

After the stope has been mucked out, a shotcrete fill barricade will be constructed across the drawpoint and the stope filled with pastefill. Rockfill may be added in certain areas to dispose of development waste.

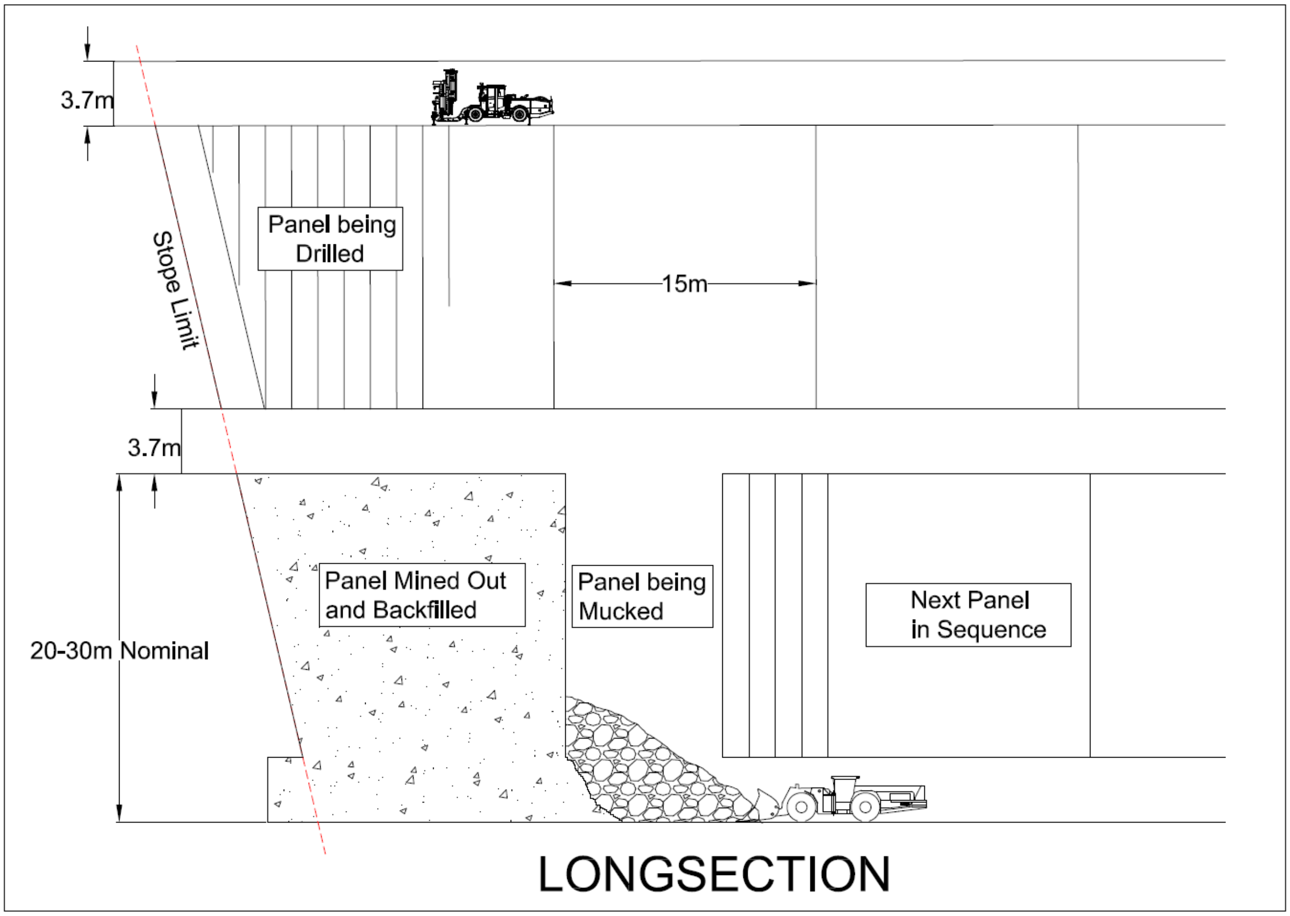


Figure 36: Longitudinal Retreat Longhole Stope 5 to 10 m Wide (SRK 2013)

15.4.5 Transverse Longhole (Greater than 10 m Wide)

Approximately 51% of the total stoping tonnes (41% of LoM tonnes) are based on mineralized zones with a horizontal width greater than 10 m wide that are amenable to longhole mining methods.

A typical stope will contain between 13,000 and 20,000 t of mineral resource with dimensions of 20 to 30.5 m height by 15 m length by 15 m width.

Advantages:

- Most productive of the methods, limited by available load-haul-dump (LHD) size;
- More common, well proven mining method;
- Lowest operating costs per tonne; and
- Development proceeds faster as it will be under survey control.

Disadvantages:

- Less selective, but stope grade estimation is more reliable when extracting larger blocks of mineralization;

- More lateral waste development than Alimak access sublevel longhole (but cost spread over larger tonnage).

The mining sequence will begin by crosscutting the mineralized zone and developing the top and bottom sill drifts at the main level elevations. Intermediate sublevels are also required on 20 to 30 m vertical spacing. These sublevels are best accessed from the ramp system, but can be developed as captive sublevels with dedicated manways and passes in isolated areas. It is envisioned that all development in the mineralized zone will be done under survey control for any stope over 5 m wide.

As the drawpoint will be developed perpendicular to the strike of the mineralized zone, additional development for slot crosscuts will be required. For the purposes of the PEA mine plan, it has been assumed that a separate slot raise will be required for each stope. The slot raises will typically be drilled and blasted as drop raises.

Once the slot raise has been opened, the remainder of the stope will be drilled off using the contractor supplied drills. During production, stope extraction will be retreated to the drawpoint that will be planned in the middle of the hanging wall or footwall of the mineralized zone. Typically the drawpoints are preferred on the footwall side, if possible. After the stope has been mucked out, a shotcrete fill barricade will be constructed across the drawpoint and the stope filled with pastefill. Rockfill may be added in certain areas to dispose of development waste.

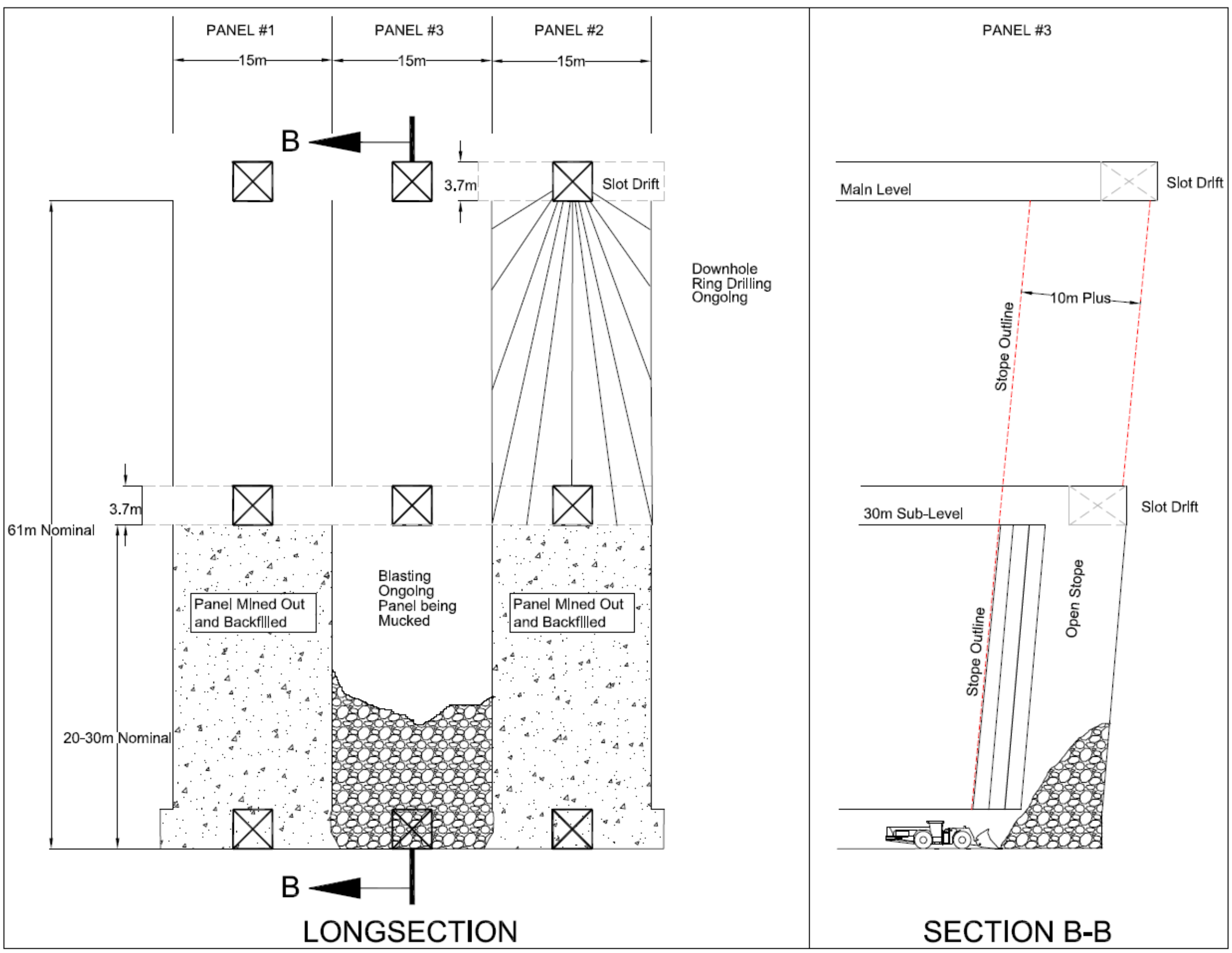


Figure 37: Transverse Longhole Stope Minimum 10 m Wide (SRK 2013)

15.4.6 Uppers Longhole Method

No tonnage has been estimated specifically for using this method, but this method can be used to recover small mineralized zones of no more than 15 m height, and to recover sill pillars and other remnants.

This simple method involves driving a drift along the strike of the mineralized zone, and positioning an inverse (slot) raise at the stope extremity, and production drilling of 15 m up holes at a 70 degree dip. Blasting and mucking will retreat towards the stope entrance. These stopes may or may not be backfilled.

15.4.7 Overhand Cut and Fill Methods

It is envisioned that two main overhand cut and fill methods will be required to mine approximately 10% of the total stope tonnes (8% of LoM tonnes) in areas that are not amenable to longhole mining methods due to irregular shapes, excessively flat or irregular dips, or being less than 1.8 m in horizontal width. These methods have the advantage of being highly selective and flexible, but are also more expensive on a cost per tonne basis due to lower productivities and increased ground support costs compared to longhole methods. The two methods are conventional captive cut and fill (CAF) and mechanized cut and fill (MCF).

The selection of the appropriate method for the situation depends on:

- Width of mineralized zone (CAF can be used at a 1.5 m width, MCF generally needs at least a 2.4 m width, depending on equipment selection); and
- Proximity to ramp system. MCF will require a ramp system located within a reasonable distance to be effective.

Conventional Captive Cut and Fill (CAF)

Variations on this mining method have been used for many years in track and trackless mines.

Advantages:

- Highly selective and flexible method;
- Requires less lateral waste development than MCF; and
- Can mine narrower than MCF or longhole methods if slushers or microscoops are used.

Disadvantages:

- Higher operating cost per tonne than MCF or longhole methods;
- Low productivity;
- Requires labour with specialized skills; and
- Mining under geology control.

The mining sequence will begin by driving one or more crosscut drifts into the mineralized zone and silling out the mineralized zone at the main level elevations (top and bottom). Once the full strike length is known at the main level elevations, a service raise will be driven from the bottom level to the top level. In this case, the service raise will most likely be driven by Alimak raise climber at an incline of 60 to 70 degrees (the length is too great to safely drive an open raise). The service raise will be equipped with a manway, power, compressed air, water, pastefill lines, and a slide compartment for lowering materials into the stope using a tugger hoist located at the top of the raise.

A sublevel will be driven off the service raise to leave a sill pillar above the level and establish a number of manways and boxholes, the quantity based on stope strike length. If required, a sill mat will be installed.

Once the stope infrastructure is established, the mining sequence will begin by drilling and blasting uppers with handheld drills, bolting the back off the muckpile and mucking to the boxholes with scrapers. Once one side is mined out, the boxholes and manways on that side will be raised, a fill wall will be constructed and that side pastefilled while mining continues on the other side of the service raise. Once both sides have been pastefilled, the central area around the service raise will be mined in a similar manner and pastefilled.

The cycle will be repeated until the stope breaks through to the upper level, unless a sill pillar is to be left. The bottom sill pillar can also be extracted after the stope is completed. Due to the lack of mechanized equipment, the stope height will be kept generally around 2.4 to 2.8 m.

Mechanized Cut and Fill (MCF)

Variations on this mining method have become common since the introduction of the LHD.

Advantages:

- Highly selective and flexible method;
- Lower operating cost per tonne than CAF, but generally higher than longhole methods;
- Higher productivity than CAF; and
- Requires same skilled labour as development crews, more flexibility in the labour pool than CAF.

Disadvantages:

- Requires more lateral waste development than CAF, especially if there is no nearby ramp system;
- Cannot mine as narrow as CAF;
- Requires more skilled labour than longhole methods; and
- Mining under geology control is required.

The mining sequence will begin by driving an attack ramp either from a level or from a nearby ramp. The attack ramp will generally be driven at a -15% gradient to access the bottom (sill) cut of the mineralized zone near the centre of the stope mass using the same development equipment as that used for ramp and level development. The mineralized zone will be developed with sill drifts to the extents of the mineralization. A sill mat will be installed, if required, prior to backfilling with pastefill or rockfill if available from the nearby development headings.

After backfilling is complete, a section of the attack ramp will be back slashed and rebolted to gain elevation for access to the next cut. The waste rock broken while doing this will be generally left in place or stored nearby to provide a road bed in the ramp and rockfill for the next cut. The cycle will be repeated until the designed number of cuts has been mined. Mining continues upward by repeating the process from a new attack ramp to access the mineralized zone at the next higher elevation.

Stope heights will generally be 3.5 m to suit the same equipment that was used for ramp and level development if the mineralized zone is wide enough. For stopes between 2.4 and 3.0 m wide, a 3.5-tonne LHD (1.5yd to 2yd) will be used.

15.5 Estimate of Potential Mineable Mineralization

15.5.1 Introduction

The portion of the Indicated mineral resources and Inferred mineral resources above the cut-off grade, including allowances for dilution and mining loss and used for evaluation purposes, is referred to as potential mineable mineralization in this report. SRK cautions that potential mineable mineralization quantities cannot be considered mineral reserves. The potential mineable mineralization quantity estimates presented in this conceptual study are based partly on Inferred mineral resources that do not support feasibility or preliminary-feasibility level work or estimates of mineral reserves. Inferred mineral resources are estimated 62% of total potentially mineable mineralization quantity estimates. There is no guarantee that further exploration will upgrade any part of the Inferred mineral resources.

The projected mining method, potential production profile and plan and mine plan are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks which include, but are not limited to, the inclusion of Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves.

This section describes the methodology used by SRK in estimating the tonnes and grade of the potential mineable mineralization and the practical targeted potential production rates for the Phoenix gold project.

SRK applied preliminary economic assessment level mine planning methods to the mineral resource block models. The metal of economic interest for this property is exclusively gold. SKR included Indicated and Inferred mineral resources in the potential underground production plans. No Measured mineral resources are defined at this time.

The mine plan is based on the completion of a representative amount of detailed stope and development design. The mine planning results obtained from areas of detailed design have been used to estimate results for the remaining areas of mineralization with similar grade and geometry characteristics.

15.5.2 Cut-Off Grade Estimate

An estimated site operating cost is required as an input to the conceptual mine planning economic cut-off criteria. As there are no actual costs or budget estimates available, site operating costs were factored considering the 2011 preliminary economic assessment and results from comparable mining operations.

The initial conceptual mine design cut-off grade was estimated at 4.0 gpt gold based on the following calculation, rounding up to the nearest gram per tonne.

The cut-off grade estimation shown in Table 38 is based on the following assumptions:

- Initial production rate estimate of 1,800 tpd;
- Underground mining costs derived from similar mining operations;

- Milling and G&A costs were provided by Rubicon and adjusted for higher production rate;
- Gold price based on US$1,500;
- 1.5% royalty assuming 0.5% of 2.0% royalty on property would be repurchased prior to production as allowed for in the royalty agreement;
- Processing recovery of 92.5% based on past recommendations by Soutex;
- Assumed 15% external dilution at a grade of 0.25 gpt gold;
- See Section 15.5.7 for details of external dilution; and
- Internal dilution is included within mining shapes using 3D mine design techniques.

Much of the initial conceptual mine design was done based on the 4 gpt mine design cut-off grade. The first assessment of project economics revealed that the design assumptions would not provide the profit margin Rubicon required from the project.

After further analysis and testing of different options, it was decided to increase the initial mine design cut-off grade to 5 gpt for the whole deposit. Therefore, the potential production profile and preliminary economic analysis presented in this report are based on a conceptual mine design cut-off grade of 5 gpt gold.

Table 38: Initial Cut-Off Grade Estimate

Parameters	Unit			COG Estimate
Production rate estimate	tpd			1,800
U/G mining	US$/t			$89.60
Processing & tailings	US$/t			$28.12
G&A	US$/t			$32.22
Site total operating cost/tonne milled	US$/t			**$149.94**
Gold price	US$/oz			$1,500
Au payable	%			99.9%
Au refining	US$/oz			$3.00
Royalty	%			1.5%
Value of Au in doré	US$/oz			$1,496
Value of Au in doré	US$/gram			48.08
Process recovery				92.5%
Value of Au in plant feed	US$/gram			44.48
Plant feed grade needed (diluted)	**Au gpt**			**3.37**
External dilution		**%**	**Au gpt**	
External dilution at grade		15.0%	0.25	0.25
Inside mining shapes COG Au gpt				**3.84**

15.5.3 Application of Cut-off Grade and Creation of Mining Shapes

To begin the mine design process, the four block models received were validated and interrogated to determine the tonnes and grade by mineral resource classification available in each of the defined main level intervals at a 4 gpt cut-off grade as shown in Table 39.

The HW zone is of no significant value to the preliminary economic assessment as most of the mineral resources are located in the crown pillar. The HW zone exhibits lower grades and it is relatively close to the historical workings. Therefore, the HW zone was not included in any mine design work.

Table 39: Mineral Resources by Level, Block Model and Classification at 4 gpt Cut-off Grade

	Main Zone Indicated			Main Zone Inferred			45 Zone Indicated			HW Zone Inferred			Level 2 Inferred			Total All Zones		
Level	**Au (gpt)**	**Tonnes**	**Contained Oz Au**	**Au (gpt)**	**Tonnes**	**Contained Oz Au**	**Au (gpt)**	**Tonnes**	**Contained Oz Au**	**Au (gpt)**	**Tonnes**	**Contained Oz Au**	**Au (gpt)**	**Tonnes**	**Contained Oz Au**	**Au (gpt)**	**Tonnes**	**Contained Oz Au**
Crown Pillar																		
183 level	15.52	92,151	45,972	9.00	213,755	61,865	4.36	2,458	345	4.40	17,051	2,411				10.57	325,415	110,593
244 level	9.19	122,891	36,298	6.96	188,295	42,159	8.17	1,777	467				4.40	12,565	1,778	7.71	325,528	80,702
305 level	8.29	142,431	37,971	8.59	253,730	70,066	8.07	18,146	4,705							8.46	414,308	112,742
366 level	9.34	166,275	49,941	5.96	142,952	27,392	5.99	28,405	5,471				8.25	20,572	5,456	7.66	358,204	88,260
427 level	8.55	200,192	55,050	8.48	217,499	59,278	6.49	97,821	20,405				9.36	15,233	4,585	8.16	530,744	139,317
488 level	8.76	150,662	42,418	7.81	255,011	64,041	7.35	146,583	34,625				11.86	162,259	61,871	8.83	714,515	202,954
549 level	8.72	150,916	42,315	8.50	253,512	69,288	5.70	49,957	9,157				10.09	195,249	63,339	8.81	649,634	184,098
610 level	10.54	106,888	36,221	9.39	221,888	67,016	5.92	18,639	3,545				8.48	244	66	9.56	347,658	106,847
671 level	9.74	208,947	65,438	6.83	252,268	55,428	5.80	68,926	12,844							7.84	530,141	133,710
732 level	8.64	240,722	66,830	5.36	284,873	49,101	6.66	55,402	11,861							6.84	580,997	127,791
793 level	8.56	232,295	63,908	5.30	183,074	31,219	5.02	9,901	1,599							7.07	425,270	96,726
854 level	9.40	217,793	65,793	6.29	208,062	42,083										7.88	425,855	107,875
915 level	9.82	232,761	73,517	7.29	222,366	52,089							6.67	44,198	9,477	8.41	499,325	135,083
976 level	9.64	377,058	116,814	8.66	199,591	55,590							7.60	245,365	59,930	8.79	822,014	232,335
1037 level	7.70	337,965	83,678	9.48	163,777	49,901	4.65	5,505	823				8.71	419,508	117,436	8.45	926,755	251,837
1098 level	6.03	333,793	64,680	11.35	231,951	84,611	4.74	38,526	5,872				6.82	96,724	21,193	7.83	700,994	176,357
1159 level	6.19	318,152	63,327	12.56	214,026	86,419	4.72	25,020	3,794							8.57	557,198	153,540
1220 level	8.46	172,958	47,016	7.63	80,001	19,625	5.27	17,199	2,912							8.01	270,158	69,553
1281 level	8.88	102,065	29,152	5.74	62,129	11,464							4.80	35,316	5,453	7.18	199,509	46,069
1342 level	11.45	139,964	51,538	17.33	227,901	126,965	4.39	1,181	167				4.83	22,381	3,475	14.47	391,426	182,145
1403 level	12.23	93,999	36,961	17.39	519,622	290,588										16.60	613,621	327,549
1464 level	13.22	7,726	3,284	11.57	220,981	82,173										11.62	228,707	85,457
1525 level				7.62	51,700	12,664							5.49	61	11	7.62	51,761	12,675
1586 level				7.64	9,682	2,377	4.71	6,030	914				5.49	6,540	1,155	6.21	22,252	4,445
Below 1586 level																		
Grand Total	**8.83**	**4,148,604**	**1,178,120**	**9.65**	**4,878,644**	**1,513,402**	**6.28**	**591,477**	**119,504**	**4.40**	**17,051**	**2,411**	**8.66**	**1,276,214**	**355,224**	**9.03**	**10,911,990**	**3,168,662**

The mineral resources contained within the crown pillar were also excluded from the mine design (Table 40). The mineral resources contained within the crown pillar are reported at a 4 gpt cut-off grade and assume the crown pillar has a flat bottom at 5,249 m elevation, which corresponds to the 122 level.

Table 40: Crown Pillar Mineral Resources by Block Model and Classification at 4 gpt Cut-Off Grade

Zone	Class	Gold (gpt)	Quantity (tonnes)	Contained Gold (ounces)
Main	Indicated	15.21	32,228	
	Inferred	11.12	639,162	
45 Zone	Indicated			
	Inferred	4.62	2,565	
HW Zone	Indicated			
	Inferred	5.31	134,242	
Level 2	Indicated			
	Inferred			
Total	**Indicated**	**15.21**	**32,228**	**15,761**
	Inferred	**10.09**	**775,969**	**251,812**

Initially stope designs were completed for the 549, 305, 244, and 183 levels at a 4 gpt mine design cut-off grade using industry standard 3D mine design software and procedures. These level intervals were chosen as these areas had the greatest density of information from diamond drilling, chip sampling (305 level only), and the bulk samples taken on the 305 level.

A minimum horizontal mining width of 1.8 m was applied during the mine design process.

During optimization of the stope designs, it was decided to not update the 549 level stope designs and therefore this level was not considered further. An analysis of the impact using the unoptimized stope designs indicated that the results would change by only 0.4% in terms of potential recovered ounces. During the optimization process, any stope designs that did not meet the 4gpt cut-off grade criteria were removed from the stope evaluation results.

After the stope designs for the 183 level, the 244 and 305 levels were optimized, the stope evaluation results (in situ stope tonnes and grades) were compared to the available mineral resources above cut-off on those levels to calculate a tonnage factor and grade factor for each block model by level. The results are summarized in Table 41.

Table 41: Tonnage and Grade Factors by Level at 4 gpt Cut-off Grade

Level	Main Zone		45 Zone		Level 2	
	Indicated + Inferred		Inferred		Inferred	
	Grade Factor	Tonnage Factor	Grade Factor	Tonnage Factor	Grade Factor	Tonnage Factor
183 level	92%	96%	0%	0%		
244 level	89%	98%	62%	129%	100%	84%
305 level	85%	93%	70%	142%		
Average	**89%**	**95%**	**73%**	**125%**	**100%**	**84%**
	Oz factor	**84%**	**Oz factor**	**91%**	**Oz factor**	**83%**

The average weighted value of these factors was then used to estimate the in situ stope tonnes and grade available from each block model by level that could be included in the LoM plan for all levels that did not have detailed stope designs completed.

For example, to determine the stope tonnes (inside mining shapes) available from the Main Zone on the 1220 level, multiply the sum of the Indicated tonnes (172,958t) and the Inferred tonnes (80,001t) (above cut-off) available on the 1220 level (see Table 39) by the average tonnage factor for the Main Zone (95%, see Table 41) to arrive at the potentially mineable stope tonnes (before external dilution) for the Main Zone on the 1220 level. The process used to estimate the grade of the potentially mineable stope tonnes for the Main Zone on the 1220 level is identical.

Note that the tonnage factors and grade factors are not the same as a simple conversion of mineral resources, as the stope evaluation results include internal dilution unavoidably included within the mining shapes.

After the first assessment of the project economics was completed using the new shaft scenario, SRK was requested to examine a series of scenarios including various applications of the cut-off grade including:

- New shaft scenario - Front loading the first five years of production with stopes having a grade in excess of 5 gpt and coming back later to mine the 4 to 5 gpt stopes. This was an exercise in sequencing the stopes designed at the 4 gpt cut-off grade;
- Deepen existing shaft scenario - Front loading the first five years of production with stopes having a grade in excess of 5 gpt and coming back later to mine the 4 to 5 gpt stopes. This was an exercise in sequencing the stopes designed at the 4 gpt cut-off grade;
- Deepen existing shaft scenario - Front loading the first six years of production with stopes having a grade in excess of 6 gpt and coming back later to mine the 4 to 6 gpt stopes. This was an exercise in sequencing the stopes designed at the 4 gpt cut-off grade; and
- Deepen existing shaft scenario – Apply 5 gpt design cut-off grade to the entire mine life.

The last scenario was selected to be the basis for the preliminary economic assessment LoM plan and economic evaluation.

If order to adjust the in situ tonnes and grade estimated using the 4 gpt design cut-off grade, the 183 level stopes were redesigned at a 5 gpt cut-off grade. Comparing the stope evaluation results for the 183 level at 4 gpt cut-off grade vs 5 gpt cut-off grade yielded the following conversion factors:

- Tonnage factor (4 to 5 gpt) = 80%
- Grade factor (4 to 5 gpt) = 114%
- Ounce factor (4 to 5 gpt) = 92%

Therefore, the stope designs prepared using a 5 gpt cut-off grade will contain 80% of the tonnes, contain 92% of the ounces and have a 14% higher grade than the same designs done using a 4 gpt cut-off grade.

In order to check that this estimate was reasonable, two other methods were evaluated, based on a global comparison of the Mineral Resource Statement data at 4 and 5 gpt cut-off grades. These both arrived at the same values by different means. The results of these two global comparison estimates were similar to the stope designed based adjustment estimate with 2% higher grade and 6% less tonnes, so the factors derived from the 183 level design work were deemed to be reasonable.

15.5.4 3D Block Models

The following figures illustrate the mineral resource block models that are the basis of the potential mineable mineralization. There are four block models for the project: Main Zone, 45 Zone, HW Zone and Level 2. Figure 38 to Figure 41 show all blocks in each of the four block models greater than the 4 gpt mineral resource cut-off grade. The existing shaft and levels are shown in the figures for reference. The cyan blocks are between 4 and 5 gpt (part of mineral resource, but below mine design cut-off grade), the yellow blocks are between 5 and 10 gpt, and the red blocks are greater than 10 gpt. More than 90% of the mineral resources used to create the LoM plan are contained within the Main Zone. In contrast, the HW Zone is characterized by:

- Lower average grade and maximum grade of 6.96 gpt;
- Mineralization scattered over a very large area; and
- Not close to other block models - most of the HW Zone is several hundred metres from the other mineral zones.

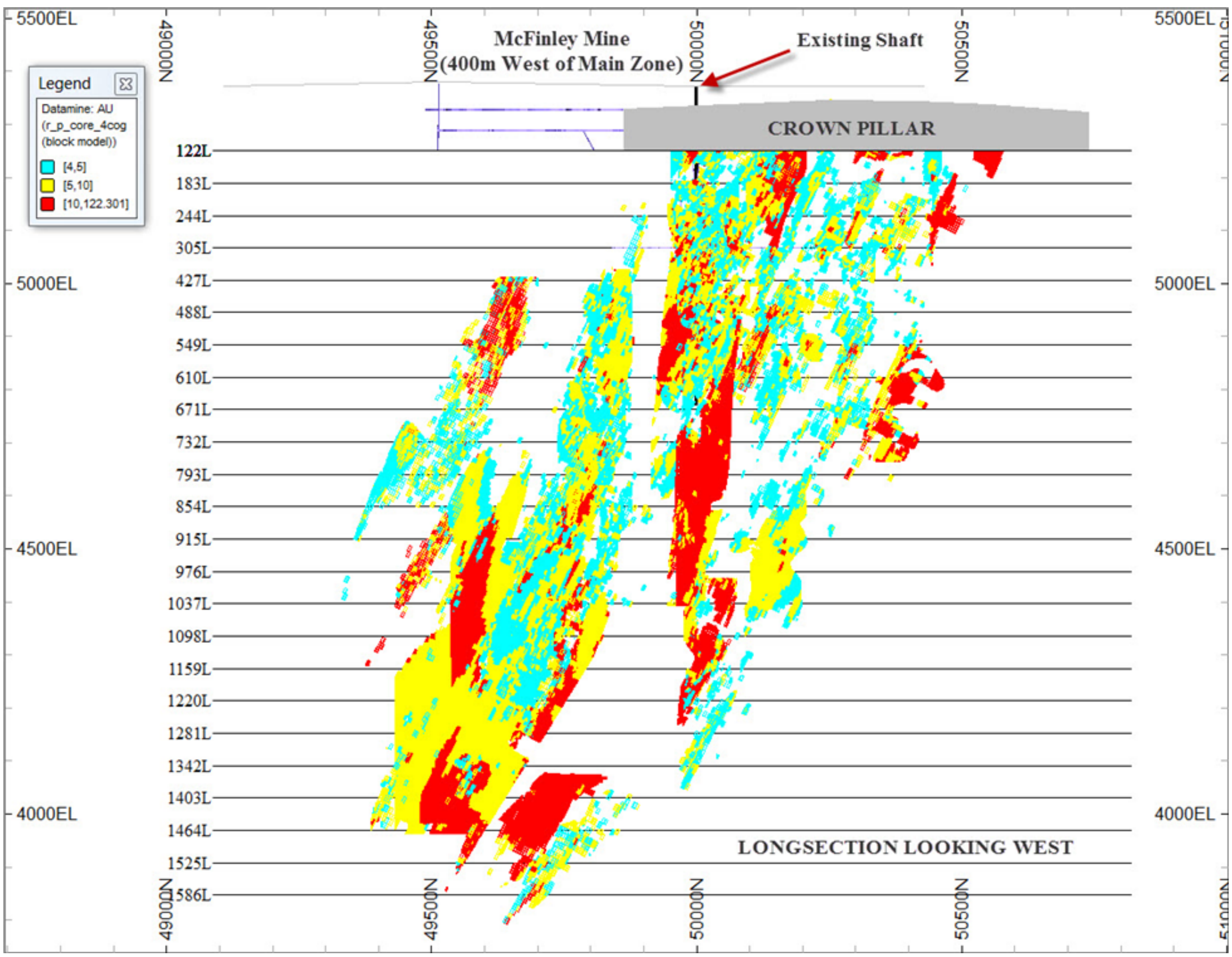


Figure 38: Longitudinal View of Main Zone Block Model (SRK 2013)

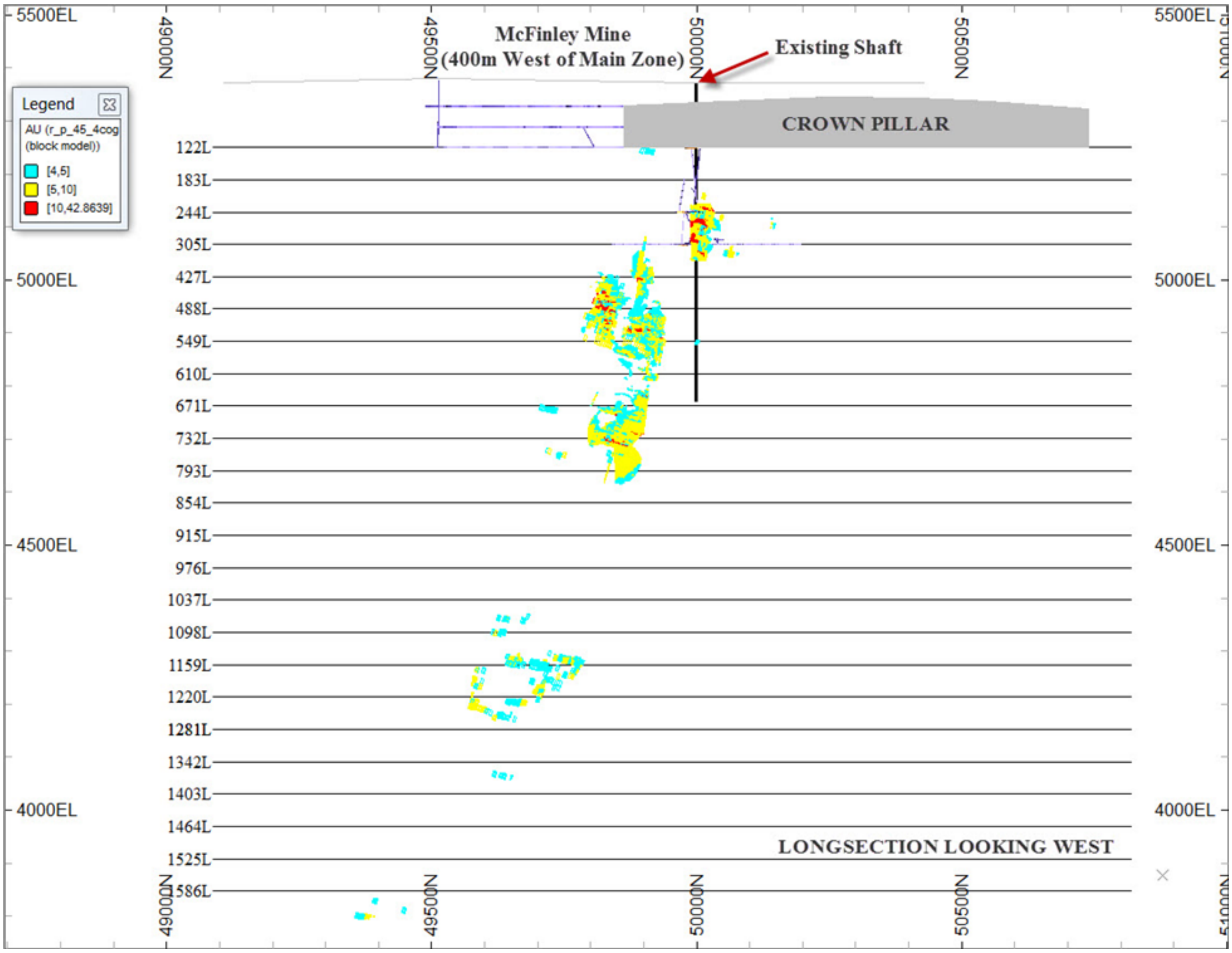


Figure 39: Longitudinal View of 45 Zone Block Model (SRK 2013)

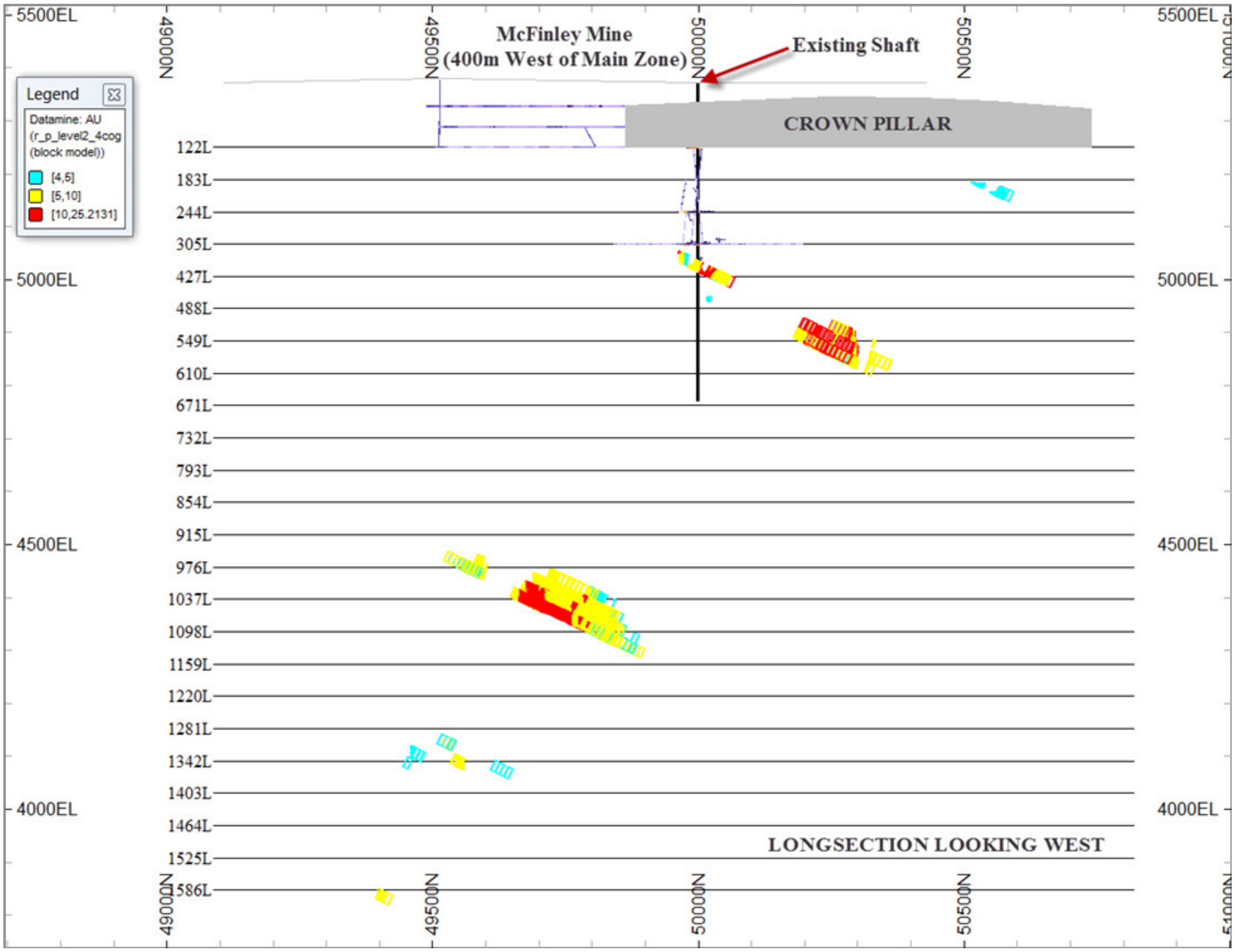


Figure 40: Longitudinal View of Level 2 Block Model (SRK 2013)

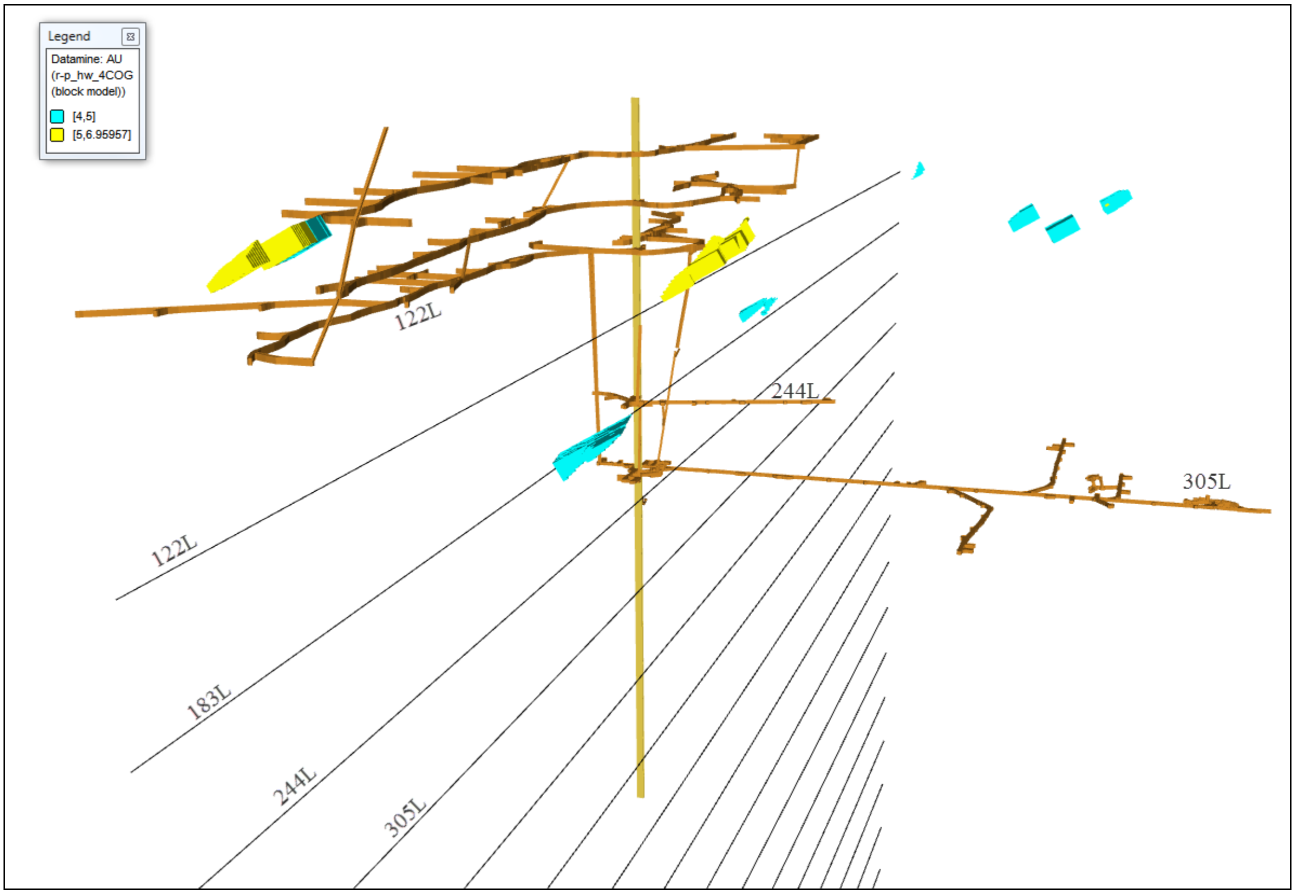


Figure 41: Isometric View of HW Block Model (Looking NNW) (SRK 2013)

15.5.5 Internal Dilution

The internal dilution was determined based on the detailed stope designs. It is comprised of the below cut-off grade material unavoidably included within the designed stope mining shapes.

The inclusion of below cut-off grade material results from:

- Mixing of above and below cut-off grade resource blocks inherent in the resource block model. Where grade continuity above cut-off is weak, internal dilution will be higher; and
- The requirement that the mining shapes designed represent stopes that are practical and workable.

For longhole stopes, the latter factor includes:

- Smoothing wall profiles to ensure the stope can be properly drilled and blasted; and
- Smoothing footwall and endwalls so that the broken muck will flow to the drawpoints.

For cut and fill stopes, this includes:

- Slashing corners and walls so the equipment can move around, especially in very narrow areas: and
- Openings for manways and mill holes in waste.

It is estimated that 26% of the potential mineable stope material is below the cut-off grade and that the average grade of this internal dilution is 1.81 gpt gold.

15.5.6 External Dilution

External dilution of the production material can be caused by overbreak of stope walls, blasting barren material, and uncontrolled sloughing of stope walls into active stopes. This external material mixes with the production material in the stope and mixed material is sent to the mill. The source of external dilution is typically subgrade material from the stope walls and backfill from the floor of both longhole and cut and fill stopes.

External dilution was applied to the stope potentially mineable material at 15% for all levels. The dilution percentage is defined as tonnes of dilution material (W) divided by tonnes of stope material (O):

Dilution % = W/O x 100.

The external dilution was estimated using the ELOS method and was based on geotechnical data collected by SRK geotechnical engineers from inspection of the 305 level, selected diamond drill core, and core photographs. The amount of geotechnical data currently available is limited.

A preliminary assessment was made for each of the mining methods described in Section 15.4 Planned Mining Methods. The resulting external dilution estimate for the individual mining methods was weighted by the potential mineable mineralization planned by each method. The resulting weighted average external dilution factor is 15%.

The average grade of the external dilution was estimated by modelling a series of 3D shapes approximately 1 m thick on the hanging wall and/or footwall of a representative number of stopes in the areas where sloughing can be expected to occur. These shapes were then evaluated in the same manner as the stope designs to estimate the tonnes and grade contained within the shapes.

External dilution grade was originally estimated to be 0.57 gpt gold using the 4 gpt cut-off grade stope designs. To estimate the average grade of dilution for the 5 gpt cut-off grade stope designs, SRK applied a 20% increase to 0.68 gpt gold.

15.5.7 Mining Recovery

A mining recovery of 95% of diluted broken tonnes has been assumed for the mining plan. This value is a recognized industry standard where no production data is available.

15.5.8 Process Recovery

A processing recovery value of 92.5% was estimated by Soutex (Section 12.3).

15.6 Estimation of Life-of-Mine Development Requirement

This section provides a description of the methodology used by SRK in estimating LoM lateral and vertical development requirements.

15.6.1 Development Types and Advance Rates

Development planning and scheduling parameters for the three types of lateral development included in the LoM plan are listed below:

- Track development:
  - Used for primary access on 122 level, 183 level and to complete 244 level;
  - Used to connect to the south return air raise as quickly as possible;
  - 305 level is already developed to mineralized zone as track drift;
  - 610 level will also be driven as track drift due to limited ventilation available;
  - Planned dimensions are 2.75 m width by 3.35 m height;
  - Single face advance rate of 3.6 m per day;
  - All track development is capitalized; and
  - Using standard track development equipment, most of which is already on site.

- Trackless development from ramp or level:
  - Used for everything on the main levels that is not track development;
  - Used for ramps and sublevels that connect to the ramps or main levels:
  - Trackless development off track drift will use 6.7-tonne LHDs to load rail cars (305 level to 122 level mainly);
  - Main levels and ramps will use 6.7-tonne LHDs and 20-tonne trucks to haul to mineralization and waste pass systems;
  - Planned dimensions are 3.7 m width by 3.7 m height;
  - Multi face advance rate of 5.5 m per day;
  - Can be capital development or operating development; and
  - Using standard trackless equipment, except that part of the fleet is specified as being battery powered LHDs and trucks. Testing of the battery powered equipment is required to verify its suitability.

- Captive trackless development:
  - Used for captive sublevels accessed by Alimak raise or independent manways and passes;
  - Used for areas where mineralized material bottoms out above or below a level and the area is too far from the ramp system;
  - Equipment can range from jackleg and slusher to longtom and 6.7-tonne LHDs depending on the situation;
  - LHDs can be tethered electric; and
  - Average advance rates range from 1.6 m to 3.8 m per day.

It is envisioned that there will be more track development scheduled early on, transitioning to all trackless development by the end of 2015.

Development planning and scheduling parameters for the two types of vertical development included in the LoM plan are listed below:

- Alimak raising:
  - The majority of the raising in the LoM plan is planned as Alimak raising at various sizes. They are used for all level to level raises for production, all return air raises and passes;
  - An advance rate of 3.6 m per day was used for raises from 2.4 m by 2.4 m to 3 m by 3 m in size; and
  - A two pass Alimak raise method (pilot raise and slash out) is specified for the larger fresh air raises (6 m by 3 m) from surface down. An overall advance rate of 1.8 m per day was used.

- Open raise:
  - Conventional open raise was specified only for relatively short raises such as manways and millholes for cut and fill stopes and captive sublevels;
  - Used the same 3.6 m per day advance rate as for Alimak raising; and
  - Open raises are generally 2.4 m width by 1.8 m height and assumed to be driven at a dip of 49.5 degrees.

15.6.2 Development Factors

In order to estimate development requirements, detailed level development layout work was done in 3D for the 305 and 244 levels as illustrated in Figure 42. Once the development design was completed, the centre line lengths were organized into 12 groups based on capital or operating, lateral or vertical, mineralized material or waste, captive or non-captive.

Capital development was estimated in the following manner:

- All capital track drifts were designed and measured by level;
- All capital lateral level development for the 610 level and above was designed and measured, then 10% was added to account for miscellaneous openings such as remuck bays, storage areas, etc.;
- All capital lateral development below the 610 level was estimated based on above and varied by strike length;
- All capital ramp development above the 305 level was designed and measured;
- All capital ramp development below the 305 level was added at a rate of 526 m per main level interval;
- Most capital raises above the 610 level were designed and measured, this includes fresh air raise, return air raises, and some O/W pass systems. The remainder was estimated;
- All capital raises below the 610 level were estimated, this includes fresh air raise, return air raises and material passes;
- With the numerous Alimak raises present it was assumed that at least one operating raise per level could be utilized as an emergency manway; and
- These values did not change with the change in mine design cut-off grade and associated block models.

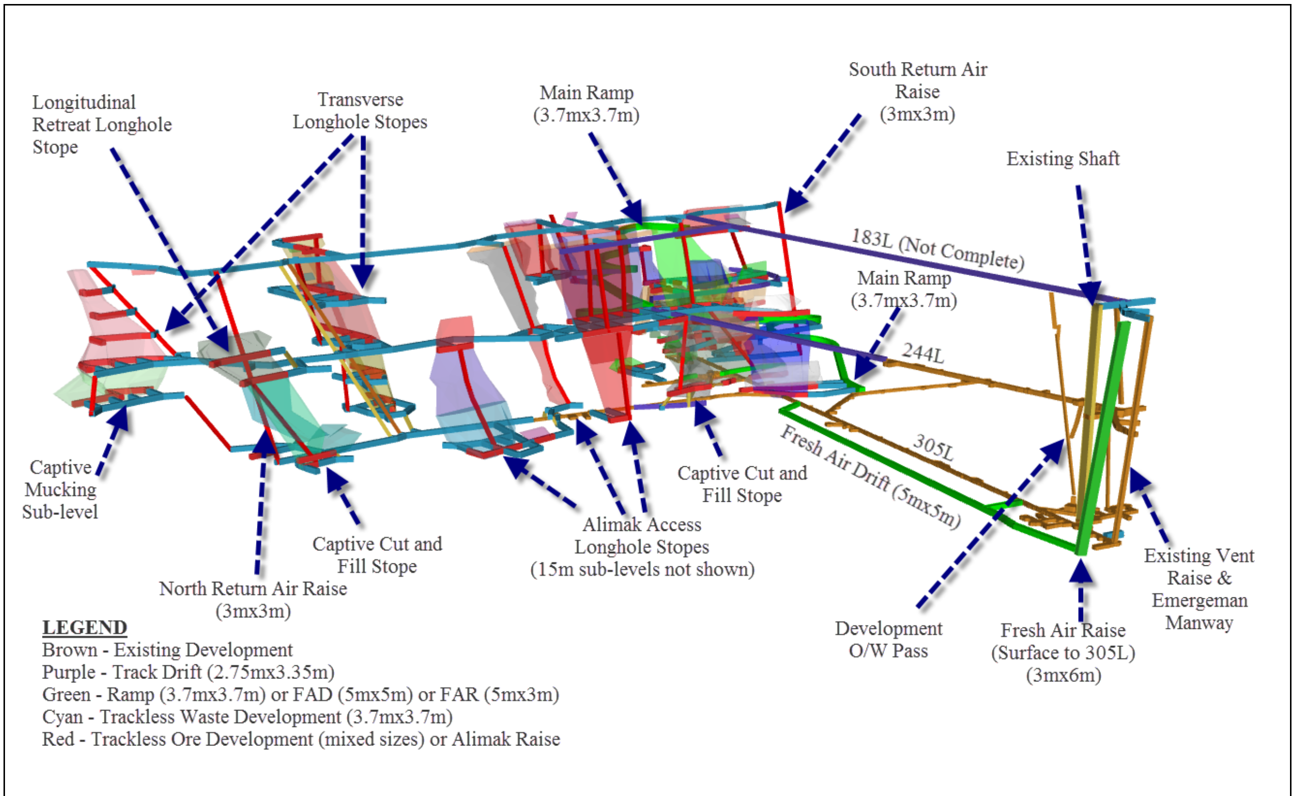


Figure 42: Isometric View of 305 Level & 244 Level Development (Looking Southeast) (SRK 2013)

Operating development was estimated in the following manner:

- The sum of the operating lateral metres of development for each type of development designed and/or estimated for the 244 and 305 levels was divided by the potential mineable mineralization available on the two levels;
- The same process was repeated for operating vertical development;
- This was expressed as a factor in metres per 1,000 tonnes milled;
- The potential mineable mineralization for each level interval was then multiplied by these factors and divided by 1,000 to estimate the operating development required on each level; and
- The factors did not change when the mine design cut-off grade and block models changed. The potential mineable mineralization changed, however the operating development also changed by the same factor as the potential mineable mineralization.

Table 42 below shows the estimated operating development factors discussed above.

Table 42: Operating Development Factors

Waste Development				Mineralized Development			
Lateral Development		Vertical Development		Lateral Development		Vertical Development	
Level	Captive	Alimak Rse	Open Rse	Level	Captive	Alimak Rse	Open Rse
(3.7mx3.7m) (m/1000t)	(3.7mx3.7m) (m/1000t)	(2.4mx2.4m) (m/1000t)	(2.4mx1.8m) (m/1000t)	(3.7mx3.7m) (m/1000t)	(2.4mx3m) (m/1000t)	(2.4mx2.4m) (m/1000t)	(2.4mx1.8m) (m/1000t)
3.80	**0.86**	**0.45**	**0.17**	**2.57**	**2.50**	**1.46**	**0.08**

15.7 Life-of-Mine Plan

The projected mining method, potential production profile and plan and mine plan referred to in this preliminary economic assessment are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks which include, but are not limited to, the inclusion of Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets and process plant designs may require additional detailed work and economic analysis and internal studies to ensure satisfactory operational conditions and decisions regarding future targeted production.

15.7.1 Conceptual Development Plan and Production Phases

The Phoenix mine is planned to be developed in five phases as illustrated in Figure 43.

Phase I and Phase II focus on establishing the infrastructure and work areas required to reach commercial production. Phase III provides the tonnage necessary to sustain this rate during the second phase of shaft deepening. Phase IV begins with the completion of the deepened shaft and increases the production capacity by providing additional stoping areas. Phase V sustains the production rate until the known mineral resources are depleted in 2027.

15.7.2 Life-of-Mine Conceptual Production Profile

Table 43 shows the planned production schedule to supply mill feed that totals 9.13 million tonnes from the start of production in H2 2014 through to Q3 2027. Before the mill is proposed to be fully commissioned in Q3 2014, some 24,000 tonnes of mineralized material are expected to be stockpiled on surface for processing during the mill commissioning period.

15.7.3 Life-of-Mine Conceptual Development Plan

Table 44 shows the total capital waste development scheduled for the Phoenix mine included in the production schedule. The total is 42.5 kilometres (km) of capital lateral and vertical development over the projected life of the mine.

Table 45 shows the total operating development scheduled for the Phoenix mine included in the production schedule. The total is 107.1 km of operating lateral and vertical development over the projected life of the mine.

Over the life of the mine, 1 m of lateral development is required for every 75.5 tonnes milled.

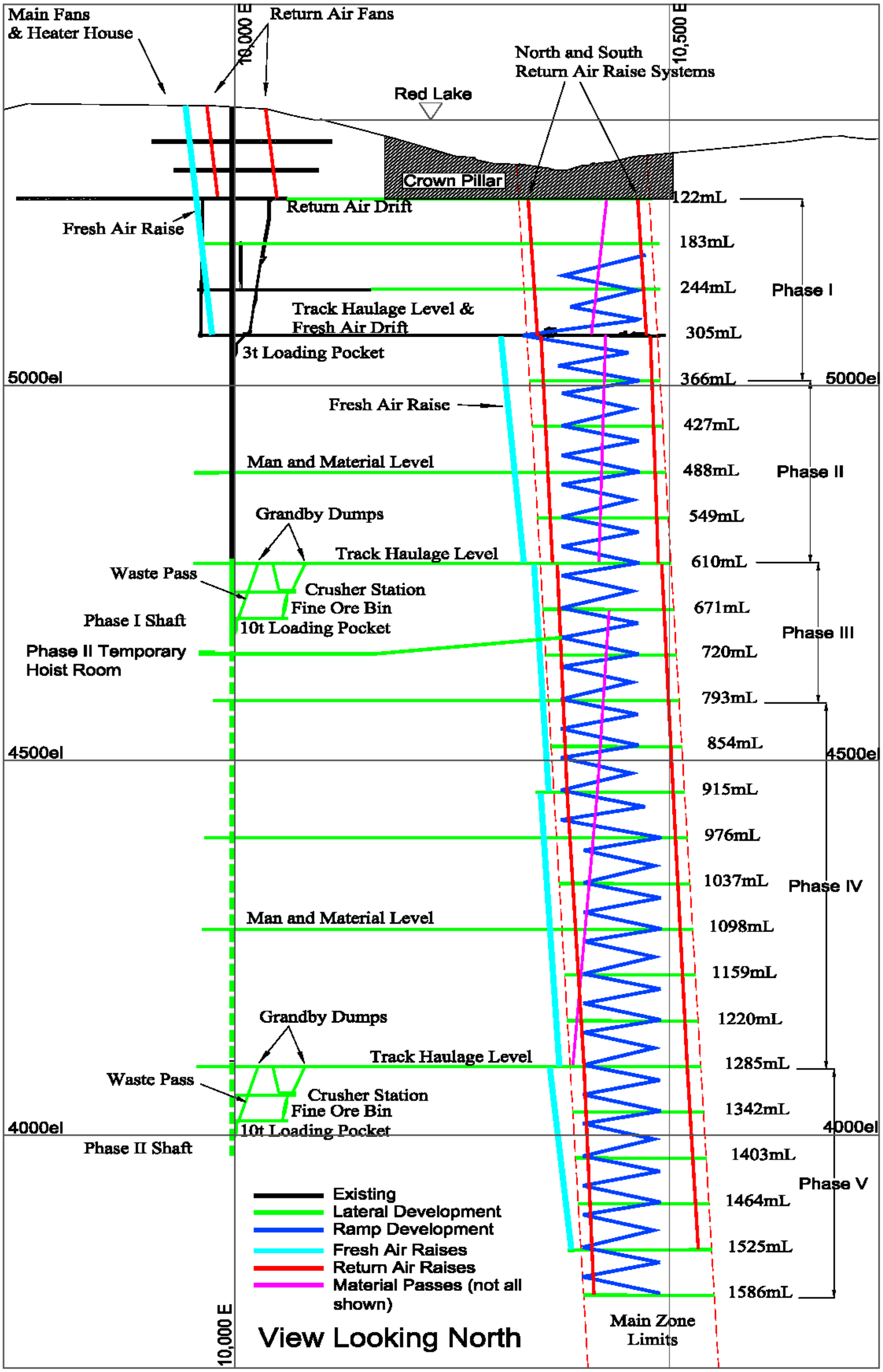


Figure 43: Mine Development Phases (SRK 2013)

Table 43: Life-of-Mine Potential Production Profile*

	Year 2013	Year 2014	Year 2015	Year 2016	Year 2017	Year 2018	Year 2019	Year 2020	Year 2021	Year 2022	Year 2023	Year 2024	Year 2025	Year 2026	Year 2027	Total
Stope muck (tonnes)		99,668	469,173	518,755	579,276	565,982	567,656	597,583	598,975	652,118	669,042	648,718	657,746	465,574	288,330	**7,378,596**
Grade (gpt)		6.94	7.45	8.02	8.04	7.54	6.79	6.86	9.85	10.16	8.28	7.61	8.04	7.87	8.03	**8.06**
Stoping (ounces)		22,242	112,446	133,737	149,795	137,276	123,960	131,820	189,703	212,938	178,177	158,697	169,980	117,814	74,483	**1,913,067**
Development muck (tonnes)		77,989	143,552	126,875	107,754	137,332	133,822	154,702	159,700	157,772	140,834	160,983	151,544	92,860	7,610	**1,753,330**
Grade (gpt)		7.25	7.81	8.11	7.73	7.18	6.74	8.06	10.10	9.56	7.54	7.90	7.93	7.93	8.12	**8.06**
Development (ounces)		18,171	36,047	33,097	26,768	31,715	29,010	40,082	51,865	48,491	34,156	40,879	38,636	23,687	1,987	**454,591**
Total mill feed (tonnes)		177,657	612,725	645,629	687,030	703,314	701,478	752,285	758,674	809,890	809,877	809,701	809,290	558,435	295,940	**9,131,926**
Grade (gpt)		7.08	7.54	8.04	7.99	7.47	6.78	7.11	9.90	10.04	8.15	7.67	8.02	7.88	8.04	**8.06**
Total (ounces)		40,413	148,493	166,834	176,563	168,992	152,970	171,902	241,568	261,429	212,332	199,577	208,616	141,500	76,471	**2,367,658**
Recovered Ounces (92.5%)		**37,382**	**137,356**	**154,321**	**163,321**	**156,317**	**141,497**	**159,009**	**223,451**	**241,822**	**196,407**	**184,608**	**192,970**	**130,888**	**70,735**	**2,190,084**
LH drilling (metres)		8,230	38,742	42,837	47,834	46,737	46,875	49,346	49,461	53,849	55,247	53,568	54,314	38,445	23,809	**609,295**
Rockfill (tonnes)		-	178,244	103,186	103,212	145,235	135,356	167,180	166,334	182,016	114,053	92,669	87,987	46,234	1,847	**1,523,553**
Pastefill (tonnes)		82,218	188,978	313,277	363,172	305,514	317,872	307,198	309,287	335,700	425,176	432,170	444,820	332,688	235,795	**4,393,867**
Mineralized material hoisted (tonnes)	-	177,657	612,725	645,629	687,030	703,314	701,478	752,285	758,674	809,890	809,877	809,701	809,290	558,435	295,940	**9,131,926**
Rock hoisted (tonnes)	73,285	201,856	178,244	103,186	103,212	145,235	135,356	167,180	166,334	182,016	114,053	92,669	87,987	46,234	1,847	**1,798,694**
Mineralized material hoisted per day		**493**	**1,702**	**1,793**	**1,908**	**1,954**	**1,949**	**2,090**	**2,107**	**2,250**	**2,250**	**2,249**	**2,248**	**1,551**	**822**	**1,913**
Waste hoisted per day[#]	**407**	**561**	**495**	**287**	**287**	**403**	**376**	**464**	**462**	**506**	**317**	**257**	**244**	**128**	**5**	**326**

* The projected mining method, potential production profile and plan, and mine plan are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks which include, but are not limited to, the inclusion of Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets and process plant designs may require additional detailed work, economic analysis, and internal studies to ensure satisfactory operational conditions and decisions regarding future targeted production.

[#] Does not include shaft sinking waste hoisted.

Table 44: Life-of-Mine Capital Development Metres

Metres of Lateral Development	Width (m)	Height (m)	Year 2013	Year 2014	Year 2015	Year 2016	Year 2017	Year 2018	Year 2019	Year 2020	Year 2021	Year 2022	Year 2023	Year 2024	Year 2025	Year 2026	Year 2027	Total
Track Drift	2.75	3.35	641	2,323	287													**3,251**
Slash 122 level track drift for RAD	1.50	3.35					630											**630**
Jackleg/longtom drifts	3.70	3.70	101	1,765	236				60									**2,163**
Jackleg crusher chambers	5.00	6.00	20						30									**50**
Jumbo drift	3.70	3.70	849	2,503	3,663	1,543	1,619	3,367	2,471	3,022	2,628	4,499	1,417					**27,582**
Jumbo drift 610 level FAD	5.00	5.00		229	151													**380**
Total Lateral Development			**1,612**	**6,821**	**4,337**	**1,543**	**2,249**	**3,367**	**2,561**	**3,022**	**2,628**	**4,499**	**1,417**	**-**	**-**	**-**	**-**	**34,056**
Fresh air raises (pilot & Slash)	6.00	3.00	140	408	62			183	122	-	373	54						**1,342**
Return air raises (Alimak)	3.00	3.00	302	181	783				630		526	354	190					**2,966**
Passes and manway raises (Alimak)	2.40	2.40	140	1,424	350				162	1,736	317							**4,129**
Open raises/drop raise for chutes	2.40	1.80	20						20									**40**
Total Vertical Development			**-**	**2,013**	**1,195**	**-**	**-**	**183**	**934**	**1,736**	**1,215**	**409**	**190**	**-**	**-**	**-**	**-**	**8,477**

Table 45: Life-of-Mine Operating Development

Metres of Lateral Development	Width (m)	Height (m)	Year 2013	Year 2014	Year 2015	Year 2016	Year 2017	Year 2018	Year 2019	Year 2020	Year 2021	Year 2022	Year 2023	Year 2024	Year 2025	Year 2026	Year 2027	Total
Jackleg/longtom drifts	3.70	3.70		204	1,073	704	25											**2,006**
Jumbo drift	3.70	3.70	20	1,744	2,258	2,333	2,763	3,090	2,930	3,957	4,069	3,621	3,487	3,932	3,686	1,846		**39,736**
Subtotal			**20**	**1,949**	**3,331**	**3,036**	**2,788**	**3,090**	**2,930**	**3,957**	**4,069**	**3,621**	**3,487**	**3,932**	**3,686**	**1,846**		**41,742**
Mineralized Material																		
Jackleg/longtom drifts	3.70	3.70		896	1,614	543	116											**3,169**
Jumbo drift	3.70	3.70		110	481	1,322	1,483	2,040	1,988	2,298	2,373	2,344	2,092	2,392	2,252	1,380	113	**22,667**
Subdrifts from Alimak	2.40	3.00		780	1,624	1,446	1,239	1,582	1,541	1,782	1,839	1,817	1,622	1,854	1,745	1,070	88	**20,028**
Subtotal			**-**	**1,786**	**3,719**	**3,311**	**2,837**	**3,622**	**3,529**	**4,080**	**4,212**	**4,161**	**3,714**	**4,246**	**3,997**	**2,449**	**201**	**45,864**
Total Lateral Development			**20**	**3,735**	**7,050**	**6,347**	**5,625**	**6,712**	**6,459**	**8,037**	**8,281**	**7,782**	**7,202**	**8,177**	**7,683**	**4,295**	**201**	**87,606**
Waste																		
Alimak LH access (& temp manways)	2.40	2.40		115	292	309	280	309	324	356	357	372	358	368	373	241	47	**4,101**
Open raises to access subs, C&F stopes	2.40	1.80		44	111	118	107	118	123	136	136	142	136	140	142	92	18	**1,564**
Subtotal			**-**	**158**	**404**	**427**	**387**	**427**	**447**	**492**	**493**	**514**	**494**	**509**	**515**	**333**	**64**	**5,666**
Mineralized material																		
Alimak LH access & slot	2.40	2.40	-	367	937	991	899	992	1,038	1,142	1,144	1,194	1,147	1,181	1,196	774	149	**13,152**
Open raise in mineralized material for C&F	2.40	1.80	-	19	49	52	47	52	54	60	60	63	60	62	63	41	8	**689**
Subtotal			**-**	**387**	**986**	**1,043**	**946**	**1,044**	**1,093**	**1,202**	**1,204**	**1,256**	**1,207**	**1,242**	**1,259**	**815**	**157**	**13,841**
Total Vertical Development			**-**	**545**	**1,390**	**1,470**	**1,333**	**1,472**	**1,540**	**1,694**	**1,696**	**1,771**	**1,702**	**1,751**	**1,774**	**1,148**	**221**	**19,507**

15.7.4 Mine Ventilation

The mine ventilation system will be a push-pull arrangement with the main supply and return air fans located on surface. Fresh air will be supplied to the levels from a single fresh air raise and return will be via two raises, one near each end of the level. Air flow on the levels will be modulated with ventilation regulators located at the return air raise connections.

The system will be developed in phases corresponding to the mine development phases. During Phase I, fresh air will be introduced on the 305 level through a 6 m by 3 m raise from surface, and across the 305 level to the F2 Zone through a 5 by 5 m fresh air transfer drift driven parallel to the shaft crosscut. From the 305 level, supply air will be distributed to the mine workings between the 366 level and 244 level via ramp and crosscuts. Fresh air will be supplied to the 183 level via a short internal fresh air raise from the 244 level.

The 122 level will be the main return air collection level for exhaust air from the south and north return air raises (3 by 3 m). Exhaust air will exit the mine via two return air transfer drifts on the 122 level driven between the active mine workings and the historic workings. The two return air raises (3 by 3 m) will deliver the exhaust air to surface. The return air drifts on 122 level will be developed in three phases during Phase I. Initially a track drift will be driven to the south return air raise to establish the development ventilation circuit. This will allow approximately 47 m^3/sec (100,000 CFM) of return for development. A return air drift (RAD) will then be driven prior to commercial production to increase the return capacity to match the initial production requirements. Finally the existing track drift will be slashed to increase the return air capacity to 245 m^3/sec (520,000 CFM) in 2017.

Subsequent ventilation phases involve modification of the ventilation system by extending the same configuration of one main fresh air raise (6 m by 3 m) and two return air raises (3 by 3 m) down to the bottom of the next phase. All raises are driven by Alimak. The fresh air raise will eventually extend to the 1525 level and the two return air raises, one near either end of the level, will extend from the 1464 and 1586 levels respectively to connect with the 305 level system.

This approach will allow fresh air to sweep from a central location across the levels to the return air raises located near the extremities of the deposit. This flow through design will reduce dependency on auxiliary ventilation systems.

The main fresh air ventilation system will initially be capable of provide 123 m^3/sec (260,000 CFM) to the active mine headings above the 366 level. During cold weather, outside supply air will be heated to +5°C with a 28 Mbtuh propane-fired heater. The final ventilation system will provide 245 m^3/sec (520,000 CFM) by twinning the supply fan, mine air heater, and return air systems.

The recommended fans and their duty cycles for the initial and final ventilation systems are presented in Table 46. Typical arrangements for the surface supply and return fans are shown in Figure 44 and Figure 45, and the final ventilation system is shown schematically in Figure 46.

Table 46: Main Ventilation Fan Requirements

Location	Fan Duty	Fan Motor Power	Initial Phase # of Fans	Final Phase # of Fans
Supply	260,000 CFM @ 16" TP 780 BHP @ 83% EFF.	746 KW (1,100 HP)	1	2
Return	250,000 CFM @ 12"TP 550 BHP @ 85% EFF.	522 KW (700 HP)	1	2

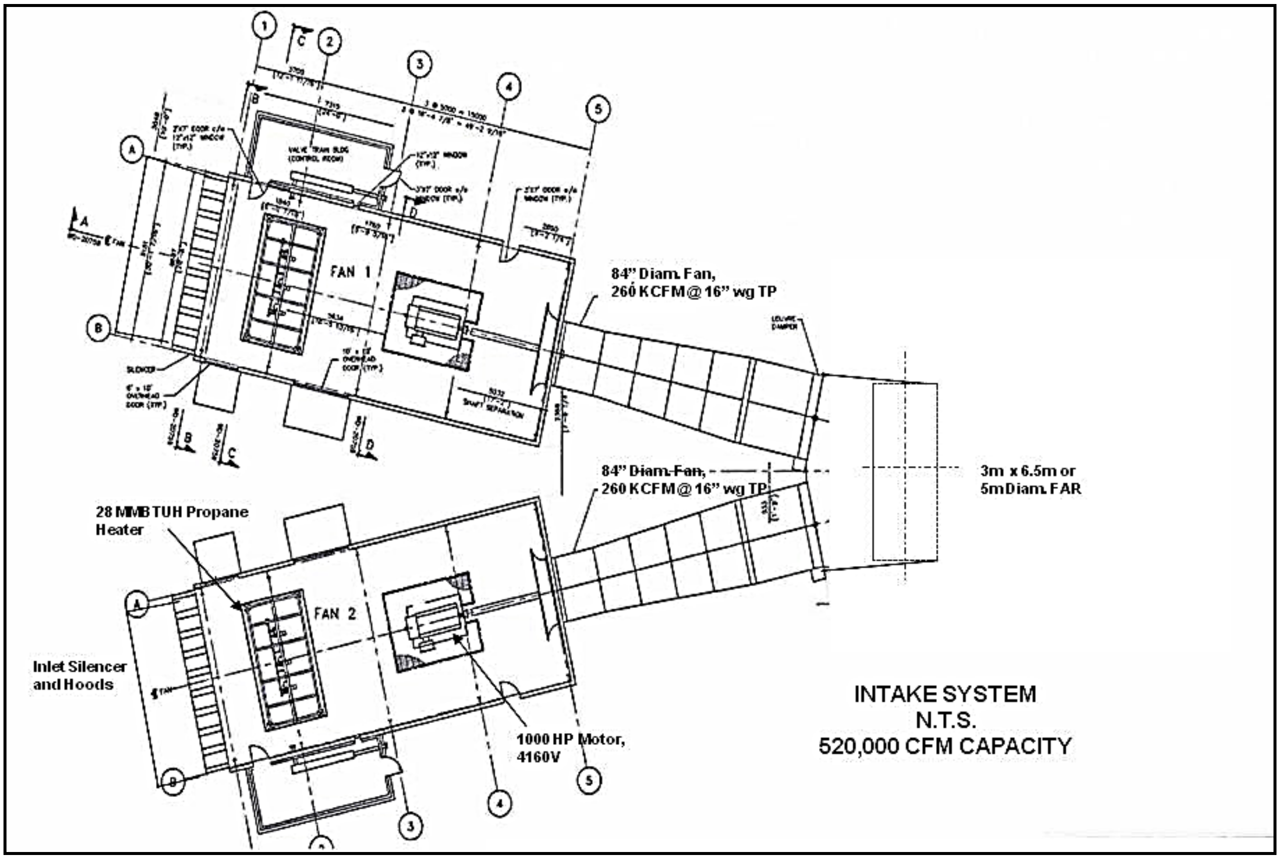


Figure 44: Fresh Air Fan and Heater House Typical Arrangement

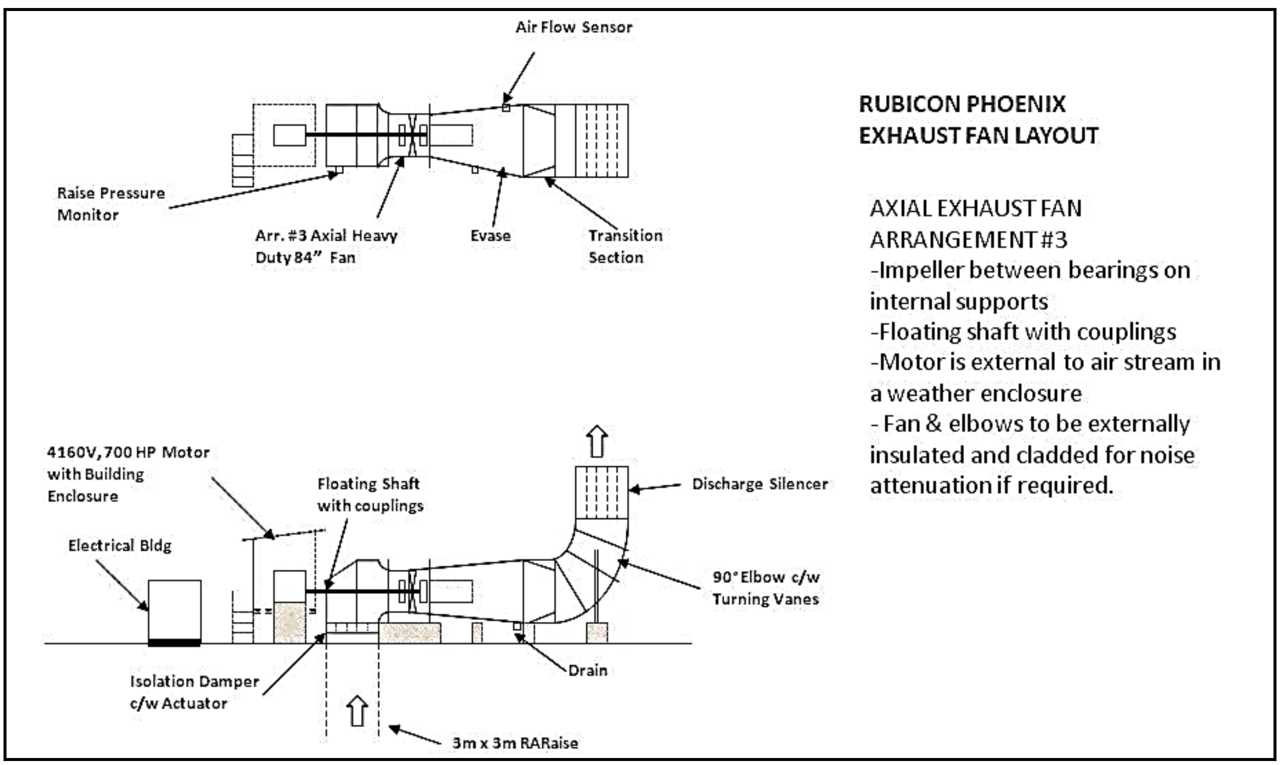


Figure 45: Return Air Fan Typical Arrangement

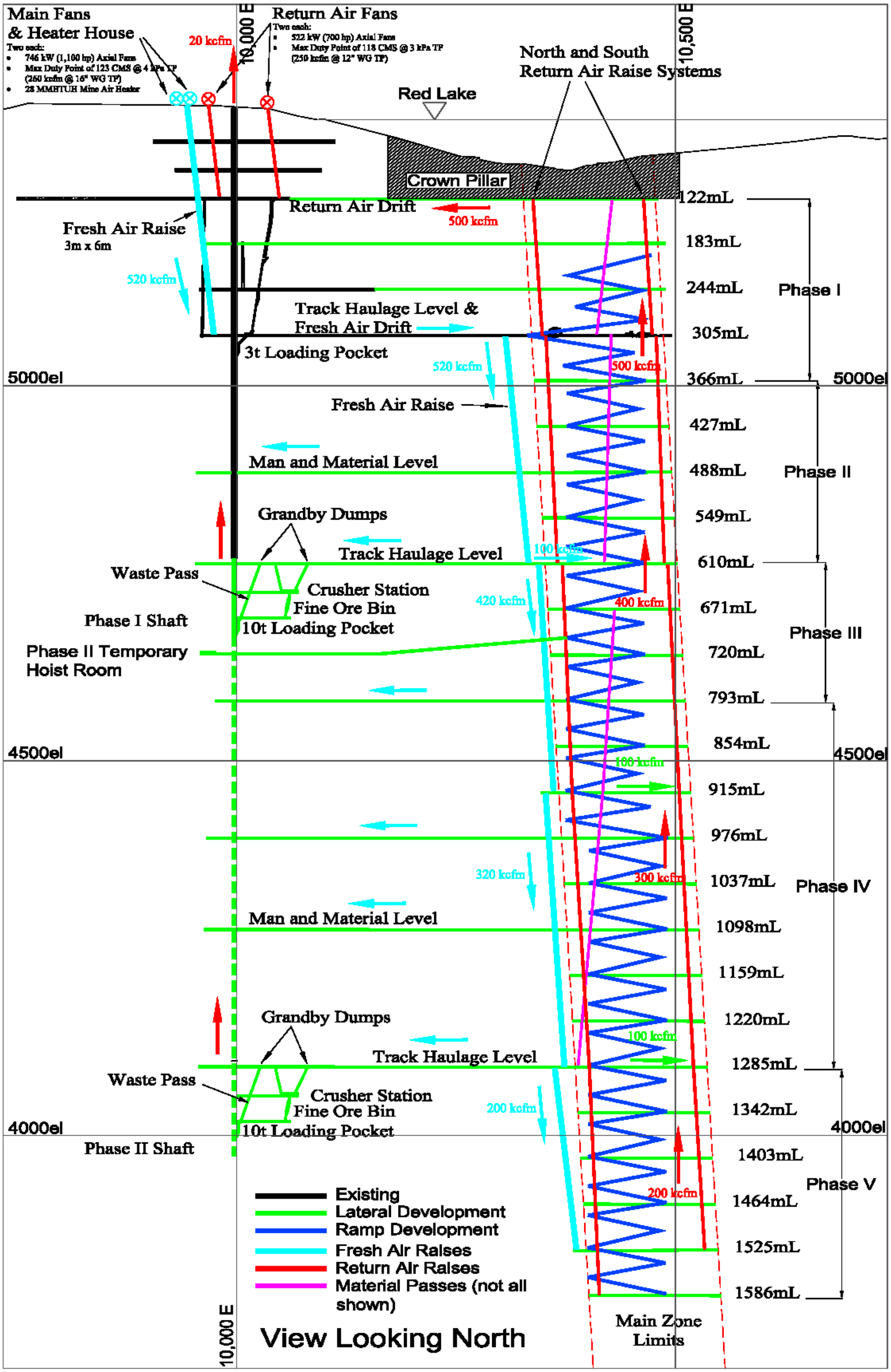


Figure 46: Schematic of Final Mine Ventilation System (SRK 2013)

15.7.5 Mine Services

The following paragraphs provide an overview of the mine services required to support the underground mine plan. The details of project infrastructure are covered in Section 17.

Man and material access to the mine will be via the existing timber shaft. During the initial phases, the 488 level will function as a man and material entrance to the mine and is expected to be developed trackless from the shaft to the mineralized zone. This level will also host the maintenance shops, storages, and main refuge station required to support operations through to 2020. The same approach will be repeated on a deeper level when the shaft is deepened.

Mineralized vein and waste handling will mainly be done using a series of passes to transfer material to the haulage levels. A combination of diesel, electric, and battery equipment will be used to move the material to the appropriate dump points at the passes. Material flow will be controlled by control chains and chutes. For the most part, material will be loaded into the rail cars by chute, and LHDs will be used as required, but mainly on the upper levels during initial development off the track drifts. On the main haulage levels, 13.5-tonne (15-ton) battery locomotives will haul material to the appropriate dumps near the shaft to be skipped to surface. The mineralized material will pass through a stationary crusher and into a fine ore bin before being skipped to surface.

The majority of the backfill used will be a pastefill product blended on surface using filtered tailings from the mill, which will be transported underground through a system of pipelines and boreholes. The paste backfill supply will be supplemented by using waste rock from development headings whenever practical. During the LoM, some 1.5 Mt of rockfill and 4.4 Mt of pastefill are expected to be placed as backfill.

The existing pumping system to the 305 level is already established as a basic clear water pumping system capable of 50 litres per second (800 USGPM). Daily pumping records for 2011/2012 indicate that the typical inflows are well under 5 litres per second (80 USGPM). Process water usage during mine operation at 1,800 tpd is estimated at 15 litres per second (238 USGPM).

Mine process water will be sourced from the lake and delivered underground through a pipeline in the shaft. This system is currently in place.

Compressed air will also be delivered underground via pipeline in the shaft. The estimated requirement is 3.3 CMS (7,000 cfm) at a production rate of 1,800 tpd. Design work is ongoing for the permanent compressor building and it is expected that the existing compressed air lines in the shaft will have to be replaced prior to reaching full production.

Electrical power will be distributed underground by 4160V feed cables installed in the shaft and distributed to mining zones from electrical substations. All planned installations are typical of underground mines.

A central blasting system will be installed to enable initiation of main blasts from surface.

Underground communications will be provided by analog phones and a leaky feeder radio system. A fibre optic system will be installed in the shaft to support future expansion and automation.

15.7.6 Equipment and Manpower

Table 47 shows the planned underground mining equipment fleet as contemplated for 2017. The equipment list accounts for the captive nature of the upper levels with no ramp connection to the 183 level and the 122 level, as well as the fact that the 122 to 305 levels are track access areas that will limit equipment size and mobility until the ramp is in place. There is also a rebuild strategy incorporated into the plan starting in 2022 for LHDs and trucks.

The equipment size has also been limited to the largest pieces of mobile equipment that can be slung down the existing shaft without having to cut the frame into many pieces. Additional means of moving larger equipment underground such as lowering units down the main fresh air raise or one of the return air raises have not been considered in the PEA.

Table 47: Estimated Equipment Fleet for 2017 (1,900 tpd)

Description	Quantity	Unit Cost
Existing Equipment U/G		
Small battery locomotives	3	
Small granby cars	9	
Hudson cars	3	
LM 56 muck machines	3	
630 crawler mucker	2	
320 cavo	1	
Longtom (track)	2	
Alimak raise climber	1	
Boart BCI2 LH drill	1	
Rock-tec rockbreaker	1	
To be Purchased		
2-boom jumbo	5	C$1,170,389
Explosive loader	3	C$435,000
3.5-tonne diesel LHD	5	C$381,000
Scissor truck	6	C$344,000
Dry shotcrete unit	3	C$65,000
6.7-tonne diesel LHD	7	C$471,800
6.7-tonne battery LHD	5	C$741,500
LHD battery/charger	3	C$265,000
20-tonne diesel truck	2	C$554,200
20-tonne battery truck	2	C$1,033,000
Truck battery/charger	2	C$285,000
Boom truck	1	C$335,000
Grader	1	C$425,000
Fuel/lube truck	1	C$340,000
Personnel carrier	1	C$280,000
Light vehicle	4	C$85,000
Forklift/backhoe	2	C$225,000
Longtom (rubber tired)	4	C$30,000
Utility vehicle	4	C$42,000
13.5-tonne battery locomotive	2	C$275,000
Large grandby cars	8	C$39,820
Locomotive battery charger	1	C$10,450

The equipment list does not account for the equipment required by the mill, site services, or contractors working on surface. Longhole drilling equipment is also not listed as this will change with the mining plan and it is expected that this equipment will be supplied by the drill and blast contractor.

Initially, all direct mining tasks required to deliver production material to the mill will be performed by mining contractors. This includes all development, stoping, mineralized material and waste haulage, backfill operations and maintenance functions such as mobile equipment, stationary equipment and electrical work.

Starting in 2015, it is expected that the majority of mining tasks will transition to company personnel with the following exceptions:

- All longhole drilling and blasting operations will be performed by a drilling contractor;
- Drilling equipment will be supplied by the drilling contractor to add flexibility to the mine plan as the range of drilling equipment required will change throughout the mine life;
- All Alimak and open raising will be by contract raise miners; and
- Surface waste haulage and disposal.

Table 48 shows the estimated manpower required on site for 2017, which is the first year that production is planned to reach the 1,900 to 2,000 tpd range.

Table 48: Estimated Site Manpower for 2017 (1,900 tpd)

Function	Staff	Hourly	Contract	Total
Surface				
Mine management	2			
Administration	4	3		
Safety & training	1	2		
Security			4	
Yard & warehouse	2	4		
Surface waste handling			2	
Subtotal Surface	**9**	**9**	**6**	**24**
Mill				
Mill Management	5			
Plant operations		16		
Refinery		2		
Assay Laboratory		4		
Maintenance		10		
Subtotal Mill	**5**	**32**	**0**	**37**
Mine				
Mine supervision	13	2		
Maintenance	8	39		
Mine technical services	13			
Safety & training	2	6		
Development		69	20	
Ore & waste handling		16		
Trammers		8		
Longhole stoping		10	12	
Cut and fill stoping		8		
Backfilling		4		
Shaft & muck circuit		16		
Mine services		6		
Construction and utility		9		
Definition diamond drilling			12	
Subtotal Mine	**36**	**194**	**44**	**0**
Total Site Personnel	**50**	**235**	**50**	**335**

16 Recovery Methods

16.1 Process Flowsheet

The simplified process flowsheet (1,250 tpd scenario) for the Phoenix gold project is presented in Figure 47.

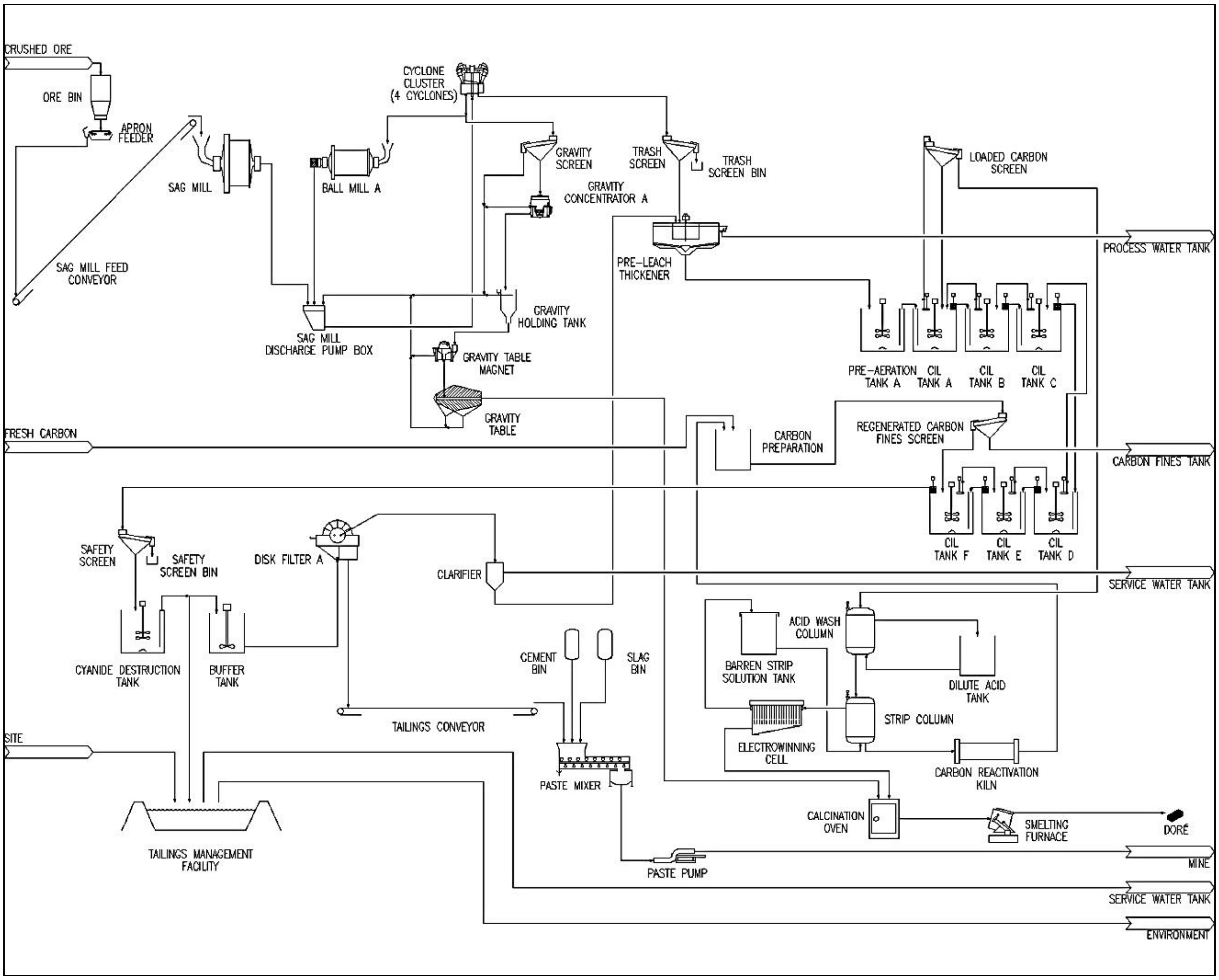


Figure 47: Simplified Process Flowsheet

16.2 Conceptual Process Flowsheet Summary

The process consists of a single line, starting with a semi-autogenous grinding (SAG) mill. The discharge from the SAG mill is sent to the ball mill circuit that uses hydrocyclones in close circuit for classification. A gravity separation circuit is included in the closed circuit with hydrocyclones to partially recover and concentrate any gravity recoverable gold. The remaining gold is extracted in a conventional carbon-in-leach (CIL) circuit. The loaded carbon is washed with hydrochloric acid solution to remove carbonates. Gold is then removed from the loaded carbon by elution (stripping) followed by electrowinning. The electrowinning and the gravity circuit both produce a high grade gold concentrate that goes for smelting of doré in an electric induction furnace. The stripped carbon is regenerated in a reactivation kiln before being reintroduced to the process. Fine carbon is constantly eliminated (and recovered) from the process to avoid gold loss, with fresh carbon being continuously added to the process.

The cyanide contained in the tailings from the CIL circuit is eliminated in a cyanide destruction tank with SO_2-air process. Once the cyanide is destroyed, the tailings are sent to the tailings pond for disposal or they are passed through the paste plant where they are filtered to lower the water content. The filter cake is then mixed with slag and cement to produce a paste. The paste produced is pumped to the mine for underground backfilling.

16.3 Process Description

16.3.1 Mineralized Material Storage

An underground grizzly screen on 305 level with, typically, 23 centimetres (cm) openings (9″ by 9″) and a rock breaker are used to reduce the ore size prior to hoisting it to the surface. A crusher will be installed below the 610 m level to appropriately size the material before it reports to the 680 m level loading pocket. The skipped mineralized material is dumped into a raw ore bin when the raw mineralized material chute lift is opened while the waste is dumped into a waste bunker when the raw ore chute lift is closed. The mineralized material is discharged from the raw mineralized material bin via a discharge chute onto a vibratory feeder, which then transfers the mineralized material onto the storage bin feed conveyor. A raw mineralized material magnet fitted on a small conveyor situated above and running perpendicular to the storage bin feed conveyor is used to remove tramp metal from the raw mineralized material. The tramp metal is collected in a bin for disposal. The remaining mineralized material is conveyed to the storage bin.

16.3.2 Grinding and Thickening

The raw mineralized material from the storage bin is reclaimed by two apron feeders and is discharged onto a first conveyor equipped with a belt scale. The first conveyor is discharged on a second conveyor, which then transfers the mineralized material to the SAG mill mobile feed chute extension.

The grinding circuit is a double-stage grinding circuit consisting of a SAG mill and a ball mill. The SAG mill operates in open circuit while the ball mill is operated in closed circuit with hydrocyclones. Process water is added to the SAG mill feed chute to achieve the correct dilution for grinding. The main portion of the hydrocyclones underflow is directed to the ball mill for regrinding while the remaining portion goes to the gravity separation circuit. The hydrocyclones overflow pulp flows to the thickening circuit.

The thickening circuit consists of one trash screen, one clarifier and one thickener. The trash screen is fed, by gravity, from the hydrocyclone cluster overflow. The screen undersize flows by gravity, via primary and secondary samplers, to the pre-leach thickener feed box. Any oversize trash is dumped into a trash bin.

The clarifier is fed by the filtrate from the disc filters and by the vacuum seal water from the vacuum pumps of the paste plant. The clarifier overflow feeds the service water tank while the underflow is pumped to the pre-leach thickener.

The pre-leach thickener is fed by the trash screen undersize, the clarifier underflow and the thickening area sump pump. Flocculent is also added to improve the settling rate. The thickener overflow feeds by gravity the process water tank while the underflow is pumped to the pre-aeration tank in the CIL circuit.

16.3.3 Gravity Separation

The gravity circuit consists of one vibrating screen, one gravity concentrator, one gravity table, and one gravity table magnet. The underflow from two of the hydrocyclones within the cluster is sent to the gravity circuit (one operational and one standby). The remaining four hydrocyclones underflow is sent to the grinding circuit (three operational and one standby).

The hydrocyclones underflow flows by gravity to the gravity screen. Dilution water is added to the screen oversize to transport the material to the gravity pump box. This material is then recirculated to the hydrocyclone feed pump box in the grinding circuit.

The gravity screen undersize flows to the gravity concentrator where gravity recoverable gold is recovered. Dilution water is added directly to the gravity screen underflow to facilitate the pulp flow into the concentrator and to adjust the feed pulp %-solids. The gravity concentrator concentrate is pumped to the gravity holding tank while the gravity concentrator tails are directed to the gravity pump box and then pumped to the hydrocyclone feed pump box in the grinding circuit.

The gravity concentrator concentrate stored in the gravity holding tank is fed to the gravity table magnet where the magnetic particles are removed and sent back to the grinding circuit. The non-magnetic portion of the stream is sent to the gravity table to produce an upgraded gold concentrate smelted into doré in the on-site refinery. The gravity table tails are recirculated into the grinding circuit via the hydrocyclone feed pump box, along with the gravity screen oversize, the gravity concentrator tails and the magnetic particles from the gravity table magnet.

16.3.4 Carbon-in-Leach

The underflow from the pre-leach thickener is pumped to the pre-aeration tank. Slurry from the pre-aeration tank overflows into the first of six agitated CIL tanks arranged in series. Cyanide solution and lime are added, as required, to the pre-aeration tank and to the first and fourth CIL tanks for gold dissolution and pH control. Lead nitrate is also added in the pre-aeration tank to improve the gold leaching kinetics. Gold in the solution is adsorbed onto the activated carbon.

The six CIL tanks have been sized to provide 36 hours of residence time at the design flow rate and solids concentration. Each CIL tank is equipped with a single interstage screen and a carbon-transfer pump and is agitated to maintain the solids in suspension. Air is injected in the bottom of the pre-aeration tank and in each CIL tank for gold dissolution. Interconnecting tank launders are arranged so that any tank in series can be bypassed without having to shut down the entire CIL circuit.

On a regular basis, loaded carbon is pumped counter current to the slurry flow through the CIL tanks in order to increase gold loading. The carbon-forwarding pump of the first CIL tank transfers the slurry onto the loaded carbon screen to recover the loaded carbon from the slurry. Screen undersize flows by gravity back to the first CIL tank while the oversize, containing the loaded carbon, flows by gravity to the acid wash column in the elution circuit. Fresh and regenerated carbon is added into the last CIL tank.

16.3.5 Elution and Carbon Reactivation

Loaded carbon recovered by the loaded carbon screen gravitates to the acid wash column of the elution circuit. The carbon elution circuit should treat a 4-tonne batch in approximately 12 hours. The circuit is designed to process one elution per day.

The acid solution is prepared in the dilute acid tank and then pumped through the acid wash column. Once the acid wash is done, the spent acid is neutralized with caustic. The carbon is transferred from the acid wash column to the strip column for gold desorption. The solution from the barren strip solution tank flows through a series of heat exchangers and a heater in order to reach the right temperature in the strip column. The solution strips the gold loaded onto the carbon which then exits through a Johnson screen from the upper side of the column. The pregnant solution then goes to the electrowinning cells in the refinery for gold recovery.

The stripped carbon is drawn from the bottom of the strip column and goes to the carbon reactivation kiln. After the reactivation, the carbon is discharged into the carbon quench tank. The carbon from the carbon quench tank is pumped and screened out to remove (and recover) fine carbon and then drops by gravity to the last CIL tank. Fresh carbon is added in the carbon quench tank on a regular basis to compensate the fine carbon removal.

16.3.6 Electrowinning and Refinery

The pregnant solution from the strip column flows first by gravity to the electrowinning flash tank and then to two parallel electrowinning cells, where the gold is plated on cathodes. The barren solution from the electrowinning cells is recovered in a pump box and pumped back to the barren strip solution tank in the carbon elution circuit.

After a certain period, the stainless steel wool cathodes are cleaned with high pressure water and the gold sludge sinks to the bottom of the cells. The gold sludge is then pumped with a diaphragm pump to a filter-press to remove excess water. The filtrate from the filter-press is recovered by the electrowinning area sump pump and pumped to the pre-aeration Tank A launder in the CIL circuit.

The filtered gold sludge from the filter-press and the concentrate from the gravity table are sent to the calcination oven to remove excessive humidity. The dried gold sludge is then mixed with suitable fluxes (typically borax, soda ash, sodium nitrate, and silica sand) and is fed into the crucible of the electric induction furnace. Once the gold is melted, it is poured into the doré moulds. The doré bar is then recovered for shipment.

16.3.7 Cyanide Destruction

The safety screen is fed by the last CIL tank overflow. It prevents the loss of carbon in the eventuality of a failure of the last CIL tank interstage screen. The carbon is recovered at the oversize.

The screen undersize flows by gravity into the cyanide destruction tank feed pump box and is pumped to the cyanide destruction tank. Air is added at the bottom of the cyanide destruction tank within a dispersion cone. Sulphur dioxide (SO_2) is added in liquid form at the bottom of the tank. The copper sulphate and the lime are added at the top of the tank.

Once cyanide destruction is complete, the tailings are discharged into the cyanide destruction discharge distributor. When the paste plant is operating, the tailings flow by gravity to the buffer tank feed pump box and are pumped to the buffer tank. When the paste plant is not operating, the tailings flow by gravity to the tailings pump box and are pumped to the tailings pond. Service water can also be added to the tailings pump box to prevent pump surging.

16.3.8 Tailings Filtration

The tailings filtration system consists of two disc filters with two filter feed pumps, two vacuum pumps, two snap blow receivers, two filtrate tanks and two filtrate pumps.

The tailings from the cyanide destruction circuit are pumped from the buffer tank feed pump box to the buffer tank. The tailings are then pumped to one of the two disc filters for filtration (one operational, one standby). The filtrate is recovered in the filtrate tank and pumped to the clarifier. The filtered tailings are discharged on the tailings conveyor which feeds the paste mixer.

16.3.9 Paste Backfill Preparation

The disc filter tailings cake is discharged on the tailings conveyor and then mixed with service water in the paste mixer to produce backfill paste. Slag and Portland cement are also added to the mixer to meet underground backfilling strength requirements. The cement and binders discharged from the storage bins are controlled to achieve the proper concentration in the backfill paste. The paste produced by the mixer is then discharged into the paste pump feed hopper.

16.3.10 Paste Backfill Distribution

Once the paste is prepared, two positive displacement pumps are used to move the paste towards the underground stopes. Each pump is equipped with a hydraulic unit.

16.3.11 Reagents

Except for the reagents used in relatively small quantities at the electrowinning and refinery sectors, the following reagents are used throughout the process:

Sodium cyanide

Sodium cyanide (NaCN) is used for gold leaching in the CIL circuit for dissolving the gold and for carbon stripping in the carbon elution circuit to improve the efficiency of the process. Sodium cyanide is supplied in bags and is added in the cyanide bag breaker hopper with fresh water. The sodium cyanide solution is then mixed in the cyanide mix tank that is the upper unit of a two-stage tank. The cyanide tank is covered and equipped with a fan. When the preparation of the sodium cyanide solution is complete, the solution is transferred to the lower unit of the two-stage tank, namely the cyanide distribution tank. One cyanide metering pump is used to distribute the sodium cyanide solution to the CIL circuit and one more pump is used to distribute the solution to the SAG mill and the barren elution solution tank.

Flocculant

Flocculant is used in the pre-leach thickener to improve the settling rate. Flocculant is supplied in bags and is added in the flocculant feed hopper. A screw feeder transfers the flocculant from the feed hopper to an eductor where fresh water is added to dilute the flocculant. The flocculant from the eductor is discharged in the flocculant mix tank where more fresh water is added. The flocculant mix tank is the upper unit of a two-stage tank. When the flocculant preparation stage is complete, the mixture is transferred to the lower unit of the two-stage tank, namely the flocculant distribution tank. The flocculant is then pumped by two metering pumps (one operating, one standby) to the flocculant static mixer where service water is added to further dilute the flocculant to the appropriate concentration, prior to being discharged into the pre-leach thickener.

Hydrochloric acid

Hydrochloric acid (HCl) is used for the carbon acid wash. The hydrochloric acid is supplied in drums and pumped to the acid tank for storage. The acid tank is covered and equipped with a fan. One acid distribution pump is used to transfer the acid to the dilute acid tank in the carbon elution circuit.

Lead nitrate

Lead nitrate ($PbNO_3$) is used to improve the gold leaching kinetics in the CIL circuit. Lead nitrate is supplied in bags and is added in the lead nitrate bag breaker hopper. The lead nitrate is then mixed with fresh water in the lead nitrate mix tank. This tank is covered and connected to a fan. One metering pump is used to transfer the lead nitrate solution to the pre-aeration Tank A in the CIL circuit.

Sulphur dioxide

Sulphur dioxide (SO_2) is used as an oxidizing agent in the cyanide destruction process. The sulphur dioxide is delivered by truck and stored in the sulphur dioxide tank. The sulphur dioxide tank is equipped with a pressure system to keep the sulphur dioxide in liquid form and to supply the sulphur dioxide to the cyanide destruction tank. The pressure system consists of one air compressor, one air receiver upstream from one air dryer, and another air receiver downstream to the air dryer.

Lime

Lime, which will be delivered as quicklime (CaO), is used to control the pH in the grinding, CIL and cyanide destruction circuits thus preventing cyanide acid (HCN) formation. The lime is delivered by truck and stored in the lime bin. The lime bin is equipped with a dust collector. A screw feed conveyor transfers the lime to the lime slaker where fresh water is added to hydrate the lime and to prepare the milk of lime. The milk of lime is stored in the lime distribution tank which is equipped with an agitator. Two lime distribution pumps (one operating, one standby) deliver the milk of lime to the CIL circuit and cyanide destruction circuits. The pumps feed a pressurized closed loop system, where time controlled and pH controlled on/off valves, located along the loop, deliver the milk of lime to the grinding, CIL, and cyanide destruction circuits. Any unused milk of lime is recirculated back to the lime distribution tank, via the closed loop.

Copper sulphate

Copper sulphate ($CuSO_4$) is used as a catalyst in the cyanide destruction process. Copper sulphate is supplied in bags and is added in the copper sulphate bag breaker hopper. The copper sulphate is then mixed with fresh water in the copper sulphate mix tank that is the upper unit of a two-stage tank. The mix tank is covered and connected to a fan. When the preparation of the copper sulphate solution is complete, the solution is transferred to the lower unit of the two-stage tank, namely the copper sulphate distribution tank. One pump is used to transfer the copper sulphate solution to the cyanide destruction tank.

Sodium hydroxide

Sodium hydroxide (NaOH) is used for carbon stripping and after the carbon acid wash to neutralize the residual acid in the dilute acid tank and the acid wash column. The caustic is supplied in drums and pumped to the covered caustic tank for storage. A caustic distribution pump transfers the caustic to the dilute acid tank and to the barren strip solution tank.

Descalant

A descalant reagent is used to reduce calcium carbonate deposits. The descalant is supplied in drums and pumped to the process water tank and barren strip solution tank.

Cement

Cement is used at the paste plant to enhance the strength of the paste backfill. Cement is delivered by truck and stored in a bin equipped with a dust collector. A screw conveyor delivers the cement to the paste mixer.

Slag

Slag is used at the paste plant to enhance the strength of the paste backfill. Slag is delivered by truck and stored in a bin equipped with a dust collector. A screw conveyor delivers the slag to the paste mixer.

16.3.12 Utilities

Fresh Water

A fresh water system is required in order to store and distribute fresh water to various areas of the mill and project site. The existing fresh water tank, situated at the highest topographical location, south of the hoist room, is used to store fresh water. The fresh water tank is fed by the existing pump system that draws water from Red Lake. Two fresh water pumps (one operational, one standby) distribute fresh water to the processing plant and various other areas at the project site. Fresh water is used for reagent preparation, cooling, gland sealing, and washbasins. Fresh water is also used to supplement the fire water required for on-site fire suppression purposes.

Reclaim Water

The water recovered in the tailings pond (reclaim water) is pumped into the service water tank. One of the three reclaim water pumps located in the tailings pond is used to supply reclaim water to the service water tank. The remaining reclaimed water pumps are used either as spares or for feeding the water treatment plant for the treatment and discharge of surplus water from the TMF to the environment.

Service Water

The service water tank is used to store reclaim water that does contain low values of cyanide. It is fed by the clarifier overflow from the thickening circuit, by reclaim water from the tailings pond, and by fresh water when required. The service water tank overflows in the process water tank and serves as make-up process water. The service water is also pumped and distributed throughout the concentrator while excess water from the service water tank is sent to the TMF.

Process Water

The process water is stored in the process water tank located on the west side of the pre-leach thickener to allow any overflow from the thickener to gravitate into the process water tank. The process water tank is also fed by the service water tank overflow and by fresh water, if additional water is required. Two process water pumps (one operational, one standby) distribute the water to various process areas. Process water is used in the grinding, gravity, and thickening circuits.

Domestic Water for Emergency Showers

Domestic water feeds the domestic water heaters. Two domestic water pumps (one operational, one standby) distribute domestic water to the emergency showers throughout the concentrator as well as the rest of the project site.

Air Service

Two air compressors supply compressed air at 125-psig pressure. The compressed air is stored in a 500-gallon air receiver. The air compressors are located in an enclosure situated to the north side of the paste plant. The air receiver supplies compressed air to the process plant as service air and to an air dryer. The air dryer supplies dry air to a dry air receiver that stores and supplies dry air for instrumentation requirements.

Service air is distributed throughout the plant via a main 100 mm (4″) pipe. This run branches off to 50 mm (2″) and 75 mm (3″) lines to various sections of the plant. Instrument air is supplied via a main 50 mm (2″) line. This also branches off to various sections of the plant with a 25 mm (1″) line. Compressed air is supplied to the cement and slag storage bins with a 50 mm (2″) pipe. Two air blowers are used for the air distribution to the CIL and cyanide destruction circuits. The main header piping for the CIL blowers is a 200 mm (8″) diameter pipe. This header branches off for each tank.

16.4 Concentrator Conceptual Design

16.4.1 Conceptual Design Criteria

Table 49 presents the main design criteria used for the concentrator design.

Table 49: Concentrator Main Design Criteria

Parameter	Value	Units
Feed Characteristics		
Gold Head Grade (Nominal)	8.06	gpt
Gold Head Grade (Maximum)	20	gpt
Mineralized Material Moisture	5	% w/w
Mineralized Material Specific Gravity	2.9	
Draw Down Angle	50	o
Repose Angle	40	o
Operating Schedule		
Scheduled Operating Days	365	day/yr
Operating Hours	24	hr/day
Plant Availability	92	%
Shifts	2	shift/day
Production Rate		
Plant Feed Rate (Nominal)	1,250	tpd
Plant Feed Rate (Operation)	1,359	tpd
Plant Feed Rate (Future Expandable)	2,500	tpd
Production Target (Dry)	456,250	t/y
Gold Recovery	92.5	%
General Characteristics		
Ambient Temperature	10 to 30	°C
Outdoor Temperature	-36 to 28	°C
Relative Humidity	20 to 100	%
Altitude Above Sea Level	600	m

16.4.2 Mass Balance

Based on a concentrator availability of 92 percent (%) and a nominal feed rate of 1,250 tpd, the throughout target is estimated to be 456,250 tpy; the mass balance is presented in Table 50.

Table 50: Concentrator Mass Balance

Stream Description	Solids (tph)	Solids (m^3/h)	Solution (tph)	Pulp (tph)	Pulp (m^3/h)	Solids (%w/w)
Grinding Circuit						
SAG Mill						
SAG Mill Feed	56.6	19.5	2.98	59.6	22.5	95
SAG Mill Discharge	56.6	19.5	23.9	80.5	43.4	70.3
Ball Mill						
Hydrocyclone Underflow to Grinding Circuit	127.4	43.9	54.6	182	98.5	70
Ball Mill Discharge	127.4	43.9	59.6	187	103.5	68.1
Hydrocyclone Feed Pump Box						
SAG Mill Discharge	56.6	19.5	23.9	80.5	43.4	70.3
Ball Mill Discharge	127.4	43.9	59.6	187	103.5	68.1
Gravity Circuit Tailings	42.5	14.6	66	108.5	80.7	39.1
Hydrocyclone						
Hydrocyclone Feed	226.4	78.1	177.9	404.4	256	56
Hydrocyclone Underflow	169.8	58.6	72.8	242.6	131.4	70
Hydrocyclone Underflow to Grinding Circuit	127.4	43.9	54.6	182	98.5	70
Hydrocyclone Underflow to Gravity Circuit	42.5	14.6	18.2	60.7	32.8	70
Hydrocyclone Overflow	56.6	19.5	105.1	161.7	124.7	35
Gravity Circuit						
Hydrocyclone Underflow to Gravity Circuit	42.5	14.6	18.2	60.7	32.8	70
Gravity Circuit Tailings	42.5	14.6	66	108.5	80.7	39.1
Gravity Table Concentrate	0.0011	0.00011	0.00006	0.001	0.0002	95
Thickening Circuit Trash Screen						
Hydrocyclone Overflow	56.6	19.5	105.1	161.7	124.7	35
Trash Screen Undersize	56.6	19.5	110.1	166.7	129.7	34
Clarifier						
Clarifier Feed (Filtrate + Vacuum Seal Water)	0.014	0.00484	31.2	31.2	31.2	0.04
Clarifier Overflow	-	-	27.9	27.9	27.9	-
Clarifier Underflow	0.014	0.00484	4.12	4.13	4.13	0.34
Pre-Leach Thickener						
Thickener Feed	56.6	19.5	115.4	172	134.9	32.9
Thickener Overflow	0.012	0.0041	58.8	58.8	58.8	0.02
Thickener Underflow	56.6	19.5	56.6	113.2	76.1	50
CIL Circuit						
Pre-Aeration Tank A Feed	56.6	19.5	58.1	114.7	77.6	49.3
Loaded Carbon Screen Undersize	7.94	2.66	8.65	16.6	11.3	47.9
CIL Tank A Feed	56.6	19.5	59	115.6	78.5	49
CIL Circuit Tailings to Safety Screen	56.6	19.5	59	115.6	78.5	49
Loaded Carbon Screen						
Pulp Transfer (with Carbon) to the Loaded Carbon Screen	8.12	2.8	8.46	16.6	11.3	49
Carbon Feed to Acid Wash Column	0.181	0.139	0.725	0.906	0.864	20
Loaded Carbon Screen Undersize	7.94	2.66	8.65	16.6	11.3	47.9
Cyanide Destruction Safety Screen						
CIL Circuit Tailings to Safety Screen	56.6	19.5	59	115.6	78.5	49
Safety Screen Oversize	0.00068	0.000523	0.00008	0.00075	0.0006	90
Safety Screen Undersize	56.6	19.5	60.5	117.1	80	48.3
Cyanide Destruction Tank						
Cyanide Destruction Tank Feed	56.6	19.5	62	118.6	81.5	47.7
Cyanide Destruction Tank Discharge	56.6	19.5	62.1	118.7	81.6	47.7
Buffer Tank Feed	31.1	10.7	35	66.1	45.7	47.1
Tailings Pond Feed	25.5	8.78	53.7	79.1	62.4	32.2

Table 50 (continued): Concentrator Mass Balance

Carbon Regeneration and Attrition Carbon Reactivation Kiln						
Carbon Reactivation Kiln Feed	0.09	0.0692	0.0047	0.095	0.0739	95
Carbon Reactivation Kiln Discharge	0.09	0.0692	-	0.09	0.0692	100
Carbon Quench Tank						
Fresh Carbon Dewatering Screen Oversize	0.0935	0.072	0.0104	0.1039	0.0823	90
Carbon Reactivation Kiln Discharge	0.09	0.0692	-	0.09	0.0692	100
Regenerated Carbon Fines Screen Feed	0.184	0.141	0.734	0.918	0.875	20
Regenerated Carbon Fines Screen						
Regenerated Carbon Fines Screen Feed	0.184	0.141	0.734	0.918	0.875	20
Regenerated Carbon Fines Screen Oversize (to CIL Tank F)	0.182	0.14	0.0321	0.214	0.172	85
Regenerated Carbon Fines Screen Undersize (to carbon fines tank)	0.00152	0.00117	0.742	0.744	0.743	0.2
Acid Wash Column						
Carbon Feed to Acid Wash Column	0.181	0.139	0.725	0.906	0.864	20
Carbon Transferred to Elution	0.181	0.139	0.725	0.906	0.864	20
Acid Wash Flow	-	-	3.03	3.03	2.72	-
Acid Solution Recirculation	-	-	3.03	3.03	2.72	-
Elution Strip Column A						
Carbon Transferred to Elution	0.181	0.139	0.725	0.906	0.864	20
Eluted Carbon Transfer to Unloaded Carbon Dewatering Screen	0.0906	0.0697	0.362	0.453	0.432	20
Eluted Carbon Transfer to Fresh Carbon Dewatering Screen	0.0906	0.0697	0.362	0.453	0.432	20
Barren Strip Solution Flowrate	-	-	8.7	8.7	8.7	-
Eluate Solution to Electro winning (electrowinning feed)	-	-	8.7	8.7	8.7	-
Refinery Electro winning						
Eluate Solution to Electro winning (electrowinning Feed)	-	-	8.7	8.7	8.7	-
Electro winning Solution Discharge Pump to Barren Strip Solution Tank	-	-	8.7	8.7	8.7	-
Sludge Filter Pump Discharge (electrowinning conc.)	0.00036	0.00002	0.0015	0.0018	0.0015	20
Paste Plant Buffer Tank						
Buffer Tank Feed	31.1	10.7	35	66.1	45.7	47.1
Filter Feed	31.1	10.7	35.8	66.9	46.5	46.5
Disc Filter	-	-	-	-	-	
Filter Feed	31.1	10.7	35.8	66.9	46.5	46.5
Cake	31.1	10.7	7.78	38.9	18.5	80
Clarifier Feed (filtrate + vacuum seal water)	0.014	0.00484	28	28	28	0.05
Mixer						
Cake	31.1	10.7	7.78	38.9	18.5	80
Water Addition to the Mixer	-	-	2.97	2.97	2.97	-
Slag Feed	0.903	0.31	-	0.903	0.31	100
Cement Feed	0.226	0.0717	-	0.226	0.0717	100
Paste Production	32.3	11.1	10.8	43	21.9	75
Water Management Tailings Pond						
Tailings Pond Feed	25.5	8.78	53.7	79.1	62.4	32.2
Reclaim Water from the Tailings Pond to the Service Water Tank	-	-	51.3	51.3	51.3	-

16.4.3 Equipment List

The equipment was selected based on design criteria outlined above for a 1,250 tpd throughput and an availability of 92%. Some major equipment items are already designed for the envisaged 2,500 tpd proposed expansion throughput. A major equipment list with a brief description of the equipment is presented in Table 51.

Table 51: Major Process Equipment

Equipment No.	Equipment Name	Equipment Description
1011-BIN-002	Ore Storage Bin	10.7 m (35 ft) diameter by 18.1 m (59.5 ft) high, 2,300 tonnes capacity
1011-CVO-002	SAG Mill Feed Conveyor A	
1011-CVO-003	SAG Mill Feed Conveyor B	
1011-FED-002	Apron Feeder A	
1011-FED-003	Apron Feeder B	
1011-FED-004	Apron Feeder C	
1011-FED-005	Apron Feeder D	
1021-CLU-001	Hydrocyclone Cluster	6 cyclones installed (each 381 mm (15 in) in diameter)
1021-MIL-001	SAG Mill	6.1 m (20 ft) diameter by 3.35 m (11 ft) (F/F), 3.0 m (10 ft) (EGL), 1,790 kW (2,400 HP)
1021-MIL-002	Ball Mill A	3.2 m (10.5 ft) diameter by 4.9 m (16 ft) (F/F), 4.7 m (15.5 ft) (EGL), 597 kW (800 HP)
1022-CLA-001	Clarifier	
1022-SCR-005	Trash Screen	linear, 1.2 m by 2.4 m (4 ft by 8 ft)
1022-THK-001	Pre-Leach Thickener	high rate, 14.0 m (46 ft) diameter
1025-GCO-001	Gravity Concentrator	
1031-SCR-006	Loaded Carbon Screen	Vibrating, 0.9 m by 1.8 m (3 ft by 6 ft)
1031-SCR-010	Regenerated Carbon Fines Screen	Vibrating, 0.9 m by 1.8 m (3 ft by 6 ft)
1031-TNK-004	Pre-Aeration Tank A	8.5 m (28 ft) diameter by 9.6 m (31.5 ft) high
1031-TNK-005	CIL Tank A	8.5 m (28 ft) diameter by 9.6 m (31.5 ft) high
1031-TNK-006	CIL Tank B	8.5 m (28 ft) diameter by 9.6 m (31.5 ft) high
1031-TNK-007	CIL Tank C	8.5 m (28 ft) diameter by 9.6 m (31.5 ft) high
1031-TNK-008	CIL Tank D	8.5 m (28 ft) diameter by 9.6 m (31.5 ft) high
1031-TNK-009	CIL Tank E	8.5 m (28 ft) diameter by 9.6 m (31.5 ft) high
1031-TNK-010	CIL Tank F	8.5 m (28 ft) diameter by 9.6 m (31.5 ft) high
1032-SCR-015	Safety Screen	linear, 1.2 m by 2.4 m (4 ft by 8 ft)
1032-TNK-011	Cyanide Destruction Tank	7.0 m (23 ft) diameter by 7.6 m (25 ft) high
1041-COL-001	Acid Wash Column	4 t
1041-COL-002	Strip Column A	4 t
1041-KIL-001	Carbon Reactivation Kiln	2 t, 7.46 kW (10 HP) (Rotation), 130 kW (heat)
1041-TNK-012	Dilute Acid Tank	
1041-TNK-013	Barren Strip Solution Tank	
1041-TNK-016	Carbon Quench Tank	2 t, 1.5 m (5 ft) diameter by 2.3 m (7.5 ft) high
1051-BIN-011	Cement Storage Bin	
1051-BIN-012	Slag Storage Bin	
1051-FIL-002	Disc Filter A	
1051-FIL-003	Disc Filter B	
1051-MIX-001	Paste Mixer	2 motors at 56 kW (75 HP)
1051-PMP-040	Paste Pump A	
1051-PMP-041	Paste Pump B	
1051-TNK-017	Buffer Tank	
1071-EWC-001	Electrowinning Cell A	
1071-EWC-002	Electrowinning Cell B	
1073-FUR-001	Smelting Furnace	340 kg (750 lb), 125 kW
1073-GTA-001	Gravity Table	shaking table

16.5 Capital Costs

The evaluation was made for a 1,250 tpd throughput concentrator and some major equipment, and provisions were also made for future expansions to 1,800 and 2,500 tpd.

16.5.1 Summary

Table 52 presents the summary of capital costs, which are divided into direct and indirect costs excluding the contingency allowance.

Table 52: Summary of Capital Costs

Description	Cost (C$)
Direct Capital Costs	62,629,613
Materials / Equipment	24,329,340
Civil Structure Architecture	10,731,801
Installation	14,026,690
Piping	4,310,412
Electricity and Control	9,231,370
Indirect Capital Costs	11,027,080
Engineering	4,003,000
Procurement & Construction Management	3,995,300
Site Supervision & Safety Equipment	130,000
Freight	1,560,000
Rentals	1,338,780
Total Pre-Contingency	**73,656,693**

16.5.2 Direct Capital Costs

Table 53 presents the summary of the direct capital costs by concentrator sector.

Table 53: Summary of Direct Capital Costs

Sector	Description	Cost (C$)
Civil, Structure and Architecture		10,731,801
100	Bin	2,697,968
200	Grinding and Thickening	8,087,967
250	Gravity Separation	213,898
300	Carbon-in-leach	2,783,346
400	Cyanide Destruction	528,858
500	Elution	2,569,316
600	Paste Plant	3,011,501
700	Reagents	2,208,274
800	Electrowinning and Refinery	702,914
900	Utilities	1,525,298
Subtotal (Material/Equipment)		**35,061,141**
Installation		14,026,690
Piping		4,310,412
Electricity and Control		9,231,370
Total of Direct Capital Costs		**62,629,613**

16.5.3 Indirect Capital Costs

Table 54 presents the summary of the indirect capital costs. Some items were not included because the evaluation is related to the concentrator operation only.

Table 54: Summary of Indirect Capital Costs

Description	Cost (C$)
Engineering	4,003,000
Procurement, Construction and Management	3,995,300
Temporary Installation During Construction	Not Included
Site Supervision & Safety Equipment	130,000
Miscellaneous Permits	Not Included
Spare Parts	Not Included
Employee Training	Not Included
Freight	1,560,000
Rentals	1,338,780
Total	**11,027,080**

16.6 Operating Costs

The operating costs for a 1,250 tpd concentrator were estimated with the selected flowsheet (as per Section 16.1). The unit operating costs only apply to the nominal throughput and head grade; any changes would have a direct effect on the unit operating costs. The operating costs have been revised according to the updated mass balance and the last equipment list. The operating costs are calculated on the basis of a processing rate of 1,250 tpd. Operating costs are based on the design criteria for the operating schedule.

16.6.1 Concentrator Operating Cost Summary

The operating costs for the concentrator are estimated to be C$26.42/t milled, inclusive of the paste plant other than for cement/binder costs. Without the paste plant operation costs, the concentrator operation costs have been estimated at C$23.63/t milled. Table 55 presents a summary of the operating costs for the concentrator.

Table 55: Summary of Concentrator Operating Costs

Description	Cost (C$)	Units
Manpower		
Plant	2,600,000	C$/y
Maintenance	1,260,000	C$/y
Management	900,000	C$/y
Supplies		
Maintenance	660,000	C$/y
Reagents	2,063,692	C$/y
Consumables	1,983,759	C$/y
Power		
Power	2,584,528	C$/y
Total		
Concentrator Operating Cost Estimate (Binders excluded)	12,051,978	C$/y
Unit Cost per Tonne Milled	26.42	C$/t
Paste Plant Operating Cost Estimate (Binders excluded)	1,272,003	C$/y
Unit Cost per Tonne Milled	2.79	C$/t
Concentrator Operating Cost Estimate without the Paste Plant	10,779,975	C$/y
Unit Cost per Tonne Milled	23.63	C$/t

16.6.2 Paste Plant and Fill System Operating Costs

Other than for cement/binder costs, direct operating costs for the paste plant in the way of labour, maintenance, power, and supplies are included in the concentrator operating costs. The operating cost directly attributable to the paste plant has been estimated at C$2.79/t milled. Fill system costs external to the paste plant, inclusive of underground piping, valving and instrumentation maintenance, fill barricade materials, installation, etc., have been estimated at C$7.17/t milled. Total fill system operation costs have been estimated at C$14.00/t milled, inclusive of cement/binder costs.

16.6.3 Manpower Costs

Given the level of instrumentation and automation of the paste plant, as well as its envisaged location adjacent to the concentrator, it is anticipated that four teams of four operators will be required to operate the concentrator, including the paste plant. In addition, two people for the refinery and four people for the assay laboratory are expected to work 40 hours per week.

Nine people are projected for maintenance to operate the overall concentrator. The maintenance people include six millwrights and three people designated to instrumentation and electrics. Finally, five people are estimated for the management of the concentrator.

The design criteria for the estimation of manpower costs are summarized in Table 56.

Table 56: Manpower Cost Design Criteria

Design Criteria	Plant			Maintenance	Management
	Operation	Refinery	Laboratory		
Annual Salary (C$/y)	110,000	140,000	140,000	140,000	180,000
Hourly Rate (C$/h)	50.37	67.31	67.31	67.31	86.54
Employees	4	2	4	9	5
Yearly Working Hours (h/y)	2,184	2,080	2,080	2,080	2,080
Yearly Man-hour (h/y)	34,944	4,160	8,320	18,720	10,400
Daily Working Hours (h/d)	24	8	8	8	8

Manpower costs are based on the number of employees and on annual salaries including benefits. All employees work 40 hours per week (2,080 hours per year) except the four teams working at the concentrator on shifts covering full time operations (2,184 hours per year).

Table 57 presents the details of the manpower operating costs, with the total of employees at 36 people. The average annual wages including benefits is estimated to be C$133,000.

Table 57: Details of Manpower Operating Costs

Description	Total	C$/y
Plant		2,600,000
Operation	16	1,760,000
Refinery	2	280,000
Assay Laboratory	4	560,000
Maintenance	9	1,260,000
Management	5	900,000
Total	**36**	**4,760,000**

16.6.4 Costs of Maintenance and Supplies

Maintenance Costs

The maintenance costs correspond to 2% of the estimated process equipment costs (C$33M) and are estimated at C$660,000 per year. This represents around C$1.45/t.

Consumables and Reagents Costs

Budget quotations for consumables and reagents were obtained from well-established suppliers and annual quantities were based on process needs as determined by the metallurgical testing and the concentrator mass balance. A rate of C$0.15/kg has been added to the reagents cost to cover freight costs. Table 58 presents the costs for concentrator consumables and reagents.

Table 58: Details of Concentrator Consumables and Reagents Costs

Description	Rate	Units	Source	Annual Quantity*	Units	Cost (C$/y)
Consumables						
Ball Consumption 5 " Grinding Ball	1,560	C$/t	2011 Budget Quote	228,125	kg/y	355,875
Ball Consumption 2 " Grinding Ball	1,305	C$/t	2011 Budget Quote	547,500	kg/y	714,488
Chrome-Moly Steel Liners (SAG Mill)	683,765	C$/unit	2011 Budget Quote	1	units/y	683,765
Rubber Liners (Ball Mill)	196,031	C$/unit	2011 Budget Quote	1	units/y	196,031
Crusher Liners	100,000	C$/unit	2012 Estimate	-	units/y	-
Filter Cloth (Paste Plant)	33,600	C$/y	2011 Estimate	33,600	$/y	33,600
Reagents (Process)						
Flocculant	5.62	C$/kg	2011 Budget Quote	4,563	kg/y	25,641
Sodium Cyanide (NaCN)	3.2	C$/kg	2011 Budget Quote	228,125	kg/y	730,000
Carbon	2.9	C$/kg	2011 Budget Quote	30,000	kg/y	87,000
Lead Nitrate ($PbNO_3$)	5.6	C$/kg	2011 Budget Quote	114,063	kg/y	638,750
Quick Lime (CaO)	0.54	C$/kg	2011 Budget Quote	228,125	kg/y	123,188
Sodium Hydroxyde (NaOH)	0.73	C$/kg	2011 Budget Quote	21,900	kg/y	15,987
Hydrochloric Acid (HCl)	0.75	C$/kg	2011 Budget Quote	68,524	kg/y	51,393
Hydrated Copper Sulfate ($CuSO_4{*}5H_2O$)	4.93	C$/kg	2011 Budget Quote	35,843	kg/y	176,708
SO_2 Liquid	0.61	C$/kg	2011 Budget Quote	325,975	kg/y	198,845
Antiscalant	6.33	C$/kg	2012 Budget Quote	2,556	kg/y	16,179
Total						**4,047,450**

* Including 92% availability

As with the concentrator operating costs, the paste plant operation costs for reagents have been estimated from budget quotations from well-established suppliers. The annual quantities were based on the projected underground fill production schedule. The paste plant consumables as well as the filter cloth are included in the operating costs for the concentrator in Table 58. The estimated cement/binder costs are presented in Table 59.

Table 59: Details of Binder Costs for the Paste Plant

Reagents for Paste Plant	Proportion of Blend (%)	Cost (C$/t)	Annual Quantity**	
3.5% Binder Proportion			(t/y)	(C$/y)
Portland Cement	20	250	1,820	454,876
Slag	80	165	7,278	1,200,872
Total Cost				**1,655,748**
Unit Cost per Tonne Milled (C$/t)				3.63

* 3.5% binder proportion

** Including 92% availability and 55% of production to backfill

The paste plant operation costs for binder/cement are estimated on a production rate that is equal to 55 % of the projected total production. An average of 3.5 % binder is estimated. Average cement/binder costs are estimated at C$3.63/t milled and C$8.80/oz of gold produced.

16.6.5 Power Costs

The estimated required unitary power of each piece of equipment was used to calculate the total power requirement of 3,404 kW. Considering that the motors are normally overdesigned and run in the range of 80% of their full load, the required unitary power for the concentrator is estimated to be 2,723 kW. Based on this number, the annual consumption is estimated to be 25,845,275 kWh for the overall concentrator at 1,250 tpd with a utilization factor of 92%. Using a unit cost of C$0.10/kWh, the annual costs are estimated to be C$2,584,528/yr. Table 60 presents the details of power costs.

Table 60: Details of Power Costs

Description	Value	Units
Total Unitary Power	3,404	kW
Required Unitary Power	2,723	kW
Annual Consumption	25,845,275	kWh/y
Overdesign Factor	80	%
Power Cost	0.1	C$/kWh
Annual Cost	2,584,528	C$/y

16.7 Capital Costs for Concentrator Upgrades to 1,800 and 2,500 tpd

The following evaluation presents the capital costs when upgrading the concentrator throughput to 1,800 and 2,500 tpd.

16.7.1 Expansion Design Criteria

Table 61 presents the expansion design criteria used for the concentrator design.

Table 61: Expansion Design Criteria

Parameter	Value 1800 tpd	Value 2500 tpd	Units
Feed Characteristics			
Gold Head Grade (Nominal)	8.06	8.06	gpt
Gold Head Grade (Maximum)	20	20	gpt
Mineralized Material Moisture	5	5	% w/w
Mineralized Material Specific Gravity	2.9	2.9	
Draw Down Angle	50	50	o
Repose Angle	40	40	o
Operating Schedule			
Scheduled Operating Days	365	365	day/yr
Operating Hours	24	24	hr/day
Plant Availability	92	92	%
Shifts	2	2	shift/day
Production Rate			
Plant Feed Rate (Nominal)	1,800	2,500	tpd
Plant Feed Rate (Operation)	1,957	2,717	tpd
Plant Feed Rate (Future Expandable)	2,500	2,500	tpd
Production Target (Dry)	657,000	912,500	t/y
Gold Recovery	92.5	92.5	%
General Characteristics			
Ambient Temperature	10 to 30	10 to 30	°C
Outdoor Temperature	-36 to 28	-36 to 28	°C
Relative Humidity	20 to 100	20 to 100	%
Altitude Above Sea Level	600	600	m

16.7.2 Summary of the Capital Costs for the Concentrator Upgrades

Table 62 presents the summary of capital costs for the concentrator upgrades. The capital costs are divided into direct and indirect costs excluding the contingency allowance.

Table 62: Summary of Capital Costs for the Concentrator Upgrades

Description	Cost to 1,800 tpd (C$)	Cost to 2,500 tpd (C$)
Direct Capital Costs	7,836,138	13,738,283
Materials / Equipment	3,745,764	6,552,472
Civil Structure Architecture	0	277,500
Installation	2,030,204	3,304,452
Piping	749,153	1,310,494
Electricity and Control	1,311,017	2,293,365
Indirect Capital Costs	1,674,656	2,909,052
Engineering	626,891	1,099,063
Procurement & Construction Management	470,168	824,297
Site Supervision & Safety Equipment	Not Included	Not Included
Freight	374,576	655,247
Rentals	203,020	330,445
Total Pre-Contingency	**9,510,795**	**16,647,335**

16.7.3 Direct Capital Costs for the Concentrator Upgrades

Table 63 presents the summary of the direct capital cost upgrades by concentrator sector.

Table 63: Summary of Direct Capital Costs for the Concentrator Upgrades

Sector	Description	Cost to 1,800 tpd (C$)	Cost to 2,500 tpd (C$)
Civil, Structure and Architecture		0	277,500
1011	Ore Handling	0	0
1021	Grinding	1,876,732	3,819,495
1022	Thickening	41,405	50,642
1025	Gravity	125,291	132,950
1031	Carbon-In-Leach	569,900	854,850
1032	Cyanide Destruction	92,754	113,446
1041	Carbon Elution	655,630	655,630
1051	Paste Fill	0	477,347
1061	Reagents	48,761	59,639
1071	Electrowinning	0	0
1073	Refinery	65,573	80,202
1081	Water & Air Distribution	269,718	308,271
Subtotal (Material/Equipment)		**3,745,764**	**6,829,972**
Installation		2,030,204	3,304,452
Piping		749,153	1,310,494
Electricity and Control		1,311,017	2,293,365
Total of Direct Capital Costs		**7,836,138**	**13,738,283**

To estimate the cost of the equipment needed for the concentrator and the paste plant upgrades, a factor of 1.625 has been applied on equipment cost at 1,250 tpd.

Factors have been applied on equipment costs to estimate the direct costs of installation, electricity, control and civil when not available. The same factors and values have already been used in Section 17.5. The civil factor has been added. The details are presented in Table 64.

Only the pebble crusher requires civil installation.

Table 64: Factorization Method to Calculate Direct Capital Costs for the Concentrator Upgrades

Description	Factorization Method				
	Installation	Piping	Electricity	Control	Civil
Factor on Mechanical Equipment	25%	20%	20%	15%	37%

16.7.4 Indirect Capital Costs for Concentrator Upgrades

Table 65 presents the summary of the indirect capital costs for the concentrator upgrades. Some items were not included because the evaluation is related to the concentrator operation only.

Table 65: Summary of Indirect Capital Costs for Concentrator Upgrades

Description	Cost to 1,800 tpd (C$)	Cost to 2,500 tpd (C$)
Engineering	626,891	1,099,063
Procurement, Construction & Management	470,168	824,297
Temporary Installation During Construction	Not Included	Not Included
Site Supervision & Safety Equipment	Not Included	Not Included
Miscellaneous Permits	Not Included	Not Included
Electricity and Control	Not Included	Not Included
Spare Parts	Not Included	Not Included
Employee Training	Not Included	Not Included
Freight	374,576	655,247
Rentals	203,020	330,445
Total	**1,674,656**	**2,909,052**

Table 66 shows factors that have been applied on equipment costs to estimate the indirect costs of engineering, procurement, construction and management, freight and rentals when not available.

Table 66: Factors used to Estimate Some Indirect Capital Costs for Concentrator Upgrades

Description	Value	Comment
Engineering	8%	Factor on Direct Costs
Procurement & Construction Management	6%	Factor on Direct Costs
Freight	10%	Factor on Mechanical Equipment
Rental	10%	Factor on Installation Cost

16.8 Concentrator Operating Costs at 1,800 and 2,500 tpd Throughputs

The unit operating costs were estimated for concentrator throughputs of 1,800 tpd and 2,500 tpd.

16.8.1 Concentrator Operating Cost for Concentrator Throughputs of 1,800 and 2,500 tpd

The operating costs for the concentrator are estimated to be C$22.68/t milled at 1,800 tpd and C$19.29/t milled at 2,500 tpd, inclusive of the paste plant other than for cement/binder costs. Without the paste plant operation costs, the concentrator operation costs have been estimated at C$20.52/t milled at 1,800 tpd and C$17.58/t milled at 2,500 tpd. Table 67 presents a summary of the operating costs for the concentrator throughputs at 1,800 tpd and 2,500 tpd.

Table 67: Summary of Concentrator Operating Costs at 1,800 and 2,500 tpd Throughputs

Description	Cost (C$)	Units
Throughput at 1,800 tpd		
Manpower		
Plant	2,600,000	C$/y
Maintenance	1,400,000	C$/y
Management	900,000	C$/y
Supplies		
Maintenance	734,915	C$/y
Reagents	2,896,670	C$/y
Consumables	3,007,416	C$/y
Power		
Power	3,361,943	C$/y
Total		
Concentrator Operating Cost Estimate (Binders excluded)	14,900,943	C$/y
Unit Cost per Tonne Milled	22.68	C$/t
Paste Plant Operating Cost Estimate (Binders excluded)	1,418,517	C$/y
Unit Cost per Tonne Milled	2.16	C$/t
Concentrator Operating Cost Estimate without the Paste Plant	13,482,426	C$/y
Unit Cost per Tonne Milled	20.52	C$/t
Throughput at 2,500 tpd		
Manpower		
Plant	2,600,000	
Maintenance	1,680,000	C$/y
Management	900,000	C$/y
Supplies		
Maintenance	791,049	C$/y
Reagents	3,956,824	C$/y
Consumables	3,925,635	C$/y
Power		
Power	3,746,454	C$/y
Total		
Concentrator Operating Cost Estimate (Binders excluded)	17,599,962	C$/y
Unit Cost per Tonne Milled	19.29	C$/t
Paste Plant Operating Cost Estimate (Binders excluded)	1,556,432	C$/y
Unit Cost per Tonne Milled	1.71	C$/t
Concentrator Operating Cost Estimate without the Paste Plant	16,043,530	C$/y
Unit Cost per Tonne Milled	17.58	C$/t

16.8.2 Paste Plant and Fill System Operating Costs for Concentrator Throughputs of 1,800 and 2,500 tpd

Other than for cement/binder costs, direct operating costs for the paste plant in the way of labour, maintenance, power, and supplies are included in the concentrator operating costs.

The operating cost directly attributable to the paste plant has been estimated at C$2.16/t milled at 1,800 tpd and C$1.71/t milled at 2,500 tpd. Fill system costs external to the paste plant, inclusive of underground piping, valving and instrumentation maintenance, fill barricade materials, installation, etc., have been estimated at C$6.73/t milled at 1,800 tpd and C$6.19/t milled at 2,500 tpd. Total fill system operation costs have been estimated at C$8.89/t milled at 1,800 tpd and C$7.90/t milled at 2,500 tpd, inclusive of cement/binder costs.

16.8.3 Manpower Costs for Concentrator Throughputs of 1,800 and 2,500 tpd

Compared to the design criteria at 1,250 tpd throughput, only the number of the maintenance employees was increased for the manpower costs at the throughputs of 1,800 tpd and 2,500 tpd. The design criteria for the estimation of manpower costs are summarized in Table 68.

Table 69 presents the details of the manpower operating costs at 1,800 tpd and 2,500 tpd.

Table 68: Manpower Cost for Concentrator Throughputs of 1,800 and 2,500 tpd

Design Criteria	Plant			Maintenance	Management
	Operation	Refinery	Assay Laboratory		
Throughput at 1,800 tpd					
Annual Salary (C$/y)	110,000	140,000	140,000	140,000	180,000
Hourly Rate (C$/h)	50.37	67.31	67.31	67.31	86.54
Employees	4	2	4	10	5
Yearly Working Hours (h/y)	2,184	2,080	2,080	2,080	2,080
Yearly Man-hour (h/y)	34,944	4,160	8,320	20,800	10,400
Daily Working Hours (h/d)	24	8	8	8	8
Throughput at 2,500 tpd					
Annual Salary (C$/y)	110,000	140,000	140,000	140,000	180,000
Hourly Rate (C$/h)	50.37	67.31	67.31	67.31	86.54
Employees	4	2	4	12	5
Yearly Working Hours (h/y)	2,184	2,080	2,080	2,080	2,080
Yearly Man-hour (h/y)	34,944	4,160	8,320	24,960	10,400
Daily Working Hours (h/d)	24	8	8	8	8

Table 69: Manpower Operating Costs for Concentrator Throughputs of 1,800 and 2,500 tpd

Description	Total	C$/y
Throughput at 1,800 tpd		
Plant		2,600,000
Operation	16	1,760,000
Refinery	2	280,000
Assay Laboratory	4	560,000
Maintenance	10	1,400,000
Management	5	900,000
Total	**37**	**4,900,000**
Throughput at 2,500 tpd		
Plant		2,600,000
Operation	16	1,760,000
Refinery	2	280,000
Assay Laboratory	4	560,000
Maintenance	12	1,680,000
Management	5	900,000
Total	**39**	**5,180,000**

16.8.4 Maintenance and Supplies Costs for Concentrator Throughputs of 1,800 and 2,500 tpd

Maintenance Costs for Concentrator Throughputs of 1,800 and 2,500 tpd

The maintenance costs at 1,800 tpd correspond to 2% of the estimated process equipment costs (C$36.7 million [M]) and are estimated at C$735,000 per year. This represents approximately C$1.12/t.

The maintenance costs at 2,500 tpd correspond to 2% of the estimated process equipment costs (C$39.6 M) and are estimated at C$791,000 per year. This represents approximately C$0.87/t.

Consumables and Reagents Costs for Concentrator Throughputs of 1,800 and 2,500 tpd

The costs for the concentrator consumables and reagents at throughputs of 1,800 tpd and 2,500 tpd are presented in Table 70.

The estimated cement/binder costs at throughputs of 1,800 and 2,500 tpd are presented in Table 71.

Table 70: Details of Concentrator Consumables and Reagents Costs for Concentrator Throughputs of 1,800 and 2,500 tpd

Description	Rate	Units	Source	Annual Quantity*	Units	Cost (C$/y)
Throughput at 1,800 tpd						
Consumables						
Ball Consumption 5 " Grinding Ball	1,560	C$/t	2011 Budget Quote	328,500	kg/y	512,460
Ball Consumption 2 " Grinding Ball	1,305	C$/t	2011 Budget Quote	788,400	kg/y	1,028,862
Chrome-Moly Steel Liners (SAG Mill)	683,765	C$/unit	2011 Budget Quote	1.5	units/y	1,025,648
Rubber Liners (Ball Mill)	196,031	C$/unit	2011 Budget Quote	2	units/y	392,062
Crusher Liners	100,000	C$/unit	2012 Estimate	-	units/y	-
Filter Cloth (Paste Plant)	33,600	C$/y	2011 Estimate	48,384	$/y	48,384
Reagents (Process)						
Flocculant	5.62	C$/kg	2011 Budget Quote	6,570	kg/y	36,923
Sodium Cyanide (NaCN)	3.2	C$/kg	2011 Budget Quote	328,500	kg/y	1,051,200
Carbon	2.9	C$/kg	2011 Budget Quote	30,000	kg/y	87,000
Lead Nitrate ($PbNO_3$)	5.6	C$/kg	2011 Budget Quote	164,250	kg/y	919,800
Quick Lime (CaO)	0.54	C$/kg	2011 Budget Quote	328,500	kg/y	177,390
Sodium Hydroxyde (NaOH)	0.73	C$/kg	2011 Budget Quote	21,900	kg/y	15,987
Hydrochloric Acid (HCl)	0.75	C$/kg	2011 Budget Quote	68,524	kg/y	51,393
Hydrated Copper Sulfate ($CuSO_4$*$5H_2O$)	4.93	C$/kg	2011 Budget Quote	51,615	kg/y	254,460
SO_2 Liquid	0.61	C$/kg	2011 Budget Quote	469,405	kg/y	286,337
Antiscalant	6.33	C$/kg	2012 Budget Quote	2,556	kg/y	16,179
Total						**5,904,085**
Throughput at 2,500 tpd						
Consumables						
Ball Consumption 5 " Grinding Ball	1,560	C$/t	2011 Budget Quote	456,250	kg/y	711,750
Ball Consumption 2 " Grinding Ball	1,305	C$/t	2011 Budget Quote	1 095,000	kg/y	1,428,975
Chrome-Moly Steel Liners (SAG Mill)	683,765	C$/unit	2011 Budget Quote	2	units/y	1,025,648
Rubber Liners (Ball Mill)	196,031	C$/unit	2011 Budget Quote	2	units/y	392,062
Crusher Liners	100,000	C$/unit	2012 Estimate	3	units/y	300,000
Filter Cloth (Paste Plant)	33,600	C$/y	2011 Estimate	67,200	$/y	67,200
Reagents (Process)						
Flocculant	5.62	C$/kg	2011 Budget Quote	9,125	kg/y	51,283
Sodium Cyanide (NaCN)	3.2	C$/kg	2011 Budget Quote	456,250	kg/y	1,460,000
Carbon	2.9	C$/kg	2011 Budget Quote	30,000	kg/y	87,000
Lead Nitrate ($PbNO_3$)	5.6	C$/kg	2011 Budget Quote	228,125	kg/y	1,277,500
Quick Lime (CaO)	0.54	C$/kg	2011 Budget Quote	456,250	kg/y	246,375
Sodium Hydroxyde (NaOH)	0.73	C$/kg	2011 Budget Quote	21,900	kg/y	15,987
Hydrochloric Acid (HCl)	0.75	C$/kg	2011 Budget Quote	68,524	kg/y	51,393
Hydrated Copper Sulfate ($CuSO_4$*$5H_2O$)	4.93	C$/kg	2011 Budget Quote	71,687	kg/y	353,417
SO_2 Liquid	0.61	C$/kg	2011 Budget Quote	651,951	kg/y	397,690
Antiscalant	6.33	C$/kg	2012 Budget Quote	2,556	kg/y	16,179
Total						**7,882,458**

* Including 92% availability

Table 71: Binder Costs for the Paste Plant for Concentrator Throughputs of 1,800 and 2,500 tpd

Reagents for Paste Plant*	Proportion of Blend (%)	Cost (C$/t)	Annual Quantity**	
Throughput at 1,800 tpd				
3.5% Binder Proportion			(t/y)	($/y)
Portland Cement	20	250	2620	655021
Slag	80	165	10,480	1,729,256
Total Cost				**2,384,277**
Unit Cost per Tonne milled(C$/t)				3.63
Throughput at 2,500 tpd				
3.5% Binder Proportion			(t/y)	($/y)
Portland Cement	20	250	3,639	909,752
Slag	80	165	14,556	2,401,745
Total Cost				**3,311,496**
Unit Cost per Tonne milled(C$/t)				3.63

* 3.5% Binder proportion

** Including 92% availability and 55% of production to backfill

16.8.5 Power Costs for Concentrator Throughputs of 1,800 and 2,500 tpd

The power costs at concentrator throughputs of 1,800 and 2,500 tpd are presented in Table 72.

Table 72: Details of Power Costs for Concentrator Throughputs of 1,800 and 2,500 tpd

Description	Value	Units
Throughput at 1,800 tpd		
Total Unitary Power	4,513	kW
Required Unitary Power	3,611	kW
Annual Consumption	33,619,425	kWh/y
Overdesign Factor	80	%
Power Cost	0.1	C$/kWh
Annual Cost	3,361,943	C$/y
Throughput at 2,500 tpd		
Total Unitary Power	5,062	kW
Required Unitary Power	4,050	kW
Annual Consumption	37,464,539	kWh/y
Overdesign Factor	80	%
Power Cost	0.1	C$/kWh
Annual Cost	3,746,454	C$/y

17 Project Infrastructure

17.1 Surface infrastructure

The Phoenix gold project site is accessed via a dedicated 8 kilometres (km) gravel road from Nungesser Road in the Municipality of Red Lake. The road is nominally 10 m wide within a 50 m right-of-way. Entry into the project facilities is controlled by a security gate with 24 hours per day service and perimeter fencing. A network of gravel roads on site provides vehicular access to the project infrastructure. A significant amount of infrastructure is already in place or under construction. The main surface infrastructure includes (Figure 48):

- Hoist, headframe and hoist house;
- Processing plant;
- Tailings management facility;
- Effluent treatment plant;
- Hydroelectric power supply and substation;
- Propane storage tanks;
- Fibre optic communications cable;
- Compressed air supply;
- Process and potable water supplies;
- Sewage works;
- Mine ventilation fans and heater house; and
- Offices, shop, warehouse, core shack, and storage buildings provide housing for related site activities.

17.1.1 Hoisting Facility

The Phoenix shaft hoist is a Canadian Ingersoll Rand double drum hoist with 4.27-metre (14 ft) diameter drums and two 932 KW (1,250 HP) motors. The shaft is currently being deepened and will conclude with the installation of the 680 level loading pocket. Upon completion of the deepening, the hoist ropes, head sheaves, and conveyances used for sinking will be changed over to permanent equipment suitable for production. The production conveyances are a combination skip-cage, operated in balance, with a skip capacity of 10 tonnes. Development rock hoisted to surface is dumped into a bin beside the headframe. Rock not used for site construction work is either stockpiled or stored in the tailings management facility (TMF).

The second phase of shaft deepening will extend the shaft to 1400 m below surface and install a new loading pocket on the 1360 level. Upon completion, the hoist, headframe, and other hoisting facilities will be upgraded to maintain hoisting capacity from the new loading pocket. The main change will be a larger double drum hoist with 4.57-metre (15 ft) diameter drums and larger motors capable of hoisting two 16-tonne capacity skips in balance.

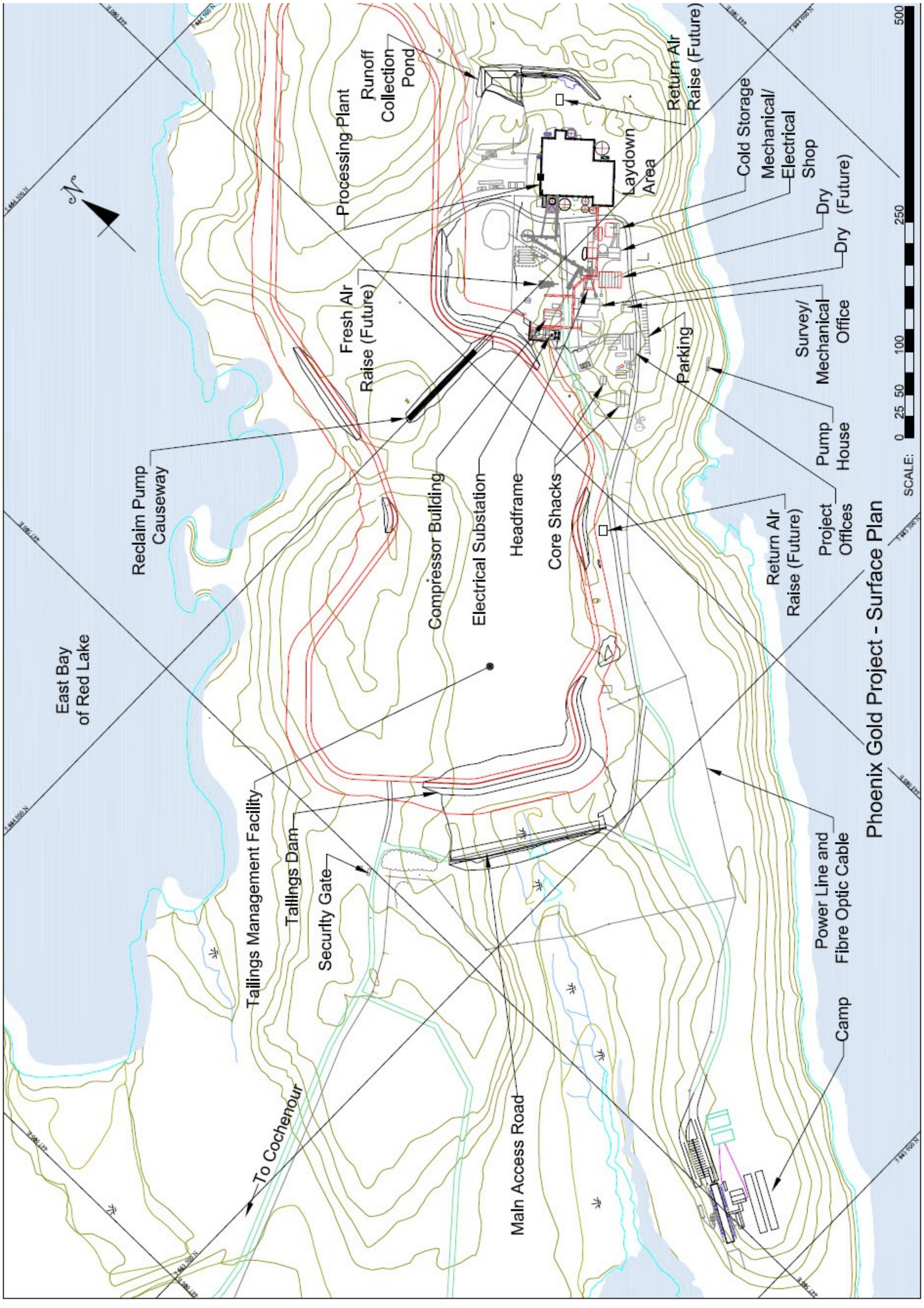


Figure 48: Project Site Plan (Source: SRK 2013)

17.1.2 Processing Plant

The mill is designed for a base processing rate of 1,250 tpd and can be upgraded incrementally to handle a processing rate of 1,800 tpd and 2,500 tpd. The mill is currently under construction and will receive the upgrades necessary to process 1,800 tpd. A further upgrade to a processing capacity of 2,500 tpd is scheduled in 2016 and 2017. Details of the processing facility design and recovery methods are presented in Sections 12 and 16.

This phased upgrade approach defers a portion of the capital cost while allowing for treatment at a lower production rate during the early years of mine life when stoping areas are first being developed and the mill feed ramps up.

The mill will house a paste fill plant that will produce a cemented paste fill product from the tailings. The paste fill will be pumped underground for placement into mined out stopes.

17.1.3 Tailings Management Facility

The historic TMF consisting of a dam and containment pond was constructed by McFinley Mines Ltd. in 1988 and operated under a Certificate of Approval. After test milling a bulk sample in 1989, the facility received minimal use. The TMF was re-activated by Rubicon and the necessary government approvals have been obtained.

The TMF and effluent treatment plant are designed to withstand a 30-day duration 1 in 100 year rain on snow event. The tailings dam will be raised in planned stages periodically over the life of the mine to increase the capacity of the TMF as more tailings are produced. Foundation investigation has been carried out for the current design. For future dam raises, similar foundation investigations will be required to refine the designs. The location of the TMF and related facilities are presented in Figure 48.

The TMF design utilizes mine rock that will be hoisted to surface for construction of the TMF dams, buttresses, etc.

17.1.4 Power and Communications

Hydroelectric Power

In 2011, Rubicon accepted an Offer to Connect from Hydro One, the electricity utility provider, for 5.3 MVA of electricity from the 44 KV grid in the Municipality of Red Lake. Title to the right-of-way for the connection to the grid was secured and power is now fed to the site via a grid connection at the corner of Nungesser Road and the project access road.

The on-site electrical substation currently has a 7.5 MVA capacity that is adequate in the short term. Additional power is available from the 44 KV grid and approximately11 MVA will be required for the operation of the mill, additional compressors, and larger mine ventilation fans. Discussions are ongoing between Rubicon and Hydro One to increase the allotment of electricity to the project.

Propane

Propane is a lower-cost alternative to electricity for certain site power requirements. Propane-fired units are used for building heat and air conditioning. Propane is also used to heat the fresh air supply to the mine during cold weather. This requirement will increase when larger mine air heaters are installed (Section 15.7.4). Propane storage tanks of a suitable size for the application are provided by the supplier and located close to the equipment. Tank sizes and locations will change as surface infrastructure construction evolves and the associated heating requirements change.

Natural Gas

Natural gas is a lower-cost power supply than hydroelectricity and propane and it may be available to the project in the future. Natural gas first became available to the Red Lake area in 2012 with the completion of the Red Lake Pipeline project. The pipeline does not currently extend to the Phoenix gold project.

Fuel Storage

A 30,000-litre above ground diesel fuel storage tank and dispensing station is currently located beside the compressor building. The facility has the requisite spill storage capacity and meets other fuel storage requirements of the Technical Standards & Safety Authority (TSSA).

A small supply of gasoline is currently kept on site for emergency use. There is no regular gasoline dispensing facility.

Communications

Site surface communication is via a VOIP telephone system. The system is connected by a fibre optic cable installed along the same route as the electrical power supply line. Radios are used for travel on the site access road.

Communication underground currently is via a leaky feeder system. A fibre optic cable is being installed in the shaft for future communications and instrumentation applications underground.

17.1.5 Compressed Air Supply

The project currently has two 261 kW (350 HP) air compressors rated at 2,675 m^3/hr (1,575 CFM) each and a small back-up unit. The estimated compressed air requirement for production is 11,961 m^3/hr (7,040 CFM) at 1,800 tpd and 15,920 m^3/hr (9,370 CFM) at 2,250 tpd. To meet this future demand, two 447 kW (600 HP), 5,097 m^3/hr (3000 CFM) compressors will be added. The compressors are currently housed in a temporary shelter and will be moved to a permanent location in the old mill.

17.1.6 Process and Potable Water Supply

Lake water is pumped from the adjacent East Bay of Red Lake to feed the process water and potable water supply systems. The authorized pumping rate from the lake is 400 l/min (106 USgpm).

Process water in the mill and water accumulating in the TMF are designed to be recirculated back into the process water supply system, thereby minimizing the amount of water pumped from the lake.

Potable water for the site is provided by a system of Nano Membrane modules and chlorine disinfection fed by water pumped from East Bay of Red Lake.

17.1.7 Sewage Treatment Facility

The project has two sewage treatment facilities. The dormitory is serviced by a 40,000 l/day (10,567 USgal/day) biofilter system. Sewage from other buildings is processed by a subsurface disposal system rated at 15,000 l/day (3,963 USgal/day).

17.1.8 Mine Ventilation Facilities

The current mine ventilation system and upgrades required for the potential underground mining activities are described in Section 15.7.4.

17.1.9 Other Site Buildings

Facilities provided by other buildings in the vicinity of the Phoenix shaft include:

- Bunkhouse and kitchen;
- Dry;
- Offices;
- Core shack and core storage;
- Maintenance shop;
- Warehouse; and
- Cold storage.

17.1.10 Waste Rock Stockpiles

The current advanced exploration program has resulted in several minor stockpiles of waste rock on the site, each less than 5,000 tonnes. Approximately half of the waste rock from the underground mine development will be hoisted to surface where it will be used for the construction of the TMF dams, buttresses, etc. The broken rock remaining underground will be used as backfill in mined out stopes.

17.1.11 Production Material Stockpiles

There is no plan to stockpile mineralized mine production material. However, it may be temporarily stockpiled on surface to maintain production. For example, after the second phase of shaft deepening is completed and hoisting is interrupted to upgrade the hoisting facilities, a temporary stockpile of mill feed will reduce a possible shutdown period.

17.1.12 Explosives Magazines

No surface explosives magazines are planned. Upon delivery to site, explosives are moved to authorized magazines underground for storage.

17.2 Underground Infrastructure

The underground infrastructure required to support production mining includes material handling facilities, the mine dewatering system, paste fill distribution system, equipment repair shops, ventilation system supply lines for compressed air and process water, electrical power supply, and miscellaneous facilities.

17.2.1 Material Handling

Blasted stope or development material will be trammed by LHD to a rock or production pass for gravity flow to the main tram levels: 305, 610, and 1285. Passes will be driven by Alimak if long or excavated conventionally if short. All passes will be 2.4 m^2 (8 ft^2) and generally dip between 65 and 75 degrees.

Three passes for handling mineralized production material on levels will be spaced to equalize LHD haulage distances on a tonnage weighted basis. Production material below the 1285 level will be trucked up the access ramp and dumped on the 1220 level, one level above the 1285 m tram level. One rock pass centrally located will handle all development rock. The haulage distance for development material will be longer than for the production material. Development crews will have a truck suited to the longer haulage when the distance makes LHD travel inefficient.

The flow of material in the passes will be controlled with anchor chain curtains at suitable transfer locations above the main tram levels and at the loadout chutes. Storage bins above the haulage levels will provide surge capacity to prevent bottlenecks for dumping into the passes. The material will be loaded into trains on the main tram levels for delivery to the shaft dumps.

The rock dump at the shaft will feed into the loading pocket. Rock storage will be provided by the waste pass below the dump. The mineralized production material will feed into a short pass above a 610 mm by 762 mm (24 in by 30 in) jaw crusher. Crushed material will be stored in an oversized pass above the loading pocket.

The Phoenix shaft will be used to move employees and materials between the surface and underground levels as well as hoist mineralized production material and waste rock. Hoisting capacity will be 3,000 tonnes from the 680 m loading pocket, based on 12 hours of hoisting per day with 10-tonne skips. There is also a 3-tonne loading pocket below the 305 level that will be used to hoist initial development rock until a rock pass to the 680 m loading pocket is established. With the second phase of shaft deepening and hoist upgrade completed, hoisting capacity will be 3,000 tonnes from the 1360 m loading pocket, based on 10 hours of hoisting per day with 16-tonne skips.

The amount of production material and development rock scheduled on an annual average daily basis is within the limits of hoisting capacity. The estimated rock hoist assumes that 50% of the development rock generated will remain underground as backfill or be stored underground in inactive headings.

17.2.2 Mine Dewatering

Main dewatering stations will be established on the 610, 976, and 1285 levels and tie into the existing stations on the 122 and 305 levels. Main pump stations will have two sumps with baffles to allow settling of suspended solids and multistage centrifugal pumps. The dual sump arrangement will allow continued water handling while periodic cleaning is done. Water will be pumped to surface in stages from one pump station to the next. Level drainage will be handled by local sumps and pumps on the level as well as by a system of drain holes to direct the drainage water to the main dewatering stations.

Conditions are dry in areas developed to date and groundwater seepage has been very low. However, in the event that water inflows occur in excess of pumping capacity, emergency water storage will be provided by installing a bulkhead, complete with valves, at the bottom of a mined out stope.

Water discharged from the mine will report to the surface water management system for recycling as process water or for treatment. The mine dewatering system will have a rated capacity of 3.0 m^3/min (800 USgpm). The current project permit allows dewatering at a rate of 2,917 l/min (771 USgpm) and a maximum of 2.1M l/day (.56M USgal/day).

17.2.3 Paste Fill Distribution System

Paste fill from surface will initially report underground via a borehole to a location near the shaft. From there, paste will be pumped through a network of pipes and boreholes.

17.2.4 Repair Shops

Two repair shops are planned, one on the 488 level to service track and mobile equipment and a larger facility on the 1098 level to service mobile equipment.

Track levels 610 and 1285 will have a car barn for repair of track equipment.

17.2.5 Miscellaneous Facilities

Other underground facilities not covered above includes but are not limited to storage bays for supplies and equipment, refuge stations, electrical substations, diamond drill stations, local electrical panels, charging stations for LHD and truck batteries, and toilet facilities located convenient to active headings.

18 Market Studies and Contracts

18.1 Market Studies

The Phoenix gold project processing facilities are anticipated to produce high grade gold doré bars at the site, which are readily marketable and are expected to be sold directly to refiners, such as the Royal Canadian Mint, at prevailing spot gold prices.

18.2 Contracts Relevant to Current and Future Project Activities

Rubicon has entered into contracts with suppliers under current market conditions to support the ongoing project activities for a range of work, including shaft deepening, underground development, surface and underground core drilling, surface and underground infrastructure construction, capital equipment, processing facility, etc. SRK considers that that similar contracts could be readily entered into as the project moves forward.

19 Environmental Studies, Permitting, and Social or Community Impact

19.1 General

From an environmental perspective, it is significant that the Phoenix gold project occupies McFinley Peninsula in Red Lake. The land and water adjoining the site is generally used for wilderness/recreation, mineral resource development, and forestry. The project is a brownfield site that was developed intensively in the 1980s prior to acquisition by Rubicon in 2002. Rubicon has assumed full ownership of the historic brownfield site conditions and all known environmental liabilities are being identified and addressed by Rubicon.

The project commenced an Advanced Exploration phase in Q1 2009 and is currently in a Production phase. Development work at the site has continued to present day and is ongoing. The project is currently permitted for commercial production at a rate of 1,250 tpd on an annual average basis.

19.2 Environmental Regulatory Setting

The environmental assessment (EA) and permitting framework for metal mining in Canada is well established. The federal and Ontario provincial EA processes provide a mechanism for reviewing major projects to assess and resolve potential environmental impacts. Following a successful EA, a project undergoes a licensing and permitting phase for the legal and environmental aspects of the project. The project is then regulated through all life cycle phases (construction, operation, closure, and post-closure) by both federal and provincial agencies.

19.2.1 Current Regulatory Status

The Advanced Exploration phase, which commenced in Q1 2009, is in accordance with regulatory approvals. In Q1 2011, a Form 1 Notice of Project Status was submitted to the Ontario Ministry of Northern Development and Mines (MNDM) to move the project from Advanced Exploration status to Production status in accordance with Section 141 of Ontario's *Mining Act*.

Approvals currently in force for the project are presented below in Table 73. The approvals generally relate to a 1,250 tpd production rate and amendments will be required for the planned increase to a 2,250 tpd production rate. It is specifically noted that title was secured to the access road and power line right-of-way for the connection to the grid through Section 21 of the *Public Lands Act* for the Crown land portion and a negotiated agreement was reached with the landowners and leaseholders for the private land portion. Also, the sewage treatment facility is installed, operated and maintained in accordance with a Certificate of Approval issued pursuant to the *Ontario Water Resources Act*.

Table 73: Current Approvals

Permit	Regulatory Agency	Relevant Legislation	Date of Issuance	Rationale
Permit to Take Water 2342-7LWRQU (amended to 7714-7TZR7D)	Ministry of Environment	Ontario Water Resources Act	December 11, 2008 (last amendment December 2, 2011)	Withdrawal of water from shaft.
Permit to Take Water 6020-7LHPX9 (amended to 3585-85KGHG)	Ministry of Environment	Ontario Water Resources Act	November 19, 2008 (last amendment May 21, 2010)	Withdrawal of water from East Bay of Red Lake.
Certificate of Approval - Sewage 4192-7JRJ3L (amended to Environmental Compliance Approval 8358-8FVRDB)	Ministry of Environment	Ontario Water Resources Act and Environmental Protection Act	January 2009 (last amendment 2 December 2011)	Approve sewage works to manage industrial waste water, including the TMF and effluent treatment plants.
Certificate of Approval – Sewage 1384-86HQR8	Ministry of Environment	Ontario Water Resources Act	July 13, 2010	Approve domestic sewage disposal system.
Certificate of Approval - Air 9500-7NGTTC (amended to Environmental Compliance Approval 6656-8RVMES)	Ministry of Environment	Environmental Protection Act	January 27, 2009 (last amendment 28 February 2012)	Approve air emissions from site.
Class Environmental Assessment pursuant to Ontario Regulation 116/01	Ministry of Environment	Environmental Protection Act	April 14, 2010	Allowed Rubicon to seek an amendment to Air Certificate of Approval 9500-7NGTTC for the operation of the supplemental diesel generators (<5MW cumulative capacity) at the Phoenix gold project site.
LRIA Approval No. RL-2009-01	Ministry of Natural Resources	Lakes and Rivers Improvement Act	January 23, 2009	Approve existing containment dams associated with historic TMF. Amendment application currently in progress for future planned TMF construction.
Easement over Crown Land; Consolidated Work Permit and Forest Resource License	Ministry of Natural Resources	Public Lands Act; Crown Forest Sustainability Act, Lakes and Rivers Improvement Act	Applications submitted March 2010 and April 2010, respectively. Letter of Authority to approve work issued January 17, 2011.	Approve easement over Crown owned surface rights; tree harvesting, power line construction and access road upgrade / extension.
Phoenix Gold Project (production) Closure Plan	Ministry of Northern Development and Mines	Mining Act	December 2, 2011	Approve development and closure of the production phase of the project.
Amendment to the Zoning By-Law 1277-10	Municipality of Red Lake	Municipal By-Law 1277-10	Process completed in February 2011	Necessary to change the zoning of the project site to Mineral Mining from Hazard Land. The requested zoning is more appropriate because the entire project site is now subject to a filed closure plan and is no longer considered an Abandoned Mine site. The amended zoning will also allow the issuance of Building Permits for the subject land.

19.2.2 Federal Environmental Assessment Process

In 2011, the Canadian Environmental Assessment Agency (CEAA) confirmed that the 1,250 tpd Production phase of the project will not trigger an EA pursuant to the *Canadian Environmental Assessment Act*. The project has been advanced since this time, and is currently regarded as a mine and is therefore subject to mining sector legislation, including the Metal Mining Effluent Regulations (MMER) that have been promulgated under the *Fisheries Act*.

In the spring of 2012, the1992 *Canadian Environmental Assessment Act* was amended and replaced by CEAA 2012. Two significant results of the updated Act were the redefinition of conditions that would trigger a federal EA and the introduction of legislated time periods within the federal EA process. With respect to the Phoenix gold project, there are two methods for which a federal EA could be required under CEAA 2012:

- A proposed project will require an EA if the project is described in the Regulations Designating Physical Activities; and
- Section 14(2) of CEAA 2012 allows the Minister of Environment to (by order) designate a physical activity that is not prescribed by regulation if, in the Minister’s opinion, either the carrying out of that physical activity may cause adverse environmental effects or public concerns related to those effects may warrant the designation.

With respect to the first method above, the Regulations Designating Physical Activities (2012) are currently being amended. The Regulations Amending the Regulations Designating Physical Activities that have been posted in Canada Gazette for public consultation purposes state:

17. The expansion of an existing

(a) metal mine, other than a rare earth element mine or gold mine, that would result in an increase in the area of mine operations of 50% or more and a total ore production capacity of 3 000 t/day or more;

(b) metal mill that would result in an increase in the area of mine operations of 50% or more and a total ore input capacity of 4 000 t/day or more;

(c) rare earth element mine or gold mine, other than a placer mine, that would result in an increase in the area of mine operations of 50% or more and a total ore production capacity of 600 t/day or more.

The increase in the potential production rate above the currently approved 1,250 tpd annual average rate to 2,250 tpd would not result in a material increase to the area of mine operations. Therefore, the proposed increased production rate would not be cause for a federal EA under the amended regulation.

With respect to the second method above, it is not anticipated that the Minister of the Environment would designate the project for EA due to the relatively minute footprint, the benign nature of concerns expressed by the public to date and the absence of discernible, off-site adverse environmental effects during the operations to date and in the foreseeable future.

19.2.3 Provincial Environmental Assessment Process

The *Ontario Environmental Assessment Act* (EA Act) is administered by the Ontario Ministry of Environment (MOE). The EA Act promotes responsible environmental decision making and ensures that interested parties have an opportunity to comment on projects that may affect them. Interested parties may make a designation request to the MOE to have a project referred to an individual EA. MOE assesses the merits of the request and may make a recommendation to the Minister, as outlined on the MOE website in the Environmental Assessments under Designating Regulations and Voluntary Agreements.

The consultation for the Advanced Exploration permits as well as the numerous other permits issued to date (Table 73) have not resulted in designation requests for an individual EA.

A Class EA was completed in 2011 for a portion of the corridor to connect the project site to Nungesser Road and the work associated therein. No negative comments were received during this process, which was conducted in accordance with the Ministry of Natural Resources (MNR) process (MNR 2003).

A Class EA was completed in 2011 pursuant to Ontario Regulation 116/01 for the use of less than 5 MW of diesel generation at the project site. No negative comments were received during the process.

19.2.4 Environmental Assessment Requirements for the Project

The project is currently permitted for a production rate of 1,250 tpd on an annual average basis.

A federal EA is not anticipated to be required for the production rate increase that is contemplated in this PEA for the reasons listed below:

- The increase in the production rate from 1,250 tpd to 2,250 tpd contemplated in this preliminary economic assessment would not result in a material increase to the bounded area of the project. Therefore, the increased production rate would not be subject to a federal EA under the amended legislation; and
- It is unlikely that the Minister of the Environment would designate the project, pursuant to Section 14(2) of CEAA 2012, due to the relatively minute footprint, the lack of public concern expressed to date and the absence of discernible, off-site adverse environmental effects during the operations to date and in the foreseeable future.

An individual provincial EA is not anticipated to be required for the production rate increase contemplated in this preliminary economic assessment based on the following:

- The EA, consultation and permitting process for the currently approved 1,250 tpd production rate did not result in designation requests for an individual EA; and
- The existing footprint for the project will not be materially affected by an increase in production rate to 2,250 tpd.

Additional Class EA processes are not anticipated for the 2,250 tpd production rate contemplated in this report.

19.3 Environmental Approvals Process

This section describes the federal and provincial approvals process for the production rate increase that is contemplated in this preliminary economic assessment.

19.3.1 Federal Approvals Process

There are no material approval requirements from the federal government for the production rate increase contemplated in this preliminary economic assessment. More specifically, no approvals are required pursuant to the *Fisheries Act*, *Migratory Birds Convention Act*, *Explosives Act*, *Navigable Waters Protection Act*, or the *Species at Risk Act*.

Permits will be sought pursuant to the *Nuclear Source Control Act* for the use of density gauges in the concentrator that utilize nuclear sources.

19.3.2 Provincial Approvals Process

Refined engineering is required for the increased production rate to determine the nature of the amendments to the provincial approvals that will be necessary. As a minimum, it is envisioned that amendments will be required to the approvals listed in Table 74.

Table 74: Anticipated Amendments to Approvals

Permit	Regulatory Agency	Relevant Legislation	Rationale for Permit Issuance	Rationale for Amendment
Permit to Take Water 6020-7LHPX9 (amended to 3585-85KGHG)	Ministry of Environment	Ontario Water Resources Act	Withdrawal of water from East Bay of Red Lake	Increased withdrawal of fresh water from East Bay.
Certificate of Approval - Sewage 4192-7JRJ3L (amended to Environmental Compliance Approval 8358-8FVRDB)	Ministry of Environment	Ontario Water Resources Act and Environmental Protection Act	Approve sewage works to manage industrial waste water, including the TMF and effluent treatment plant.	Increased production rate (administrative amendment), potential changes associated with changes to water balance, approve engineering design for TMF modifications during late stages of the mine life.
Certificate of Approval - Air 9500-7NGTTC (amended to Environmental Compliance Approval 6656-8RVMES)	Ministry of Environment	Environmental Protection Act	Approve air emissions from site	Modifications to mine ventilation and increased return air volume; additional potential sources of fugitive dust and gas.
LRIA Approval No. RL-2009-01	Ministry of Natural Resources	Lakes and Rivers Improvement Act	Approval of TMF dams prior to construction.	On-going TMF dam construction.
Phoenix Gold Project (production) Closure Plan	Ministry of Northern Development and Mines	Mining Act	Approve development and closure of the production phase.	Increased production rate and modified dimensions of the TMF upon closure, along with modified financial assurance requirement. The spatial extent of the project footprint will not be materially affected by the increased production rate.

19.4 Environmental Studies and Management

19.4.1 Environmental Studies

The project closure plan describes current conditions at the property. Baseline monitoring activities and areas of study to date are listed below and have been incorporated into the closure plan:

- Monthly surface water monitoring since 2007 in the vicinity of the project site;
- Semi-annual sampling of groundwater monitoring wells since 2009;
- Archaeological assessment by Ross Associates;
- Annual species at risk assessment by Northern Bioscience;
- Background conditions study by BZ Environmental;
- Aquatic biological assessment by EAG;
- Effluent mixing and plume delineation study by EAG;
- Assessment of risks to the downstream environment from the project by Novatox;
- Hydrogeological characterization by AMEC Earth and Environmental;
- Phase 1 and Phase 2 environmental site assessments by True Grit Consulting;
- Risk assessment of the groundwater and soils at the project site in accordance with O. Regulation 153/04 by Novatox;
- Geochemical characterization of development rock associated with the Advanced Exploration phase by AMEC Earth and Environmental;
- Geochemical characterization of development rock, ore, tailings and quarried surface rock by Chem-Dynamics;
- Geotechnical assessments of underground workings by AMEC Earth and Environmental and AMC Mining Consultants; and
- Project reviews by WESA Consultants and ArrowBlade Consulting Services.

No biological values, i.e., species at risk, ecologically significant features, regionally significant wetlands, significant wildlife habitat, environmentally sensitive areas, etc., that would preclude the re-development of the project site have been identified to date. Ongoing field studies are being conducted with input from the MNR to ensure adherence to the provincial *Endangered Species Act* and the Provincial Policy Statement that has been issued pursuant to Section 3 of the *Planning Act*.

Consultation to date with Aboriginal communities has not identified the presence of cultural heritage values in the vicinity of the project site. In addition, the desktop and field work by Ross Archaeological Research Associates did not identify any areas with a high potential to host cultural heritage values on McFinley Peninsula (Ross Associates 2010). As the project involves the re-development of the existing footprint with only moderate expansion, the potential for impacts to cultural heritage values as a result of the re-development of the brownfield project site are considered to be negligible.

19.4.2 Environmental Management

Rubicon has developed and adheres to an environmental management system (EMS) for the project (Rubicon 2012). The EMS is a simple, plain language tool that has been prepared internally to identify and help manage environmental compliance obligations for the Phoenix property. The extent of the property covered by the EMS includes the project site on McFinley Peninsula as well as off-site areas within the larger Phoenix lands and along the access corridor.

The elements of the EMS are:

- Lists of the relevant legislation, approvals, agreements and documents that contain Rubicon's environmental obligations;
- Division of the property into discrete environmental management areas, each area having a description of the environmental obligations and the corresponding inspection frequency;
- Designated inspectors and documented inspection protocols;
- Procedures to deal with non-compliance issues and conditions; and
- Guidance for documentation requirements, regular updates, and regular internal reporting on performance and auditing.

The EMS identifies the project's compliance obligations and outlines inspection/audit protocols to ensure compliance issues are identified, reported, mitigated and documented. The EMS also addresses community engagement/consultation obligations and includes a commitments registry of Aboriginal agreements, community commitments, etc. The EMS is expected to evolve into a tool to manage corporate social responsibility commitments and obligations.

19.5 Social Setting

This section summarizes Rubicon's consultation and outreach program, which began on a formal basis in 2008.

19.5.1 Aboriginal Consultation

Rubicon has undertaken consultation with Aboriginal communities under the guidance of government agencies. To supplement the guidance, Rubicon commissioned an independent traditional use study that concluded the project site is within the traditional territory of Lac Seul First Nation (LSFN) and Wabauskang First Nation (WFN) (Forbes 2011).

An archaeological study of the McFinley Peninsula was commissioned by Rubicon, comprising a desktop study as well as field work. The study did not identify any sites with a high potential to host cultural heritage value within the development footprint (Ross Associates 2010). Also, as the project involves the re-development of the existing footprint with only moderate expansion, the potential for impacts to cultural heritage value sites as a result of the re-development of the area is considered to be negligible. Accordingly, it has been deemed reasonable to solely engage LSFN, WFN, and the Métis Nation of Ontario (MNO) to further discuss and identify potential cultural heritage values within the development footprint that may warrant protection.

Rubicon commissioned an independent conservative risk assessment to quantify the potential risks to valued environmental components (VECs) identified in Forbes (2011) and to human habitations downstream of Red Lake. The study identified effluent discharge as the sole credible pathway for exposure of the downstream VECs and communities to potential contaminants of concern. The study concluded that the additional, incremental ecological and human health risk that the planned operation of the project poses to the environment downstream of Red Lake is not significant (Novatox 2011). Accordingly, Rubicon has not engaged Aboriginal communities with traditional territory downstream of Red Lake regarding potential impacts as a result of the project.

Rubicon believes in establishing and maintaining meaningful relationships with Aboriginal communities in the Red Lake district where the project is located. In January, 2010 Rubicon became the first public company in the Red Lake district to sign an Exploration Accommodation Agreement with the LSFN. In January of 2012 Rubicon signed a Letter of Intent with the MNO and continues to grow its relationship with the Métis citizens particularly in Region 1, where the project is located. Rubicon has established a successful history of consultation with the local Aboriginal communities and is committed to continued consultation over the life of the project. Rubicon has set a goal to establish benefits agreements with neighbouring Aboriginal communities as the project moves forward.

Rubicon's Aboriginal Policy

Rubicon formalized its Aboriginal policy in 2008, which is reproduced as follows:

> *Rubicon management endeavours to develop and operate sustainable projects that meet high economic, environmental and social standards.*
>
> *We respect and value the communities that neighbour our projects, and recognize the unique status of the First Nations and the Métis Nation of Ontario as the original members of those communities.*
>
> *Whenever our operations might affect First Nations or the Métis Nation of Ontario, Rubicon seeks to develop enduring relationships with those First Nations and the Métis Nation of Ontario built upon trust and respect.*
>
> *Rubicon will:*
>
> - *Develop and maintain mutually beneficial relationships with First Nations and the Métis Nation of Ontario communities in the areas of our operations*
> - *Initiate and maintain ongoing, transparent and good faith communications with First Nations and the Métis Nation of Ontario in the area of our exploration projects.*
> - *Consult with First Nations and the Métis Nation of Ontario communities in the areas of our projects to inform them of our plans and to listen and understand their interests.*
> - *Respect the traditional knowledge, cultural practices, and culturally-significant sites of First Nations and the Métis Nation of Ontario.*

Additional details regarding Rubicon's First Nation agreements, related economic development, capacity funding and outreach efforts will be available on Rubicon's website.

Lawsuit Brought by Wabauskang First Nation in 2012

In December of 2012, Rubicon became aware of the lawsuit brought by WFN against the project. This lawsuit was brought by WFN despite Rubicon's incorporation of all of the environmental mitigation measures proposed by WFN's independent environmental consultant in the closure plan. WFN has chosen to pursue this legal action against the Province of Ontario and Rubicon and discussion of terms of a benefits agreement have been delayed. The most current developments in this matter are:

- The decision by the Ontario Court of Appeal to uphold the jurisdiction of Province of Ontario to issue mining permits (Court of Appeal for Ontario 2013);
- Rubicon's documented record of consultation with WFN; and
- Consent by Rubicon and the Province of Ontario to WFN's request to postpone application for judicial review and further postponement under motion brought by WFN.

19.5.2 Public Consultation

Public information sessions have been held annually in the Red Lake community since 2008. No unresolved negative comments have been received to date during these sessions. Rubicon maintains an open door policy to proactively identify and address stakeholder concerns regarding the project. Formal public consultation to date is summarized in Table 75.

Table 75: Summary of Public Consultation

Date	Summary of Public Consultation that was Undertaken	Summary of Information Provided	Summary of Comments that were Received (if any)
December 2008	Public information session in Cochenour, in accordance with Section 140 Mining Act and Section 8 O. Regulation 240/00.	Overview PowerPoint presentation of the project, including the diesel generator aspect.	No comments received in relation to any aspect of the project. There was a general discussion regarding the modernization of the *Mining Act*.
December 2009	Voluntary Annual Public Information Session. Notice was in general accordance with Section 8 of O. Regulation 240/00.	Overview PowerPoint presentation of the project, including the diesel generator aspect.	No comments received in relation to any aspect of the project.
2008 to 2010	Class EA in accordance with MNR (2003) and Environmental Registry postings.	The Environmental Registry postings include that associated with Air Certificate of Approval 9500-7NGTTC, which included diesel generators.	One comment was received by MNR as part of their Class EA process in March – April 2010. The comment was positive, in support of the project.
September 2010 to March 2011	Notice of Commencement of Screening and Notice of Completion, Class EA process pursuant to O. Regulation 116/01.	Publish newspaper article, mail notices to nearby landowners, notify relevant government agencies.	No comments received in relation to the supplemental diesel generators or the project.
December 2010	Public information session in Red Lake, in accordance with Section 141 Mining Act and Section 8 O. Regulation 240/00. This session was also held as part of the Class EA process required pursuant to O. Regulation 116/01.	Publish newspaper article, mail notices to nearby landowners, notify relevant government agencies.	No written comments. The sole question posed following the session was to inquire if water sampling would be conducted in East Bay and in the future TMF.
December 2011	Voluntary Annual Public Information Session. Notice was in general accordance with Section 8 of O. Regulation 240/00.	Overview PowerPoint presentation of the project, the potential production phase, road upgrades and the PEA.	No comments received in relation to any aspect of the project.
2012	Voluntary Annual Public Information Session. Notice was in general accordance with Section 8 of O. Regulation 240/00.	Published newspaper notice of meeting. Overview PowerPoint presentation of the Project highlighting infrastructure updates (mill foundation and camp), consultation and anticipated update and optimization of the PEA.	No comments received in relation to any aspect of the project.

Public complaints received to date are summarized below:

- One complaint was received by Rubicon in relation to noise from the current construction activities at the project site. Rubicon has planned the project features to mitigate noise emissions and expects that noise emissions will be within government criteria during routine operation of the project site; and

- One comment was received regarding noise from Rubicon's regional exploration activities in close proximity to the project site. The nuisance noise has been effectively mitigated and no subsequent comments have been received.

Rubicon maintains an issues tracking matrix as part of its EMS to effectively track and manage potential concerns as they arise.

19.6 Tailings Disposal

A TMF (tailings management facility) consistent with contemporary regulatory requirements was constructed at the project site by McFinley Mines Ltd. in 1988 in preparation for a bulk-sampling program. The site chosen was an extensive topographic depression lying immediately west of the shaft site, and a retaining dam was constructed to impound tailings and effluents prior to ultimate drainage south into the waters of East Bay. The disposal area received a Certificate of Approval in 1988. The termination of activities on the project in 1989, after test-milling of an estimated 2,500 tons of the bulk sample, resulted in minimal use of this area.

The TMF, and other sewage works, have been re-activated and approved by a Certificate of Approval issued pursuant to the Ontario *Water Resources Act*. The existing dam has also been authorized via an approval issued pursuant to the Ontario *Lakes and Rivers Improvement Act*.

19.7 Environmental Sensitivities

The project site is situated on a peninsula in a valued recreational lake. As such, emphasis has been placed on potential off-site discharges of water, fugitive dust, and noise.

19.7.1 Water Discharge

Responsible management of water discharges will be a priority during production and closure. Project features related to mitigating potential risks to local water quality are summarized in the bullets below.

- An engineered runoff collection system founded in native clay soils or competent bedrock is being constructed around the perimeter of the project site to effectively collect runoff from the operations area where ore, tailings, and waste rock will be handled. Collected runoff will be pumped to the TMF prior to use as process water or treated and discharged to the environment in accordance with regulatory requirements. The effluent treatment system combined with the storage capacity in the TMF will contain and manage a robust environmental design flood;
- The effluent treatment system that treats surplus water from the TMF is regarded as best-in-class and has been proven to be effective for the removal of metals and suspended solids at other sites in Canada;
- The TMF is being designed in accordance with the most robust design criteria in Ontario that are available (MNR 2011; CDA 2007);
- Cyanide will be destroyed using the proven SO_2-air process prior to the tailings being discharged from the mill building envelope;
- Ammonia in mine water due to blasting practices will be managed by worker education/good housekeeping practices, product selection, biological treatment, and other approved treatment methods; and
- Mine water pumped from underground and water reclaimed from the TMF will be recycled for use in the mill to the maximum extent practical to reduce water intake from East Bay.

Rubicon has adopted the precautionary principle regarding the management of tailings and mine rock, as outlined in the Phoenix Project Closure Plan. There are no waste rock stockpiles related to historic mine operations. Tailings and mine rock that poses a potential risk of chemical instability will be consolidated within the TMF footprint during the Operational phase of the project, dewatered to the extent practical and covered with an engineered dry cover at the end of the mine life. These management strategies are described further in the closure plan.

19.7.2 Fugitive Dust

Air emission sources will comprise diesel-fired equipment, diesel generators, propane- and natural-gas-fired combustion heating units, return air from the underground workings, and fugitive dust emissions from vehicle operation, the TMF, crushing and material handling typically associated with an underground mining and milling operation. Practices to minimize fugitive dust are listed in the bullets below:

- Minimize vehicle speed and travel time, utilize dust suppressants on travelled roads, minimize track-out of fines from material handling areas;
- Minimize stockpile size and utilize buildings and treelines as windbreaks to the maximum extent practical;
- Frequent re-location of the tailings discharge location in order to maintain a wetted tailings surface;
- Tackifier and/or binder (cement or fly ash) could be added to deposited tailings to bind together the tailings solids and prevent entrainment by wind;
- Enclose material transfer points and utilized water sprays to suppress dust; and
- Other applicable best practices listed in MOE (2009) and Environment Canada (2009).

Rubicon has developed and implements a best management practices plan for the control of fugitive dust.

19.7.3 Noise

There are permanent and seasonal residential interests on East Bay with potential for exposure to noise. Rubicon is designing infrastructure for the project so that, once constructed, noise emissions from the site will be largely controlled in order to protect the residential interests. Modern noise abatement measures are being integrated into the project design.

19.8 Closure Plan

Rubicon has planned and intends to execute the project in a manner that is consistent with industry best practices and conducive to a “walk-away” closure condition. Chemical and physical stability requirements will be satisfied and monitored in accordance with regulatory requirements and the closure plan, which was filed by MNDM on December 2, 2011 in accordance with Section 141 of the *Mining Act*.

Close-out rehabilitation activities will be completed within approximately 36 months of project closure; major activities are presented below in general chronological order:

- Buildings, trailers, intermodal shipping containers, storage tanks, equipment and any chemicals/consumables will be removed and salvaged, recycled or disposed of in accordance with applicable legislation. Concrete foundations will be demolished to grade as is necessary and used to backfill local depressions;
- Hydrocarbon contaminated soil will be identified and remediated in accordance with applicable legislation (Ontario *Environmental Protection Act*);
- Equipment in the underground workings will be purged of all operating fluids and salvaged to the maximum extent practicable. Consumables will be removed from the underground workings and salvaged;
- Mine openings will be sealed to prevent access, in accordance with O. Regulation 240/00;
- Impounded water within the TMF will be partially treated to remove metals and directed to the underground workings. The dewatered tailings surface will be covered with a dry cover and native topsoil from the established stockpiles and re-vegetated. Downstream embankments will be progressively rehabilitated during the production phase to the extent practical to reduce work that will be required at closure. Post-closure, the spillway channel will be lowered to prevent ponding of runoff water. An engineered overflow channel will be constructed to direct runoff from the surface of the TMF to the downstream toe of the existing dam to effectively return the local drainage pattern to the pre-development condition. While the dry cover is being constructed, the small volume of residual seepage that is expected to be collected in the TMF seepage collection system will be pumped underground. The operation of the TMF seepage collection system will cease in consultation with MOE and MNDM post-closure, once the seepage rate decreases and is demonstrated that it does not pose an environmental risk;
- Ancillary areas within the closure plan area that are overlain with development rock will be scarified and any modest embankments will be sloped for long-term physical stability. These prepared areas will be re-vegetated after placement of native soil from the established stockpiles on McFinley Peninsula. Accumulations of soil-sized particles in rock embankment crevices will be planted with native tree seedlings in accordance with established silvicultural practices;
- The pump in the porous decant towers with the TMF will be operated on a continuous basis to remove the residual pore water to the underground workings until demobilization is completed. The decant towers will be backfilled with development rock when they are decommissioned. The surface drainage sump will be operated on a continuous basis to direct surface runoff and subsurface seepage into the underground workings until demobilization is completed and the dry cover is placed over the plant site and vegetated;
- Site roads will be rehabilitated in general accordance with MNR (1995);
- Pipelines (water, compressed air) on the site will be purged and left in place. Fuel pipelines (propane / natural gas) will be decommissioned as per legislative requirements and Technical Standards and Safety Association standards as applicable;
- Domestic sewage disposal system components will be salvaged. The septic tank will be purged of its contents and backfilled with locally available soil and/or rock;
- Remaining liquid and solid waste at the project site will be removed for recycling or disposal with licensed contractors in accordance with legislative requirements. No mineralized material will be left on site at mine closure; and
- The long term chemical and physical stability monitoring program will be continued to completion, in accordance with the closure plan.

The access road and utility corridor from Nungesser Road to the property are outside the scope of the closure plan. It is envisaged that Rubicon's interest in the right of way, with the road and utilities contained therein, will be transferred to a third party to secure long term access to the on-site accommodations at the south end of peninsula. It is anticipated that these will be sold to a third party and operated as an independent commercial enterprise. The subject parcels where the accommodations are located will be re-zoned as necessary to comply with future requirements of the Municipality of Red Lake.

19.8.1 Closure Cost Estimate

Approximately C$3.3M of financial assurance has been provided to MNDM as part of the closure plan. Prior to generating tailings, Rubicon is required to provide additional financial assurance to provide for the placement of an engineered dry cover over the TMF and plant site at the end of the mine life.

According to the preliminary design that has been prepared in accordance with Ontario Regulation 240/00 promulgated under the *Mining Act*, this is anticipated to be approximately C$4M in 2014. This amount is included in the financial model although Rubicon is evaluating alternatives to cash for the financial assurance provision.

Assuming a conservatively low salvage value of approximately 20 to 30 percent of the capital (supply) cost, the sale of the material assets at closure is anticipated to generate approximately C$9M (not discounted).

20 Capital and Operating Costs

20.1 Capital Costs

Costs incurred during the project's pre-production period have been scheduled and accounted as project capital whereas capital costs during production years are considered sustaining capital. The estimated project capital and sustaining capital costs are summarized in Table 76. The average contingency is 20 percent (%). All costs are in 2013 Canadian dollars (C$) and no escalation was applied to the estimates.

It is expected that the pre-production capital from June 2013 to June 2014 would be C$224M and sustaining capital from July 2014 through to 2026 would be C$426M. There is expected to be no sustaining capital in 2027, which would be the last year of potential production. Sunk capital is not included in the economic model.

The capital cost totals include an estimated C$57M for surface infrastructure, C$113M for the mill, C$248M for underground infrastructure and the shaft, C$203M for underground development, and C$28M for other items.

The average unit capital cost over the life of the mine is C$71.14 per tonne milled. By comparison, the average unit sustaining capital cost while in production is C$46.62 per tonne milled and C$194 per ounce of gold recovered.

Table 76: Capital Cost Estimate Summary

Capital Cost Item	Project Capital	Sustaining Capital	Total Capital
Surface infrastructure	C$42.7	C$14.6	C$57.2
Mill infrastructure and initial supply inventory	C$94.5	C$18.5	C$113.0
Underground infrastructure and shaft	C$15.8	C$232.1	C$247.8
Development – ramps and lateral	C$32.1	C$117.1	C$149.2
Development - raises	C$10.6	C$43.4	C$53.9
Project indirect capital	C$22.9	-	C$22.9
Pre-production definition diamond drilling	C$4.8	-	C$4.8
Royalty purchase	C$0.7	-	C$0.7
Total CAPEX	**C$224.0**	**C$425.7**	**C$649.7**
Total CAPEX per tonne milled		**C$46.62/t**	**C$71.14/t**
Total CAPEX per ounce gold recovered		**C$194/oz**	

Source: SRK (Phoenix_MineEconomics_max2250tpd_5CR008001_v42_dwh_2013-06-20.xls).
Note: Totals may appear to not add up due to rounding.

The project capital costs cover preparations required for production. The pre-production capital work items include:

- Completion of the processing plant;
- TMF construction;
- Deepening the shaft to 710 metres (m);
- Underground development and construction to establish the material handling system and ventilation circuit;

- Development of stopes for initial production; and
- Diamond drilling to define the extent of mineralization in the initial stoping areas.

Sustaining capital expenditures during the production years are attributed mainly to:

- A planned upgrade to the mill to increase throughput from 1,800 to 2,250 tpd;
- TMF dam raises to increase capacity as capacity becomes depleted;
- Ongoing underground development to access additional stoping areas in order to sustain production;
- Equipment purchases and rebuilds for the underground fleet; and
- Deepening the shaft to the 1,400 m for efficient mining of deeper mineralization.

Capital cost has been deferred where possible to improve project economics:

- The purchase of underground equipment for employee crews is reduced by having contractor crews provide the initial underground development with their own equipment;
- The mill upgrade to 2,250 tpd capacity is postponed until required to meet production schedule;
- The second phase of shaft deepening is scheduled for completion in time to develop stopes in the lower area of the deposit; and
- Completion of the final ventilation system is scheduled to coincide with increased ventilation demands of the underground fleet.

20.1.1 Royalty

The water covered portions of the Project, except the small staked claim, are still subject to a NSR royalty to Franco-Nevada Corporation of 2.0%. Advance royalties of US$50,000 are due annually (to a maximum of US$1,000,000 prior to commercial production), of which US$450,000 has been paid to 30 June 2012.

Rubicon has the option to buy back 0.5% NSR of the royalty for US$675,000 at any time, in which case the original 2.0% NSR royalty to Franco-Nevada Corporation would be reduced to a 1.5% NSR royalty.

In this report, the financial model assumes that Rubicon exercises the buy-back option and thus a 1.5% NSR royalty has been applied. Upon a positive production decision, Rubicon would be required to make an additional advance royalty payment of US$675,000 as well as certain of the maximum US$1,000,000 in advance royalty payments described above. Rubicon has confirmed that the annual payments are up to date.

20.1.2 Surface Costs

Surface facilities covered by the project capital costs in Table 76 include all buildings and other facilities for the project listed in Section 17. The pre-production facilities will be supplemented as sustaining capital include a warehouse and a permanent dry.

Specific capital costs are described in the following sections.

Tailings and Water Management

The TMF is a large part of the project site and much of the site infrastructure is associated with the TMF. The project capital for the initial TMF and associated water management is C$7.2M and the sustaining capital for periodic dam raises is C$7.1M.

Project Indirect Costs

Indirect costs are comprised of project management, administration, accommodations, travel, insurance, safety, hydro and fuel. In the pre-production years Jun 2013 to Q2 2014 these costs amount to C$22.9M.

Process Plant Capital Cost

The capital cost for the process plant was estimated by Soutex based on quotes for the processing plant which is under construction.

Table 77 summarises the capital costs for the 1,800 tpd mill, which is under construction, and the subsequent upgrade to 2,500 tpd. Costs are based on the process methodology, flowsheet and detailed capital costs in Section 16.

Table 77: Process Plant Capital Cost Summary

Description	Capital Cost		Total
	Project	Sustaining	
Mill upgraded to 1,800 tpd	C$94.5		C$94.5
Mill expansion from 1,800 to 2,500 tpd		C$18.5	C$18.5
Total			**C$113.0**

Freight and Contingency

An allowance for freight was applied for all capital and sustaining capital equipment and infrastructure costs except underground development. Freight was estimated at 2% of mobile equipment capital costs to cover delivery from point of purchase to site. Freight was included in the estimate for the processing plant in Section 16.

Contingency was added to capital costs to allow for estimating error and to cover additional items that may be required in future but are unknown at this time:

- 20% for underground infrastructure, mobile equipment, construction of the 1,800 tpd mill and mine development;
- 22% for surface infrastructure;
- 25% for project indirect costs; and
- 19% for the mill capital costs.

The low contingency reflects the level of detail in itemizing the capital requirements as well as the level of confidence in the cost estimates. Most estimates were quoted, based on actual site costs or derived from reliable factors.

20.1.3 Underground Costs

Mobile Equipment

The mobile fleet consists of underground track and mobile equipment required for both development and production. The estimates for the equipment were provided largely as budgetary quotes by equipment manufacturers and vendors. The selection of the equipment and the quantities were provided by SRK with input from Rubicon. The initial purchase of each piece of equipment was scheduled depending on its requirement for development or production.

The cost of replacements and rebuilds was scheduled according to each equipment type and the expected wear and tear. The fleet list and total estimated costs for the producing life of the mine are shown in Table 78. The estimate covers company owned equipment only and does not include contractor's equipment.

Table 78: Mobile Equipment Cost Estimate

Description	Quantity	Unit Cost (C$)	Project Capital (C$)	Sustaining Capital (C$)	Total Capital (C$)
2-Boom jumbo	6	1,170,389		7,022,333	7,022,333
Explosive loader	2	435,000		870,000	870,000
3.5 tonne diesel LHD	5	381,000	381,000	1,524,000	1,905,000
Scissor truck	11	344,000		3,784,000	3,784,000
Dry shotcrete unit	3	65,000	130,000	65,000	195,000
6.7 tonne battery LHD	11	741,500	1,483,000	5,783,700	7,266,700
20 tonne battery truck	3	1,033,000	1,033,000	1,756,100	2,789,100
6.7 tonne LHD	15	471,800		6,227,760	6,227,760
20 tonne diesel truck	5	554,200		2,438,480	2,438,480
Boom truck	1	335,000		335,000	335,000
Fuel/lube truck	2	340,000		680,000	680,000
Personnel carrier	5	280,000		1,400,000	1,400,000
Light vehicle	18	85,000		1,530,000	1,530,000
Forklift/backhoe	10	225,000		2,250,000	2,250,000
LHD battery/charger	3	265,000	530,000	265,000	795,000
Truck battery/charger	2	285,000	285,000	285,000	570,000
Longtom (rubber tired)	4	30,000	120,000	-	120,000
Utility vehicle	16	42,000		672,000	672,000
15 ton battery locomotive	3	275,000		825,000	825,000
160-200 ft3 cars	16	39,820		637,120	637,120
Battery charger	2	10,450		20,900	20,900
Subotal	143		**C$3,962,000**	**C$38,371,393**	**C$42,333,393**
Freight at 2%			79,240	767,428	846,668
Contingency at 20%			792,400	7,674,279	8,466,679
Total			**C$4,833,640**	**C$46,813,099**	**C$51,646,739**

Source: SRK

Note: Some equipment sustaining costs are lower than simple multiplication of number of units x unit cost where rebuilds are budgeted in place new equipment.

20.2 Capital and Operating Development Costs

Development advance was estimated according to type and size of excavation and sorted by contractor crew and employee crew. The development cost estimate was based on the unit costs summarized in Table 79 and Table 80. Where unit costs are blank, work was not scheduled to be performed by that crew. Where there are no unit costs for employee crews, all work is done by contractor crews.

The contract crew costs are based on contractor's rates in Red Lake. The employee costs are a 80% of the contractor's costs which is the general factor in Red Lake.

Development was scheduled to have stopes ready for production at least 6 months ahead of start of production. This was to allow flexibility to re-assign production should a development area or production mining front run into an unforeseen difficulty which could otherwise result in a production delay.

Cost estimates for pre-production and sustaining capital for underground construction of pumping stations, refuge stations, electrical substations, ventilation controls, etc., were included in the development design allowance of 15% for ancillary excavations, e.g., refuge stations, electrical substations and storage bays.

Table 79: Capital Development Unit Cost

Capital Development	Size, m		Unit Cost by Crew, (C$/m)	
	Width	Height	Contractor	Employee
Lateral				
Track Drifting	2.75	3.35	C$4,100	C$3,280
Slash 122 level Track Drift for RA	1.50	3.35	C$1,500	C$1,200
Jackleg/Longtom Drifts	3.70	3.70	C$4,200	C$3,360
Jackleg Crusher Chambers	5.00	6.00	C$6,000	-
Jumbo Drift and Ramp	3.70	3.70	C$4,200	C$3,360
Jumbo Drift (air transfer on 610 level)	5.00	5.00	C$5,080	C$4,064
Vertical				
Main FA Raises (pilot & slash)	6.00	3.00	C$7,860	-
Main RA Raises (Alimak)	3.00	3.00	C$4,800	-
Manway Raises (Alimak)	2.40	2.40	C$4,800	-
Open or Drop Raises for Chutes	2.40	2.40	C$2,000	-

Table 80: Operating Development Unit Cost

Operating Development	Size, m		Unit Cost by Crew, (C$/m)	
	Width	Height	Contractor	Employee
Lateral - Waste				
Jackleg/Longtom Drifts	3.70	3.70	C$4,200	C$3,360
Jumbo Drift	3.70	3.70	C$4,200	C$3,360
Jumbo Drift	4.50	5.00	C$5,080	C$4,064
Vertical Waste Development				
LH Access in Waste (Alimak)	2.40	2.40	C$3,200	C$2,560
Lateral - In Stope				
Jackleg/Longtom Drifts	3.70	3.70	-	C$2,800
Vertical In Stope Development				
LH Stope Access & Slot Raise (Alimak)	2.40	2.40	C$3,200	-

20.3 Operating Costs

The project operating cost covers operating development, in stope production drilling, blasting and removal of broken material, underground and surface services and supplies to support mining, operation of the backfill system and processing costs.

The unit costs per tonne for a steady state 1,800 tpd production rate are shown in Table 81. The 1,800 tpd was chosen for the operating cost table to correspond to the initial mill capacity. The production profile ranges from about 1,800 tpd in the early years and peaks at 2,250 tpd. The higher production rates are handled through planned upgrades to the mill and hoist scheduled for completion in 2019 when production increases above 1800 tpd.

The in-stope costs include development in rock and ore to establish levels and sub-levels within the stope boundary for drilling, blasting, removal, ventilation and secondary egress. Longhole drilling for production and blasting are assumed to be performed by contractors. Costs for all mining methods have been blended into a single cost for the purpose of this report. Note that the operating costs shown in Table 81 exclude costs for operating development. In the financial model, total operating cost per tonne includes the operating development cost plus the mining cost.

The general and administrative costs include but are not limited to surface, yard and project support requirements.

Individual operating cost items were made more adaptable to the changing production schedule by assigning a fixed and variable component. In the economic model, the fixed amount is independent of production rate while the variable portion is prorated by tonnage produced each year. This method was also representative of the processing operating costs where the algorithm is within 3% of the operating cost estimates for different milling rates in Section 16.

Table 81: Mine Operating Cost for 1800 tpd Production

Cost Item	Operating Cost (C$/tonne)
LH stoping (90% of production)	14.77
LH stope material removal	2.80
Waste rock haulage	0.28
Cut and Fill (10% of production)	2.19
Secondary ground support	1.20
Backfill	6.73
In Stope Mining Cost	27.96
Power cost	6.00
Ventilation air heating	1.29
Dewatering, process water, electrical	6.20
Definition diamond drilling	3.50
Mobile maintenance	2.60
Track haulage material	1.25
Hoisting	5.42
Waste haulage	0.67
Fleet haul trucks	1.04
Crushing	0.89
Underground Indirect Cost	28.86
Subtotal Underground Cost	**56.82**
Surface and road maintenance	2.00
Water treatment	1.50
Pastefill plant operation	2.79
Tailings facility operation	1.00
Progressive decommissioning	0.20
Employee accommodation/travel	12.21
General and administrative cost	13.72
Subtotal Surface Cost	**33.42**
Processing cost	20.52
Total Operating Cost	**110.76**

Process Costs

Processing operating costs from Section 16 are summarized in Table 82. The labour cost is for a total of 37 process employees (staff + hourly) at 1,800 tpd and 39 at 2,500 tpd. Supplies include maintenance, reagents and consumables. Power cost for the base 1,250 tpd mill was based on an estimated unitary power for the mill of 2,723 kW, 92% utilization, 80% overdesign and annual consumption of 25.8M kWh/yr at C$0.10 per kWh. Costs for the base 1,250 tpd mill were factored up to estimate the costs for the upgraded 1,800 tpd and 2,250 tpd plants. Factors were developed for various components associated with mechanical equipment as well as engineering, procurement, construction management, freight and rentals (Table 64 and Table 66). Although the dollar amounts increase with higher processing rate, the unit costs decrease due to the ability to process relatively more tonnage.

Table 82: Mill Operating Cost

Description	Units	Cost	
		1,800 tpd	2,500 tpd
Manpower	C$/yr	4,900,000	5,180,000
Supplies	C$/yr	6,639,001	8,673,508
Power	C$/yr	3,361,943	3,746,454
Concentrator Operating Cost Estimate (Binders excluded)	C$/yr	14,900,943	17,599,962
Unit Cost per Tonne milled	C$/t	22.68	19.29
Paste Plant Operating Cost Estimate (Binders excluded)	C$/yr	1,418,517	1,556,432
Unit Cost per Tonne milled	C$/t	2.16	1.71
Concentrator Operating Cost Estimate without the Paste Plant	C$/yr	13,482,426	16,043,530
Unit Cost per Tonne milled	C$/t	20.52	17.58

Source: Soutex

21 Economic Analysis

21.1 Assumptions

No escalation was applied to the revenues or costs in the economic model. The net present value (NPV) calculation was based on a discount rate of 5 percent (%) with annual cash flow amounts assumed as occurring at mid-year.

The price of gold for evaluation purposes was US$1,385 per ounce and the exchange (C$/US$) rate of 1.05 was consistent with both the historical rates and consensus forecast.

Franco-Nevada Corporation holds a 2.0% royalty based on payable value of recovered gold after processing costs. The royalty agreement allows for Rubicon to purchase 25% of the royalty for C$708,750 (US$675,000). The purchase lowers the royalty from 2.0% to 1.5%, which increases LoM revenue by C$15.9 million (M) (Table 83). It is assumed that the purchase is made in Q2 2014 before the start of production.

Table 83: Justification to Purchase 25% of Royalty

Royalty %	Basis of Royalty	Royalty Amount
2.0%	C$3,174,852,564	C$63,497,051
1.5%	C$3,174,852,564	C$47,622,788
Increase in revenue from royalty purchase		**C$15,874,263**
Cost to purchase 25% of 2.0% royalty		**C$708,750**
		(US$675,000)

21.2 Financial Evaluation

This preliminary economic assessment is not adequate to confirm the economics of the Phoenix gold project. A preliminary-feasibility study, or feasibility study, as defined in Canadian Securities Administrators National Instrument 43-101, containing mineral reserve estimates is required for this purpose.

Readers are cautioned that the projected mining method, potential production profile and plan and mine plan referred to in this preliminary economic assessment are conceptual in nature. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks which include, but are not limited to, the inclusion of Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets and process plant designs may require additional detailed work and economic analysis and internal studies to ensure satisfactory operational conditions and decisions regarding future targeted production.

This preliminary economic assessment is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the Inferred mineral resources will be converted to the Measured and Indicated categories, that the Measured and Indicated mineral resources will be converted to the proven or probable mineral reserves and there is no certainty that this preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability; the estimate of mineral resources in this report may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The Phoenix gold project's indicative base case economics were evaluated on a post-tax cash flow basis. Highlights of the evaluation are:

- Cumulative cash flow of C$897 M;
- Internal rate of return (IRR) of 27.0% calculated on a mid-period basis; and
- NPV of C$531 M at a discount rate of 5% calculated on a mid-period basis.

The project is expected to generate estimated revenue after refining cost and royalty of C$3,127 M, incur C$1,378 M in operating costs and expend C$650 M in project and sustaining capital. Project capital during the pre-production period from June 2013 through to June 2014 accounts for C$224 M of the total capital. The remaining C$426 M is sustaining capital expended during the production period from July 2014 through to 2026.

There are no capital expenditures scheduled in 2027, the last year of production. Canadian and provincial tax payments amount to C$203 M.

The average daily processing rate is 1,914 tonnes per day over the 13.25 year mine operating life. The estimated total operating and sustaining capital costs and average unit costs over the life of the mine are presented in Table 84. For this base case scenario, the project has a payback period of approximately 3.7 years from the start of commercial production in H2-2014.

Table 84: Unit Operating and Sustaining Capital Costs

Item	LoM Total	LoM Average	
	C$M	C$/t	C$/oz
Operating cost	C$1,378	C$151	C$629
Sustaining capital	C$426	C$47	C$194

The estimated NPV of the Phoenix gold project using a discount rate of 5% was C$650 M on a pre-tax basis and C$531 M after tax. A contingency of 20% was applied to project capital and sustaining capital. There was no contingency on operating cost. The IRR was 28.7% before tax and 27.0% after tax.

The inputs to the financial model and base case financial results are summarized in Table 85. The financial model uses a detailed tax calculation based on Canadian federal and Ontario provincial corporate tax rules. The detailed financial results as well as schedules of production, revenue, costs, and cash flow are presented in Table 86.

Table 85: Financial Evaluation - Summary of Inputs and Results

Description	Unit	Value
Total LoM production tonnes	tonne	9.13M
Total LoM recovered ounces	oz	2.19M
LoM average production rate	tonnes/day	1,914
Peak production rate	tonnes/day	2,250
Peak production rate	tonnes/year	809,890
Mine production life	years	13.25
Gold price*	US$/oz	1,385
Gold payable	%	99.9%
Gold refining	US$/oz	$3.00
Royalty	%	1.5%
Discount rate	%	5.0%
Process recovery (gold)	%	92.5%
Exchange rate**	C$/US$	1.05
Operating days per year	days	360
Mining method	90% longhole; 10% C&F	
LoM average operating cost per tonne	C$/t	$151/t
Total revenue	C$	$3,127M
Operating cost	C$	$1,378M
Project capital (with 20% contingency)	C$	$224M
Sustaining capital (with 20% contingency)	C$	$426M
Cash Flow, pre-tax	C$	$1,100M
Pre-tax IRR	%	28.7%
Pre-tax NPV 5%	C$	$650M
Taxes	C$	$203M
Cash Flow, post-tax	C$	$897M
Post-tax IRR	%	27.0%
Post-tax NPV 5%	C$	$531M

* 30-day average of the London PM fix spot gold price of US$1,385 per ounce as of June 14, 2013.

** C$/US$ consensus exchange rate of 1.05/1.00 (Source: Bloomberg C$/US$ FX Forecast 2013 through 2017 as of June 18, 2013).

Table 86: Conceptual Production and Cash Flow Schedule, Rubicon Gold Project

Item	Units	Inputs	Total	Project Period	Operating Period													
				Year -2	Q1/2-CAP Q3/4-OP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
				2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
Days in Year	360	360		0	180	360	360	360	360	360	360	360	360	360	360	360	360	270
Annual Production	t/year		9,131,926	0	177,657	612,725	645,629	687,030	703,314	701,478	752,285	758,674	809,890	809,877	809,701	809,290	558,435	295,940
Avg daily production processed	tpd			0	987	1,702	1,793	1,908	1,954	1,949	2,090	2,107	2,250	2,250	2,249	2,248	1,551	1,096
Au Grade	gpt		8.06	0.00	7.08	7.54	8.04	7.99	7.47	6.78	7.11	9.90	10.04	8.15	7.67	8.02	7.88	8.04
Au in mill feed	g/year		73,642,455	0	1,256,971	4,618,638	5,189,106	5,491,723	5,256,228	4,757,902	5,346,755	7,513,620	8,131,364	6,604,269	6,207,534	6,488,685	4,401,158	2,378,504
Au in mill feed	oz/year		2,367,658	0	40,413	148,493	166,834	176,563	168,992	152,970	171,902	241,568	261,429	212,332	199,577	208,616	141,500	76,471
Process recovery	%	92.5%		92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%
Au Recovered	g/year		68,119,271	0	1,162,698	4,272,240	4,799,923	5,079,844	4,862,011	4,401,059	4,945,749	6,950,098	7,521,511	6,108,949	5,741,969	6,002,033	4,071,071	2,200,116
Au Recovered	oz/year		2,190,084	0	37,382	137,356	154,321	163,321	156,317	141,497	159,009	223,451	241,822	196,407	184,608	192,970	130,888	70,735
Total Revenue	C$		**$3,127,229,775**	**0**	**53,377,313**	**196,130,633**	**220,355,595**	**233,206,221**	**223,205,950**	**202,044,478**	**227,050,169**	**319,066,168**	**345,298,674**	**280,450,559**	**263,603,157**	**275,542,258**	**186,895,329**	**101,003,269**
Gold price	US$/oz	$1,385		$1,385	$1,385	$1,385	$1,385	$1,385	$1,385	$1,385	$1,385	$1,385	$1,385	$1,385	$1,385	$1,385	$1,385	$1,385
Exchange Rate	C$/US$	$1.05		1.050	1.050	1.050	1.050	1.050	1.050	1.050	1.050	1.050	1.050	1.050	1.050	1.050	1.050	1.050
Recovered metal value	C$		$3,184,929,358	$0	$54,362,162	$199,749,381	$224,421,311	$237,509,040	$227,324,257	$205,772,340	$231,239,404	$324,953,163	$351,669,677	$285,625,069	$268,466,820	$280,626,206	$190,343,679	$102,866,850
Au Payable	%	99.9%		99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%
Au Payable metal	oz/year		2,187,894	-	37,344	137,218	154,167	163,157	156,161	141,356	158,850	223,227	241,580	196,211	184,424	192,777	130,757	70,665
Payable metal value	C$		$3,181,744,429	$0	$54,307,800	$199,549,631	$224,196,889	$237,271,531	$227,096,932	$205,566,568	$231,008,165	$324,628,209	$351,318,007	$285,339,444	$268,198,354	$280,345,579	$190,153,336	$102,763,983
Au Refining	C$	US$3.00	$6,891,865	$0	$117,634	$432,237	$485,625	$513,946	$491,907	$445,271	$500,379	$703,166	$760,978	$618,064	$580,935	$607,247	$411,884	$222,593
Basis for royalty deduction	C$		$3,174,852,564	$0	$54,190,165	$199,117,394	$223,711,264	$236,757,585	$226,605,026	$205,121,297	$230,507,786	$323,925,044	$350,557,030	$284,721,380	$267,617,419	$279,738,333	$189,741,451	$102,541,390
Royalty	C$	1.5%	$47,622,788	$0.00	$812,852	$2,986,761	$3,355,669	$3,551,364	$3,399,075	$3,076,819	$3,457,617	$4,858,876	$5,258,355	$4,270,821	$4,014,261	$4,196,075	$2,846,122	$1,538,121
Operating Cost	C$		**$1,377,538,296**		**$43,177,064**	**$94,208,958**	**$102,020,566**	**$101,251,540**	**$107,224,801**	**$106,332,067**	**$115,826,868**	**$117,155,683**	**$117,917,832**	**$115,260,758**	**$119,451,098**	**$117,470,074**	**$89,014,748**	**$31,226,240**
Mining OPEX - Mineralized Material	C$		$957,747,038		$30,473,181	$62,830,675	$71,655,610	$73,700,139	$74,504,297	$74,413,642	$76,922,654	$77,238,193	$79,767,393	$79,766,758	$79,758,047	$79,737,754	$67,349,605	$29,629,089
Operating Development Waste	C$		$182,790,604	$0.6909	$7,113,452	$14,554,406	$13,429,601	$12,318,226	$13,645,650	$13,057,423	$17,322,345	$17,774,957	$16,030,753	$15,434,662	$17,265,949	$16,294,462	$8,370,132	$178,586
Operating Development Mineralized Material	C$		$237,000,654		$5,590,431	$16,823,876	$16,935,354	$15,233,175	$19,074,854	$18,861,002	$21,581,869	$22,142,533	$22,119,686	$20,059,338	$22,427,102	$21,437,858	$13,295,010	$1,418,565
OPEX per tonne Milled	C$/t		$151/t		$243/t	$154/t	$158/t	$147/t	$152/t	$152/t	$154/t	$154/t	$146/t	$142/t	$148/t	$145/t	$159/t	$106/t
OPEX per ounce Recovered	C$/oz		$629/oz		$1,155/oz	$686/oz	$661/oz	$620/oz	$686/oz	$751/oz	$728/oz	$524/oz	$488/oz	$587/oz	$647/oz	$609/oz	$680/oz	$441/oz
Capital Cost			**$649,668,769**	**$126,204,546**	**$146,390,294**	**$41,539,559**	**$22,949,468**	**$37,520,426**	**$42,783,883**	**$78,602,426**	**$64,726,014**	**$36,230,089**	**$32,532,930**	**$10,261,410**	**$5,055,127**	**$3,325,461**	**$1,547,136**	**$-**
Surface Infrastructure	C$		$57,242,209	$20,018,955	$23,235,605	$4,133,099	$3,645,099	$0	$701,302	$0	$1,772,477	$0	$1,869,430	$0	$1,866,240	$0	$0	$0
UG Infrastructure and shaft	C$		$247,841,574	$6,170,076	$24,207,925	$12,712,716	$7,371,668	$28,657,484	$26,780,838	$62,419,204	$40,770,108	$17,267,107	$9,971,177	$3,451,787	$3,188,887	$3,325,461	$1,547,136	$0
UG Develop. - ramps, drifts, raises	C$		$203,159,542	$12,101,451	$52,572,717	$24,693,744	$6,222,701	$7,435,607	$15,301,742	$16,183,222	$22,183,428	$18,962,982	$20,692,323	$6,809,623	$0	$0	$0	$0
Mill Infrastructure	C$		$113,043,445	$74,827,613	$31,078,497	$0	$5,710,000	$1,427,335	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Pre-production definition drilling	C$		$4,800,000	$1,200,000	$3,600,000													
Project Indirects	C$		$22,873,250	$11,886,450	$10,986,800	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Royalty purchase - 25% of 2% NSR	C$		$708,750		$708,750													
Total project capital	C$		$223,962,428	$126,204,546	$97,757,882													
Total sustaining capital	C$		$425,706,341		$48,632,412	$41,539,559	$22,949,468	$37,520,426	$42,783,883	$78,602,426	$64,726,014	$36,230,089	$32,532,930	$10,261,410	$5,055,127	$3,325,461	$1,547,136	$0
Project + sustaining CAPEX per t milled	C$/t		$71.14/t															
Sustaining CAPEX per t milled	C$/t		$46.62/t			$67.79/t	$35.55/t	$54.61/t	$60.83/t	$112.05/t	$86.04/t	$47.75/t	$40.17/t	$12.67/t	$6.24/t	$4.11/t	$2.77/t	$0.00/t
Sustaining CAPEX per oz recovered	C$/oz		$194/oz															
Net Cash Contribution Pre-Tax	C$		**$1,100,022,710**	**$(126,204,546)**	**$(136,190,045)**	**$60,382,116**	**$95,385,562**	**$94,434,255**	**$73,197,267**	**$17,109,985**	**$46,497,288**	**$165,680,396**	**$194,847,912**	**$154,928,392**	**$139,096,932**	**$154,746,723**	**$96,333,445**	**$69,777,029**
Cumulative cash flow before tax	C$		1,100,022,710	$(126,204,546)	$(262,394,591)	$(202,012,475)	$(106,626,913)	$(12,192,658)	$61,004,609	$78,114,594	$124,611,882	$290,292,278	$485,140,189	$640,068,581	$779,165,513	$933,912,236	$1,030,245,681	$1,100,022,710
Internal Rate of Return (IRR)	28.7%																	
Net Present Value (NPV)			$649,960,397	$(124,662,013)	$(129,678,801)	$54,757,379	$82,375,609	$77,665,337	$57,332,811	$12,763,468	$33,031,506	$112,086,561	$125,541,952	$95,068,105	$81,283,623	$86,116,949	$51,056,962	$35,220,949
Discount rate	5.0%																	
Total corporate and Ontario mining taxes			$203,072,239	$0	$0	$0	$0	$0	$0	$0	$0	$6,654,326	$36,353,068	$42,247,467	$36,066,912	$41,959,808	$22,012,947	$17,777,710
Net Cash Contribution Post-Tax	C$		**$896,950,472**	$(126,204,546)	$(136,190,045)	$60,382,116	$95,385,562	$94,434,255	$73,197,267	$17,109,985	$46,497,288	$159,026,069	$158,494,844	$112,680,925	$103,030,020	$112,786,915	$74,320,499	$51,999,319
Cumulative Cash Flow after Tax	C$			$(126,204,546)	$(262,394,591)	$(202,012,475)	$(106,626,913)	$(12,192,658)	$61,004,609	$78,114,594	$124,611,882	$283,637,951	$442,132,794	$554,813,719	$657,843,739	$770,630,654	$844,951,153	$896,950,472
Internal Rate of Return (IRR)	27.0%																	
Net Present Value (NPV)			**$531,044,381**	**$(124,662,013)**	**$(129,678,801)**	**$54,757,379**	**$82,375,609**	**$77,665,337**	**$57,332,811**	**$12,763,468**	**$33,031,506**	**$107,584,758**	**$102,119,402**	**$69,143,956**	**$60,207,318**	**$62,766,208**	**$39,390,046**	**$26,247,396**
Discount Rate	5.0%																	

Source: SRK

Readers are cautioned that the projected mining method, potential production profile and plan and mine plan referred to in this preliminary economic assessment are conceptual in nature. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks which include, but are not limited to, the inclusion of Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets and process plant designs may require additional detailed work and economic analysis and internal studies to ensure satisfactory operational conditions and decisions regarding future targeted production.

This preliminary economic assessment is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the Inferred resources will be converted to the Measured and Indicated categories and that the Measured and Indicated mineral resources will be converted to the Proven and Probable mineral reserve categories and there is no certainty that this preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability; the estimate of mineral resources in this report may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

21.3 Sensitivity

The post-tax NPV for the Phoenix gold project of C$531 M was evaluated for sensitivity to changes in gold price, gold grade, operating cost, capital expenditures, and process recovery. The sensitivity of NPV to these parameters is presented in Table 87, Figure 49, and Figure 50.

Sensitivity analysis indicates that NPV is most sensitive, and almost equally sensitive, to changes in gold price and gold grade, which are both related to revenue. The gold price and gold grade curves in Figure 49 overlap as they have almost equal effect on NPV.

A 20% increase in gold price from C$1,385/oz to C$1,662/oz results in a 59% increase in post-tax NPV to C$845 M. This compares to a 20% decrease in gold price to C$1,108/oz which yields a 61% decrease in post-tax NPV to C$205 M.

With respect to gold grade, a 20% increase to 9.68 gpt gold results in a 59% higher post-tax NPV of C$845 M while a 20% decrease to 6.45 gpt gold yields a 61% lower post-tax NPV of C$206 M.

Table 87: Sensitivity of NPV with Economic Variables (Post-Tax)

Variable	Range	Value	NPV (C$M)
Base case post-tax NPV at 5% discount rate			531.0
	20%	1,662	845.8
	10%	1,524	689.5
Long term gold price ($US/ounce)	0%	1,385	531.0
	-10%	1,247	371.5
	-20%	1,108	205.4
	20%	9.68	845.1
	10%	8.87	689.1
Gold grade (grams/tonne)	0%	8.06	531.0
	-10%	7.26	371.8
	-20%	6.45	206.2
	20%	780	430.9
	10%	715	481.4
Capital cost (C$ million)	0%	650	531.0
	-10%	585	581.1
	-20%	520	631.3
	20%	755	387.7
	10%	692	459.9
Operating cost (C$ million)	0%	629	531.0
	-10%	566	602.3
	-20%	503	673.1
	4%	96.2%	594.4
	3%	95.3%	578.6
Process recovery (percentage)	2%	94.4%	562.7
	0%	92.5%	531.0
	-5%	87.9%	451.8
Limits	**Max Worse Case**		**205.4**
	Max Best Case		**845.8**

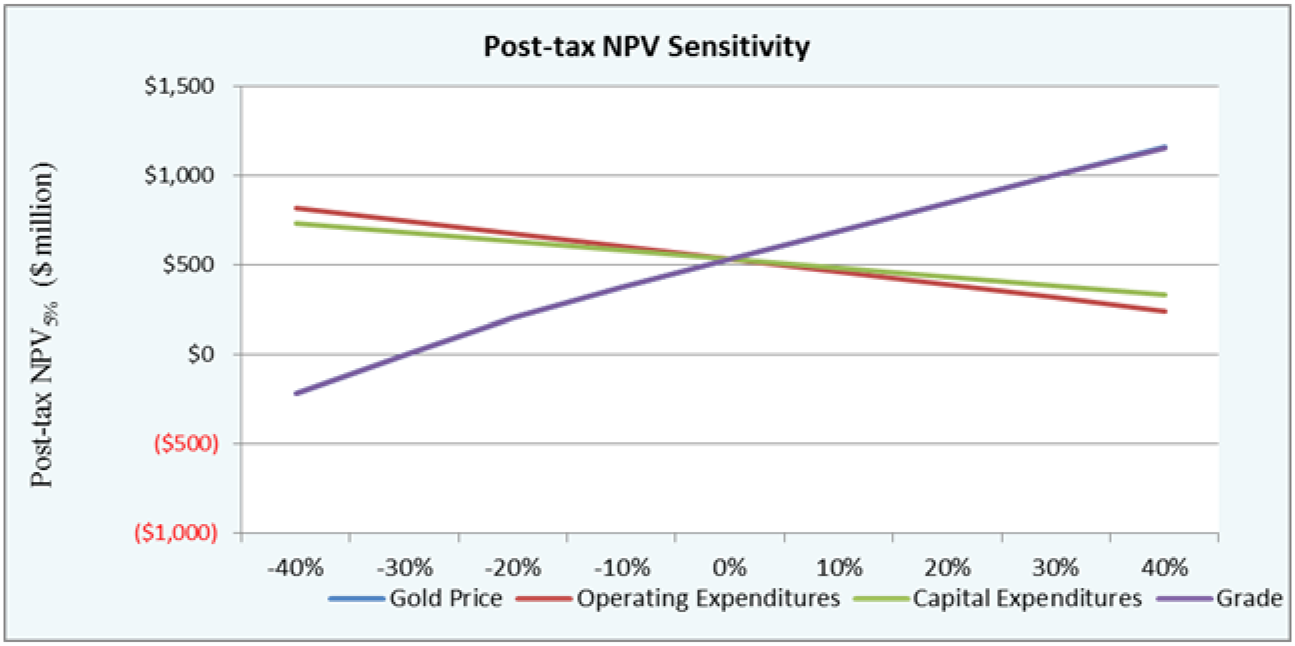


Figure 49: Sensitivity of NPV to Economic Variables (Post-Tax)

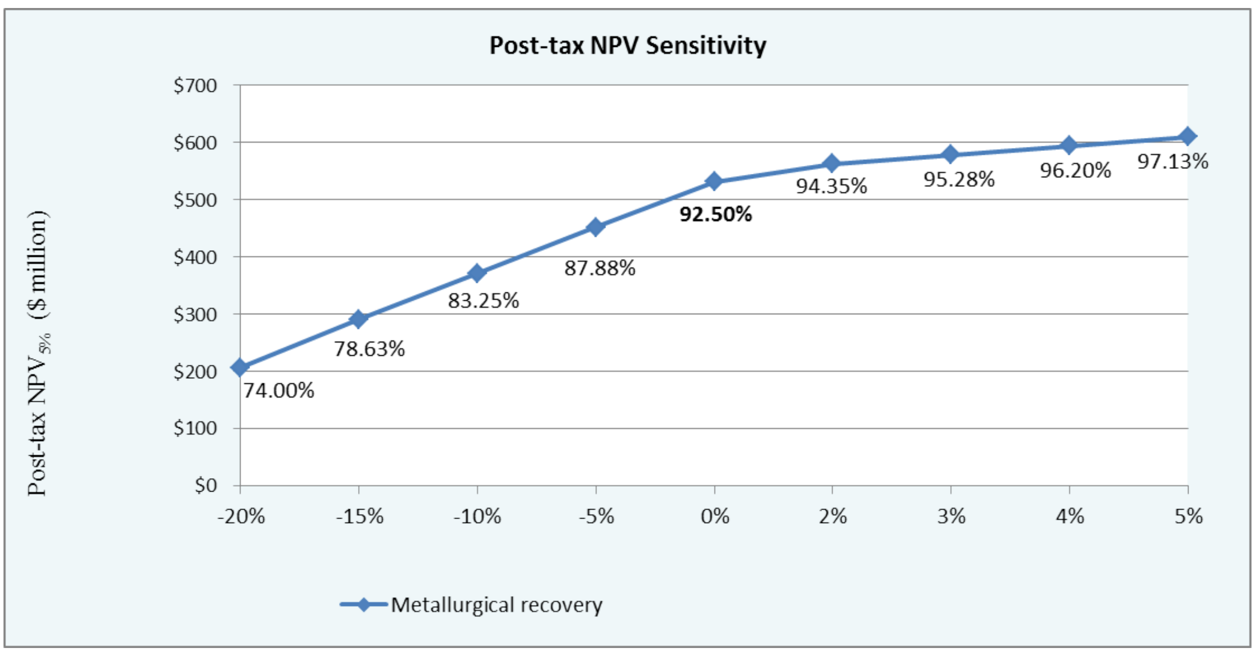


Figure 50: Sensitivity of NPV to Metallurgical Recovery (Post-Tax)

21.4 Payback Period

The payback for the Phoenix gold project is approximately 3.7 years from the assumed targeted start of production in Q3 2014 to mid-2018 (Figure 51). The payback period is defined as the period of time from the start of production of the process plant to the date when the initial expenditures incurred in developing the project are recovered, i.e., the time when cumulative cash flow reaches zero. The payback period is calculated using the undiscounted, post-tax, net cash flow (NCF).

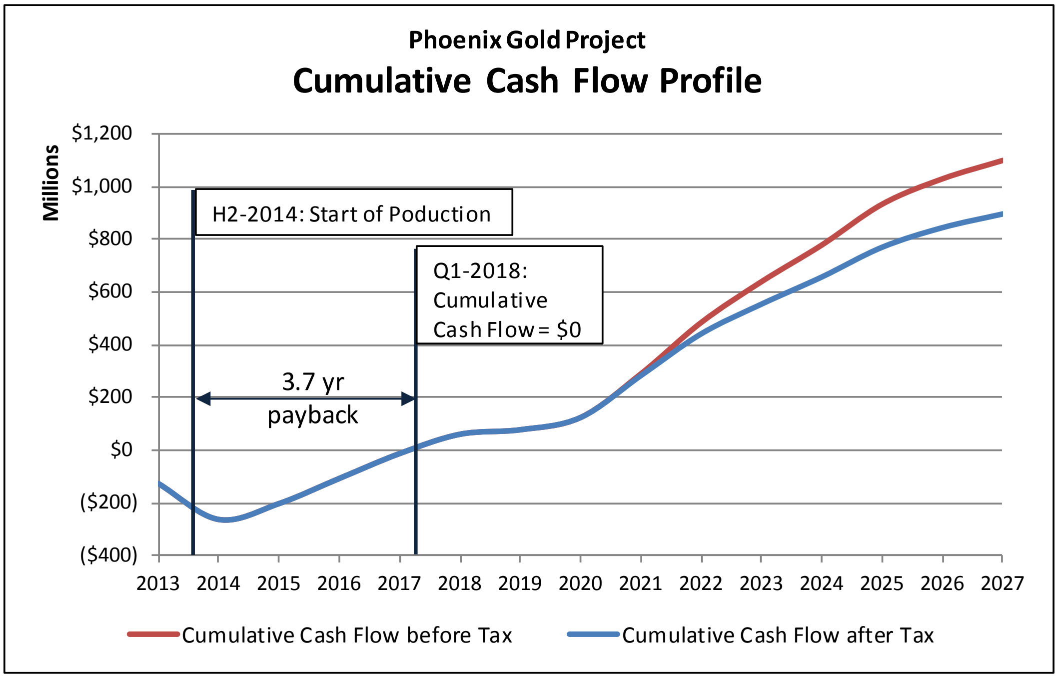


Figure 51: Cash Flow Profile and Payback Period

22 Adjacent Properties

There are no adjacent properties that are considered relevant to this technical report.

23 Other Relevant Data and Information

SRK is not aware of any other data or information that is relevant to the Phoenix gold project.

24 Interpretation and Conclusions

This preliminary economic assessment is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. The quantity and grade of reported Inferred mineral resources in this preliminary economic assessment are uncertain in nature and there has been insufficient exploration to define these Inferred mineral resources as an Indicated or Measured mineral resource and it is uncertain if further exploration will allow conversion to the Measured and Indicated categories or that the Measured and Indicated mineral resources will be converted to the Proven or Probable mineral reserves. There is no certainty that this preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability; the estimate of mineral resources in this preliminary economic assessment may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The projected mining method, potential production profile and plan and mine plan referred to in this preliminary economic assessment are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a feasibility study carries additional potential risks discussed herein. Mine design and mining schedules, metallurgical flow sheets and process plant designs may require additional detailed work and economic analysis and internal studies to ensure satisfactory operational conditions and decisions regarding future targeted production.

There is no preliminary-feasibility or feasibility study with respect to the Phoenix gold project. The results of this preliminary economic assessment support continued advancement of the project and work related to further technical studies. Underground development at the project is currently in progress to support the ongoing underground exploration drilling program. In addition, significant surface and underground facilities and infrastructure to support future potential production are either installed or under construction.

The following sections present the conclusions of this study and associated project risks and opportunities.

24.1 Sampling Method, Approach and Analyses

It is SRK’s opinion that Rubicon used industry best practices to collect, handle, and assay core samples collected during the 2008 to 2012 period. All drilling and sampling was conducted by appropriately qualified personnel under the direct supervision of appropriately qualified geologists.

Rubicon has partly relied on the internal quality control measures of the accredited laboratory; however, it has also implemented external analytical quality control measures, consisting of inserting control samples (blanks and certified reference material, and field duplicates) with each batch of core drilling samples submitted for assaying.

In the opinion of SRK, the field sampling and assaying procedures used by Rubicon meet industry best practices.

24.2 Data Verification

It is SRK's opinion that gold grades can be reasonably reproduced, suggesting that the assay results reported by the primary assay laboratories are sufficiently reliable for the resource estimation used in this preliminary economic evaluation.

On completion of the validation procedures, SRK concludes that the digital database for the Phoenix gold project is also reliable for resource estimation

24.3 Sample Preparation, Analyses and Security

In the opinion of SRK, Rubicon exploration personnel used care in the collection and management of field and assaying exploration data. In addition, the sampling preparation, security and analytical procedures used by Rubicon are consistent with generally accepted industry best practices and are therefore adequate for the purpose of mineral resource estimation.

24.4 Mineral Resources

SRK reviewed and audited the exploration data available for the Phoenix gold project. SRK is of the opinion that the exploration data are sufficiently reliable to interpret with confidence the boundaries of the gold mineralization and support evaluation and classification of mineral resources in accordance with generally accepted CIM *Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines* and CIM *Definition Standards for Mineral Resources and Mineral Reserves*.

The drilling database includes information from 820 core boreholes (355,611 metres [m]) and considers drilling information available to November 1, 2012. Unsampled intervals were assigned a 0.0 gpt gold value.

The mineral resources were evaluated using a geostatistical block modelling approach constrained by 56 gold mineralization wireframes. High grade subdomains were defined only in areas of high drilling density. It is possible that additional drilling may permit the extension of currently defined high grade subdomains.

The Mineral Resource Statement effective June 24, 2013 is tabulated in Table 88.

The Mineral Resource Statement documented herein represents a significant change relative to the previous Mineral Resource Statement released in June 2011. It is significant to note the material increase in tonnages reported in 2013 at reduced grades compared to the tonnages reported in 2011. Reported tonnages and grades are the product of contrasting mineral resource estimation methodologies applied during the two studies. The additional infill drilling since 2011 contributed to the significant increase in the reported Indicated mineral resources in 2013.

There is an opportunity to expand the currently reported mineral resource in areas adjacent to mineral resource blocks where although boreholes are present with elevated gold grades, borehole density is insufficient to satisfy the applied mineral resource criteria. Targeting these areas for follow-up drilling has a high probability to increase the mineral resource.

SRK considers that the mineral resource model document herein is sufficiently reliable to support engineering and design studies to evaluate the potential viability of a mining project at a conceptual level.

Table 88: Mineral Resource Statement*, Phoenix Gold Project, Ontario, SRK Consulting (Canada) Inc., June 24, 2013

Domain	Resource Category	Quantity (000't)	Grade Au (gpt)	Contained Gold (000'oz)
Main#	Measured	-	-	-
	Indicated	4,120	8.52	1,129
	Measured + Indicated	4,120	8.52	1,129
	Inferred	6,027	9.49	1,839
HW	Measured	-	-	-
	Indicated	-	-	-
	Measured + Indicated	-	-	-
	Inferred	151	5.21	25
External	Measured	-	-	-
	Indicated	-	-	-
	Measured + Indicated	-	-	-
	Inferred	1,274	8.66	355
Combined	Measured	-	-	-
	Indicated	4,120	8.52	1,129
	Measured + Indicated	4,120	8.52	1,129
	Inferred	7,452	9.26	2,219

* Mineral resources are not mineral reserves and do not have a demonstrated economic viability. All figures have been rounded to reflect the relative accuracy of the estimates. Reported at a cut-off grade of 4.0 gpt gold and assuming an underground extraction scenario, a gold price of US$1,500 per ounce, and metallurgical recovery of 92.5%.

The Main domain includes the Main 45 domain.

24.5 Mine Geotechnical and Hydrogeological

Additional data on rock mass properties is required and presents an opportunity to improve the mine design.

Additional geotechnical data is also required to improve the mine design. Stopes may be more or less stable than currently estimated and external dilution may be highly variable as well as dilution grade.

In the event of water inflow from an ungrouted core borehole or geological structure, provisions have been made for emergency water storage behind bulkheads in mined out stopes until the borehole can be properly grouted. Mining near surface could weaken the crown pillar to the extent that mining induced cracks become water bearing. Currently, a 45 m crown pillar has been planned and other measures such as conventional grout curtains will mitigate this risk.

The risk of uncontrolled stope failures due to unknown structural weaknesses in the rock mass cannot be predicted. These conditions can cause short-term production losses.

The crown pillar was not included in the proposed mining plan due to lack of information about the extent of mineability. The geotechnical properties of the crown pillar should be assessed to determine the safe limits of mineability with respect to stope dimensions.

24.6 Conceptual Mining

SRK considers that a key challenge for the project will be to thoroughly understand the character of the gold mineralization and, from this, to develop the ability to readily locate and efficiently mine the deposit. SRK believes that this aspect of the project probably presents both its greatest risk and greatest reward potential.

A part of the above key challenge is the uncertainty related to grade control and stope shape. A procedure for identifying the outline of gold mineralization prior to and during stope development and mining will need to be devised and followed diligently. Once a longhole stope has been designed, development and longhole drilling will be done under survey control. Cut and fill stopes will likely be mined under geology control. Block model reconciliation to actual performance will be critical to ongoing improvements in block model accuracy at the stope level.

A further feature of this style of deposit is that the generation of mineral reserves may require a much greater degree of delineation drilling and, therefore, infill drilling will be more expensive than for other more uniformly distributed gold mineralization. The planned infill drilling and definition drilling programs are expected to improve the understanding of the deposit and provide a basis to improve the mine design.

Due to the lack of clear visual indicators, drifting in the mineralized zones will tend to be slower than other deposits. Experienced geologists will be required in many areas to sample and mark up faces for advance. This generally limits the advance rate to one round per day while waiting for samples to be processed.

Due to the variable and complex nature of the mineralization found in the deposit, a variety of mining approaches will be required to efficiently mine the deposit. The level of detailed planning required to understand this variability is beyond the scope of a preliminary economic assessment. This leads to the possibility of:

- Requiring significantly different equipment fleets to deal with different situations over time;
- Potential increases (or reductions) in capital and maintenance expenses if the relative ratios of the different mining methods change significantly over time;
- Significant variation in the operating costs and revenues if the relative ratios of the different mining methods changes significantly over time; and
- These can present both a risk and an opportunity.

A large portion of the potential mineable mineralization considered in this preliminary economic assessment is based on Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the Inferred mineral resources will be converted to the Measured and Indicated categories, that the Measured and Indicated mineral resources will be converted to the Proven or Probable mineral reserve categories. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Accordingly there is a significant risk that the potential production schedule quantity and grade considered in this preliminary economic assessment are not realized. It is possible that additional exploration drilling and mineral resource modelling will improve the confidence of the mineral resources and that additional technical studies will provide a better definition of the potential mining quantities.

There are several more general risks associated with developing and operating the mine:

- Attracting and retaining key technical staff who have the ability to work effectively with the complex nature of deposit;
- Attracting, training, and retaining the required company workforce; and
- Securing of additional required project financing.

24.7 Processing

Potential risks related to processing include:

- The metallurgical test work was done on drill cores representing the two main lithologies that host gold mineralization. However, the current average metallurgical performances, especially the ore grindability and the gold recovery, might not show the variations that can be encountered throughout the short to medium term of the mine life. To decrease the processing uncertainties and have a more accurate estimation of the range of possible gold recovery, additional metallurgical test work should be considered on drill core samples throughout the deposit;
- Presence of sulphide in some feed samples has been taken into account with design of the effluent treatment facility and a dry cover for the TMF; and
- The cyanide concentration was not optimized during the CIL testwork. The cyanide will be destroyed with the SO_2-air process while producing cyanate ions that will be degraded, thus producing ammonia. As ammonia is a regulated effluent discharge parameter, it is necessary to keep it within the allowable limit. Further testwork is needed aimed at reducing the cyanide concentration.

24.8 Power Supply

There is currently sufficient capacity to operate at 1,250 tpd. To operate above 1,250 tpd, additional hydroelectric power will be required to operate the upgraded mill and new mine ventilation fans. Securing the electrical allotment from Hydro One will provide the power needed for future operations.

The use of natural gas would reduce reliance on electricity and/or propane. Natural gas is not currently supplied to the project site but may be brought in by tanker trucks if proven to be cost-effective.

24.9 Environmental

Environmental risk is low except in the area of Aboriginal relations where a lawsuit was brought by Wabauskang First Nation in 2012. Rubicon is open to discussions to formulate an impact and benefits agreement. Other areas of environmental involvement indicate low risk:

- Public consultations have resulted in two complaints regarding noise from construction and exploration activities. All consultations are documented and the comments are on record;
- The site EMS is in good order and no issues relating to monitoring results have been identified;
- It is unlikely that the planned increase in production rate from 1,250 to 2,250 tpd would trigger a federal EA. The change would not increase the area of the project site, there is no expressed public concern for the project, and there have been no reported adverse environmental effects from the project thus far;

- The prospect of a provincial EA is also unlikely. The project has undergone two provincial class EAs with no negative comments received. The permitting process to date has not resulted in a request for an individual EA;
- Recognized environmental sensitivities include water discharge, fugitive dust, and noise. Mitigation plans are in place; and
- A closure plan for the project is on file with the Ontario Ministry of Northern Development and Mines. A progress payment has been made as financial assurance for closure activities. Further assurance will be required prior to tailings disposal in the TMF.

24.10 Project Opportunities

Opportunities that may benefit the Phoenix gold project include:

- Improvement of resources and mine design will be gained from additional information about the extent of the mineral resource. Infill drilling will provide this information;
- The Alimak horizontal longhole stoping method should be investigated for possible reduction in operating costs. This method has the potential to reduce or eliminate a portion of the operating sublevel development as well as to improve productivity by improving stope cycle time;
- A new shaft or ramp would increase available hoisting time for the Phoenix shaft by sharing the movement of personnel and material. Consideration should be given to developing a larger modern shaft or a properly sized ramp connection to surface. Either facility could provide opportunities to improve overall mining efficiency, enable use of larger equipment underground if required, reduce overall mine operating cost, and alleviate congestion in the Phoenix shaft; and
- Exploration of the depth extensions of currently defined gold mineralization trends and the feasibility of deepening the shaft to access this deeper gold mineralization.

24.11 Risks Common to Mines in the Red Lake Area

Mines in the Red Lake area have developed techniques to work in cold weather over many decades of operation. Production delays as a result of cold weather may occur but are not expected to be a concern.

Forest fire smoke has been known to cause evacuation from the area. Depending on the intensity of smoke or location of fire, access to underground may be prohibitive due to poor air quality or loss of hydroelectric power, whether possible or actual. Forest fire smoke may also interrupt supply lines.

24.12 Synopsis of Results

The main results of this preliminary economic assessment are:

- Mineral resource at 4 gpt gold cut-off grade: indicated tonnage of 4.1 million (M) tonnes at 8.52 gpt containing 1.1 M ounces gold and inferred tonnage of 7.5 M tonnes at 9.26 gpt containing 2.2 M ounces gold;
- Proposed mining method: five longhole and cut and fill mining methods were successfully adapted to the variable shape and thickness of the deposit to achieve an average resource extraction of 78%;
- External dilution: weighted average external dilution from all proposed mining methods was estimated to be 15% grading 0.68 gpt gold;

- Potential mineable mineralization at 5 gpt gold cut-off grade estimated at 9.1 M tonnes at a grade of 8.06 gpt gold and containing 2.4 M ounces of gold;
- Potential recovered ounces: 2.2 M ounces of gold at 92.5 percent (%) processing recovery;
- Conceptual production profile and plan: production period is 13.25 years; projected potential production rate reaches a maximum of 2,250 tpd in the latter half of the mine life;
- Potential recovered ounces of gold per year averaging 165,000;
- Total projected operating cost estimated at C$1,378 M, averaging C$151 per tonne or C$629 per ounce of gold; and
- Capital costs:
  - Total C$650 M; averaging C$71.14 per tonne milled or C$297 per ounce of gold;
  - Pre-production C$224 M; and
  - Post-production C$426 M; averaging C$46.62 per tonne milled or C$194 per ounce of gold.

The indicative post-tax financial projected potential results for the Phoenix gold project are:

- Cumulative cash flow of C$897 M;
- NPV(5%) of C$531 M on a mid-period calculation basis;
- IRR of 27%; and
- 3.7 years payback from targeted start of commercial production in Q3-2014.

25 Recommendations

The results of this preliminary economic assessment support the continued advancement of the Phoenix gold project and work related to further technical studies. No production decision has been made at this time. Such a decision, if reached, will require such additional technical studies and ongoing evaluation by Rubicon of the construction and advancement of the Phoenix gold project and would not be based solely on the results of this report.

The following recommendations are made considering the results of this preliminary economic study and the project risks identified. The proposed work program includes additional exploration drilling, geotechnical studies, metallurgical testing, and other various studies aimed at completing the characterization of the project in preparation for further engineering studies.

The recommended work program includes the following components:

- Infill drilling and resource modelling to potentially upgrade the classification of resources;
- Initiation of a pre-production definition drilling program;
- Continuation of underground mine design optimization;
- Continuation of environmental studies;
- Continuation of First Nation and public consultations;
- Continuation of geotechnical and hydrogeological studies in specific areas;
- Consideration of additional underground bulk sampling to further characterize the gold mineralization and to evaluate the precision of the mineral resource model;
- Additional metallurgical testing; and
- Revisions to the mine plan and updates to the project financial model to incorporate the results of the above investigations.

Geology and Exploration

Additional core drilling is required to achieve three objectives:

- Infill the remnant gaps in the drilling data with the potential to increase the mineral resources and to potentially improve resource classification;
- Step-out drilling to test to the lateral and depth extensions of the gold mineralization; and
- In preparation for production, delineation drilling, in particularly the shallow levels, will be required for short term mine planning.

SRK considers that drilling to 20-metre centres will be appropriate for delineation drilling purposes, whereas drilling to 45-metre centres will be adequate for resource delineation and classification to the Indicated category. SRK understand that Rubicon has budgeted C$4.8 million (M) for 40,000 metres (m) of pre-production delineation drilling program between the 244 and 366 levels. An additional amount of C$2.6 M (22,000 m) has been assigned to sustaining capital for continued infill and delineation drilling.

SRK also recommends further geological studies aimed at improving the understanding of the geological and structural setting of the deposit. The results of the proposed infill and step-out exploration drilling programs should be incorporated in an updated deposit geological interpretation and resource model.

Mine Geotechnical and Hydrogeology

Based on the interpretation in the previous section, improved confidence in the geotechnical information is required to optimize potential mining plans and opportunities. SRK (SRK 2013) recommends the following geotechnical testing and studies in the crown pillar and selected areas of the deposit:

- A program of targeted geotechnical drilling, field testing, and laboratory tests to understand the rock mass conditions in key areas; and
- A hydrogeology investigative program, combined with the geotechnical program, to determine the hydraulic conductivity and pore pressure of major geologic units and structures.

Mine Design and Planning

Recommendations for effective mine design and planning are:

- Undertake additional mine planning work in order to optimize the conceptual mine plans for Phase I and Phase II of the mine plan. The estimated cost for Rubicon to do this work in-house with existing resources is C$150,000;
- Establish a test stope program to test and refine the proposed mining methods, especially in terms of the geotechnical performance and reconciliation of the block model with cavity surveys. The estimated additional in-house labour by a junior mining engineer for six months is C$60,000;
- Perform a trade-off study to examine the relative LoM capital and operating costs for various shaft and ramp access options. A key conclusion of this study would be the operational efficiencies realized by being able to bring in larger equipment and reducing the congestion in the Phoenix shaft. The estimated cost of the study is C$50,000 to assess the following proposed options:
  - Deepening the existing shaft as envisioned in the preliminary economic assessment mine plan;
  - Deepening the existing shaft and establishing ramp access from surface;
  - Sinking a new, larger shaft to the 1400 level;
  - Sinking a hoisting winze closer to the deposit; and
  - Sinking a hoisting winze closer to the shaft.
- Revise the mining plan based on the revised mineral resource model(s) resulting from the proposed exploration drilling and resource modelling program. Cost is estimated to be C$100,000.

Mineral Processing

Two recommendations relating to mineral processing testwork have been identified:

- Perform additional metallurgical testwork on drill core samples throughout the deposit to decrease the processing uncertainties and have a more accurate estimation of the range of possible gold recovery; and
- Further testwork aimed at reducing cyanide concentration is recommended to keep ammonia within the regulated limit for discharge.

Environment

SRK recommends that Rubicon maintains diligence with all aspects of the project's environmental management system. Issues in the forefront are review of permits to identify amendments required for the increased production rate outlined in this report and consultation with Wabauskang First Nation regarding the lawsuit brought against the project.

Power Supply

Negotiations with Hydro One for an increase in hydroelectric power allotment are to be maintained in order to secure the power necessary for the future operation of the mill and mine ventilation fans.

Cost Estimate

The estimated costs to carry out the recommended work programs are presented in Table 89.

Table 89: Cost Estimates for Recommended Work Programs*

Work Program	Estimated Cost (C$)
Diamond Drilling	
Infill drilling for mineral resource upgrade (40,000 m)	4,800,000
Delineation drilling between 244 and 366 levels (22,000 m)	2,600,000
Subtotal Drilling	7,400,000
Engineering Studies	
Geotechnical studies	450,000
Metallurgical testwork	80,000
Mine design / planning (combined)	360,000
Subtotal Engineering	890,000
Other Studies	
Environment	250,000
Subtotal Other Studies	250,000
Contingency (20%)	1,800,000
Administration	300,000
Total	**10,640,000**

* All costs are included in the project cost estimate.

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APPENDIX A

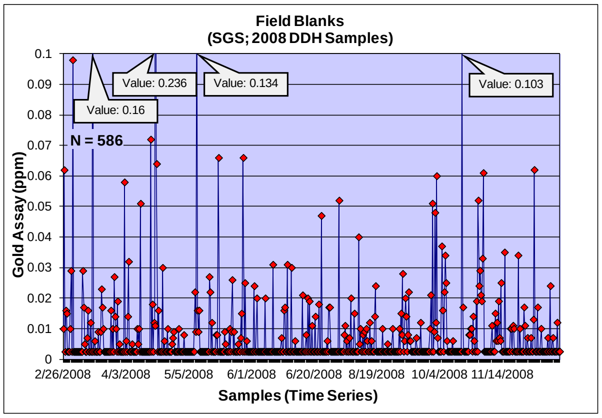
Analytical Quality Control Charts

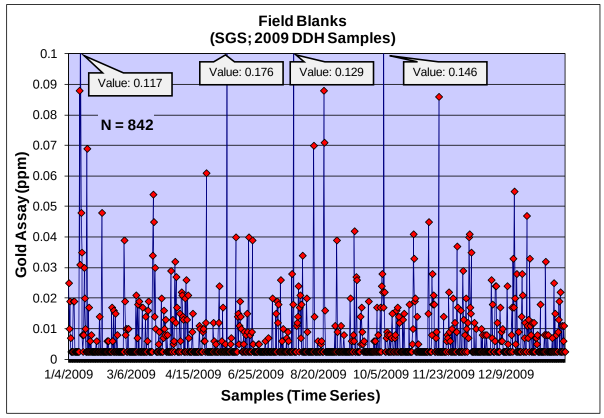
Time Series Plots for Blank Samples Assayed by ALS, Vancouver and SGS, Red Lake between 2008 and 2012

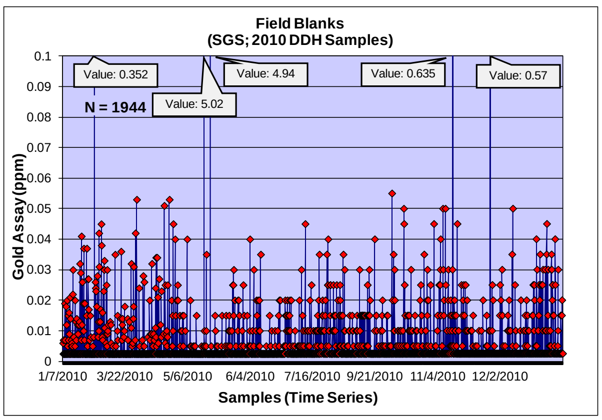
srk consulting

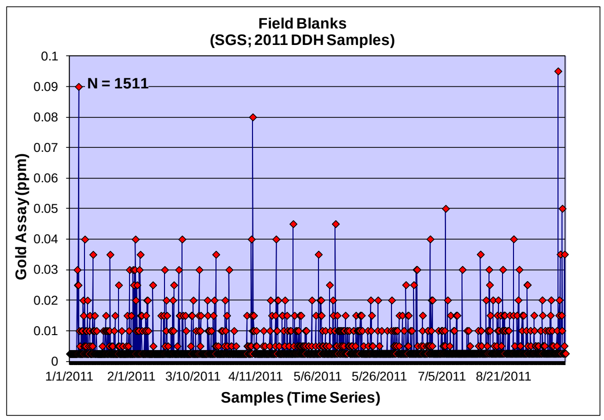
Project	F2 Zone, Phoenix Gold
Data Series	2008-2012 Blanks
Data Type	DDH Samples
Commodity	Au in ppm
Laboratory	SGS
Analytical Method	Fire Assay, AAS Finish
Detection Limit	0.005 ppm

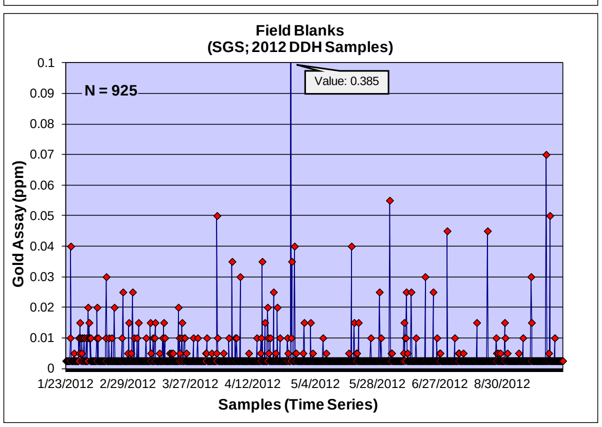
Statistics	2008	2009	2010	2011	2012
Sample Count	586	842	1,944	1,511	925
Mean	0.008	0.008	0.012	0.006	0.005
Min	0.003	0.003	0.003	0.003	0.003
Max	0.236	0.176	5.020	0.095	0.385
Mode	0.003	0.003	0.003	0.003	0.003









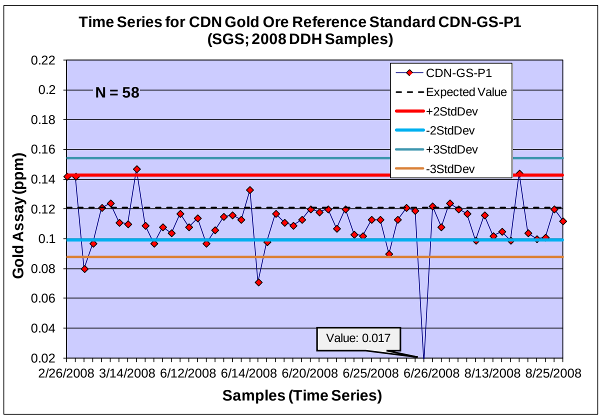


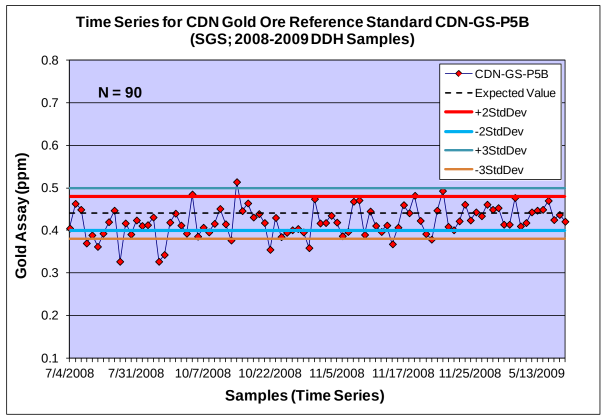
Time Series Plots for Gold Ore Reference Standard Samples Assayed by SGS, Red Lake between 2008 and 2012

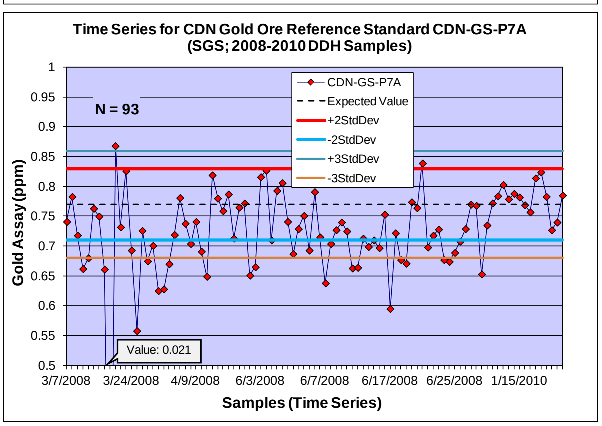
srk consulting

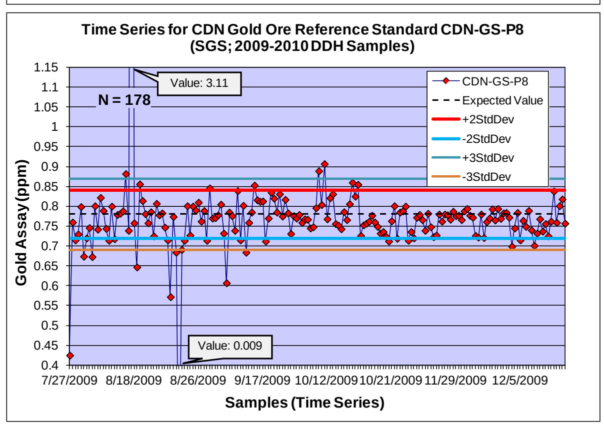
Project	F2 Zone, Phoenix Gold
Data Series	2008-2012 CDN Gold Standards
Data Type	DDH Samples
Commodity	Au in ppm
Laboratory	SGS
Analytical Method	Fire Assay, AAS Finish
Detection Limit	0.005 ppm

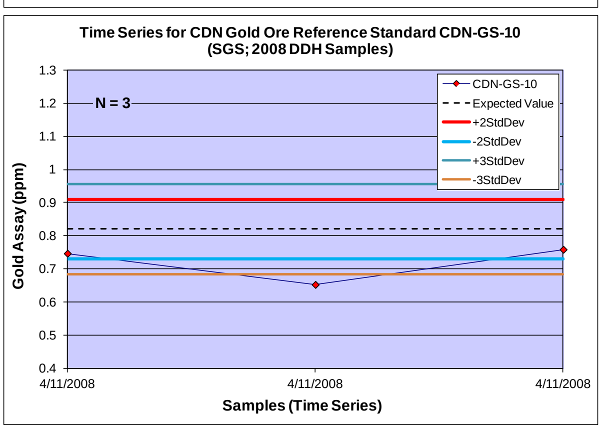
Statistics	P1	P5B	P7A	P8	10	1J
Sample Count	58	90	93	178	3	170
Expected Value	0.121	0.44	0.77	0.78	0.82	0.946
Standard Deviation	0.011	0.02	0.03	0.03	0.05	0.051
Mean	0.110	0.42	0.72	0.77	0.72	0.973
Outside 2StdDev	17.2%	31.1%	37.6%	18.5%	33.3%	8.2%
Below 2StdDev	8	24	33	24	1	8
Above 2StdDev	2	4	2	9	0	6

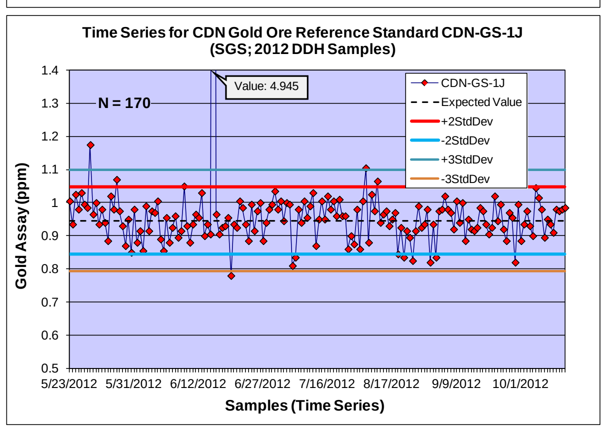










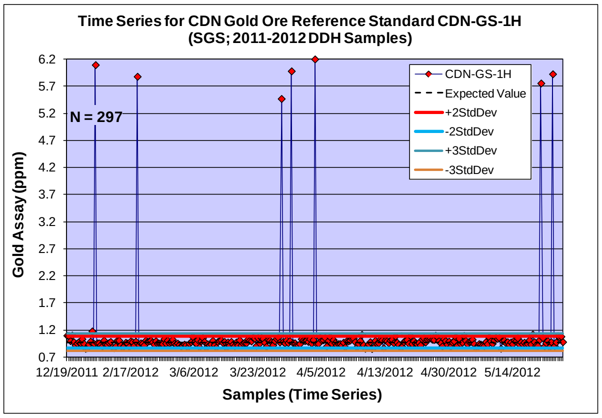


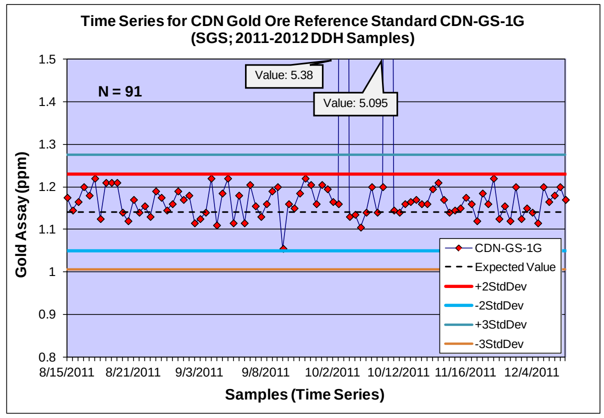
Time Series Plots for Gold Ore Reference Standard Samples Assayed by SGS, Red Lake between 2008 and 2012

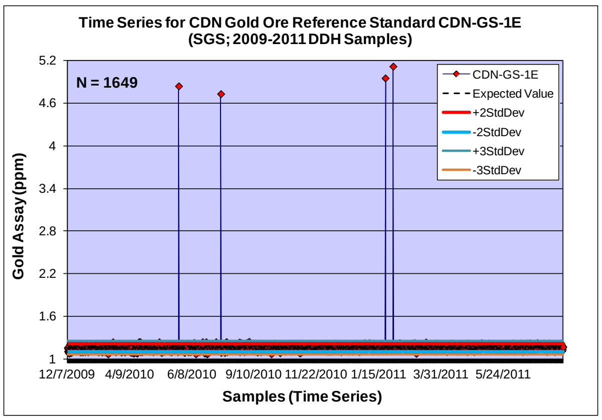
srk consulting

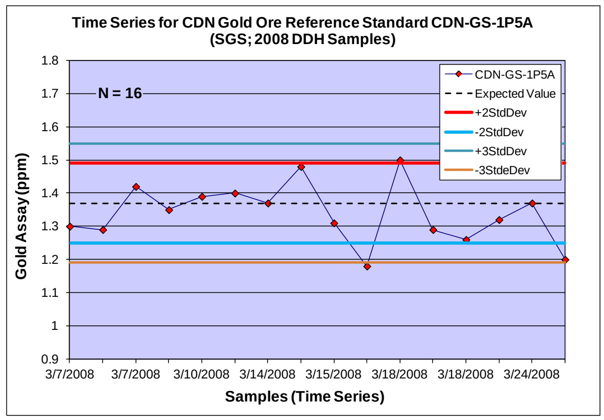
Project	F2 Zone, Phoenix Gold
Data Series	2008-2012 CDN Gold Standards
Data Type	DDH Samples
Commodity	Au in ppm
Laboratory	SGS
Analytical Method	Fire Assay, AAS Finish
Detection Limit	0.005 ppm

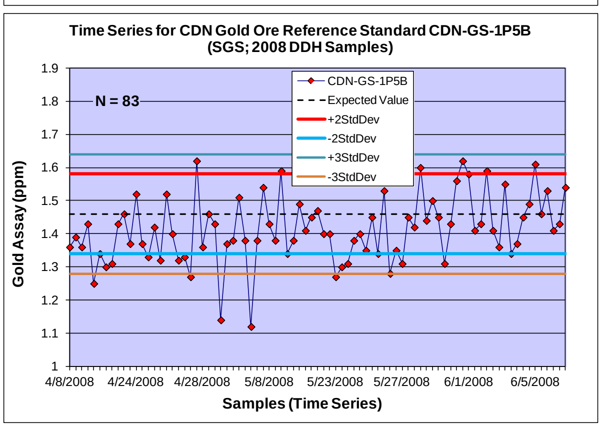
Statistics	1H	1G	1E	1P5A	1P5B	9
Sample Count	297	91	1,649	16	83	123
Expected Value	0.972	1.14	1.16	1.37	1.46	1.75
Standard Deviation	0.054	0.05	0.03	0.06	0.06	0.07
Mean	1.090	1.25	1.17	1.34	1.41	1.72
Outside 2StdDev	7.4%	2.2%	3.0%	18.8%	26.5%	22.0%
Below 2StdDev	4	0	22	2	16	21
Above 2StdDev	18	2	27	1	6	6

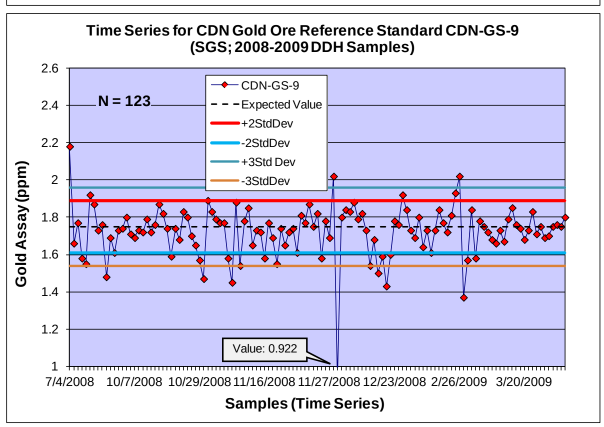












Time Series Plots for Gold Ore Reference Standard Samples Assayed by SGS, Red Lake between 2008 and 2012

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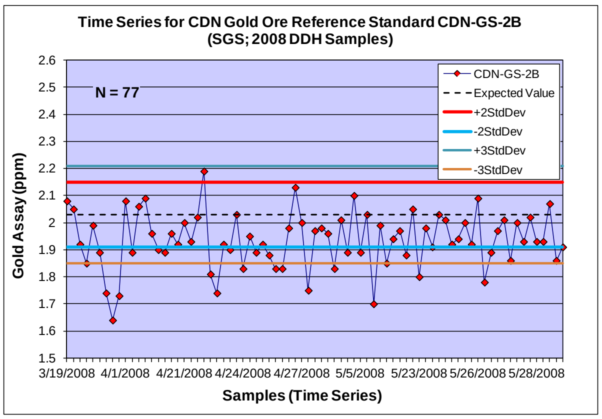
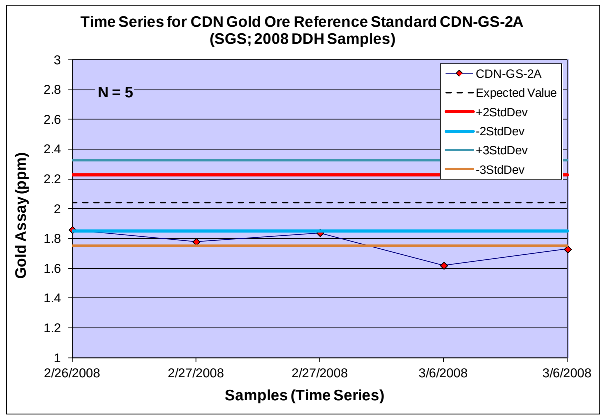
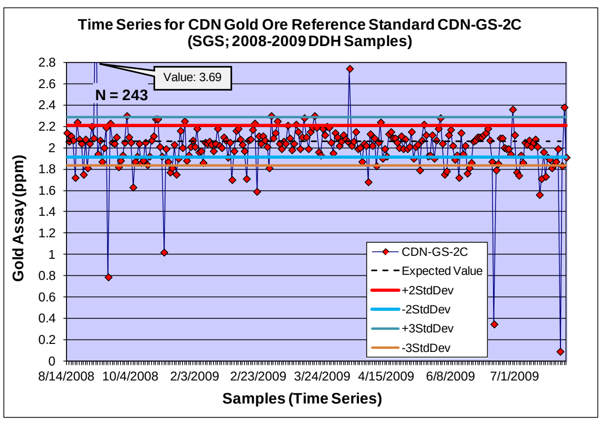
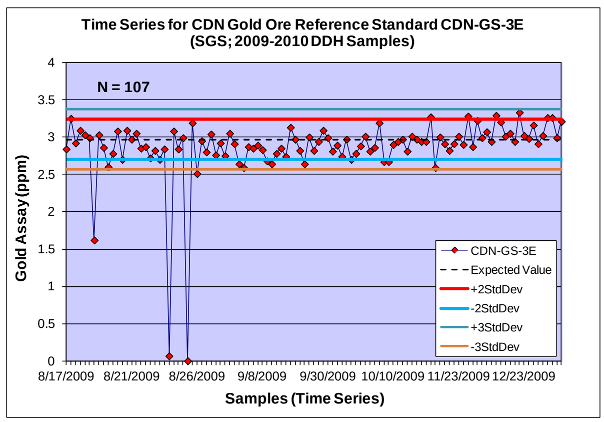
Project	F2 Zone, Phoenix Gold
Data Series	2008-2012 CDN Gold Standards
Data Type	DDH Samples
Commodity	Au in ppm
Laboratory	SGS
Analytical Method	Fire Assay, AAS Finish
Detection Limit	0.005 ppm

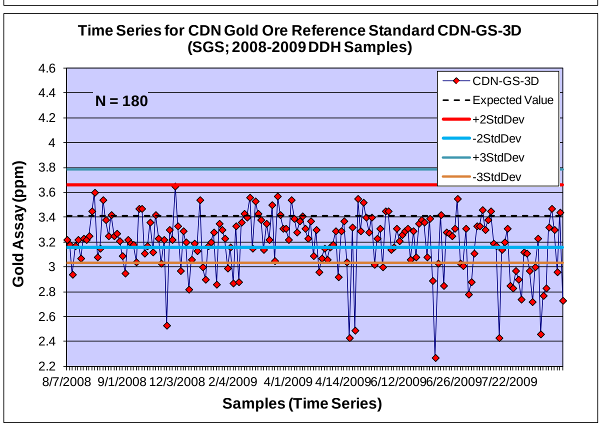
Statistics	**2B**	**2A**	**2C**	**3E**	**3D**	**5C**
Sample Count	77	5	243	107	180	1
Expected Value	2.03	2.04	2.06	2.97	3.41	4.74
Standard Deviation	0.06	0.095	0.075	0.135	0.125	0.14
Mean	1.93	1.77	2.00	2.86	3.19	4.81
Outside 2StdDev	37.7%	80.0%	30.0%	18.7%	36.1%	0.0%
Below 2StdDev	28	4	54	13	65	0
Above 2StdDev	1	0	19	7	0	0

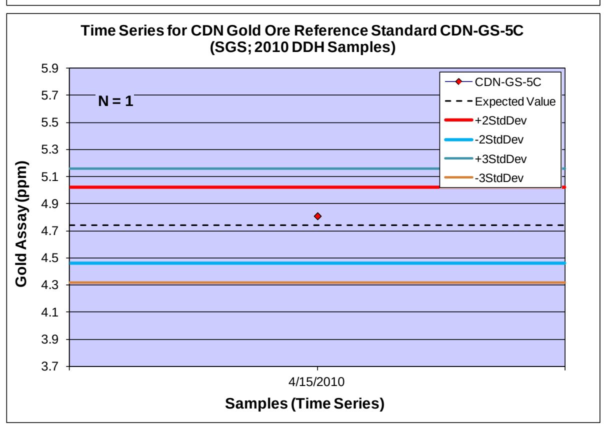












Time series plots for Gold Ore Reference Standard Samples Assayed by SGS, Red Lake between 2008 and 2012

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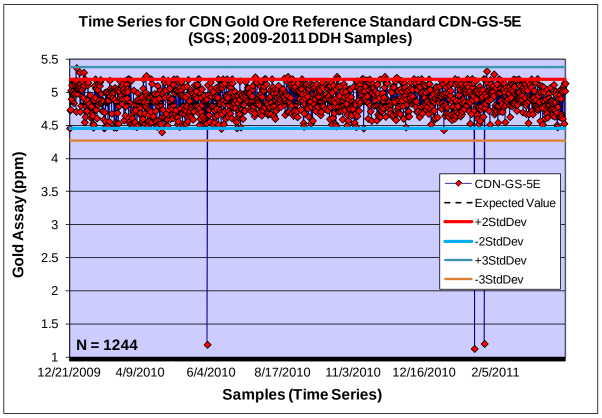
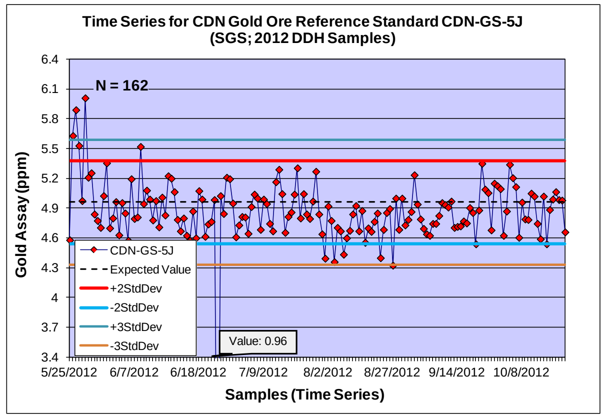
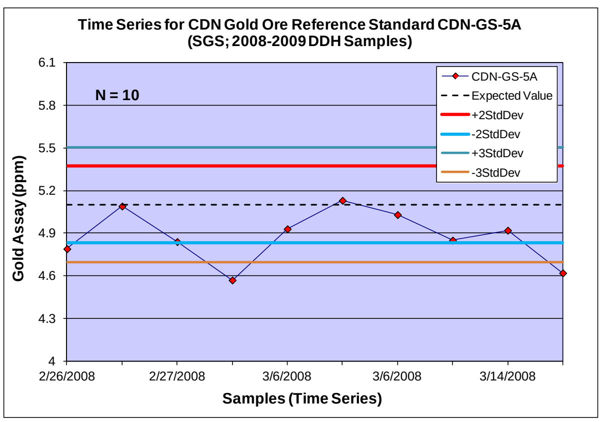
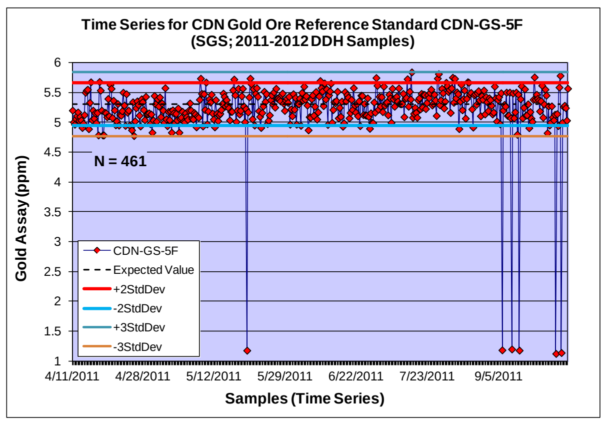
Project	F2 Zone, Phoenix Gold
Data Series	2008-2012 CDN Gold Standards
Data Type	DDH Samples
Commodity	Au in ppm
Laboratory	SGS
Analytical Method	Fire Assay, AAS Finish
Detection Limit	0.005 ppm

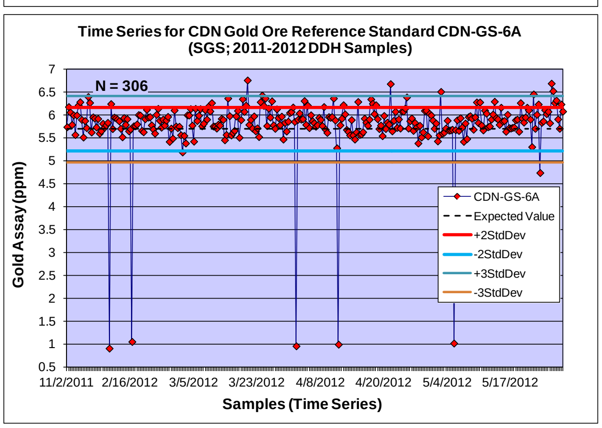
Statistics	**5E**	**5J**	**5A**	**5F**	**6A**	**7A**
Sample Count	1,244	162	10	461	306	121
Expected Value	4.83	4.96	5.10	5.30	5.69	7.20
Standard Deviation	0.185	0.21	0.135	0.18	0.24	0.30
Mean	4.88	4.86	4.88	5.22	5.80	7.26
Outside 2StdDev	1.5%	6.8%	30.0%	10.6%	15.4%	14.0%
Below 2StdDev	7	6	3	30	7	3
Above 2StdDev	12	5	0	19	40	14

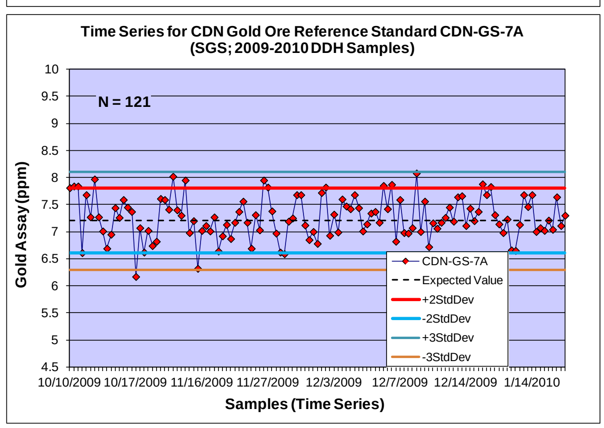










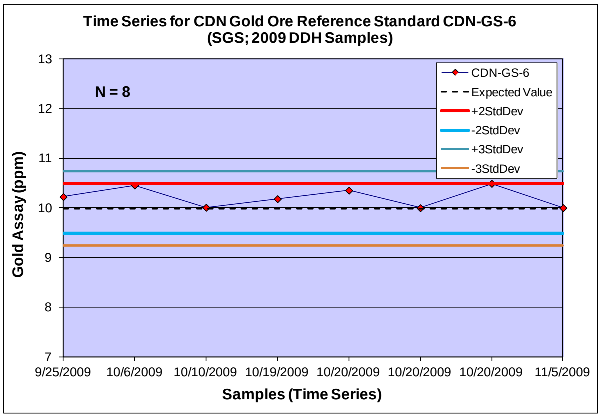


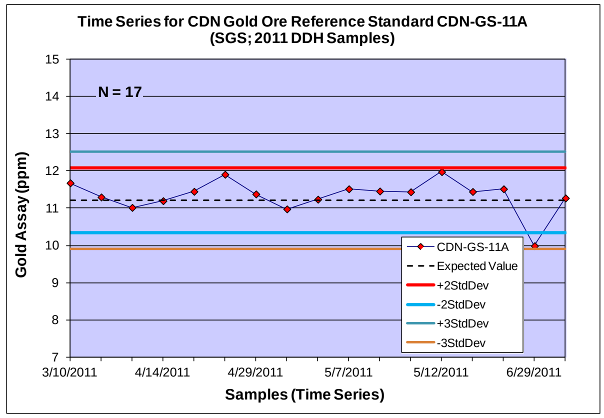
Time series plots for Gold Ore Reference Standard Samples Assayed by SGS, Red Lake between 2008 and 2012

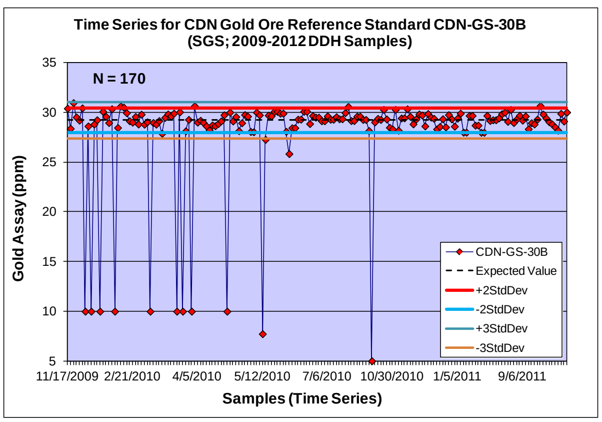
srk consulting

Project	F2 Zone, Phoenix Gold
Data Series	2008-2012 CDN Gold Standards
Data Type	DDH Samples
Commodity	Au in ppm
Laboratory	SGS
Analytical Method	Fire Assay, AAS Finish
Detection Limit	0.005 ppm

Statistics	**6**	**11A**	**30B**
Sample Count	8	17	170
Expected Value	9.99	11.21	29.21
Standard Deviation	0.25	0.435	0.615
Mean	10.22	11.34	27.96
Outside 2StdDev	12.5%	5.9%	12.9%
Below 2StdDev	0	1	16
Above 2StdDev	1	0	6





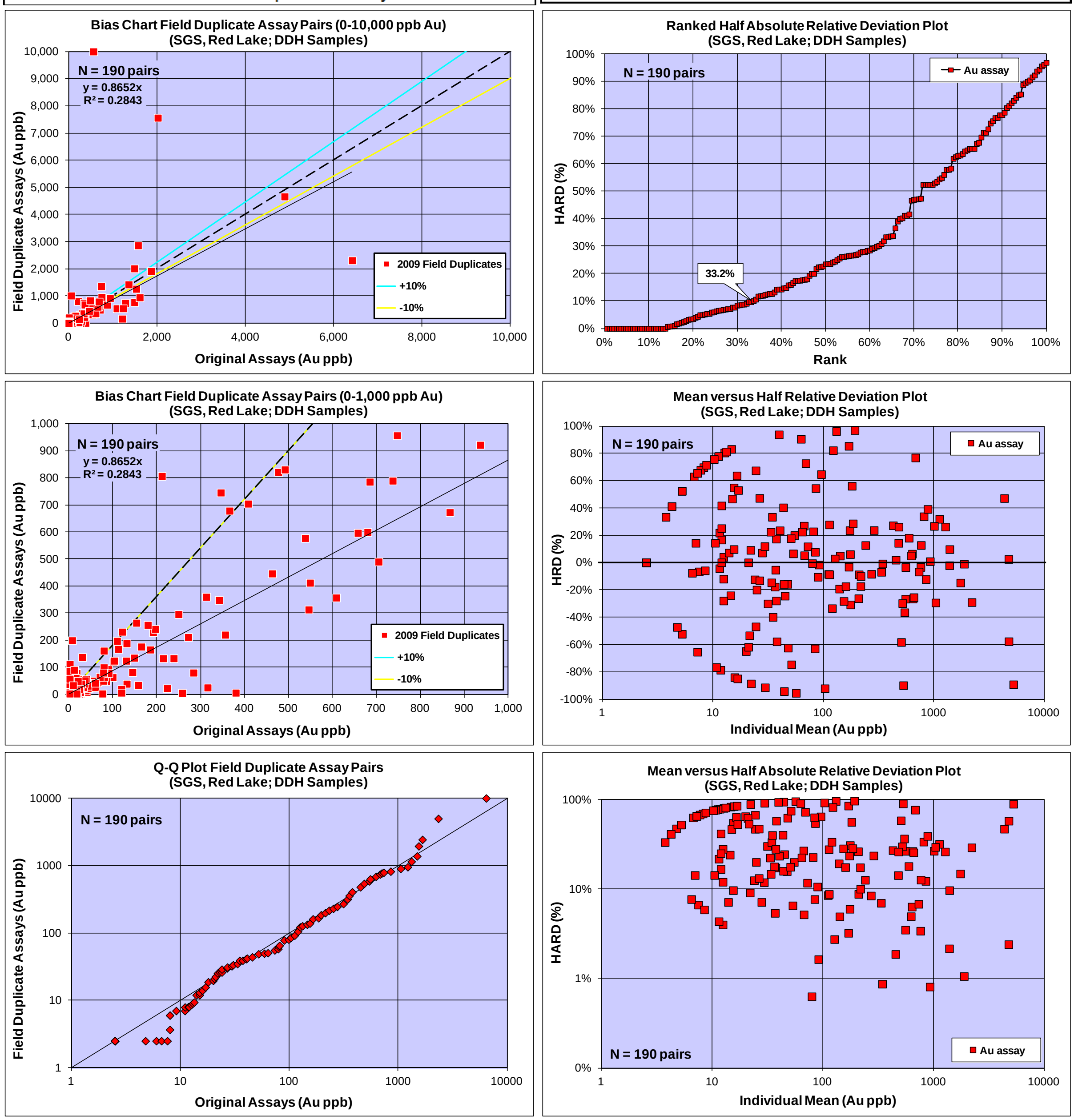


Bias Charts and Precision Plots for Field Duplicate Samples Assayed by SGS, Red Lake in 2009

srk consulting

Project	F2 Zone, Phoenix Gold
Data Series	2009 Field Duplicates
Data Type	DDH Samples
Commodity	Au in ppb
Analytical Method	Fire Assay, AAS Finish
Detection Limit	5 ppb
Original Dataset	Original Assays
Paired Dataset	Field Duplicate Assays

Statistics	Original	Field Duplicate
Sample Count	190	190
Minimum Value	2.50	2.50
Maximum Value	6,420	10,000
Mean	264.58	317.72
Median	30.50	33.50
Standard Error	49.59	74.64
Standard Deviation	683.56	1,028.85
Correlation Coefficient	0.5412	
Pairs ≤ 10% HARD	33.2%	

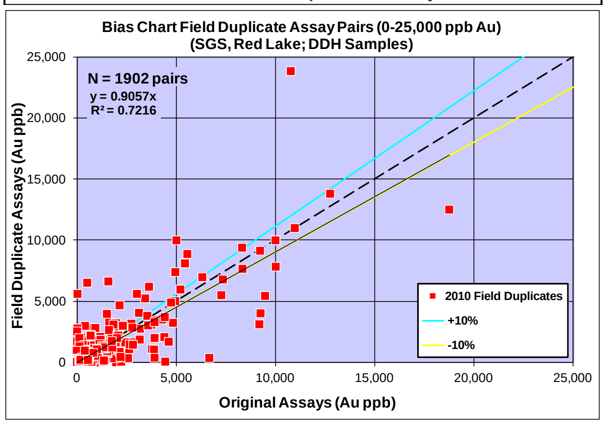


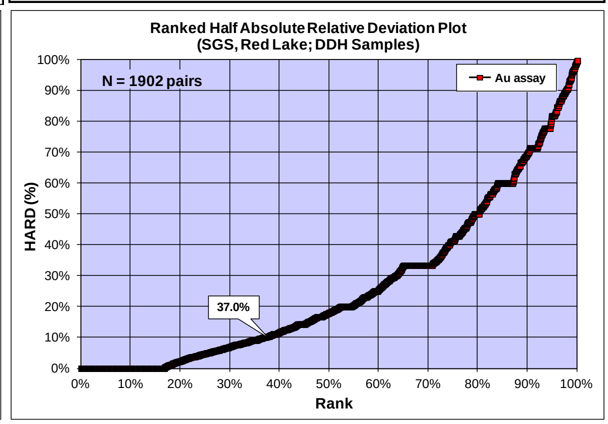
Bias Charts and Precision Plots for Field Duplicate Samples Assayed by SGS, Red Lake in 2010

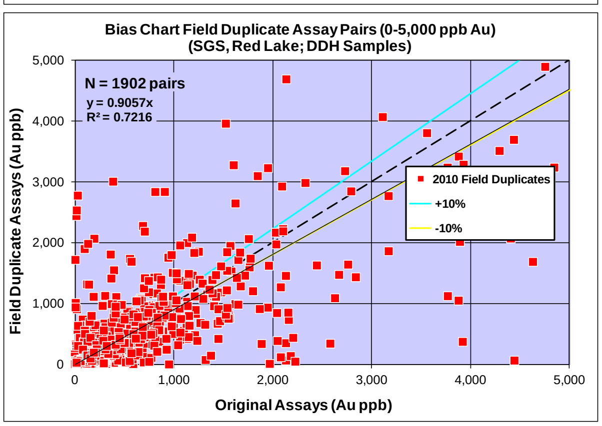
srk consulting

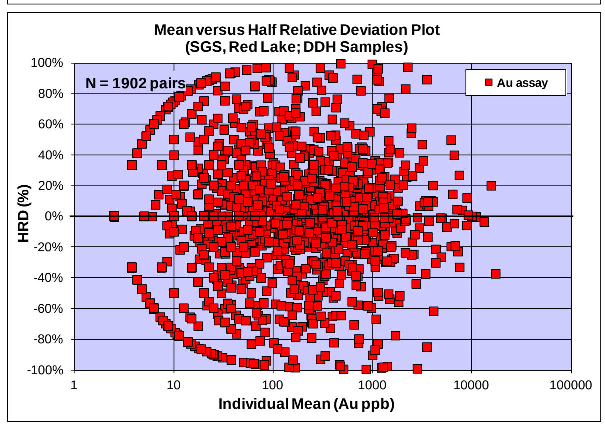
Project	F2 Zone, Phoenix Gold
Data Series	2010 Field Duplicates
Data Type	DDH Samples
Commodity	Au in ppb
Analytical Method	Fire Assay, AAS Finish
Detection Limit	5 ppb
Original Dataset	Original Assays
Paired Dataset	Field Duplicate Assays

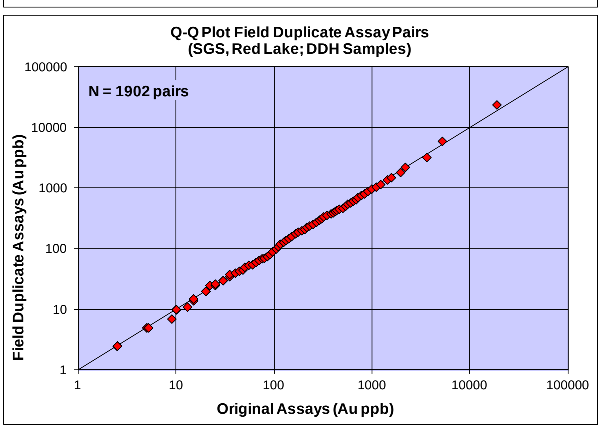
Statistics	**Original**	**Field Duplicate**
Sample Count	1,902	1,902
Minimum Value	2.50	2.50
Maximum Value	18,756	23,835
Mean	398.80	397.80
Median	65.00	60.00
Standard Error	25.82	27.16
Standard Deviation	1,126.19	1,184.68
Correlation Coefficient	0.8501	
Pairs ≤ 10% HARD	37.0%	

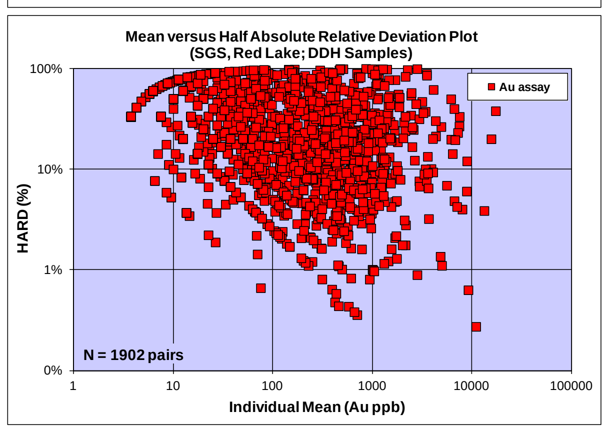










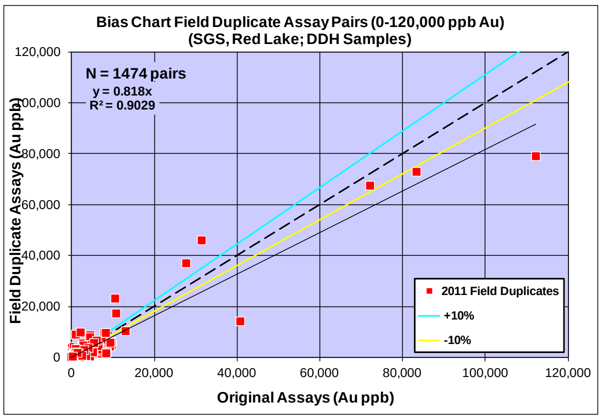


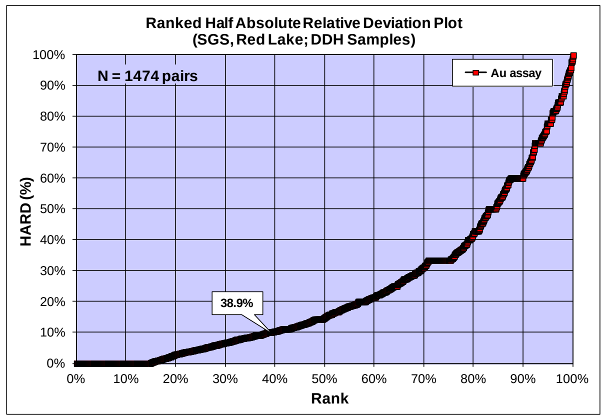
Bias Charts and Precision Plots for Field Duplicate Samples Assayed by SGS, Red Lake in 2011

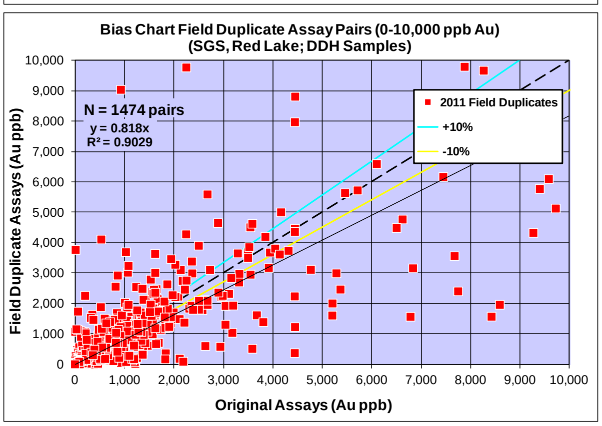
srk consulting

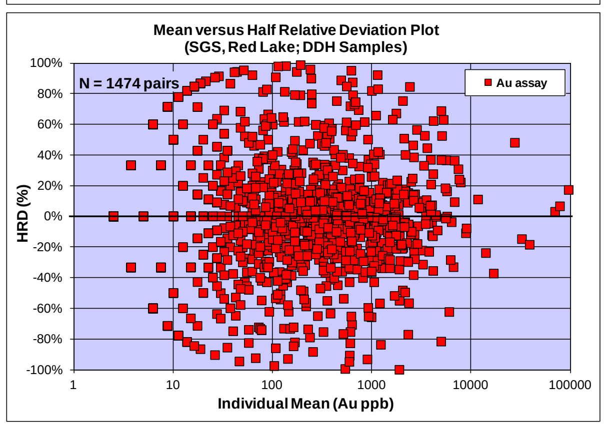
Project	F2 Zone, Phoenix Gold
Data Series	2011 Field Duplicates
Data Type	DDH Samples
Commodity	Au in ppb
Analytical Method	Fire Assay, AAS Finish
Detection Limit	5 ppb
Original Dataset	Original Assays
Paired Dataset	Field Duplicate Assays

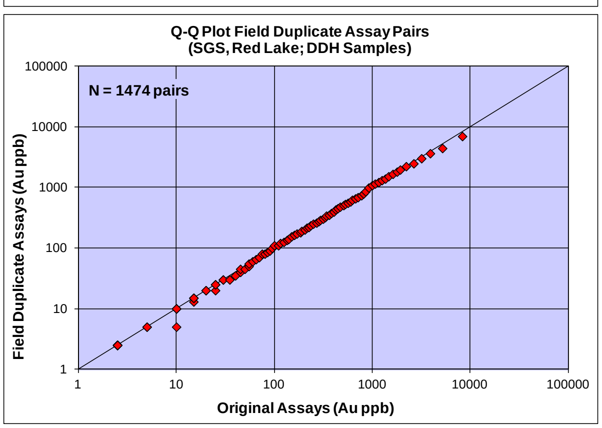
Statistics	**Original**	**Field Duplicate**
Sample Count	1,474	1,474
Minimum Value	2.50	2.50
Maximum Value	112,214	79,122
Mean	804.50	762.05
Median	135.00	135.00
Standard Error	117.26	100.40
Standard Deviation	4,501.91	3,854.57
Correlation Coefficient	0.9506	
Pairs ≤ 10% HARD	38.9%	

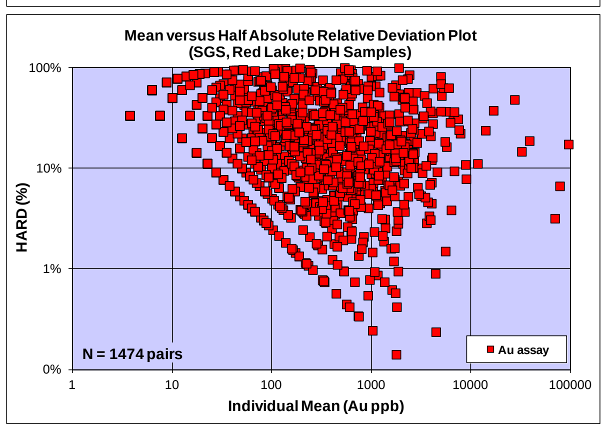










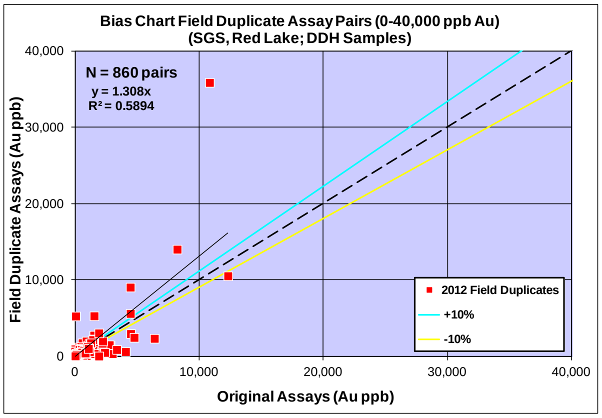


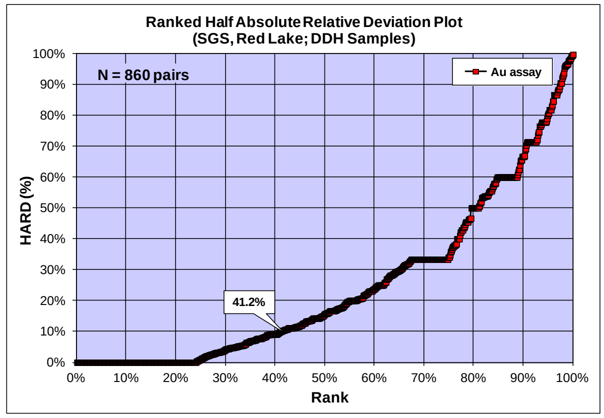
Bias Charts and Precision Plots for Field Duplicate Samples Assayed by SGS, Red Lake in 2012

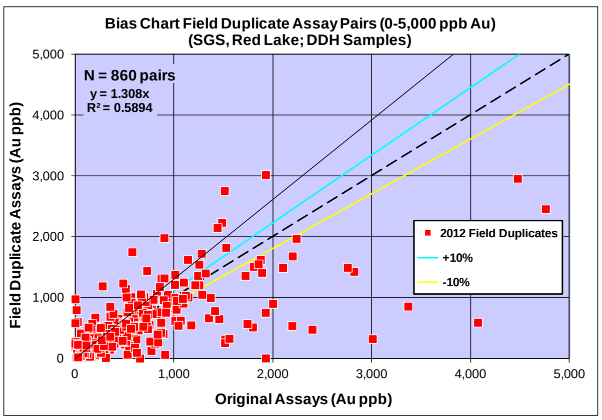
srk consulting

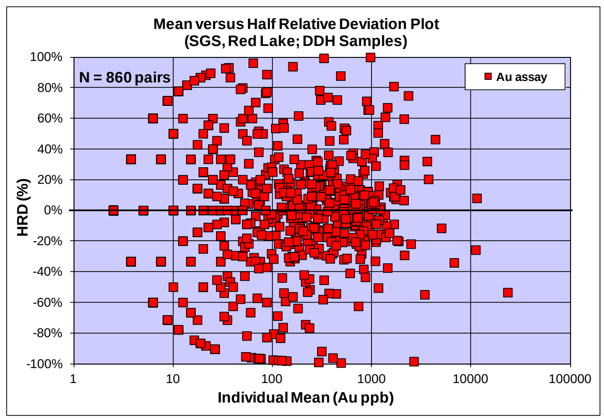
Project	F2 Zone, Phoenix Gold
Data Series	2012 Field Duplicates
Data Type	DDH Samples
Commodity	Au in ppb
Analytical Method	Fire Assay, AAS Finish
Detection Limit	5 ppb
Original Dataset	Original Assays
Paired Dataset	Field Duplicate Assays

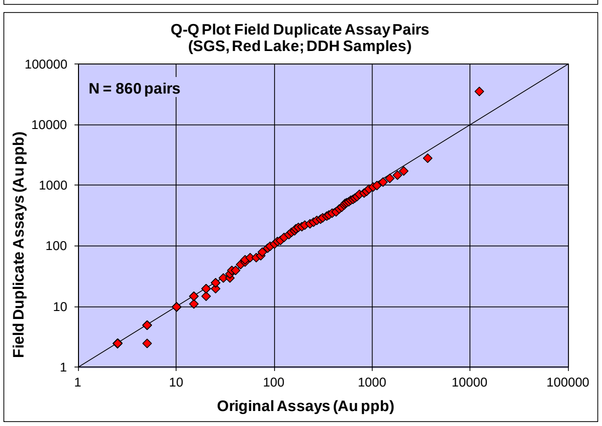
Statistics	**Original**	**Field Duplicate**
Sample Count	860	860
Minimum Value	2.50	2.50
Maximum Value	12,323	35,852
Mean	318.32	337.28
Median	45.00	50.00
Standard Error	28.70	50.08
Standard Deviation	841.63	1,468.78
Correlation Coefficient	0.7699	
Pairs ≤ 10% HARD	41.2%	

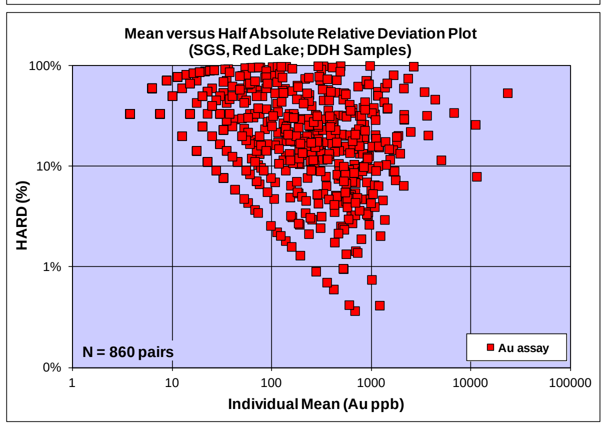










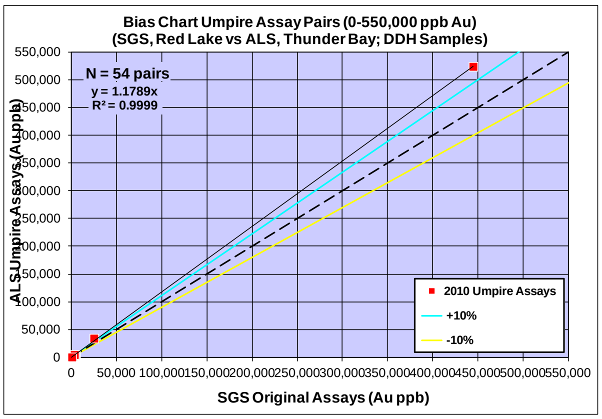


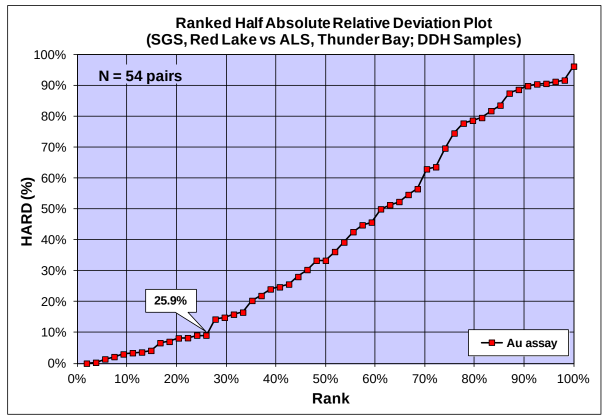
Bias Charts and Precision Plots for Check Assay Samples (SGS, Red Lake versus ALS, Thunder Bay) Assayed in 2010

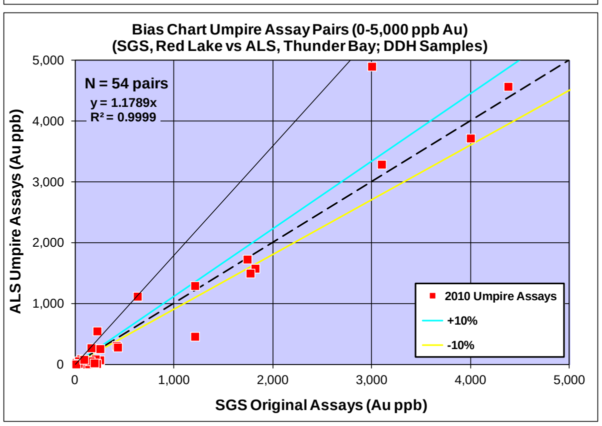
srk consulting

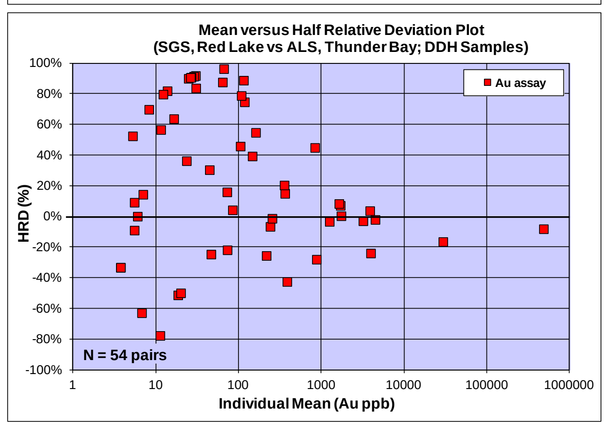
Project	F2 Zone, Phoenix Gold
Data Series	2010 Umpire Assays
Data Type	DDH Samples
Commodity	Au in ppb
Analytical Method	Fire Assay, AAS Finish
Detection Limit	5 ppb
Original Dataset	SGS Original Assays
Paired Dataset	ALS Umpire Assays

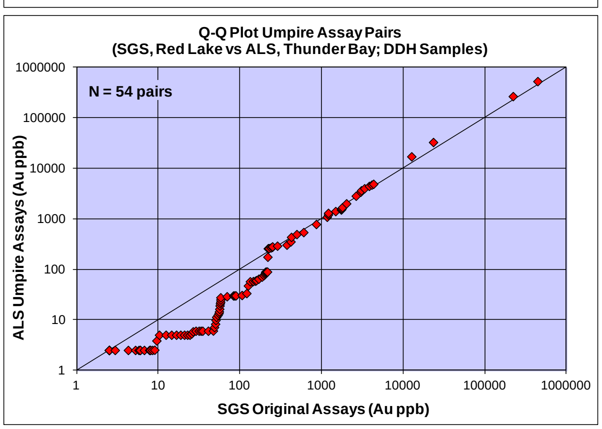
Statistics	Original	Umpire
Sample Count	54	54
Minimum Value	2.50	2.50
Maximum Value	444,718	524,000
Mean	9,189.31	10,838.63
Median	86.00	30.00
Standard Error	8,230.78	9,703.80
Standard Deviation	60,483.65	71,308.11
Correlation Coefficient	0.9999	
Pairs ≤ 10% HARD	25.9%	

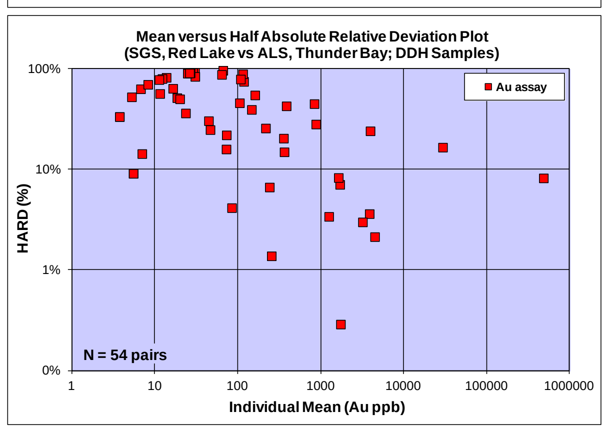










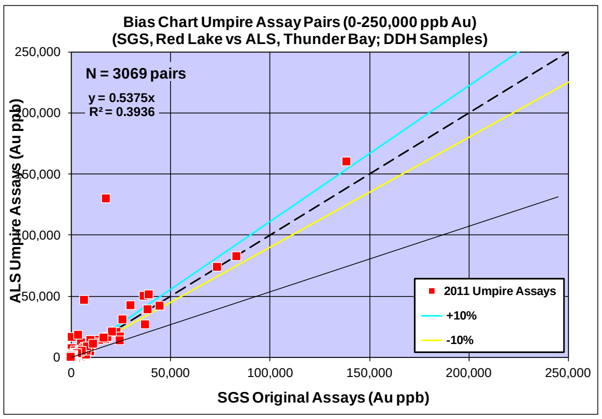


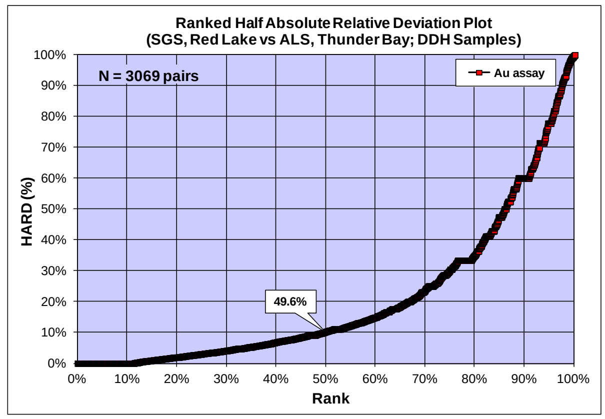
Bias Charts and Precision Plots for Check Assay Samples (SGS, Red Lake versus ALS, Thunder Bay) Assayed in 2011

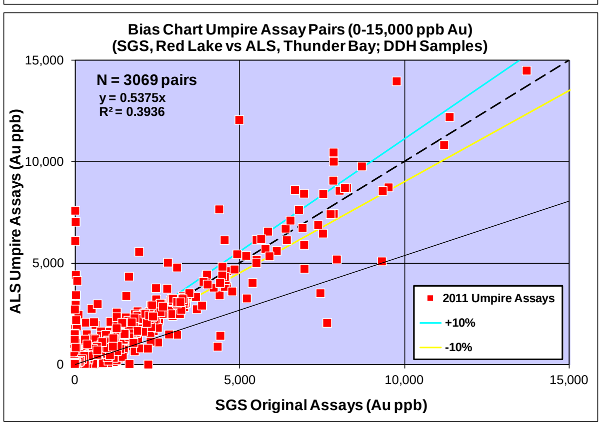
srk consulting

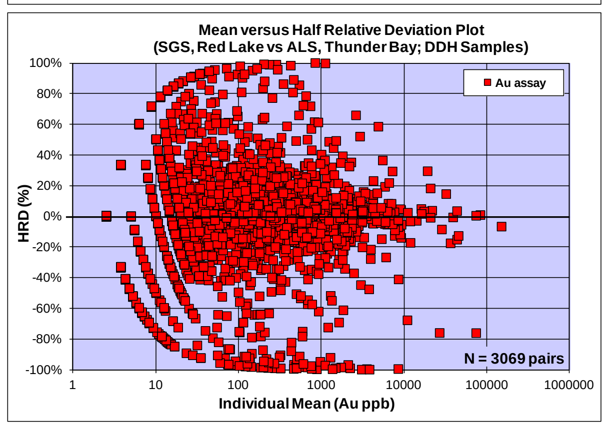
Project	F2 Zone, Phoenix Gold
Data Series	2011 Umpire Assays
Data Type	DDH Samples
Commodity	Au in ppb
Analytical Method	Fire Assay, AAS Finish
Detection Limit	5 ppb
Original Dataset	SGS Original Assays
Paired Dataset	ALS Umpire Assays

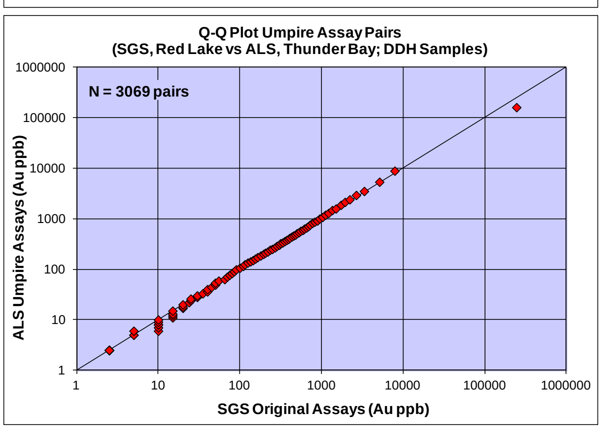
Statistics	**Original**	**Umpire**
Sample Count	3,069	3,069
Minimum Value	2.50	2.50
Maximum Value	245,008	160,000
Mean	784.12	798.11
Median	100.00	106.00
Standard Error	106.45	89.08
Standard Deviation	5,897.37	4,935.17
Correlation Coefficient	0.6321	
Pairs ≤ 10% HARD	49.6%	

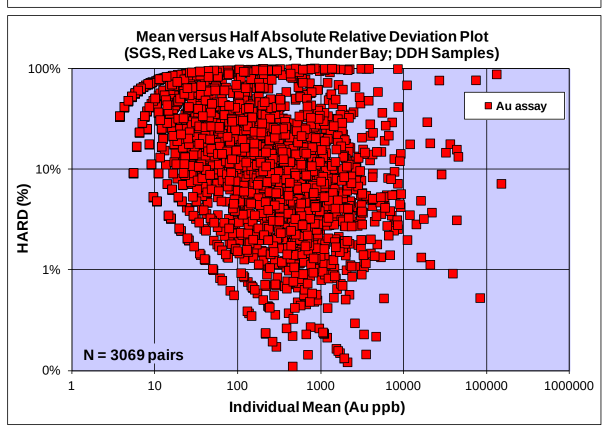










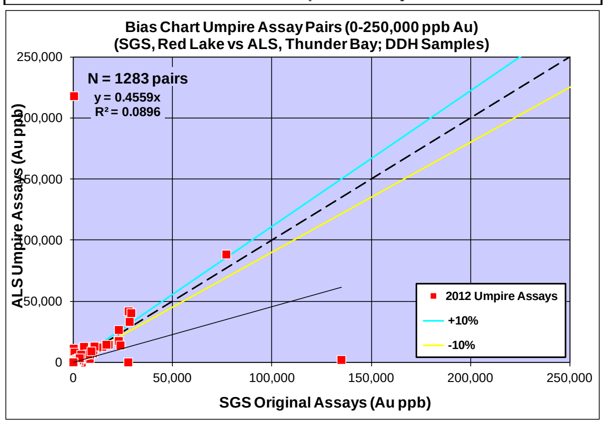


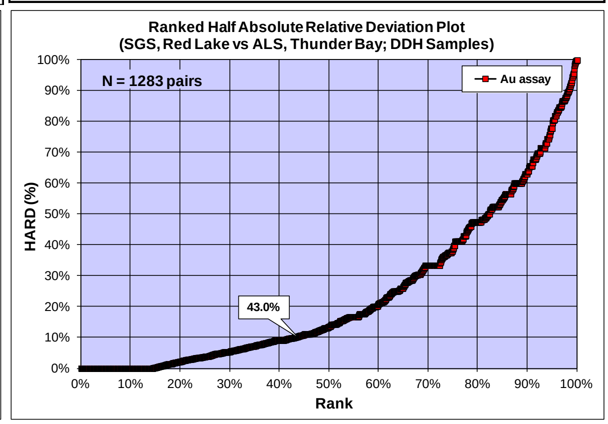
Bias Charts and Precision Plots for Check Assay Samples (SGS, Red Lake versus ALS, Thunder Bay) Assayed in 2012

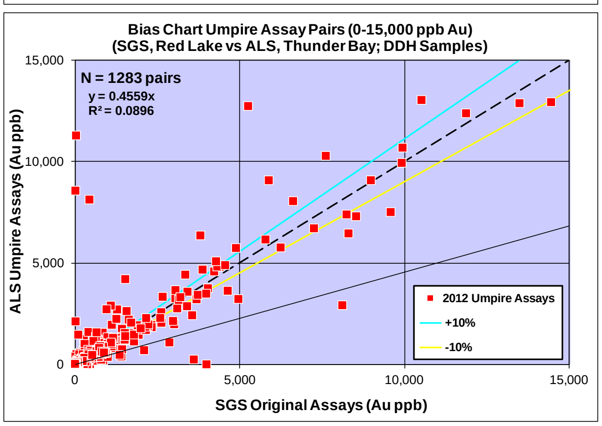
srk consulting

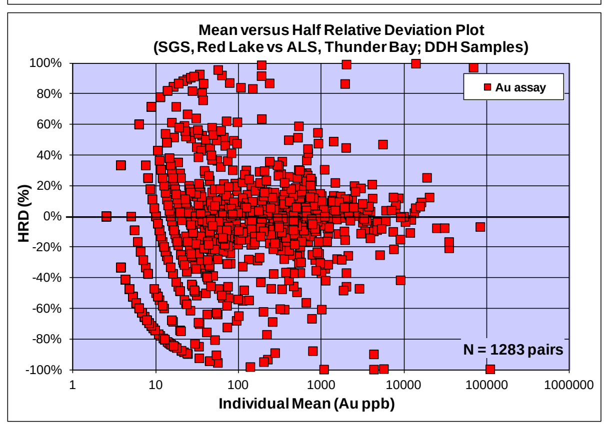
Project	F2 Zone, Phoenix Gold
Data Series	2012 Umpire Assays
Data Type	DDH Samples
Commodity	Au in ppb
Analytical Method	Fire Assay, AAS Finish
Detection Limit	5 ppb
Original Dataset	SGS Original Assays
Paired Dataset	ALS Umpire Assays

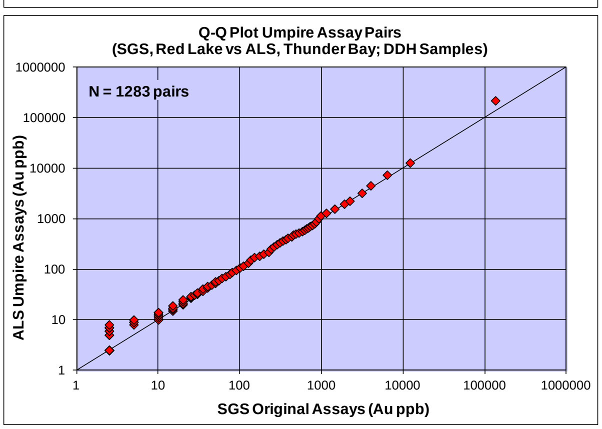
Statistics	**Original**	**Umpire**
Sample Count	1,283	1,283
Minimum Value	2.50	2.50
Maximum Value	134,973	218,000
Mean	754.28	850.05
Median	35.00	37.00
Standard Error	137.25	196.06
Standard Deviation	4,916.10	7,022.76
Correlation Coefficient	0.3081	
Pairs ≤ 10% HARD	43.0%	

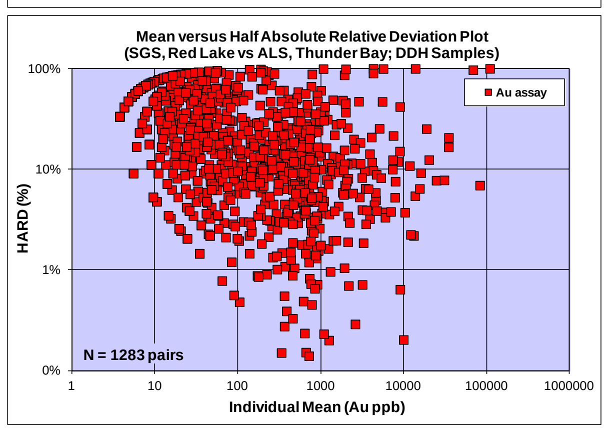






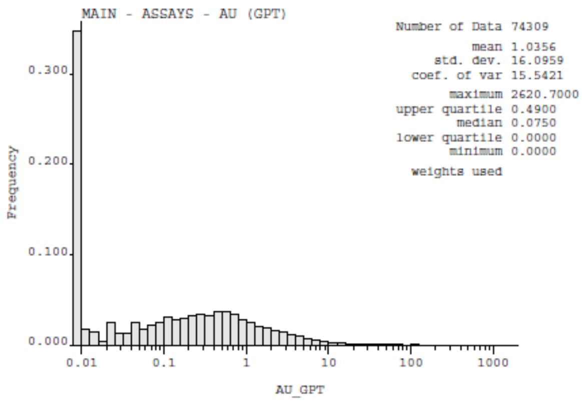


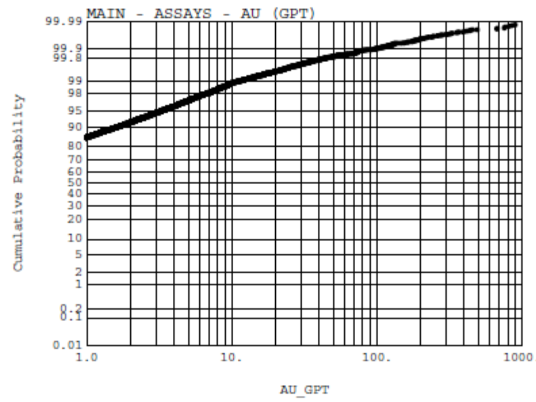


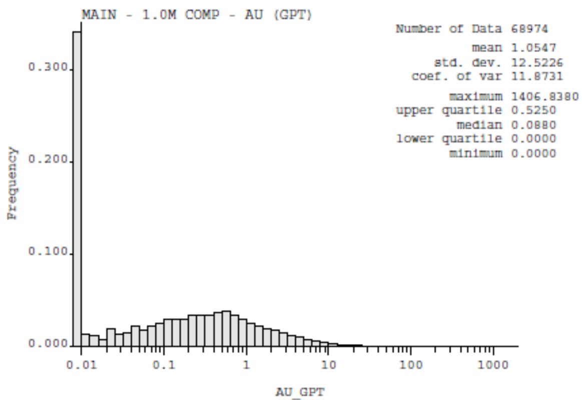


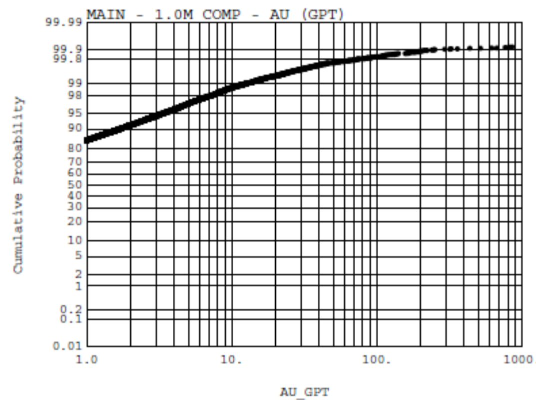
APPENDIX B

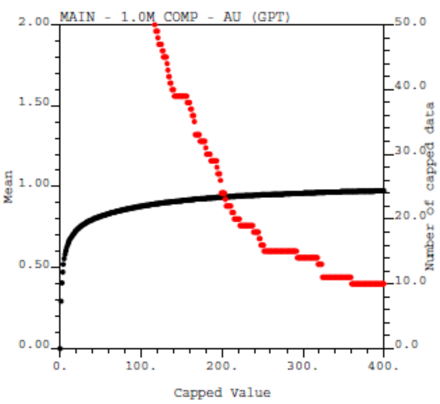
Base Statistics, Histograms, and Cumulative Probability Curves

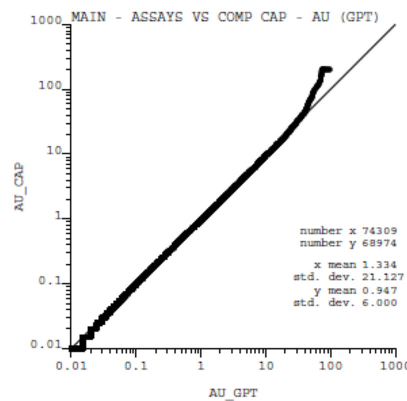

MAIN - ASSAYS - AU (GPT)
Number of Data 74309
mean 1.0356
std. dev. 16.0959
coef. of var 15.5421
maximum 2620.7000
upper quartile 0.4900
median 0.0750
lower quartile 0.0000
minimum 0.0000
weights used
Frequency
AU_GPT

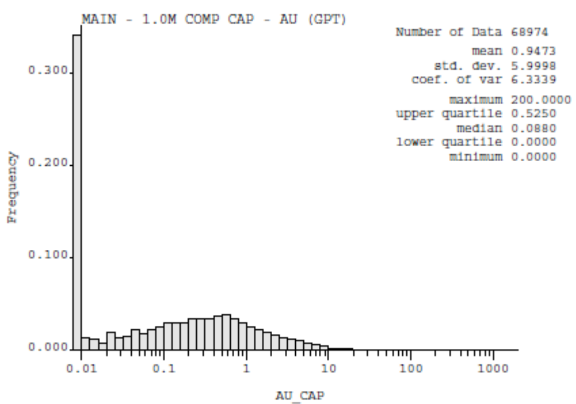

MAIN - ASSAYS - AU (GPT)
Cumulative Probability
AU_GPT

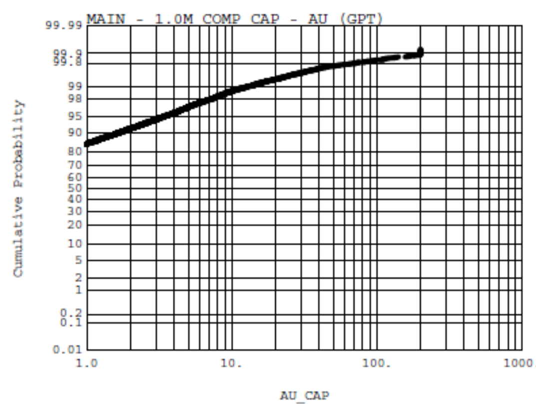

MAIN - 1.0M COMP - AU (GPT)
Number of Data 68974
mean 1.0547
std. dev. 12.5226
coef. of var 11.8731
maximum 1406.8380
upper quartile 0.5250
median 0.0880
lower quartile 0.0000
minimum 0.0000
Frequency
AU_GPT

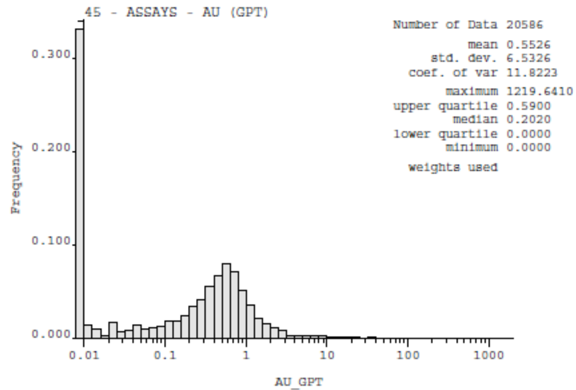

MAIN - 1.0M COMP - AU (GPT)
Cumulative Probability
AU_GPT

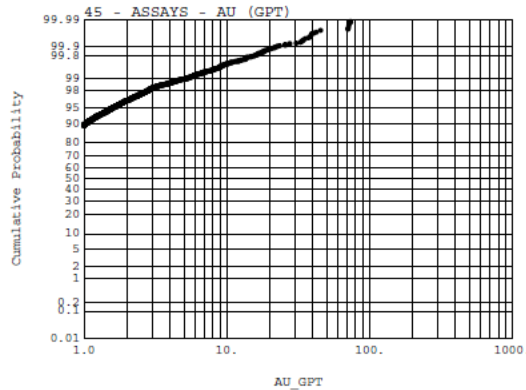

MAIN - 1.0M COMP - AU (GPT)
Mean
Number of capped data
Capped Value

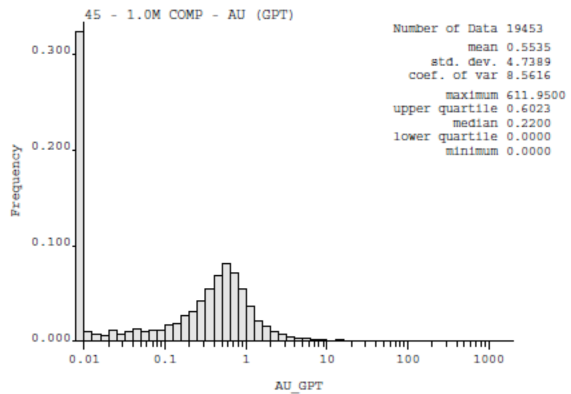

MAIN - ASSAYS VS COMP CAP - AU (GPT)
number x 74309
number y 68974
x mean 1.334
std. dev. 21.127
y mean 0.947
std. dev. 6.000
AU_CAP
AU_GPT

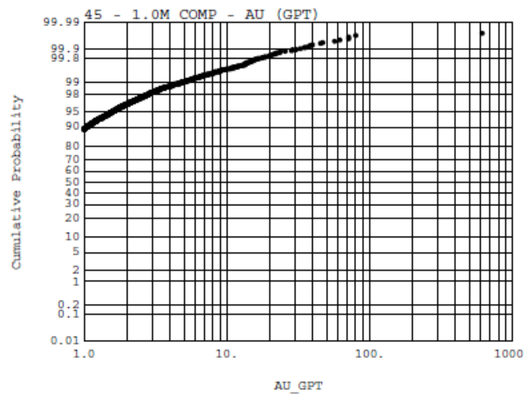

MAIN - 1.0M COMP CAP - AU (GPT)
Number of Data 68974
mean 0.9473
std. dev. 5.9998
coef. of var 6.3339
maximum 200.0000
upper quartile 0.5250
median 0.0880
lower quartile 0.0000
minimum 0.0000
Frequency
AU_CAP

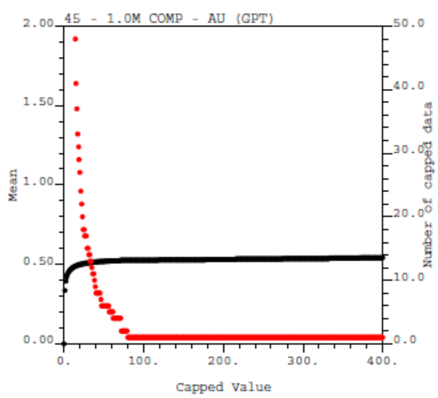

MAIN - 1.0M COMP CAP - AU (GPT)
Cumulative Probability
AU_CAP

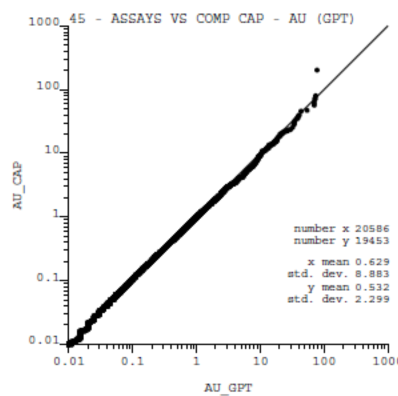

45 - ASSAYS - AU (GPT)
Number of Data 20586
mean 0.5526
std. dev. 6.5326
coef. of var 11.8223
maximum 1219.6410
upper quartile 0.5900
median 0.2020
lower quartile 0.0000
minimum 0.0000
weights used
Frequency
AU_GPT

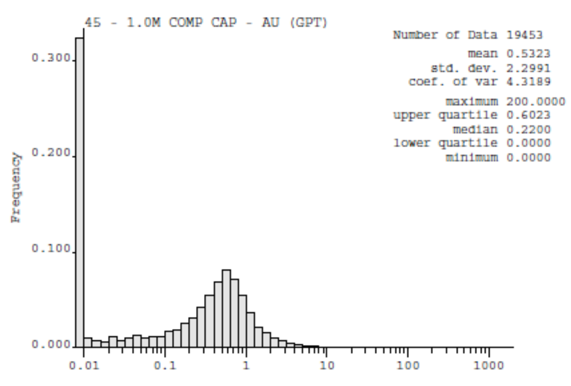

45 - ASSAYS - AU (GPT)
Cumulative Probability
AU_GPT

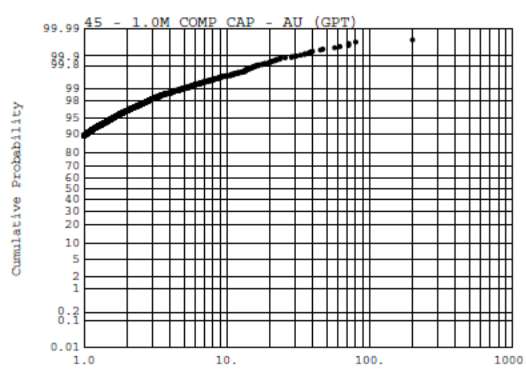

45 - 1.0M COMP - AU (GPT)
Number of Data 19453
mean 0.5535
std. dev. 4.7389
coef. of var 8.5616
maximum 611.9500
upper quartile 0.6023
median 0.2200
lower quartile 0.0000
minimum 0.0000
Frequency
AU_GPT

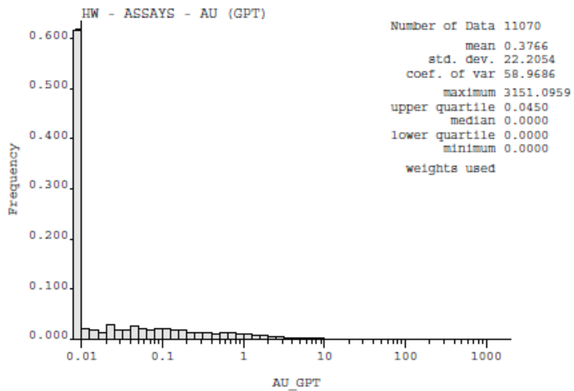

45 - 1.0M COMP - AU (GPT)
Cumulative Probability
AU_GPT

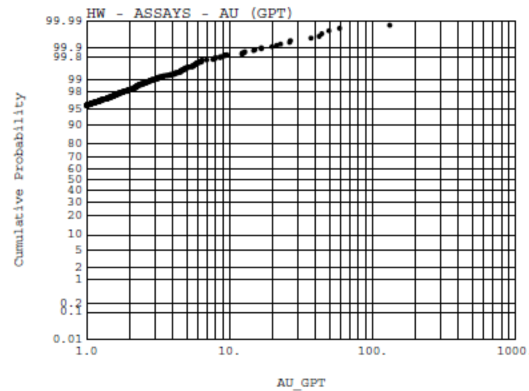

45 - 1.0M COMP - AU (GPT)
Mean
Number of capped data
Capped Value

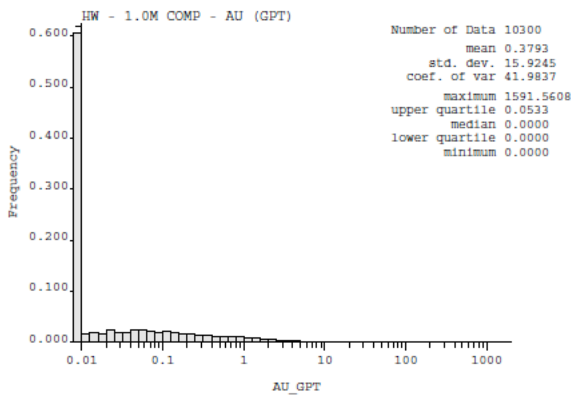

45 - ASSAYS VS COMP CAP - AU (GPT)
number x 20586
number y 19453
x mean 0.629
std. dev. 8.883
y mean 0.532
std. dev. 2.299
AU_CAP
AU_GPT

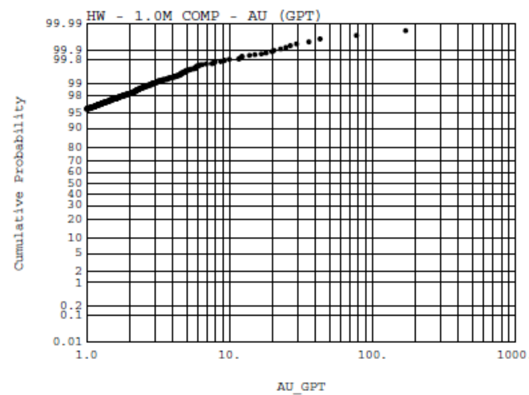

45 - 1.0M COMP CAP - AU (GPT)
Number of Data 19453
mean 0.5323
std. dev. 2.2991
coef. of var 4.3189
maximum 200.0000
upper quartile 0.6023
median 0.2200
lower quartile 0.0000
minimum 0.0000
Frequency
AU_CAP

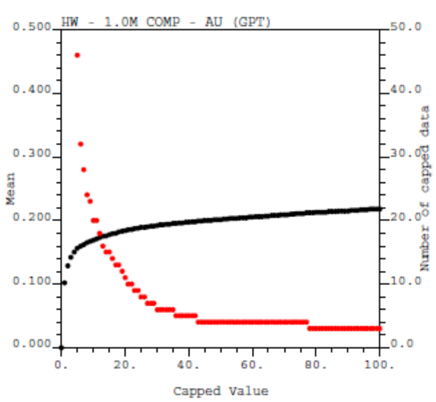

45 - 1.0M COMP CAP - AU (GPT)
Cumulative Probability
AU_CAP

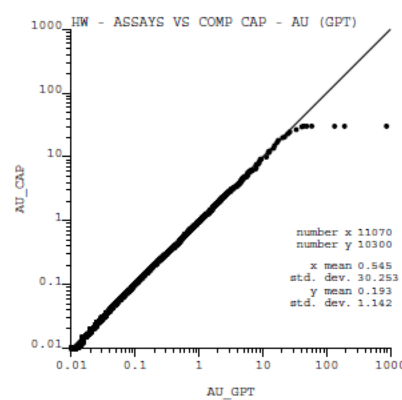

HW - ASSAYS - AU (GPT)
Number of Data 11070
mean 0.3766
std. dev. 22.2054
coef. of var 58.9686
maximum 3151.0959
upper quartile 0.0450
median 0.0000
lower quartile 0.0000
minimum 0.0000
weights used
Frequency
AU_GPT

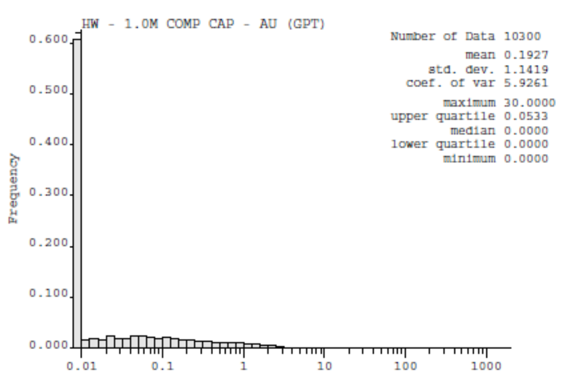

HW - ASSAYS - AU (GPT)
Cumulative Probability
AU_GPT

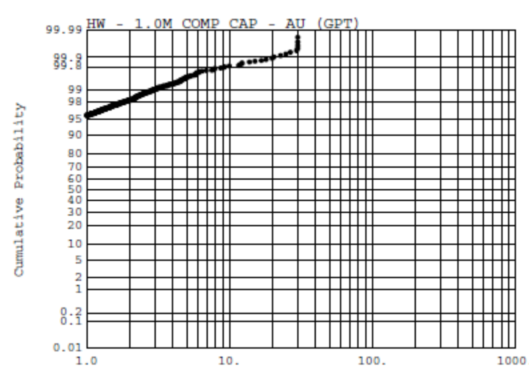

HW - 1.0M COMP - AU (GPT)
Number of Data 10300
mean 0.3793
std. dev. 15.9245
coef. of var 41.9837
maximum 1591.5608
upper quartile 0.0533
median 0.0000
lower quartile 0.0000
minimum 0.0000
Frequency
AU_GPT

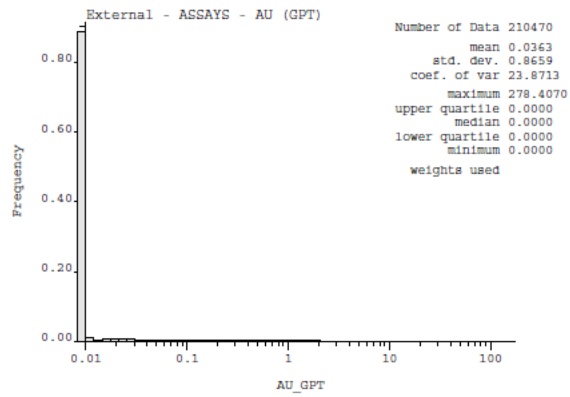

HW - 1.0M COMP - AU (GPT)
Cumulative Probability
AU_GPT

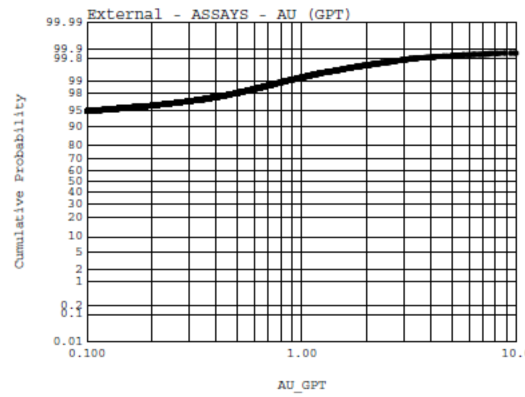

HW - 1.0M COMP - AU (GPT)
Mean
Number of capped data
Capped Value

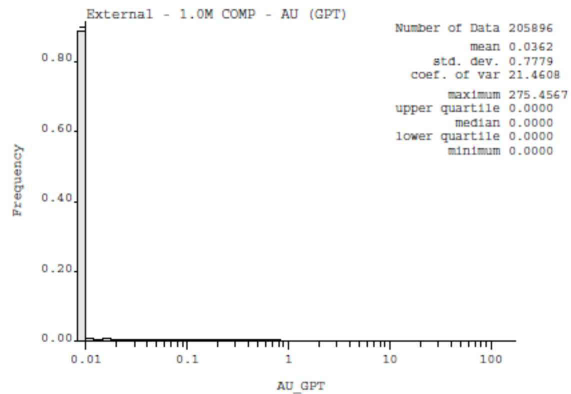

HW - ASSAYS VS COMP CAP - AU (GPT)
number x 11070
number y 10300
x mean 0.545
std. dev. 30.253
y mean 0.193
std. dev. 1.142
AU_CAP
AU_GPT

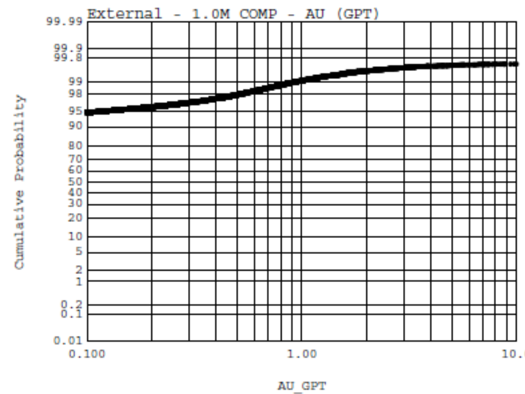

HW - 1.0M COMP CAP - AU (GPT)
Number of Data 10300
mean 0.1927
std. dev. 1.1419
coef. of var 5.9261
maximum 30.0000
upper quartile 0.0533
median 0.0000
lower quartile 0.0000
minimum 0.0000
Frequency
AU_CAP

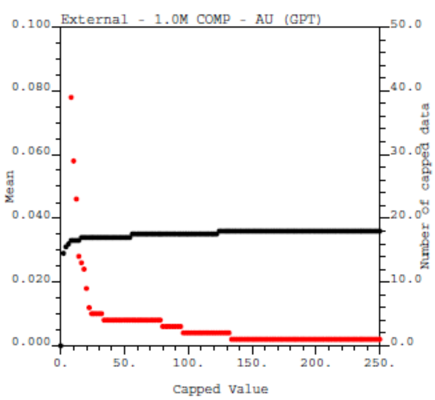

HW - 1.0M COMP CAP - AU (GPT)
Cumulative Probability
AU_CAP

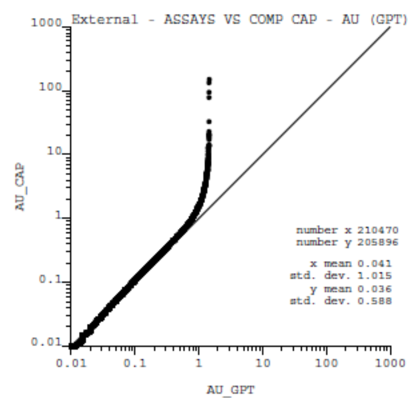

External - ASSAYS - AU (GPT)
Number of Data 210470
mean 0.0363
std. dev. 0.8659
coef. of var 23.8713
maximum 278.4070
upper quartile 0.0000
median 0.0000
lower quartile 0.0000
minimum 0.0000
weights used
Frequency
AU_GPT

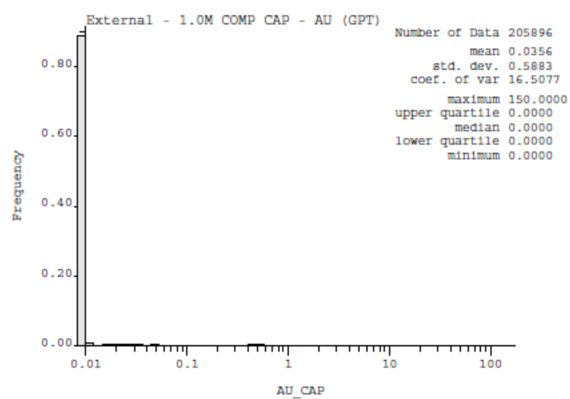

External - ASSAYS - AU (GPT)
Cumulative Probability
AU_GPT

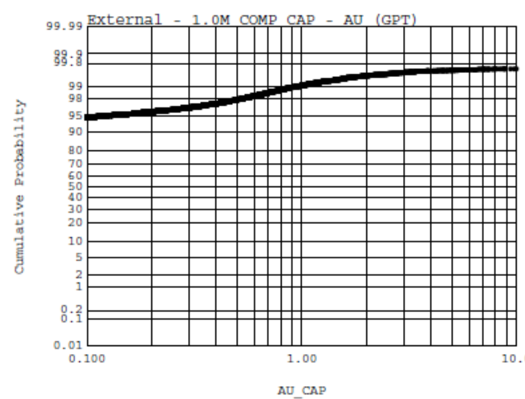

External - 1.0M COMP - AU (GPT)
Number of Data 205896
mean 0.0362
std. dev. 0.7779
coef. of var 21.4608
maximum 275.4567
upper quartile 0.0000
median 0.0000
lower quartile 0.0000
minimum 0.0000
Frequency
AU_GPT


External - 1.0M COMP - AU (GPT)
Cumulative Probability
AU_GPT


External - 1.0M COMP - AU (GPT)
Mean
Number of capped data
Capped Value

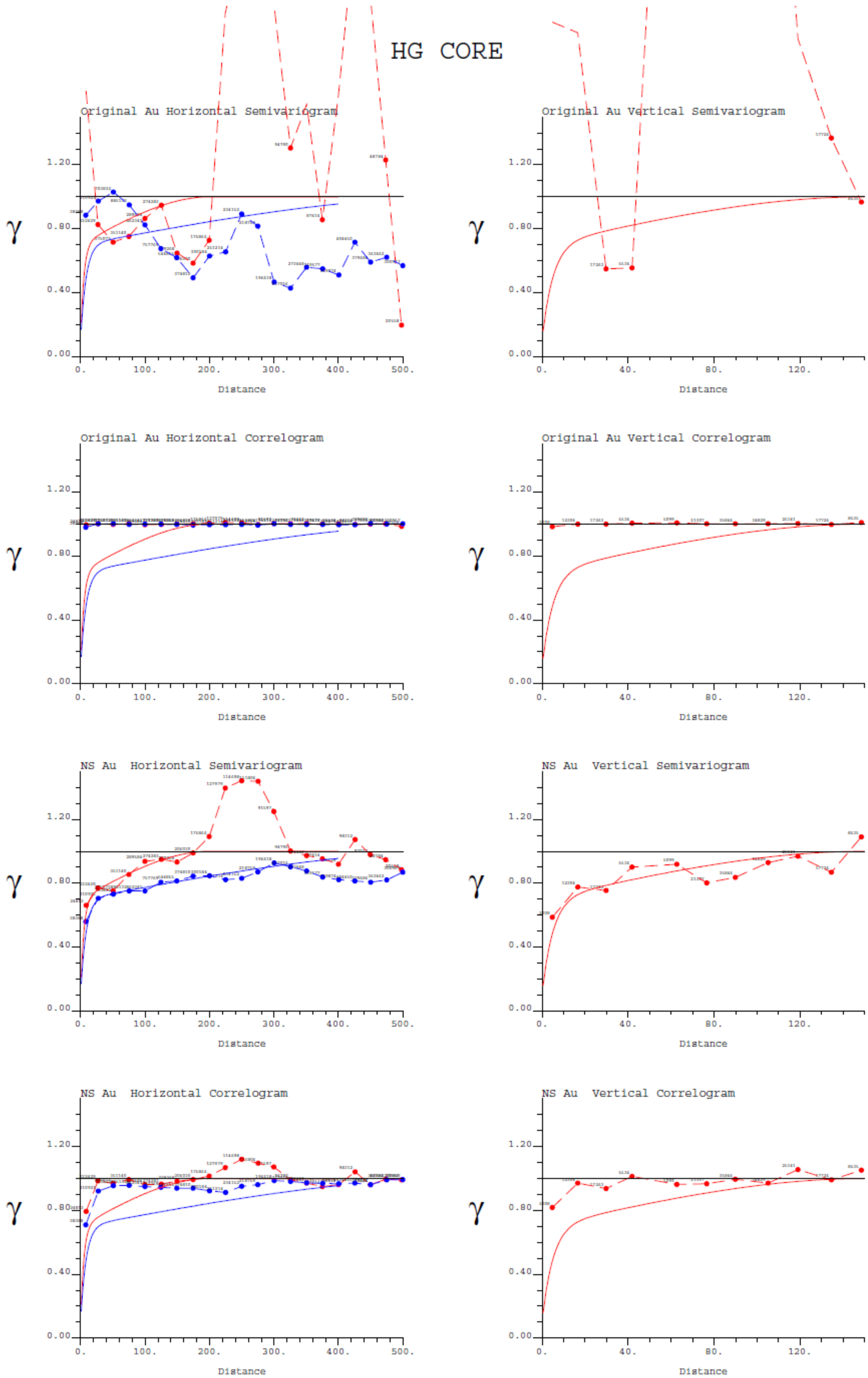

External - ASSAYS VS COMP CAP - AU (GPT)
number x 210470
number y 205896
x mean 0.041
std. dev. 1.015
y mean 0.036
std. dev. 0.588
AU_CAP
AU_GPT


External - 1.0M COMP CAP - AU (GPT)
Number of Data 205896
mean 0.0356
std. dev. 0.5883
coef. of var 16.5077
maximum 150.0000
upper quartile 0.0000
median 0.0000
lower quartile 0.0000
minimum 0.0000
Frequency
AU_CAP

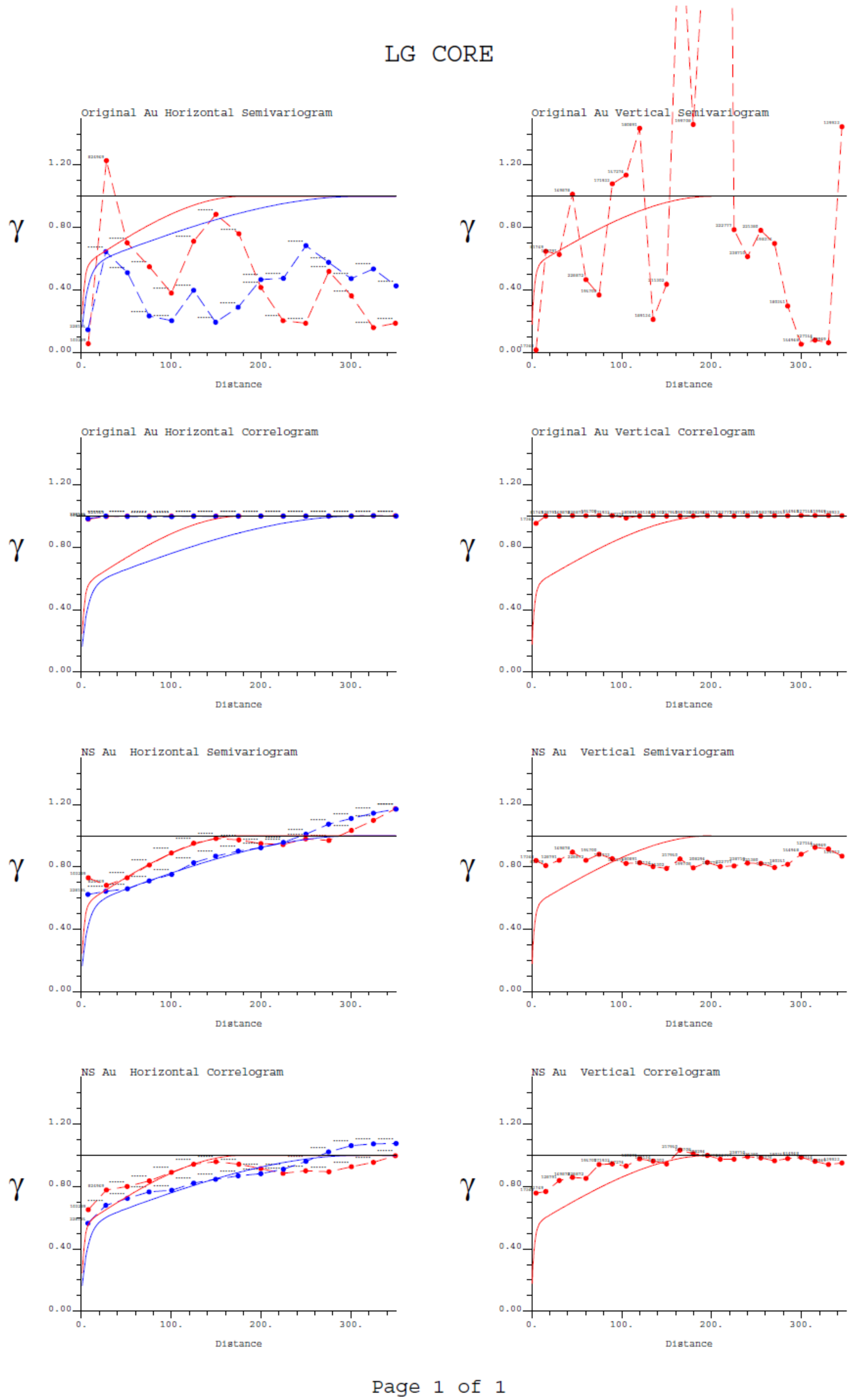

External - 1.0M COMP CAP - AU (GPT)
Cumulative Probability
AU_CAP

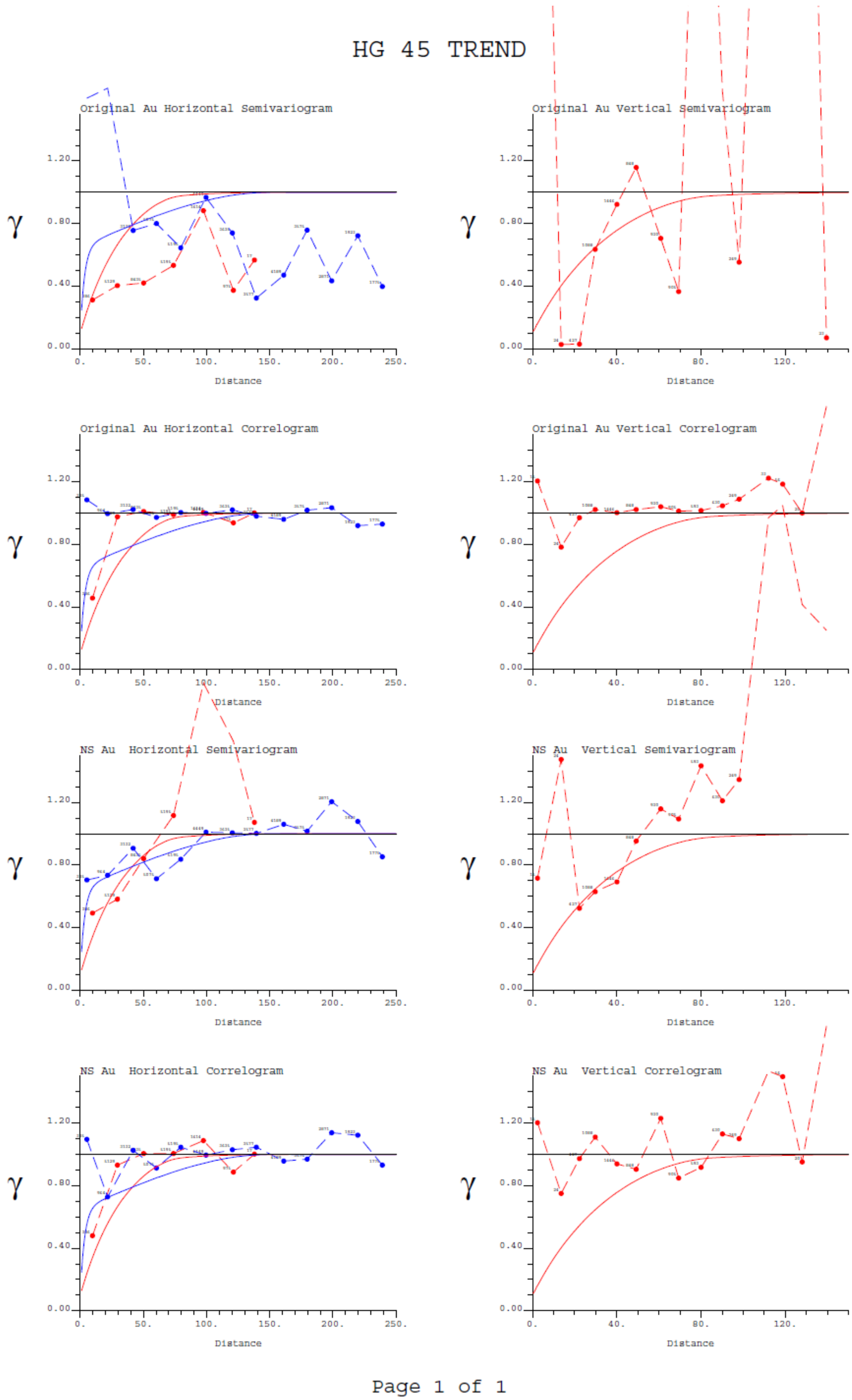
APPENDIX C

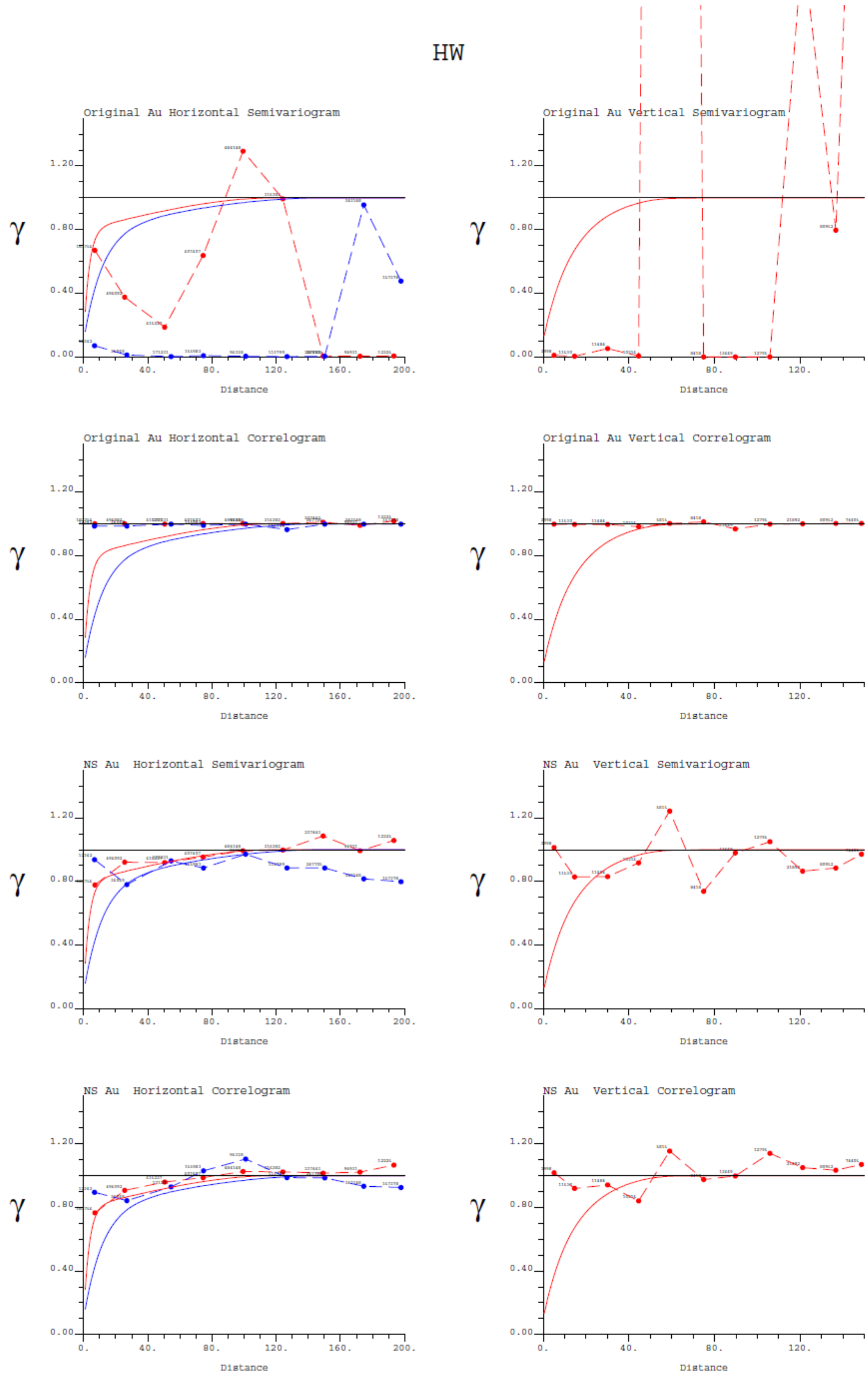
Modelled Variograms


HG CORE
Original Au Horizontal Semivariogram
Original Au Vertical Semivariogram
Original Au Horizontal Correlogram
Original Au Vertical Correlogram
NS Au Horizontal Semivariogram
NS Au Vertical Semivariogram
NS Au Horizontal Correlogram
NS Au Vertical Correlogram
γ
0.00
0.40
0.80
1.20
0.
100.
200.
300.
400.
500.
40.
80.
120.
Distance

Page 1 of 1


LG CORE
Original Au Horizontal Semivariogram
Original Au Vertical Semivariogram
Original Au Horizontal Correlogram
Original Au Vertical Correlogram
NS Au Horizontal Semivariogram
NS Au Vertical Semivariogram
NS Au Horizontal Correlogram
NS Au Vertical Correlogram
γ
0.00
0.40
0.80
1.20
0.
100.
200.
300.
Distance
Page 1 of 1


HG 45 TREND
Original Au Horizontal Semivariogram
Original Au Vertical Semivariogram
Original Au Horizontal Correlogram
Original Au Vertical Correlogram
NS Au Horizontal Semivariogram
NS Au Vertical Semivariogram
NS Au Horizontal Correlogram
NS Au Vertical Correlogram
γ
Distance
Page 1 of 1

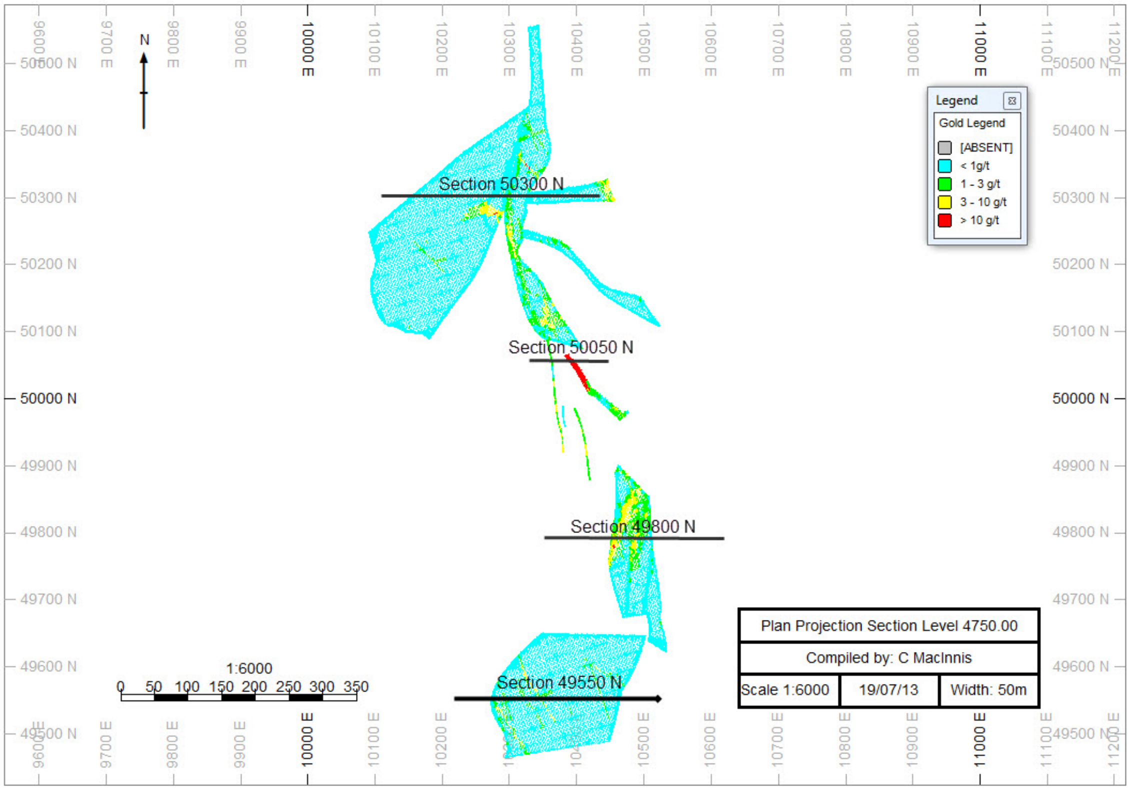

HW
Original Au Horizontal Semivariogram
Original Au Vertical Semivariogram
Original Au Horizontal Correlogram
Original Au Vertical Correlogram
NS Au Horizontal Semivariogram
NS Au Vertical Semivariogram
NS Au Horizontal Correlogram
NS Au Vertical Correlogram
γ
Distance
0.00
0.40
0.80
1.20
0.
40.
80.
120.
160.
200.

Page 1 of 1

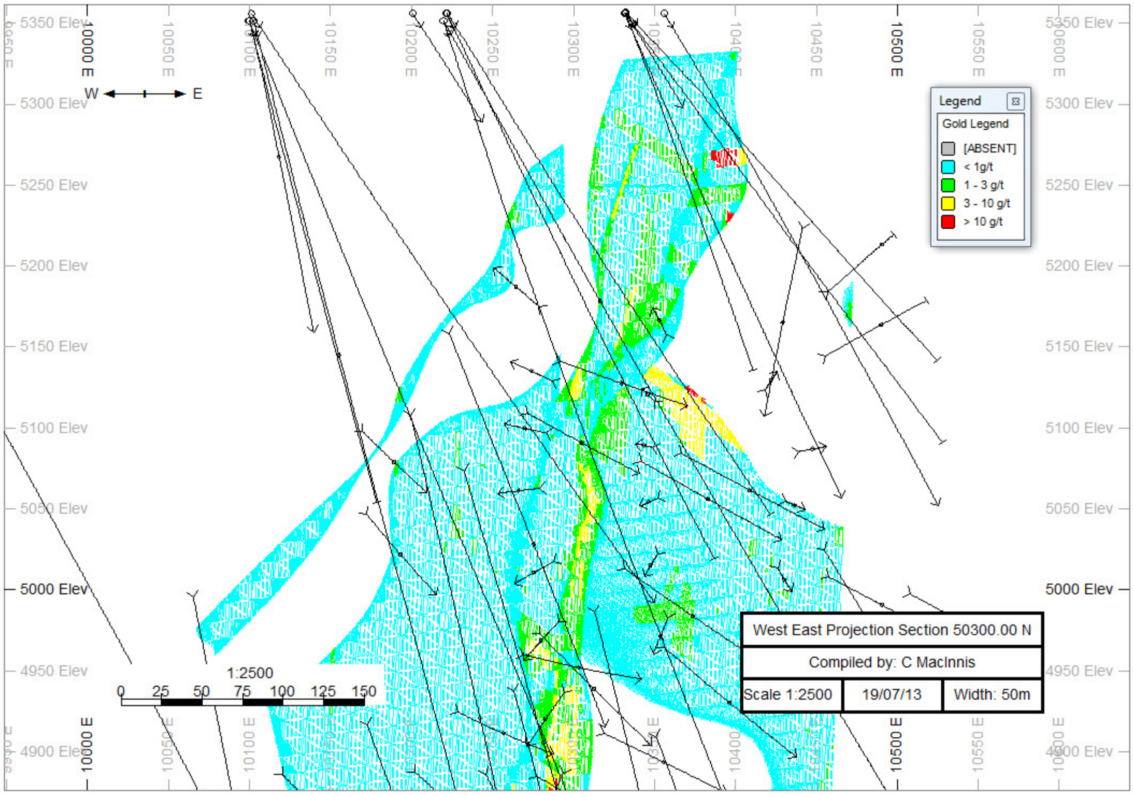
APPENDIX D

Plan and Sections across the Phoenix Gold Deposit

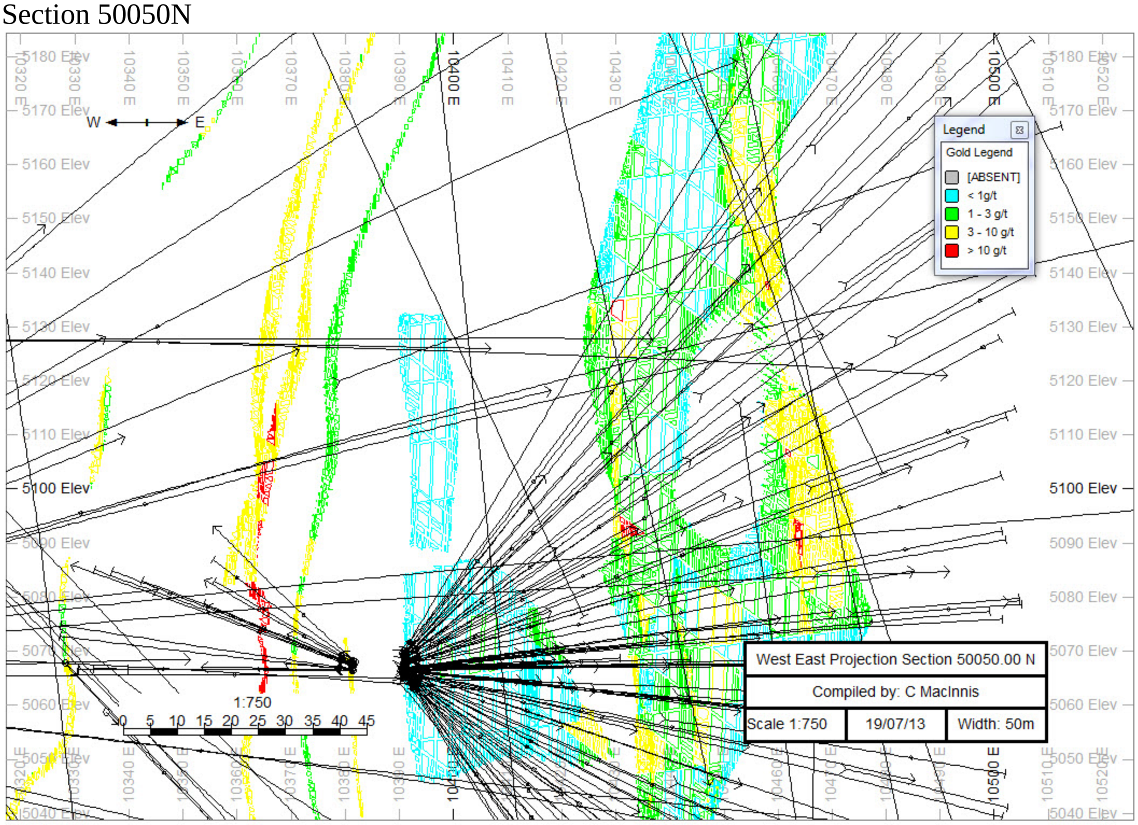
Plan showing location of sections across the Phoenix Gold deposit



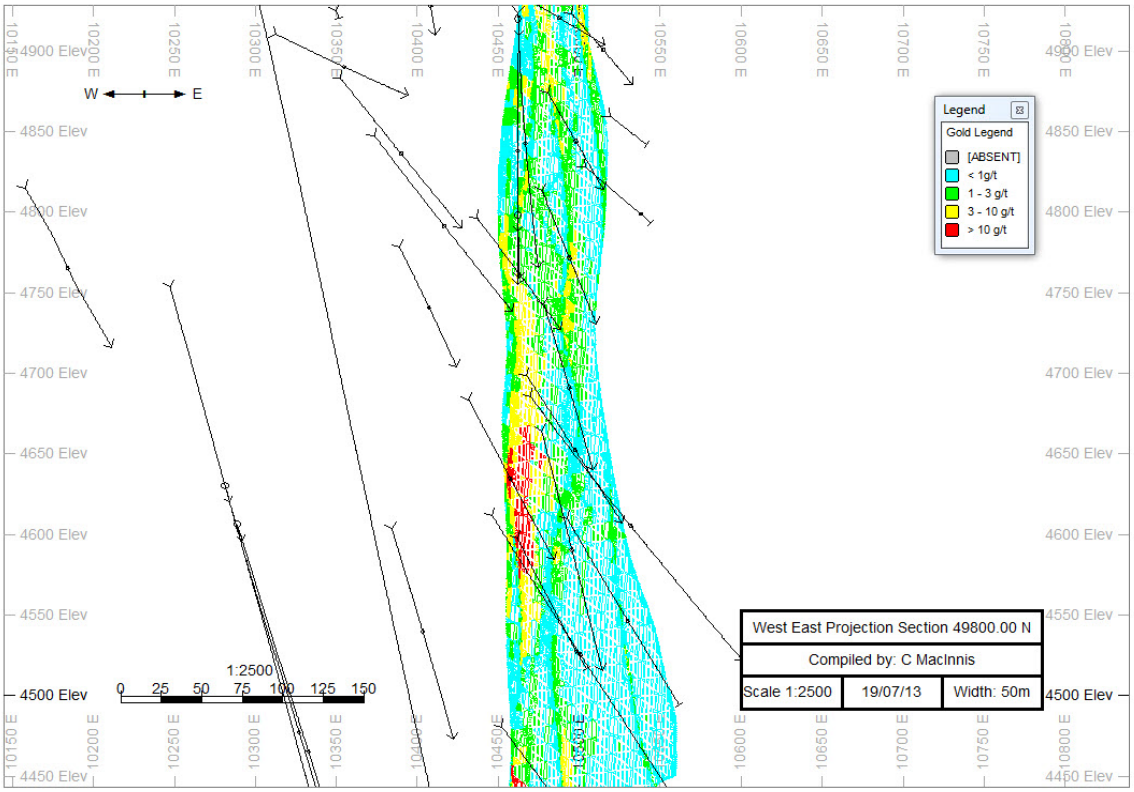
Section 50300N



Section 50050N



Section 49800N



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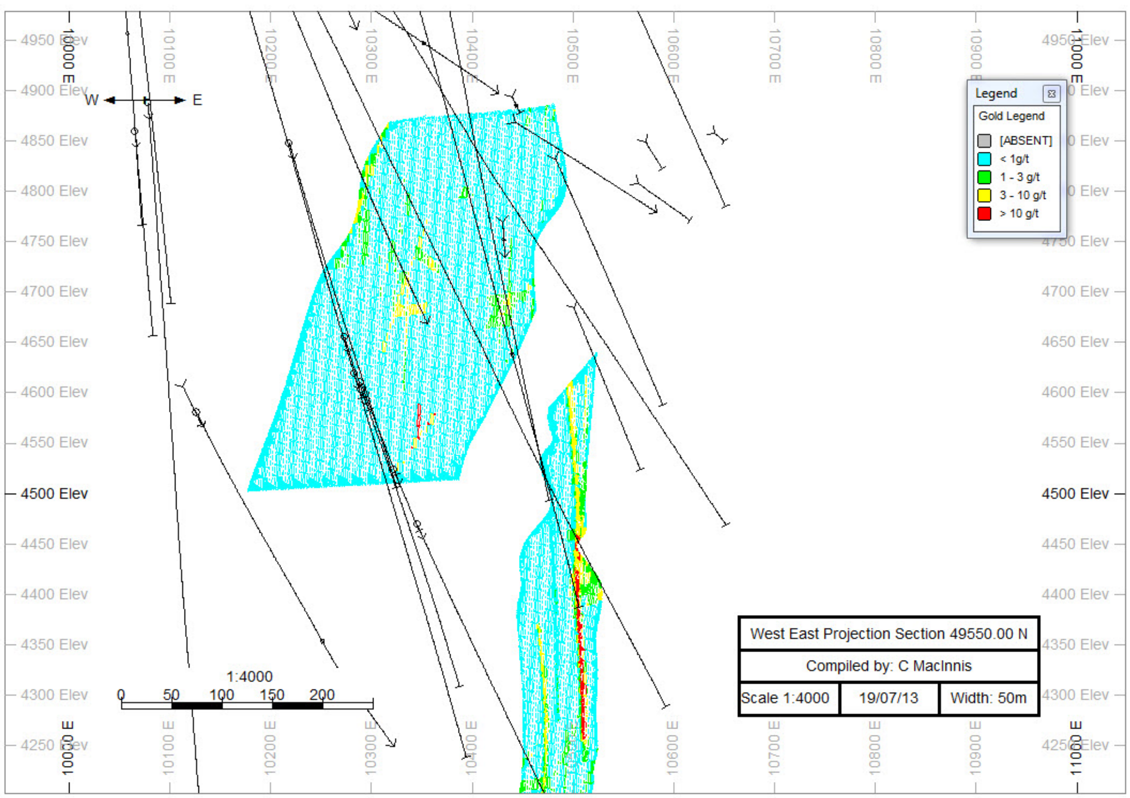

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: "Preliminary Economic Assessment of the F2 Gold System, Phoenix Gold Project, Red Lake, Ontario" dated February 28, 2014.

I, Sébastien B. Bernier, residing at 54 Bayside Crescent, Sudbury do hereby certify that:

1) I am a Principal Consultant (Resource Geology) with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 101, 1984 Regent Street South, Sudbury, Ontario, Canada;
2) I am a graduate of the University of Ottawa in 2001 with B.Sc. (Honours) Geology and I obtained M.Sc. Geology from Laurentian University in 2003. I have practiced my profession continuously since 2002. I worked in exploration and commercial production of base and precious metals mainly in Canada. I have been focussing my career on geostatistical studies, geological modelling and resource modelling of base and precious metals since 2004;
3) I am a professional Geoscientist registered with the Association of Professional Geoscientists of Ontario (# 1847);
4) I have not personally inspected the Phoenix Gold Project, but have relied on site visits by Mr. Stephen Taylor, PEng (PEO # 90365834) on January 22, 2013 and on April 17, 2013.
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) I, as a Qualified Person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the co-author of this report and responsible for Section 13 and accept professional responsibility for that section of this technical report;
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) SRK Consulting (Canada) Inc. was retained by Rubicon Mineral Corporation to prepare a technical audit of the Phoenix project. In conducting our audit, a gap analysis of project technical data was completed using CIM "*Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines*" and Canadian Securities Administrators National Instrument 43-101 guidelines. The preceding report is based on a site visit, a review of project files and discussions with Rubicon Minerals Corporation personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Phoenix project or securities of Rubicon Minerals Corporation; and
12) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

["signed and sealed"]

Sudbury, Ontario
February 28, 2014

Sébastien B. Bernier, PGeo (APGO#1847)
Principal Consultant (Resource Geology)

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: “Preliminary Economic Assessment of the F2 Gold System, Phoenix Gold Project, Red Lake, Ontario” dated February 28, 2014.

I, Glen Cole residing at 15 Langmaid Court, Whitby, Ontario do hereby certify that:

1) I am a Principal Consultant (Resource Geology) with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 1300, 151 Yonge Street, Toronto, Ontario, Canada;
2) I am a graduate of the University of Cape Town in South Africa with a B.Sc (Hons) in Geology in 1983; I obtained an M.Sc (Geology) from the University of Johannesburg in South Africa in 1995 and an M.Eng in Mineral Economics from the University of the Witwatersrand in South Africa in 1999. I have practiced my profession continuously since 1986. Between 1986 and 1989 I worked as a staff geologist on various Anglo American mines. Between 1989 and 2005 I have worked for Goldfields Ltd at several exploration projects, underground and open pit mining operations in Africa and held positions of Mineral Resources Manager, Chief Mine Geologist and Chief Evaluation Geologist, with the responsibility for estimation of mineral resources and mineral reserves for development projects and operating mines. Since 2006, I have estimated and audited mineral resources for a variety of early and advanced base and precious metals projects in Africa, Canada, Chile and Mexico.
3) I am a Professional Geoscientist registered with the Association of Professional Geoscientists of the Province of Ontario (APGO#1416), the Association of Professional Engineers and Geoscientists of the Province of Saskatchewan (PEGS#26003) and am also registered as a Professional Natural Scientist with the South African Council for Scientific Professions (Reg#400070/02);
4) I have not personally inspected the Phoenix Gold Project, but have relied on site visits by Mr. Stephen Taylor, PEng (PEO # 90365834) on January 22, 2013 and on April 17, 2013.
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) I, as a Qualified Person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the co- author of this report and responsible for sections 1 to 11, 13, 24, 25 and Appendix A to D of this technical report and accept professional responsibility for those sections of this technical report;
8) I have had no prior involvement with the subject property.
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) SRK Consulting (Canada) Inc. was retained by Rubicon Minerals Corporation to prepare a technical audit of the Phoenix gold project. In conducting our audit, a gap analysis of project technical data was completed using CIM “*Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines*” and Canadian Securities Administrators National Instrument 43-101 guidelines. The preceding report is based on a site visit, a review of project files and discussions with Rubicon Minerals Corporation personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Phoenix gold project or securities of Rubicon Minerals Corporation; and
12) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

["signed and sealed"]

Toronto, Ontario
February 28, 2014

Glen Cole, P.Geo (APGO#1416)
Principal Consultant (Resource Geology)

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: "Preliminary Economic Assessment of the F2 Gold System, Phoenix Gold Project, Red Lake, Ontario" dated February 28, 2014.

I, Daniel Hewitt, residing at 1960 Latimer Crescent, Sudbury, Ontario do hereby certify that:

1) I am a Principal Consultant (Mining) with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 101, 1984 Regent Street South, Sudbury, Ontario, Canada;
2) I am a graduate of McGill University in Montreal with a BEng (Mining Engineering) in 1976 and studied business administration at York University in Toronto in 1983 and 1984. I have practiced my profession continuously since 1976. Between 1976 and 1980, I worked as a mine planner and mine foreman for Vale SA at the Copper Cliff South underground nickel mine in Sudbury, ON, and from 1980 to 1983 I was mine engineer at the underground tungsten mine in Tungsten, NT for Canada Tungsten Mining Corporation. From 1984 to 1989 I provided site engineering services and technical reports for advanced underground exploration gold projects in Ontario and Saskatchewan as a partner of Caribou Resources. Between 1989 and 2004, I worked with Vale SA as a mine planner (1989 to 1991), project engineer (1992 to 1995) and environmental coordinator (1996 to 2004) for several underground and open pit base metal mine sites in Sudbury, ON. During 2005 and 2006, I provided mine engineering and environmental services to mines and quarries in Ontario as a partner of Flat River Consulting. From 2006 to present, I have worked for SRK to design and provide site QA services for several mine reclamation projects, notably the Tundra and Giant gold mines in Northwest Territories, and for technical reports for underground gold projects in Red Lake, ON and Kirkland Lake, ON;
3) I am a professional engineer registered with Professional Engineers Ontario of the Province of Ontario (PEO#19465012) and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG#1032);
4) I have personally inspected the subject project on April 15, 2013. The purpose of the visit was to review the status of the project and possible data gaps with respect to geotechnical conditions in the proposed mining areas, existing underground openings, mining plans and schedule, processing and backfill facilities, environmental programs and compliance, property boundary, and closure planning;
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) I, as a Qualified Person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the co-author of this report and responsible for Sections 15 and 17 to 25, and accept professional responsibility for those sections of this technical report;
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) SRK Consulting (Canada) Inc. was retained by Rubicon Minerals Corporation to prepare a technical audit of the Phoenix gold project. In conducting our audit, a gap analysis of project technical data was completed using CIM "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" and Canadian Securities Administrators National Instrument 43-101 guidelines. The preceding report is based on a site visit, a review of project files and discussions with Rubicon Minerals Corporation personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Rubicon Minerals Corporation or securities of Rubicon Minerals Corporation; and
12) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

["signed and sealed"]

Sudbury, Ontario
February 28, 2014

Daniel Hewitt, PEng (PEO#19465012)
Principal Consultant (Mining)

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: “Preliminary Economic Assessment of the F2 Gold System, Phoenix Gold Project, Red Lake, Ontario” dated February 28, 2014.

I, Stephen Taylor, residing at 1352 Hastings Crescent, Sudbury, Ontario do hereby certify that:

1) I am a Principal Consultant (Mining Engineering) with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 101, 1984 Regent Street South, Sudbury, Ontario, Canada;
2) I am a graduate of Laurentian University in Sudbury, Ontario with a B.Eng in Mining Engineering in 1992 and I also obtained an M.Sc (Mining Engineering) from the University of Nevada-Reno, Mackay School of Mines in 1995. I have practiced my profession continuously since 1995. Between 1992 and 1995 I worked as a contract Engineer for Bharti Engineering, as a temporary Engineer at the Coeur Rochester mine in Nevada and as teaching assistant at the Mackay School of Mines. Between 1995 and 2002 I worked for Royal Oak Mines Ltd and Placer Dome Inc in a variety of underground and open pit positions in Newfoundland and Ontario. When the Porcupine Joint Venture was created in 2002, I was transferred to the Hoyle Pond mine in Timmins as Senior Engineer where I oversaw the mine engineering department and all capital projects. In 2006, I joined INCO Ltd (now Vale S.A.) as Senior Project Engineer where I oversaw many mine development and construction projects. Since joining SRK in 2010, I have been involved in a number of mining studies, including PEA’s and Feasibility studies of underground mines in north and south America;
3) I am a professional engineer registered with Professional Engineers Ontario of the Province of Ontario (PEO#90365834);
4) I have personally inspected the subject project on two occasions, visiting the site on January 22nd, 2013 and April 17th, 2013. The purpose of the January site visit was to collect data on ground conditions encountered previously, review the results of the bulk sampling program, and observe shaft sinking development. In April, the purpose of the site visit was to review existing infrastructure and equipment as well as discuss gaps in infrastructure to be addressed in the ongoing mine design work;
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) I, as a Qualified Person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the co-author of this report and responsible for Sections 14, 15, 24 and 25, and accept professional responsibility for those sections of this technical report;
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) SRK Consulting (Canada) Inc. was retained by Rubicon Minerals Corporation to prepare a technical audit of the Phoenix gold project. In conducting our audit, a gap analysis of project technical data was completed using CIM “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” and Canadian Securities Administrators National Instrument 43-101 guidelines. The preceding report is based on a site visit, a review of project files and discussions with Rubicon Minerals Corporation personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Rubicon Minerals Corporation or securities of Rubicon Minerals Corporation; and
12) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

[“signed and sealed”]

Sudbury, Ontario — Stephen Taylor, PEng (PEO#90365834)
February 28, 2014 — Principal Consultant (Mining Engineering)

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: "Preliminary Economic Assessment of the F2 Gold System, Phoenix Gold Project, Red Lake, Ontario" dated February 28, 2014.

I, Pierre Roy, residing at 2451 Avenue De Vitré, Quebec, Quebec, do hereby certify that:

1) I am a Senior Metallurgist and Mineral Processing Specialist with the firm of Soutex Inc. (Soutex) with an office at 357 Jackson, Quebec, Quebec, Canada;
2) I graduated with a Bachelor in Mining from Laval University, Canada in 1986 and with a Master of Science from Laval University, Canada in 1989. I have practiced my profession continuously since 1986 (including a master's degree), and have been involved in mineral processing for a total of 25 years since my graduation from University. This has involved working in Canada. My experience is principally in ore processing and in environment management;
3) I am a registered member of the "Ordre des Ingenieurs du Quebec" (OIQ#45201) and of the "Professional Engineers of Ontario"(PEO#100110987);
4) I have personally inspected the subject project on February 16 to 18, 2011, on March 10, 2011 and on April 18 to 20, 2011. The purpose of the visits was related to the sub-sampling of the two bulk samples that were extracted from underground and a review of surface infrastructure was also done for the confirmation of the planned mineral processing facility implementation. An underground visit of the place where the bulk samples were taken from was also performed;
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) I, as a Qualified Person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the co-author of this report and responsible for Sections 10.3, 11.3, 12, 13.12.2, 16, 24.7 and 25, and accept professional responsibility for those sections of this technical report;
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) Soutex Inc. was retained by Rubicon Minerals Corporation to prepare a technical audit of the Phoenix gold project. In conducting our audit, a gap analysis of project technical data was completed using CIM "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" and Canadian Securities Administrators National Instrument 43-101 guidelines. The preceding report is based on a site visit, a review of project files and discussions with Rubicon Minerals Corporation personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Rubicon Minerals Corporation or securities of Rubicon Minerals Corporation; and
12) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make the technical report not misleading.

Pierre Roy "signed and sealed"

Quebec, Quebec
February 28, 2014

Pierre Roy, PEng (PEO#100110987)
Senior Metallurgist, Mineral Processing Specialist