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F2 GOLD SYSTEM - PHOENIX GOLD PROJECT

BATEMAN TOWNSHIP

RED LAKE, CANADA

TECHNICAL REPORT

for

RUBICON MINERALS CORPORATION

Prepared by AMC Mining Consultants (Canada) Ltd. In accordance with the requirements of National Instrument 43-101, "Standards of Disclosure for Mineral Projects"

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1 EXECUTIVE SUMMARY

Introduction

This Technical Report on the F2 Gold System, Phoenix Gold Project (the Project) within the larger Phoenix Gold Property (the Property) in Bateman Township, Red Lake, Canada has been prepared by AMC Mining Consultants (Canada) Ltd (AMC) of Vancouver, Canada on behalf of Rubicon Minerals Corporation (Rubicon) of Vancouver, Canada. It has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101), “Standards of Disclosure for Mineral Projects”, of the Canadian Securities Administrators (CSA) for lodgement on CSA’s “System for Electronic Document Analysis and Retrieval” (SEDAR). This report is the disclosure of a Preliminary Economic Assessment (PEA) carried out by AMC. It is an also update to the mineral resources stated in a previously published Technical Report entitled “Technical Report, Mineral Resource and Geological Potential Estimates, F2 Gold System - Phoenix Gold Project”, prepared for Rubicon by GeoEx Limited, and effective 11 April 2011.

This PEA is preliminary in nature, includes Inferred 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, and there is no certainty that the preliminary economic assessment will be realized.

History, Location and Ownership

The Property was initially staked by McCallum Red Lake Mines Ltd in 1922. Ownership changed several times until Rubicon optioned the Property from Dominion Goldfields Corporation (DGC) (Water Claims) and 1519369 Ontario Ltd (Land Claims) in 2002.

It is located in the southwestern part of Bateman Township within the Red Lake Mining Division of northwestern Ontario, at the northern end of the McFinley Peninsula and partly under the East Bay of Red Lake. The site incorporates the former McFinley underground mine, a shaft and hoist system, and other surface infrastructure. The town of Red Lake is approximately 265 km NE of Winnipeg and is serviced by road and air, with daily scheduled flights from Winnipeg and Thunder Bay. The closest railway is approximately 160 km south. Temperatures range from summer highs of 35°C to winter lows of -40°C. Weather conditions allow drilling from the ice of Red Lake during January to early April.

The Property consists of 38 contiguous blocks covering 509.4 ha that are comprised of 16 patented mining claims (Land Portion), one unpatented staked claim, one mining lease and 25 licenses of occupation (Water Portion). The Water Portion claims are subject to a 2% Net Smelter Return (NSR) royalty with a requirement to pay advance royalties and an option to acquire a 0.5% NSR royalty for US$675,000. The Land Portion claims are subject to a sliding scale, 2-3% NSR royalty depending on the price of gold, with a requirement to pay advance royalties and an option to acquire a 0.5% NSR royalty for $1,000,000.

Rubicon holds all permits required to allow it to carry out its current drilling and underground exploration program and is in the process of acquiring additional permits required in contemplation of future production.

Geology and Mineralization

The Red Lake Greenstone Belt is located in the western portion of the Canadian Shield, and consists of an EW trending sequence of volcanic and sedimentary rocks with volcanic intrusives that span a period of 300 million years.

The local area is underlain by three sequences of primarily tholeiitic mafic volcanic rocks, separated by marker horizons of felsic and ultramafic volcanic rocks. A strong NNE trending structural fabric through the area is considered part of the East Bay Deformation Zone (EBDZ), which dominates the geology of the area.

The F2 Gold System comprises a NE-trending, steeply W-dipping sequence with numerous lode gold shoots typical of the Red Lake Camp containing high grade gold mineralization, hosted within 'Hi-Ti' Basalt and Felsic Intrusive lithologies within a larger body of ultramafic and mafic talc-rich lithologies. The geological sequence is reasonably consistent along the length of the Property for over 4 km. Rock types hosting mineralization have been correlated over vertical distances of approximately 1,500m and horizontal distances of approximately 1,200 m.

Delineation drilling of the F2 Core Zone has defined the presence of NNE-trending gold mineralization associated with silicification, quartz veining and strong alteration within, and adjacent to, favourable host rock types. Gold mineralization also occurs in NW-trending structures that are generally confined within, or immediately adjacent to, NE-trending bounding geological units and parallel to the regional F2 fold trend direction. Typically, this mineralization occurs as local quartz veining and brecciation.

Exploration

Rubicon has conducted extensive exploration programs since acquiring the property. Work has included geological mapping, re-logging of selected historic drillholes, digital compilation of all historical data available, high resolution airborne magnetic survey, ground magnetic survey, seismic lake bottom topographic survey, Titan 24 geophysical survey and numerous drilling programs, all of which have been covered in this or the earlier Technical Reports.

Since 2002, Rubicon has completed 313,030 m of diamond drilling (182,802 m surface drilling and 130,228 m underground drilling) to February 28, 2011. In the fourth quarter of 2010, a 27,000 m underground drilling program was initiated, designed to test a 150 m (horizontal) by 200 m (vertical) area within the F2 Core Zone. This program identified discrete sub-zones within the F2 Core Zone and returned some high-grade gold intercepts. The total system has now been expanded to a strike length of approximately 1,240 m and to depths of around 1,460 m vertically.

AMC's mineral resource estimate is based on approximately 259,000 m of drilling completed on the F2 Gold system between 2008 and 28 February 2011.

Data Quality and Management

Collar surveying of proposed and completed drillholes, and down-hole surveying of holes, has been undertaken to good industry standards. Core recovery has generally been excellent. RQD measurements are completed on the core, as well as representative specific gravity and magnetic susceptibility measurements.

Core handling, splitting and sampling procedures are standard and the facilities are good with 24 hour on-site security including personnel and video surveillance. All analytical or testing laboratories used by Rubicon are independent. While a number of analytical laboratories have been used over time, assaying since January 2008 has been conducted by SGS in Red Lake.

Blank and standards assay protocols were developed in 2003 and revisited in 2009 with input from Dr. B Smee, Ph.D., P.Geo, Independent Geochemist. Since May 2010, ioGlobal Pty Ltd has taken over the management of the project assay data, providing independent quality control and quality assurance reporting and database auditing.

AMC is satisfied that the sample preparation, security, analytical procedures and application of Quality Assurance/Quality Control (QA/QC) analysis is performed in accordance with industry good practice and that the database is appropriate for resource modelling and estimation.

Mineral Resources and Mineral Reserves

AMC has estimated mineral resources using a block modelling approach based on all drilling up to 28 February 2011. The estimate is not constrained vertically by a potential crown pillar and extends to incorporate all data at depth. Summary results at a cut off grade of 5.0 g/t Au are shown in Table 1 below.

Table 1 Summary of Mineral Resource Estimates as of 15 June 2011

Classification	M Tonnes	g/t Au	M oz Au
Indicated	1.028	14.5	0.477
Inferred	4.230	17.0	2.317

Notes: 1. CIM definitions used for mineral resources
2. Cut off grade applied 5.0 g/t Au
3. Capping value of 270g/t Au applied to composites
4. Based on drilling results to 28 February, 2011

A total of 511 drillholes were used in the modelling. Rubicon's interpretations of lithologies, mineralization controls and geology domains were reviewed and accepted by AMC. Mineralized domains (total of 12) were interpreted by AMC using a low gold threshold (0.1 g/t Au), and were further expanded to incorporate all significant mineralized zones.

A composite length of 1.0 m was chosen and grades were capped at 270 g/t Au after compositing. The parent block size was 2 m by 8 m by 12 m, with sub blocking utilized. Search parameters were 8 m by 24 m 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. Grade interpolation was by inverse distance cubed. An average bulk density value of 2.90 t/m^3 was used for all rock types.

Resource classification was carried out using data support as a main criteria with a manual review creating volumes based on drillhole density and number of samples to inform a block.

There are currently no mineral reserves quoted for the Project.

Underground Activities and Potential Mining

Rubicon has rehabilitated parts of the existing McFinley Mine workings, including the shaft, which it has deepened to just below 305 Level (305L). It has undertaken lateral and raise development to enable the establishment of diamond drilling stations on 122L, 244L and 305L, and has recently begun construction of a new hoist and headframe with the capability to exceed the envisaged, steady-state hoisting rate of 1,250 tonnes per day (tpd).

In AMC's view, economic mineralization in the F2 Zone is likely to be relatively discontinuous, (as typical for lode gold style mineralization), requiring a flexible mining approach and a high and sustained geological effort, including closely spaced definition drilling, to facilitate understanding localized mineralization trends

The selected primary mining method is conventional, captive cut and fill, with paste fill using mill tailings. Mining widths will be typically around 2 m, ranging from 1.5 m or less to 3 m or more. Ore will be mined using stopers, jacklegs, slushers and mucking machines, dumped to an ore pass, loaded into track cars, passed to a shaft loading pocket via a grizzly and then hoisted to surface. Waste development will also generally be done using stopers, jacklegs and mucking machines.

Cross-cut and access development from the shaft at nominal 2.4 m width by 2.7 m height is envisaged on levels at 61 m intervals between 183L and 1403L, complementary to the present 305L development. Necessary raising for waste, fresh-air and mine egress will be undertaken together with development of loading pockets for waste adjacent to the shaft. Ore passes at 2.4 m by 2.4 m will be developed at points central to the stoping areas. The required lateral and raising development is projected to continue over the first twelve years of the project.

The shaft bottom is currently at around 30 m below 305L, with eventual deepening of the shaft projected to just below 1464L. For the projected initial mining stage and prior to completion of shaft sinking, ore immediately below 305L may be accessed via ramp development.

AMC has undertaken regular geotechnical inspections and audits over a two year period and has developed provisional ground support standards for ongoing site activities in conjunction with Rubicon. AMC's geotechnical investigations have included a recommendation that a minimum crown pillar thickness of 45 m be maintained for the typical mining widths envisaged.

Groundwater flows appear unlikely to be a material issue for underground mining, but AMC has included a hydrological assessment in its recommendations. A third-party ventilation report has been reviewed by AMC, and the system proposed is considered appropriate for the mining methodology, equipment and production rates envisaged. Additional ventilation capability would be required if significant development were to occur using diesel scoops.

As the Project is at a PEA level, no mineral reserves have been estimated. To assess the preliminary economics of the Project, AMC has applied a cut-off grade of 6.0 g/t Au to the resource estimate lying between 1464L and the base of the planned crown pillar (approximately at 122L), based on an initial operating cost estimate of $200 /tonne and a gold price of US$1,040 /oz. Final cost estimation and use of a gold price of $1,100/oz for the PEA has confirmed that a mining cut-off grade of 6.0 g/t Au is reasonable at this stage of the Project. Mining dilution and recovery have been considered as have the possibly anomalously high grades of the deepest resource blocks, which are relatively poorly supported in terms of drillhole data.

Production is currently planned to start at 750 tpd, increase to around 970 tpd in the fourth year and reach a steady-state level of 1,250 tpd in the fifth year. A 14-year project life is envisaged for the resources scheduled in the PEA, with the first two years being devoted to key surface and underground infrastructure construction, including mill and paste plant, shaft sinking, lateral and raise development, etc. Total mine production over the 12-year life of mine is projected at 4.5 Mt, for total gold mined of around 2M ounces.

In early 2011, Rubicon extracted two 1,000 t bulk samples representing two underground areas on the 305L. Metallurgical testwork was conducted on the samples under the supervision of a processing consultant. Weighted average grades from drilling of the broad zones from within which the bulk samples were extracted are compared with metallurgical test results in Table 2.

Table 2 Comparison of Bulk Sampling to Drill Data

Item	WLB2 grade - g/t Au	F2 Core grade - g/t Au
Delineation Drilling Weighted Average	5.8	9.1
Milled Bulk Sample Testing Results	7.1	8.2

Processing

Proposed metallurgical processing consists of a single line, starting with a semi-autogenous grinding (SAG) mill. The discharge from the SAG mill is pumped to hydrocyclones for classification. A gravity separation circuit is included in closed circuit with hydrocyclones to recover any gravity recoverable gold prior to regrinding in a ball mill. Gold is extracted in a conventional carbon-in-leach (CIL) circuit. The loaded carbon is washed with hydrochloric acid solution to remove carbonate. Gold is then removed from the loaded carbon by stripping (elution) followed by electrowinning and 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 from the process to avoid gold loss, with fresh carbon being continuously added to the process.

The cyanide in the tailings from the CIL circuit is removed in a cyanide destruction tank with SO_2 and air diffuser placed at the bottom of the tank. Once the cyanide is destroyed, the tailings pass through the paste plant to produce a paste product. The paste produced can be sent to the tailings management facility (TMF) or used in the mine for backfill after the addition of cement and/or other binder to meet fill strength requirements.

Gold feed tonnage is projected to be 450,000 tonnes per annum (tpa) at steady state (Project years 05 through 11). Gold recovery is projected at 92.5%, with an average of 180,000 ounces of saleable gold per annum resulting over the steady state period. The mining scenario average head grade of 13.9 g/t gives 1.86 M saleable ounces of gold over the LOM.

Services and Infrastructure

Electrical power at the Project site is currently supplied by diesel generators. Rubicon has accepted an Offer to Connect from Hydro One for 5.3MW of electricity from the 44KV grid in the Municipality of Red Lake, via a 10.4 km power line. It has constructed a connection to the grid and is securing title to the required right-of-way. A fibre-optic line will be installed along the same route as the power-line to provide communication capability for the site.

Compressed air is currently supplied to underground via a surface compressor set-up that AMC believes will be adequate for the underground mining activity envisaged. Mine water is pumped to a holding tank at the site from the nearby East Bay of Red Lake. Use of paste fill means that there should be no significant source of waste water in the mine other than groundwater.

Process water will be a combination of water from East Bay, clarified water recycled from the TMF, water from the plant site sump and treated water from the mine and mill.

A TMF was constructed at the Project site in 1988 in preparation for a bulk-sampling program. The termination of activities in 1989 resulted in minimal use of this area. The TMF, and other sewage works, have been re-activated and approved under various Ontario government regulations. The TMF and effluent treatment plant have been designed to withstand a 30 day duration, 1 in 100 year rain or snow event.

There are no waste rock piles at the Project site related to historic development activities. Waste rock from future development activities will be consumed by roadbed, used to fill stopes or stored in the TMF. Mineralized material will be temporarily stockpiled on surface before processing. There will be no significant stockpiles of mineralized material at closure.

Environmental and Social

Rubicon has been advancing environmental and social aspects of the Project, including baseline monitoring and closure plans, since 2007. All are progressing satisfactorily with necessary permits being obtained or in progress and, to date, there have been no issues identified that post a material risk to project development and envisaged production.

Any waste rock not used for project purposes will be stored in the TMF, which will be monitored during project operation and rehabilitated at the end of mine life. Tests on representative samples suggest that tailings may ultimately turn acidic. A supplemental characterization program is continuing and will determine the need to implement a management plan to address potential chemical stability concerns. At closure, the TMF and plant sites will be covered with an engineered, low-permeability dry cover that will minimize infiltration of water.

Previous environmental work identified soil metal concentrations in parts of the 1980s brownfield project site that were above regulatory criteria. A risk assessment concluded that there was no unacceptable risk to the environment or to human health provided the project site is managed in accordance with applicable government legislation.

Surveys to date have not identified any endangered plants, animals or aquatic life that would be adversely affected by project development. Further field studies are being or will be conducted to ensure adherence to relevant government regulations.

There are no concerns with respect to instability at surface due to the underground workings that were developed in the 1980s. AMC has evaluated crown pillar requirements in accordance with government regulations and made recommendations for crown pillar dimensions to prevent instability at surface.

Environmental liabilities associated with the Project site are described in the February 2009 Phoenix Advanced Exploration Project Closure Plan, filed with the provincial government. In summary, there are no material chemical or physical stability liabilities associated with the historical development or the advanced exploration phase of the Project. Rubicon intends to identify and manage any liabilities associated with the production phase of the Project in order to mitigate potential impacts Financial assurance has been provided to the Government of Ontario to rehabilitate all identified features of the Project site.

A Form 1 Notice of Project Status was submitted in Q1 2011 to move the Project from advanced exploration to production status. The submission of this Notice did not give indication of commencement of commercial production. Approvals that relate to the advanced exploration phase of the project generally apply to the production phase.

Postings on the provincial Environmental Registry for various permits have not resulted in negative comments or requests for an individual environmental assessment. The Canadian Environmental Assessment Agency has confirmed that the production phase of the Project will not trigger an Environmental Assessment (EA) pursuant to the Canadian Environmental Assessment Act. A Class EA that was required, pursuant to Ontario Regulation 116/01 was completed and circulated for public comment in March 2011. No comments were received and the Class EA process was completed in April 2011.

Annual public information sessions were held in the Red Lake community in December 2008, 2009 and 2010. No negative comments have been received during the public information sessions to date. Rubicon has undertaken consultation with aboriginal communities and commissioned an independent traditional use study. This study concluded that the project site is within the traditional territory of Lac Seul First Nation and Wabauskang First Nation. Rubicon has consulted with these groups since 2007 / 2008.

An archaeological study of the McFinley Peninsula commissioned by Rubicon did not identify any sites with a high potential to host a cultural heritage value within the development footprint. Also, as the project involves the re-development of the existing footprint with only moderate expansion, the potential for impacts to cultural heritage values is considered to be negligible.

Rubicon has developed a Project Closure Plan to guide the closure of the project in accordance with industry best practices. Close-out rehabilitation activities will be completed within approximately 36 months of project closure.

Capital and Operating Costs

Capital costs for the envisaged two-year pre-production period are summarized in Table 2.

Table 2 Pre-Production Capital Summary

Pre-Production Capital ($M)	Total	Y01	Y02
Surface Infrastructure	98,000,000	32,500,000	65,500,000
U/G Infrastructure	51,462,000	25,485,000	25,977,000
Power & Utilities & Mine Supplies	4,140,000	2,070,000	2,070,000
Rubicon Project Team	8,138,000	4,069,000	4,069,000
Engineering (civil/mech/elec/geotech/mill)	2,000,000	1,000,000	1,000,000
Project Administration	1,000,000	500,000	500,000
Environmental Disbursements - sampling	180,000	90,000	90,000
Sub-Total Pre-Production Capital	164,920,000	65,714,000	99,206,000
Contingency at 30%	49,476,000	19,714,000	29,762,000
Total Pre-Production Capital	**214,396,000**	**85,428,000**	**128,968,000**

90% of the envisaged pre-production capital is shared between two areas, surface infrastructure including the mill (59%), and underground infrastructure including underground development (31%). The major component (75%) of the surface infrastructure cost is the mill, which includes construction of a paste fill plant. Excavation items account for 76% of the underground infrastructure.

Sustaining capital is estimated to total approximately $52 M over the life of the Project. The major component (75%) is related to underground infrastructure development.

An average total operating cost of $214 /t has been estimated. The estimated average operating cost per recovered Au ounce is $519. A category breakdown of operating costs is shown in Table 3.

Table 3 Operating Costs

Operating Costs ($'000)	Total	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14
Labour	398,889	25,298	27,536	36,645	36,636	36,627	36,628	36,652	36,627	36,641	33,215	31,244	25,139
Contractor	35,164	4,149	3,010	3,984	2,894	3,276	3,760	3,218	3,640	3,377	2,356	1,112	390
Material	254,057	16,279	20,534	25,151	25,248	25,417	25,383	25,095	25,417	25,194	19,086	15,750	5,503
Milling	98,991	5,940	7,700	9,900	9,900	9,900	9,900	9,900	9,900	9,900	7,590	6,270	2,191
Fill Plant/System	62,994	3,780	4,900	6,300	6,300	6,300	6,300	6,300	6,300	6,300	4,830	3,990	1,394
Reclamation	4,500	270	350	450	450	450	450	450	450	450	345	285	100
Delineation Drilling	54,588	5,000	6,481	5,000	5,000	5,000	5,000	5,000	5,000	5,000	3,833	3,167	1,107
G&A	8,203	751	973	752	751	752	751	752	751	752	576	476	166
Housing	44,996	2,700	3,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	3,450	2,850	996
Total	962,381	64,167	74,984	92,682	91,680	92,220	92,673	91,867	92,584	92,113	75,282	65,143	36,986
Cost $/t	213.88	237.7	214.2	206	203.7	204.9	205.9	204.1	205.7	204.7	218.2	228.6	371.4
Cost $/oz Au	518.6	589.0	536.9	516.5	524.4	525.4	592.6	503.1	495.7	450.8	453.7	484.6	787.7

Total mining cost averages about $150 /t, with the labour portion (including contractors) at around $95 /t reflecting the manpower-intensive nature of captive cut and fill mining. It is assumed that contractor labour will be used for all raising, including raiseboring, and that Rubicon personnel will be responsible for capital and operating development and stoping.

Economic Analysis

It should be noted that this PEA is preliminary in nature, includes Inferred 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, and there is no certainty that the preliminary economic assessment will be realized.

AMC has assessed the pre-royalty and pre-tax economics of the project using base case parameters of: gold price - $US1,100 /oz; exchange rate - $CAN1 = $US1; discount rate - 5%; gold recovery from mined ounces - 92.5%. Table 4 summarizes production, cost, revenues and economics for the base parameters.

Table 4 Production, Cost, Revenues and Economics for Base Case Parameters

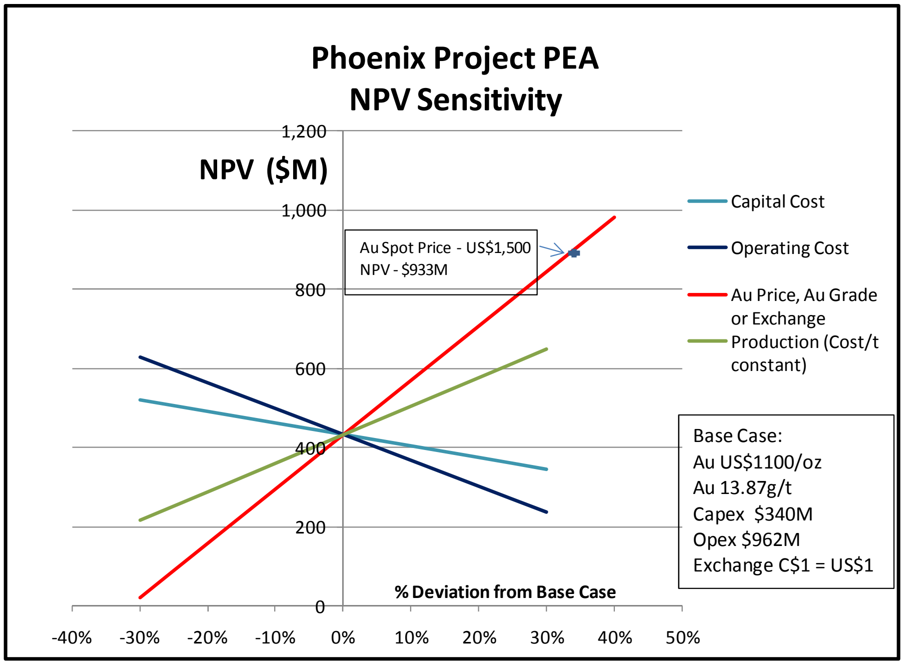
		Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14
Production Tonnes	4,499,588			270,000	350,000	450,000	450,000	450,000	450,000	450,000	450,000	450,000	345,000	285,000	99,589
Mining Rate (TPD)				750	972	1,250	1,250	1,250	1,250	1,250	1,250	1,250	958	792	277
Gold Recovery				92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%
Diluted Grade	0.45			0.44	0.43	0.43	0.42	0.42	0.38	0.44	0.45	0.49	0.52	0.51	0.51
Net Oz/year				108,935	139,650	179,444	174,824	175,516	156,396	182,605	186,769	204,323	165,921	134,437	46,955
Cumulative Oz	1,855,774			108,935	248,585	428,029	602,853	778,370	934,765	1,117,370	1,304,139	1,508,463	1,674,383	1,808,820	1,855,774
Gold Price ($/OZ)	$1,100			$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100
Exchange Rate	1.00			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
Revenue (CDN$ - 000's)	$2,041,352		0	119,828	153,615	197,388	192,307	193,068	172,035	200,865	205,446	224,756	182,513	147,880	51,650
Costs - (CDN$ - 000's)															
Pre-Production Capital	$214,415	85,443	128,971												
Sustaining Capital	$51,802			13,079	2,041	20,573	4,284	2,821	1,300	1,560	2,048	2,048	2,048		
Permanent Development	$73,538			4,865	5,835	2,620	24,203	5,047	14,259	2,422	10,440	3,019	829		
Operating /Stoping Cost	$962,381			64,167	74,984	92,682	91,680	92,220	92,673	91,867	92,584	92,113	75,282	65,143	36,986
Cash Flow Pre Royalties	$739,217	-85,443	-128,971	37,717	70,755	81,514	72,140	92,979	63,804	105,016	100,374	127,576	104,354	82,737	14,664
Cum. CF Pre Royalties		-85,443	-214,415	-176,697	-105,942	-24,428	47,711	140,690	204,494	309,511	409,885	537,461	641,816	724,553	739,217
NPV	$432,699														
IRR	28%														
Payback	5.3														
Total Cost/Tonne	$289.39														
Total Opex Cost/Tonne	$213.88														

NB. All $ Canadian other than for Gold Price

As per Figure 1, Project NPV has the following sensitivities to variation in key parameters (all other factors remaining constant):

- $NPV_{5\%}$ ranges from $345M to $520M with +/- 30% variation in capital cost
- $NPV_{5\%}$ ranges from $237M to $628M with +/- 30% variation in operating cost
- $NPV_{5\%}$ ranges from $650M to $216M with +/- 30% variation in production
- $NPV_{5\%}$ ranges from $845M to $20M with +/- 30% variation in Au price, grade or exchange rate

Figure1 NPV Sensitivity



The Project is therefore most sensitive to variations in gold price, grade and exchange rate.

Conclusions

The PEA indicates that the Phoenix Project has significant potential to become an economically viable mining operation.

The resource modelling used as the basis for the PEA employed a cut off grade of 5.0 g/t and has resulted in estimates of 1.03 Mt at 14.5 g/t of Indicated Resources (477,000 ounces Au) and 4.23 Mt at 17.0 g/t Au of Inferred Resources (2,317,000 ounces Au) for the F2 Zone lode-style mineralization.

The scenario for a potential mining operation envisages a two-year pre-production period with a 12-year Life of Mine (LOM) using a captive cut and fill method with up to six horizons being mined simultaneously. Around 450,000 tonnes would be produced annually at steady state. Average mined grade over the LOM is projected at 13.9 g/t.

Pre-production capital expenditures of $214 M have been estimated inclusive of a 30% contingency. Total sustaining capital over the LOM is projected to be $52 M. AMC notes that some aspects of the capital estimation have been done to a much greater degree of detail than may be regarded as typical for a PEA estimate.

Average operating costs of $214 /t and $519 /oz have been estimated. Mining operation costs make up over 70% of the envisaged total operating expenditure which, to a large degree, is a reflection of the labour-intensive mining method. The estimated manpower numbers also reflect provision for the degree of uncertainty that lode-style mining can present from an operations point of view.

Using the base case parameters of gold price US$1,100 /oz, exchange rate of Can$1 = US$1, discount rate of 5%, and gold processing recovery of 92.5%, the PEA shows (pre-royalty and pre-tax) an NPV of $433 M and an IRR of 28%. Using a gold price of US$1,500 /oz and with other parameters unchanged, the PEA shows an NPV of $933 M and an IRR of 48%.

As is not unusual for this style of gold deposit, the average grades estimated for the resources (and, therefore, for the mining scenario) have a significant dependency on higher grade drillhole intercepts. AMC considers that a key challenge for the Project will be to thoroughly understand the character of the mineralization and, from this, to develop the ability to readily locate and mine, with optimum dilution, such higher grade areas. AMC believes that this aspect of the Project probably presents both its greatest risk and greatest reward potential.

An enhanced degree of understanding of the zone and mineralization characteristics may also further facilitate the interpretation of exploration drilling data in terms of quantification of mining potential.

A further feature of lode-style deposits is that the generation of significant quantities of reserves may require a much greater degree of delineation drilling and, therefore, expense than for other more uniformly distributed mineralization.

Efforts made by Rubicon to develop site infrastructure have resulted in an excellent platform on which to move the Project towards a potential mining operation.

Mine design parameters have included a 45 m thick bedrock crown pillar between the uppermost workings and the sediment base in the East Bay of Red Lake. A stable geotechnical environment with little or no major faulting, structure or stress issues has been assumed. AMC believes that these are reasonable assumptions for the rock types and conditions observed in the ongoing advanced exploration operations to-date, but further geotechnical and hydrological assessment will be required in the next phase of the Project.

Relative to the location uncertainty associated with lode-style deposits, a variety of mining approaches, with possibly significant equipment capital and maintenance expense, may be necessary. This aspect can present both a risk and an opportunity.

Achievement of an average production rate of 1250 tpd will be dependent on having development sufficiently advanced and ore location sufficiently understood to provide a large number of available and viable stoping areas.

Consequent with the above aspects of lode-style mining, high operating costs can be a significant risk.

Gold recovery estimates to date are dependent on a limited number of samples. Additional testing over a much broader sample range will be required in the next project phase. Gold recovery can present both a potential risk and a potential opportunity for the Project.

There do not appear to be any permitting, community or environmental issues that would be a major constraint to the project.

The historical McFinley Mine area may provide opportunity for additional resources and early production.

Drilling in targeted areas below 305L should allow upgrading of resources that are currently in the Inferred category.

Further exploration down dip and along strike may identify additional resource potential. In the case of the area immediately above the 1464L, more drilling is warranted to provide greater support for the higher grade Inferred resources currently identified. This may allow a higher-grade mining scenario to be envisaged in those areas.

In the later stages of the conceived mining program, and subject to gaining additional geotechnical knowledge and understanding of the crown pillar area in the context of an operating mine, and a very rigorous analysis and risk assessment, it is possible that some of the lower portion of the crown pillar area could be mined.

Recommendations

AMC recommends that the Project work continue on several levels that will serve to better understand the deposit and its practical implications for conceived future mining. Several specific recommendations are made by AMC, These are outlined below, together with others that are based on Rubicon's advice that it considers it important at this stage of the project to test opportunities for increasing the tonnage of mineral resources and to continue its progress towards a position where it can readily move to a production status,

1. Engage a specialist to undertake a structural study of the deposit; this would include undertaking some oriented core drilling.
2. Improve geological understanding of the key characteristics of the deposit as they relate to potential mining by:
    a. developing drifts for local diamond drilling in at least two areas, with drives nominally parallel to, and around 25 m from, the mineralization to be drilled. The drifting would be configured to serve both for definition drilling and for possible, future mining access.
    b. from the above drifts, conducting fan drilling in a vertical plane across the mineralization on sections at nominal 10 m spacing.
    c. drifting on and across mineralization in each area in a systematic manner and relating to above fan drilling via mapping, face sampling and subsequent assaying.

3. Improve knowledge of the resource as a whole by:
    a. executing a delineation diamond drilling program in known zones focusing on Indicated Resources.
    b. executing a diamond drilling program on Inferred Resources aimed at upgrading to Indicated Resources.
    c. executing a deep diamond drilling program of the broader Phoenix area.

Each of the above programs would be carried out in two phases with commencement and execution of the second phase being dependent on results from the first.

4. Conduct further mining studies with respect to:
    a. assessing opportunities for early ore production by optimizing the schedule, including ramp from surface and possibilities in the historical McFinley area
    b. considering alternative mining methods such as mechanized cut and fill and long-hole stoping relative to improved understanding of the resource.
5. Undertake further geotechnical assessment as a greater understanding is gained of the mineralization and the impact that may have on mining geometry, extraction sequence and crown pillar size requirements. In parallel with this work, also complete relevant hydrogeological studies.
6. Undertake further metallurgical testwork on samples from different areas of mineralization to improve understanding of the range of possible gold recoveries. As part of this exercise, implement a metallurgical test program aimed at defining a fully optimized processing circuit that considers ore blends anticipated to be delivered to the mill.
7. Undertake further study on the characteristics of likely mill tailings with respect to use for paste fill.
8. Update the ventilation study to reflect projected production rate and increased depth.
9. Continue working with Aboriginal Groups, undertake further environmental studies and proceed with related procurement as required.
10. Based on Rubicon's advice that it considers it important at this stage of the project to test opportunities for increasing the tonnage of mineral resources, extend the shaft 200 m below the current elevation, develop additional accesses and drill stations, install a rockbreaker station and refuge station and undertake further exploratory diamond drilling from and around the 488 L.
11. Based on Rubicon's advice that it considers it important to continue to move towards a production ready status, purchase long lead items for the anticipated mill.

The total estimated cost of these recommendations is approximately $61 M.

AMC advises that these recommendations be implemented in a phased manner. AMC anticipates that the total program would take 12-18 months to complete, with each phase being of the order of 6-9 months in length.

CONTENTS

TABLES

FIGURES

Distribution list:

3 copies to Mr. David Adamson, Rubicon Vancouver office
2 copies to Mr. Claude Bouchard, Phoenix Gold Project site
2 copies to Mr. Matt Wunder, Phoenix Gold Project site
1 copy to AMC Vancouver office

2 INTRODUCTION

2.1 General and Terms of Reference

Rubicon Minerals Corporation ("Rubicon") controls 100% of the Phoenix gold property (the "Property" or the "Project"), which is situated in the long-established mining area of Red Lake in north-western Ontario. The Property is in close proximity to active mining and exploration operations and the area provides access to skilled mining personnel and mine supply companies.

Rubicon has requested that AMC Mining Consultants (Canada) Ltd. ("AMC") prepare, with the assistance of Soutex Inc. ("Soutex"), a Preliminary Economic Assessment ("PEA") on the Property, together with an associated NI 43-101 compliant Technical Report ("the Technical Report"). This document constitutes the Technical Report and includes the results of the PEA and the resource estimation upon which the PEA is based.

The focus of the Technical Report is the Phoenix F2 Gold System on which previous resource estimates and associated Technical Reports have been generated, the most recent of which was authored by P.T. George of GeoEx Ltd (GeoEx) and is entitled 'Technical Report, Mineral Resource and Geological Potential Estimates, F2 Gold System – Phoenix Gold Project, NTS 52N/04, Red Lake, Ontario for Rubicon Minerals Corporation' (dated April 11, 2011). The resource estimation in that report was based on drillhole information collected as of July 31, 2010. The resource estimation that AMC has prepared and used for the PEA and that, therefore, forms part of this report, incorporates previous data but also includes drillhole information gained as of February 28 2011. A somewhat different resource estimation methodology from that used previously has been employed for the current work, the details of which are given in Section 14 of this report.

AMC has not used the entire estimated Project resource as the basis for its mining and economic assessment. It has not included mineralization in the designated crown pillar area or mineralization that is below the 3907 m elevation (approximately 1464 m below surface).

2.2 Sources of Information

The authors of the Technical Report have each visited the Project site and have been provided access to what AMC understands are all technical data available for the Project, including but not limited to:

- digital files on all historical and more recent drilling
- drill core and Quality Assurance and Quality Control ("QA/QC") protocols
- all technical reports that are relevant to the Project, including metallurgical reports
- as-built files for historical and more recent underground development
- Rubicon's future aims for the project and associated key details, including cost information

2.3 Report Authors

A listing of the main authors of the Technical Report, together with the sections for which they are responsible, is given in Table 2.1 .

Table 2.1 List of Qualified Persons

Qualified Person	Position	Employer	Independent of Rubicon	Date of Site Visit	Professional Designation	Sections of Report
QUALIFIED PERSONS RESPONSIBLE FOR THIS REPORT						
Mr. H A Smith	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd	Yes	8 – 9 July 2009 and 9 Feb 2010	B.Sc., M.Sc. P.Eng.	Sections 1-3, 15, 16, 18-22, 24-27
Mr M Shannon	Principal Geologist	AMC Mining Consultants (Canada) Ltd	Yes	29 - 31 Mar 2011	B.A. (Mod)., M.A., P.Geo.	Sections 1, 14, 24-27
Ms D Nussipakynova	Senior Geologist	AMC Mining Consultants (Canada) Ltd	Yes	29 Nov -1 Dec 2010, and 11 – 12 Jan 2011	B.Sc., M.Sc., P.Geo.	Section 14
Ms C. Pitman	Senior Geologist	AMC Mining Consultants (Canada) Ltd	Yes	29 – 31 Mar 2011	M Sc., P.Geo.	Sections 4-12, 23, 27
Mr S Caron	Senior Metallurgist, Director	Soutex Inc.	Yes	16 – 18 Feb and 18 – 20 April 2011	B.Eng, M.Sc. ing,	Sections 13, 17
Mr P Roy	Senior Metallurgist Mineral Processing Specialist	Soutex Inc.	Yes	16 – 18 Feb and 18 – 20 April 2011	B.Eng, M.Sc., P.Eng.	Sections 13, 17

The authors are all Qualified Persons as that term is defined in National Instrument 43-101, *Standards of Disclosure for Mineral Projects* (“NI 43-101”) and are independent of Rubicon.

2.4 Units of Measure and Currency

Throughout the Technical Report, measurements are in metric units, although, in some instances, both metric and imperial units are used, or, if appropriate in a historic context, imperial units only may be cited. Table 1.2 includes key terms used and their abbreviations. Regional maps are in Universal Transverse Mercator (“UTM”) co-ordinates, North American Datum (“NAD”) 83, Zone 15 N.

Drill plans and sections are related to a metric mine grid with “Mine Grid North” oriented along a True azimuth of 045°.

The mine elevation datum of 5500 is equivalent to 500 metres above sea level.

Currency amounts are quoted in Canadian dollars (‘$’) unless otherwise noted.

Table 2.2 Terms and Abbreviations

Unit/Term	Abbreviation	Unit/Term	Abbreviation
Percent	%	Long-hole open stoping	LHOS
Per pound (avdp)	/lb	Metre(s)	m
Per ounce (avdp)	/oz	Square meters	m^2
Per kilowatt hour	/kW.hr	Cubic meters	m^3
Per cubic metre	/m3	Cubic meters per hour	m^3/hr
Per stope tonne	/stope t	Millimetres	mm
Per tonne kilometre	/t.km	Million ounces (avdp)	Moz

Unit/Term	Abbreviation	Unit/Term	Abbreviation
One millionth of a meter	µm	Megapascals	MPa
Above mean sea level	AMSL	Megatonnes	Mt
Bond work index	BWI	Megatonnes per annum	Mtpa
Cut and Fill	C&F	Megawatts	MW
Cyanide	CN	Modified Stability Number	N'
Diameter	dia	Project net present value	NPV
dry metric tonnes	dmt	Per annum	pa
Earnings before interest, tax, depreciation, and amortization	EBITDA	Per tonne	/t
Engineering, procurement, and contract management	EPCM	Preliminary Economic Assessment	PEA
General and administration	G&A	Pre-feasibility study	PFS
Grams /t	g/t	Acidity or basicity	pH
Grams /t of gold	g/t Au	Pyrite	Py
Hydraulic Radius	HR	Pyrite doré	Py doré
Hangingwall	HW	Rock Mass Rating	RMR
Internal rate of return	IRR	Rock quality designation	RQD
Kilogram(s)	kg	Rock work index	RWI
Kilograms per cubic metre	kg/m^3	Semi-autogenous grinding	SAG
Kilometre(s)	km	Tonne(s)	t
Square kilometres	km^2	Tonnes per cubic metre	t/m^3
Cubic kilometres per annum	km^3/a	Tonnes per day	tpd
Kilopascal	kpa	Tonnes per hour	tph
Kilotonne per annum	kt/a	Tailings management facility	TMF
Kilowatt	kW	Universal transverse mercator	UTM
Kilowatt-hours	kWh	Volt(s)	V
Litre	l	Weight for weight	w/w
Pound (avdp)	lb	Wet metric tonnes	wmt
Fire Assay	FA	Gold	Au
Atomic Absorption	AA	Inductively-Coupled Plasma	ICP
Atomic Absorption Spectrometry	AAS	Tonnes per year	tpa
Natural Heritage Information Centre	NHIC	Ounces per year	oz/y
Natural Resources and Values Information System	NRVIS	Environmental Effects Monitoring, a requirement under the federal *Metal Mining Effluent Regulations.*	EEM
Metal Mining Effluent Regulations, promulgated under the federal *Fisheries Act*	MMER	Provincial Water Quality Objectives for the protection of surface water resources	PWQO

3 RELIANCE ON OTHER EXPERTS

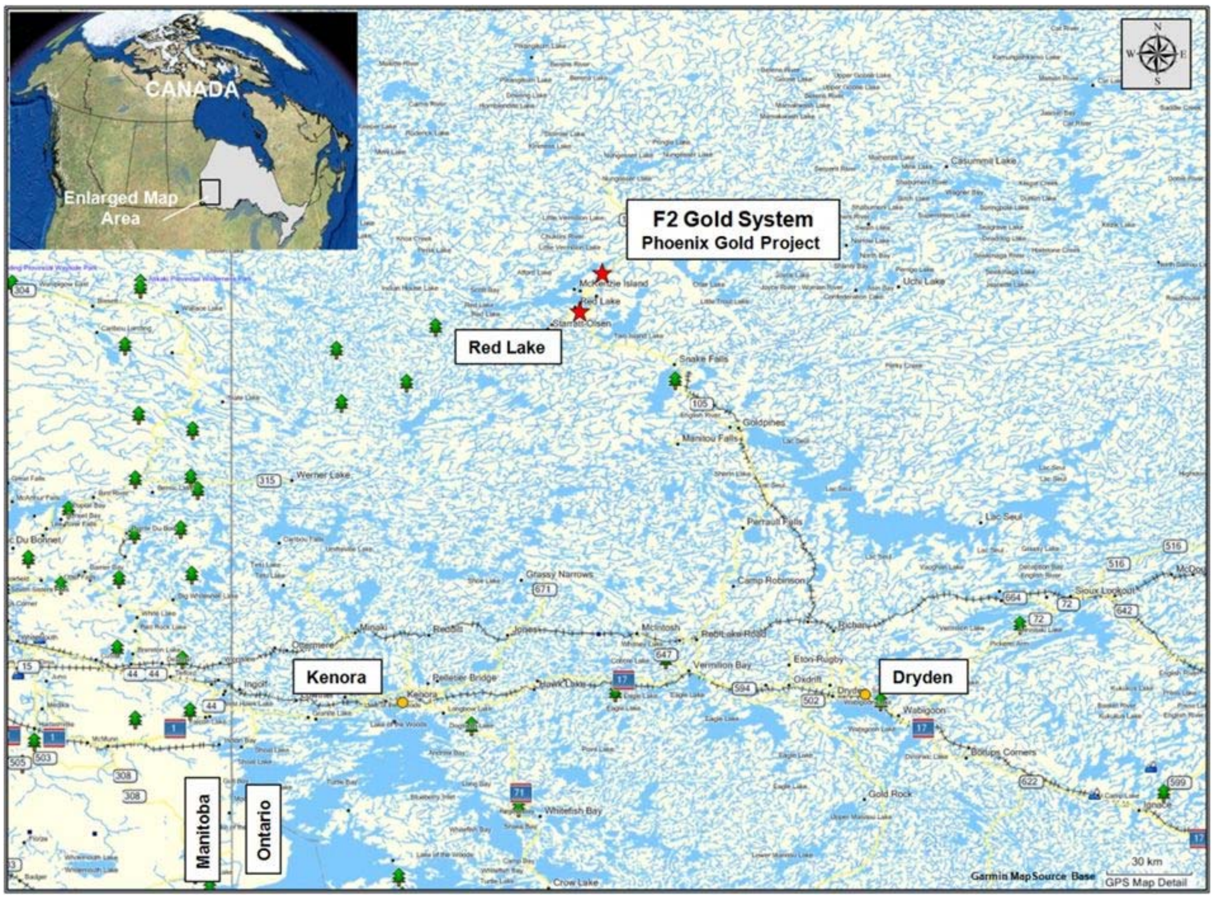
While AMC has reviewed information provided by Rubicon relating to its holding of title to the Property and on other legal, land tenure, corporate structure, permitting and environmental matters, AMC does not offer an opinion in these areas. AMC, and its individual consultants, are not experts in land, legal, permitting, environmental, and related matters and therefore it has relied (and believes there is a reasonable basis for this reliance) in this report on Rubicon, who contributed the information regarding legal, land tenure, corporate structure, permitting, environmental issues and specifics of the property description and location.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Project is located in the south western part of Bateman Township within the Red Lake Mining Division of north western Ontario, Canada. The town of Red Lake is approximately 265 km NE of Winnipeg and 150 km NW of Dryden (which is on the TransCanada Highway) and is serviced by both road and air. The historical McFinley Shaft (now called the Phoenix Shaft), on which the Project is centred, is located at UTM coordinates 448167E, 5663962N at an elevation of 369 m (above sea level).

Figure 4.1 Location of Phoenix Gold Project within Red Lake District



4.2 Mineral Tenure

The Property consists of 38 contiguous blocks covering 509.4 ha that are comprised of 16 patented mining claims (Land Portion), one unpatented staked claim, one mining lease and 25 licenses of occupation (water portion), see Figure 4.2 for all blocks. Table 4.1 lists the claims, leases and licenses by type, along with their expiration dates. A single KRL or K numbered block can consist of a patented land portion and associated water portion (license of occupation containing a separate License of Occupation number) when it covers land and water within its boundaries. A single KRL or K number can also consist of solely land or solely water. The

Mining Lease 108126 consists of four separate KRL numbered blocks, one of which is not contiguous to the other three.

Figure 4.2 Claim Map for the Phoenix Gold Project

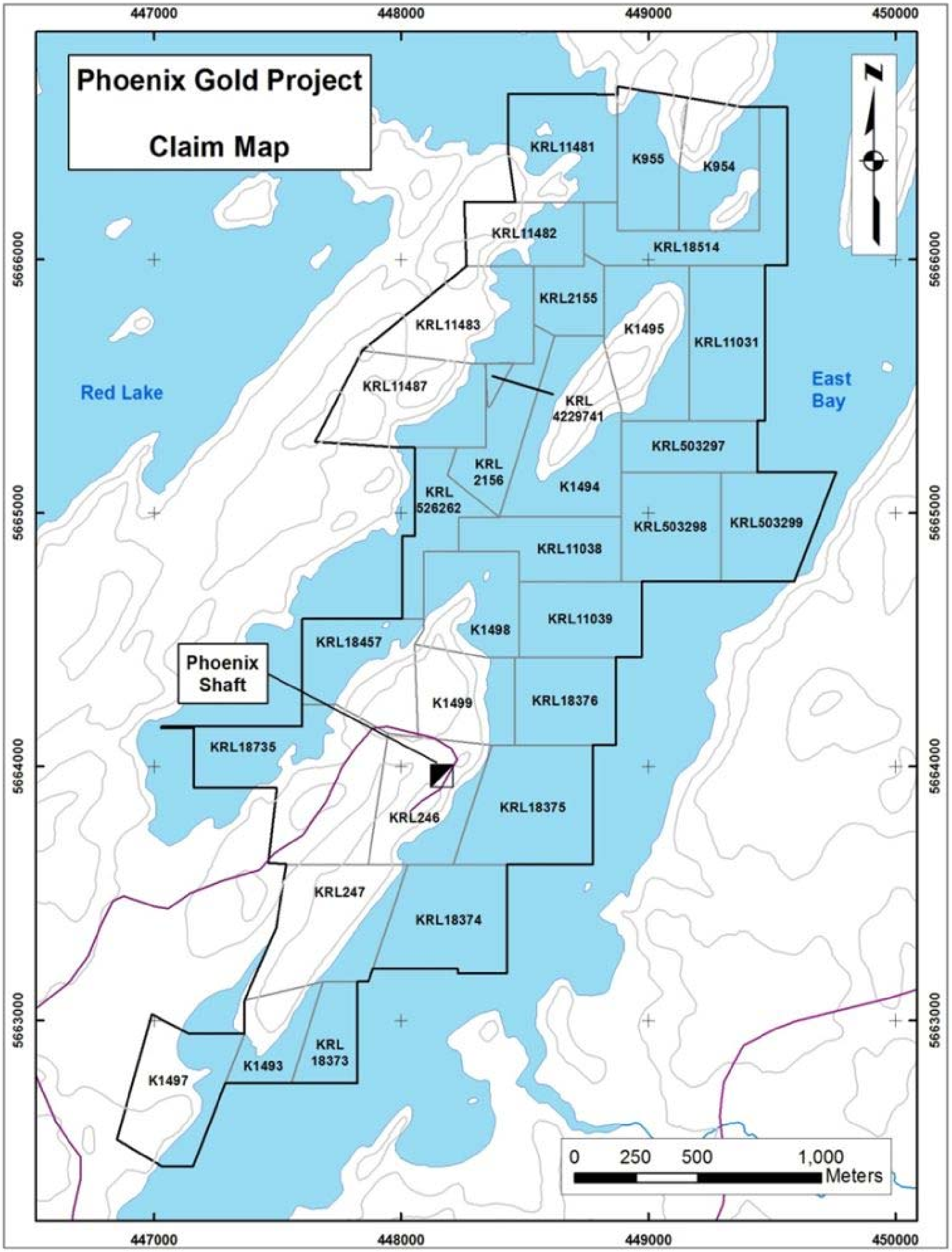


Table 4.1 Mineral Tenure for Phoenix Gold Project

License	Mining Lease	Township	Expiry Date	Hectares
KRL503297, KRL503298, 503299, and 526262	**104721 (renewed as 108126)**	**Bateman**	**2028-Oct-31**	**56.03**
Claim No.	**License of Occupation**		**Start Date**	
KRL2155	3186	Bateman	1945-Aug-01	9.9153
KRL2156	3187	Bateman	1945-Aug-01	13.678
K1498	3289	Bateman	1945-Oct-01	11.048
K1499	3290	Bateman	1945-Oct-01	2.428
K1493	3370	Bateman	1946-Mar-01	5.018
K1494	3371	Bateman	1946-Mar-01	18.737
K1495	3372	Bateman	1946-Mar-01	10.117
K1497	3380	Bateman	1946-Mar-01	6.111
KRL246	3381	Bateman	1946-Mar-01	4.33
KRL247	3382	Bateman	1946-Mar-01	4.532
KRL11038-39	10830	Bateman	1947-Jan-01	28.672
K11487	10499	Bateman	1941-Nov-01	5.738
KRL11031	10834	Bateman	1947-Jan-01	17.887
K954 (rec. as KRL18152)	10835	Bateman	1947-Jan-01	9.267
K955 (rec. as KRL18515)	10836	Bateman	1947-Jan-01	9.955
KRL18514	10952	Bateman	1947-Oct-01	17.478
KRL18735	11111	Bateman	1950-Jan-01	12.226
KRL18457	11112	Bateman	1950-Jan-01	10.967
KRL18373	11114	Bateman	1950-Jan-01	7.734
KRL18374	11115	Bateman	1950-Jan-01	19.688
KRL18375	11116	Bateman	1950-Jan-01	22.869
KRL18376	11117	Bateman	1950-Jan-01	15.018
KRL11483	10495	Bateman	1941-Nov-01	6.718
K11482	10496	Bateman	1948-Nov-01	5.637
K11481	10497	Bateman	1941-Nov-01	14.148
Patent Claim No.	**Parcel No.**			
K1498	992	Bateman	n/a	3.04
K1499	993	Bateman	n/a	11.45
K1493	994	Bateman	n/a	5.1
K1494	995	Bateman	n/a	8.38
K1495	996	Bateman	n/a	10.4
KRL246	997	Bateman	n/a	15.01
KRL247	998	Bateman	n/a	17.93
K1497	999	Bateman	n/a	13.48
KRL11481	1446	Bateman	n/a	4.24
Patent Claim No.	**Parcel No.**			
KRL11482	1447	Bateman	n/a	6.94
KRL11483	1448	Bateman	n/a	12.18
KRL11487	1452	Bateman	n/a	15.31
K954 (recorded as KRL 18152)	1977	Bateman	n/a	6.92
K955 (recorded as KRL 18515)	1978	Bateman	n/a	4.29
KRL18457	2449	Bateman	n/a	7.86
KRL18735	2450	Bateman	n/a	20.93
Staked Claim No.		Township	Recording Date	Hectares
KRL 4229741		Bateman	2009-Jun-22	1 unit
Total Area				**509.4 ha**

4.3 Property Title

The Project is subject to option agreements under which Rubicon has earned a 100% interest. The Property was acquired in two separate agreements during 2002. The water covered areas, held as 25 Licenses of Occupation and one Mining Lease, were optioned from Dominion Goldfields Corporation ("DGC") in January 2002. Land portions of the Project, held as 16 Patented Claims, were later optioned from the same vendor by agreement in June 2002. The mining rights of Patented Claims were optioned from DGC and the rights pertaining to surface claims of the same Patented Claims were optioned from DGC subsidiary 1519369 Ontario Ltd. and subsequently transferred to Rubicon or its 100% wholly owned subsidiary. Collectively, all of these titles are now referred to as the Project.

Rubicon confirms that the various Licenses of Occupation, Mining Lease and Patents have been legally surveyed and are in good standing, and that the property taxes are paid to date.

Titles to the Licenses of Occupation, the Mining Lease, staked claim and 16 Patented Claims (within which the F2 Gold System is situated) are held by Rubicon and its subsidiary and are registered with the Land Title Office, Kenora, ON and with the Ontario Ministry of Northern Development, Mines and Forestry ("MNDMF"). Surface rights covering all material parts of the Project, most of which are on the McFinley Peninsula, including those where mine buildings and tailings management facilities ("TMF") are situated, are owned by 691403 BC Ltd., a 100% owned subsidiary of Rubicon. Property taxes related to the surface parcels of some patented claims were written off by the Red Lake Municipality in early 2002 and Rubicon proceeded to purchase these surface parcels by way of public auction and all taxes are currently up to date. Rubicon has full right of access to all areas of the Project either as title holders or under contractual agreements.

4.4 Rubicon Obligations

4.4.1 Licenses of Occupation and Mining Lease

Rubicon optioned 25 licenses of occupation and one mineral lease (Water Portion) in January 2002 from DGC by agreeing to pay $800,000, issue 260,000 shares and complete US$1,300,000 of exploration prior to March 31, 2006. During 2004, Rubicon completed its acquisition of these Water Claims after meeting all the required payments and expenditures. The licences of occupation have been subsequently transferred to Rubicon.

The Water Portion claims are subject to a NSR royalty (to DGC) of 2%, for which advance royalties of US$50,000 are due annually (to a maximum of US$1,000,000 prior to commercial production) of which US$400,000 have been paid to 31 July 2011. Rubicon has the option to acquire a 0.5% NSR royalty for US$675,000 at any time. Upon a positive production decision the Company would be required to make an additional advance royalty payment of US$675,000, which would be deductible from commercial production royalties, 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, and it retains a right of first refusal on any sale of the remaining royalty interest on certain terms and conditions.

4.4.2 Patented Claims

Rubicon purchased the mining rights to 16 patented claims (Land Portion) from DGC in July 2002 for $500,000 ($425,000 paid as of December 31, 2002 and $75,000 paid prior to June 2003) and issuance of 500,000 shares (completed). The Company was also to issue to the vendor 100,000 stock options (issued). The Land Claims are subject to a sliding scale NSR royalty ranging between 2-3% subject to the price of gold, for which advance royalties of $75,000 are due annually (to a maximum of $1,500,000 prior to commercial production), of which $675,000 has been paid to 31 July 2011. Rubicon has the option to acquire a 0.5% NSR royalty for $1,000,000 at any time. Upon a positive production decision Rubicon would be required to make an additional advance royalty payment of $1,000,000, which would be deductible from commercial production royalties. Rubicon retains a right of first refusal on any sale of the remaining royalty interest on certain terms and conditions.

4.5 Environmental Liabilities

The current environmental liabilities associated with the project site are described in the *Phoenix Advanced Exploration Project Closure Plan (February 27, 2009)*, filed with the Ontario provincial government pursuant to Part VII of the Ontario *Mining Act* (“Mining Act”). This will be updated by the forthcoming production Closure Plan submission by Rubicon later in 2011 (see Section 20.3.2). AMC understands that there are no significant chemical or physical stability liabilities associated with the project site and financial assurance has been provided to the Government of Ontario by Rubicon to rehabilitate all identified features of the project site in accordance with the Mining Act.

4.6 Permits and Authorizations

Rubicon currently holds all permits required for allowing it to carry out its current drilling and underground exploration program on the Phoenix Gold Project and is in the process of acquiring additional permits required in contemplation of future production. The permits that are required for the production phase of the Project have been identified in collaboration with the federal, provincial and municipal levels of government and are either obtained or are in progress.

A full list of permits and their status, as advised by Rubicon, is given in Section 20.

4.7 Other Significant Factors or Risks

AMC is not aware of any significant factors or risks beyond those that are referenced within the PEA report.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography, Elevation and Vegetation

The Project is an area of subdued topography with less than 15 m elevation above the level of Red Lake. Land areas are largely covered with spruce, poplar and birch trees with minor swamp. 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. The Property is covered by 2 to 10 m of glacial overburden with bedrock outcrop mostly restricted to shoreline exposures. Lake depth is generally relatively shallow at less than 15 m, with the maximum depth of Red Lake being 46 m. Recent seismic surveys of lake areas indicate average accumulations of 10 to 20 m of lake sediments and overburden at the lake bottom, with troughs up to 80 to 100 m deep along the structural trend underlying East Bay.

5.2 Access

The Project is accessible via an eight-kilometre gravel road, accessed from paved roads servicing the village of Cochenour and the surrounding communities of Balmertown and Red Lake. Situated on East Bay, the Project is also easily accessible via the waters of Red Lake.

5.3 Proximity to Population Centre and Transport

The Red Lake municipal area comprises three small towns (Red Lake, Balmertown and Cochenour) and surrounding communities (Madsen and McKenzie Island) making up a population of approximately 5000. The next largest towns are Dryden (2.5 hrs by road) and Kenora (3 hrs by road), both located on the TransCanada Hwy #17 via 170 km connection to the south on Hwy 105. The area has daily scheduled bus services from Kenora and also daily scheduled flights from Winnipeg in Manitoba and Thunder Bay in Ontario. The closest railway lines are approximately 160 km south on Hwy 105.

5.4 Climate and Operating Seasons

Annual mean precipitation for the region is 640 mm, which includes mean average snowfall of 378 mm. The annual mean temperature is 0.9°C, with mean winter temperatures (October to April) of -9°C and mean summer temperatures of +14°C. Temperatures can reach summer highs of 35°C and winter lows of -40°C. Weather conditions allow surface drilling from the ice of Red Lake during January to early April, from a barge May through early October and from land year round. Municipal winter snow clearance extends to the end of paved roads near Cochenour and the site access road is easily maintained by local road contractors.

5.5 Surface Rights and Mining Operation Infrastructure

Electrical power at the Phoenix site is currently supplied by diesel generators. Approval has been received to allow the construction of a 10.4 km power transmission line to connect to the 44KV grid in the Municipality of Red Lake. This line will initially provide 5.3MVA of electricity and will connect to a 10MVA substation that has recently been set up on site. Power to underground will be supplied at 4160V via a shaft power cable and level sub-stations will be installed as required. Power underground will be available at 4160V, 550V and 120V.

Compressed air is currently supplied to underground via a surface compressor set-up that AMC understands will be adequate for the underground mining activity envisaged.

Mine water is pumped to a holding tank at the site from the nearby East Bay of Red Lake. The water is piped underground via a 100 mm water line for drilling use, muckpile watering, etc. Potable water for drinking is provided on surface and underground in 15L bottles.

Use of paste fill means that there should be no significant source of waste water in the mine other than ground water, which is currently handled by conventional sumps and pumping to a 100 mm shaft line. That will continue with additional pumping capability added as required.

A three-compartment exploration shaft was developed on the McFinley Peninsula in 1955 to a depth of 130 m (428 ft) but abandoned in 1956. New facilities including head frame, hoisting facilities, 150 tpd mill complex and camp infrastructure were developed during a later program of development and exploration during 1983 to 1988. Underground development was focused on the 150, 275 and 400-foot elevations (45 m, 84 m and 122 m levels respectively). The workings were allowed to flood in 1989 after the onset of legal disputes. The mill, hoist and head frame are intact, with the hoist and headframe being used in the current phase of the Project.

As part of the current advanced exploration phase, the shaft has been rehabilitated and deepened to a depth of approximately 335 m and extensive development has been completed on the 305 m level. This consists of two refuge stations, one permanent and four temporary pumping stations, six diamond drill stations, one rock breaker station and rock pass to shaft bottom, one second egress up to the 400 foot (122 m) level, electrical infrastructure, and two explosives storage magazines. These support the drilling platforms and the drift towards the mineralized envelope. In addition, stubs have been established by Rubicon on the 800’ and 600’ (244 m and 183 m) levels in preparation for stations for future level development, with some recent additional development achieved on 244 Level. The current advanced exploration phase has provided an opportunity to establish the infrastructure required for the envisaged initial production phase of the project. In addition, the electrical power sub-station and power line have been purchased and installed (connection to the grid pending), and a new hoist has been procured that is capable of achieving the production rate and shaft depth conceived in the PEA.

New core logging/cutting buildings, secure core storage buildings, generator building and office trailer complex have been constructed and access to the site has been restricted with a gatehouse that is staffed on a 24 hours per day basis. Infrastructure and facilities have been rehabilitated to facilitate the on-going underground and surface exploration programs.

Rubicon is currently evaluating the existing mill equipment and other existing infrastructure in preparation for the envisaged production phase of the Project. A TMF consistent with regulatory requirements was constructed on the McFinley Peninsula in 1988 in preparation for a bulk-sampling program. The site chosen was an extensive topographic depression lying immediately west of the McFinley Shaft site (now called the Phoenix Shaft site), and a retaining dam was constructed to impound tailings and effluents prior to their drainage south into the waters of East Bay. The disposal area received a Certificate of Approval in 1988. The termination of activities on that project in 1989, after test-milling of an estimated 2,500 tons of the bulk sample, has resulted in minimal use of this area. The permits on the TMF and other sewage works have been re-activated. The existing dam has also been approved.

The Project location is in an active mining district and affords access to mining service companies and skilled mining personnel.

6 HISTORY

6.1 Prior Ownership

The Property was initially staked and owned by McCallum Red Lake Mines Ltd. in 1922. Ownership was registered in the name of McFinley Red Lake Gold Mines Ltd. during the period of 1944 to 1974. In 1974, Sabina Industries Ltd. ("Sabina") earned a 60% interest in the Property. McFinley Red Lake Gold Mines Ltd. changed its name to McFinley Red Lake Mines Limited in 1975 and a plan of arrangement between McFinley Red Lake Mines Limited and Sabina in 1983 transferred title to McFinley Red Lake Mines Limited ("McFinley Red Lake Mines"). In 1984, the Project was joint ventured with Phoenix Gold Mines Ltd. (42.9%) and Coniagas Mines Ltd. (7.1%). This 50% joint venture interest was subsequently repurchased in 1986 with financial backing from Alexandra Mining Company (Bermuda) Ltd and McFinley Red Lake Mines continued underground exploration and development.

Financial difficulties experienced by McFinley Red Lake Mines in 1989 led to a long period of dispute with creditors, and ownership issues existed between 1990 and 2002. DGC was awarded title to the Licenses of Occupation and Mining Lease of the Project in 1999 and 2002 through a vesting order from the Superior Court of Ontario. DGC and a wholly-owned subsidiary, 1519369 Ontario Ltd., were subsequently granted ownership of the mining rights and surface rights respectively, to the McFinley Patents by a vesting order of the Superior Court of Ontario in 2002. Rubicon Minerals optioned the property from DGC (water title), and DGC and 1519369 Ontario Ltd. (land title), respectively, in two agreements in 2002 (see Sections 4.3 and 4.4 of this Technical Report for details).

6.2 Historic Exploration and Development

The extensive history of exploration activities on the Project have been described in detail in two previous reports prepared by G.M. Hogg in 2002 (Hogg, 2002a, 2002b). One report covered the Patented Claims, with the second document discussing historical work completed on the water titles, the 'Licenses of Occupation' 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 6.1.

Table 6.1 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 Red Lake area. Trenching, sampling and shallow drilling was undertaken by McCallum Red Lake Mines Ltd. Wide-spread but erratic gold mineralization was noted in cherty metasediments on both McFinley Peninsula and McFinley Island.
1941-42	Mineral occurrences were drilled as part of the Wartime Minerals Evaluation program.
1944-46	McFinley Red Lake Gold Mines Ltd. carried out ground magnetic surveys, a 48 hole drill program consisting of 167m (548 ft) of drilling over the McFinley Peninsula, and a 1,487m (4,877 ft) drilling program from the ice of Red Lake.
1946-55	Fourteen holes (M Series) were completed for a total of approximately 1,585m (5,200 ft) of diamond drilling.
1955-56	Little Long Lac Gold Mines sank a 130m (428ft) vertical shaft on claim KRL 246 and completed 414m (1,358 ft) of exploratory underground development on two levels. Work terminated in 1956.

Year	Description of Work
1974-75	Sabina Industries completed 25 diamond drillholes for approximately 3,048m (10,000 ft) of drilling on the Project; ground magnetic and electromagnetic surveys and ten holes in approximately 735m (2,410 ft) of diamond drilling over a portion of the lake properties.
1981-83	Sabina Industries and McFinley Red Lake Mines completed a magnetic/electromagnetic geophysical survey over the McFinley Peninsula area, surface bulk sampling and 3,672m (12,046 ft) of surface diamond drilling in 33 holes.
1983-84	McFinley Red Lake Mines Ltd. and Sabina Industries completed seven holes for a total of approximately 646m (2,120 ft) of diamond drilling.
1984-85	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 479m (1,570 ft) of drifting and crosscutting on the 150ft (46m) and 400ft (122m) levels. Metallurgical work and mineral processing were carried out. Eighty underground drillholes totalling 1,829m (6,000 ft) and sixty-nine surface holes totalling 10,628m (34,870 ft) of diamond drilling were completed. Funding difficulties resulted in the project being placed on temporary standby in February 1985.
1985-87	A total of 1,151m (3,775 ft) of drifting and crosscutting was carried out on the 150 and 400 levels. 7,111m (23,333 ft) of underground drilling, 9.14m (30 ft) of raising and an extensive chip-sampling program were completed. A program of 12,763m (41,874 ft) of diamond drilling was also completed in 61 surface holes.
1987-89	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 (2,890m/9,482 ft) continued on 150 and 400 levels, a new 275 level (at 84m) and on a ventilation raise from the 400 level to surface. Additional sampling, diamond drilling (8,730m/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 oz Au/ton (8.23 g/t) 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 over 5,791m (19,000 ft). Total historical diamond drilling focused on the Peninsula area is estimated to be 45,110m (148,000 ft) from surface and 35,814m (117,500 ft) from underground. An estimated 54,864m (180,000 ft) of core is stored on the property.
1999-2002	DGC foreclosed on the Licenses of Occupation and Mining Lease and was awarded title to the lake-covered 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.

6.3 Historical Estimates

6.3.1 McFinley Red Lake Mines

This section contains a description of historic mineral resource estimation. The historic estimates were prepared prior to the implementation of NI 43-101. The authors have neither audited these estimates nor made any attempt to classify them according to NI 43-101 standards or the Canadian Institute of Mining, Metallurgy and Petroleum Standing Committee on Reserve Definitions (CIM Standards). They are presented because they are considered relevant and of historic significance. The reader should not rely on these estimates. A qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. AMC has not treated the historical estimate as current mineral resources or mineral reserves.

A non-compliant NI 43-101 resource estimate was completed by McFinley Red Lake Mines staff in 1986 and was reported and discussed in Hogg (May, 2002, 2003). The estimate refers to the shaft area on the McFinley Peninsula where historic underground exploration and development and extensive sampling were carried out. The area is in stratigraphic units separate to the current F2 Gold System hosted stratigraphy. The 1986 'resource estimate' was developed using underground sampling results augmented with closely spaced drillhole data. Standard methods of resource block development were employed to a depth of 122m (400 feet), and an in-place grade calculated on the basis of sampling information. The 1986 'resource estimate' is presented in Table 6.2, with the metric equivalents given in the two columns to the right.

Table 6.2 Historic Inferred Resources on the Property*

Zone	Tons	Grade (oz/ton Au)	Tonnes	Grade (g/t)
FWC-3	3,875	0.50	3,515	17.1
C Zone	10,520	0.87	9,544	29.8
FWC-1 & 2	30,600	0.24	27,760	8.2
C-2	128,700	0.11	116,755	3.8
C-3	36,562	0.19	33,168	6.5
WL Zone	10,500	0.49	9,525	16.8
403 Zone	5,000	0.80	4,536	27.4
BX Zone	2,000	0.84	1,814	28.8
D Zone	106,250	0.15	96,388	5.1
Total Estimated Undiluted Resource	**334,007**	**0.20**	303,006	**6.9**

(Modifed from Hogg May 2002)

*The reader is cautioned that this historical mineral resource was prepared before the development of National Instrument 43-101 guidelines and that the figures reported in this table should not be relied upon. This historical mineral resource estimate is superseded by the mineral resource statement reported herein.

Diamond drilling below 122 m (400 ft) in 1986 led to encouraging results at depth across the Peninsula. On the basis of results from these holes, the 'resource estimate' at McFinley was re-calculated by the McFinley staff in 1986, and increased to 890,000 tons at an in-situ grade of 0.19 oz/ton gold (6.51 g/t gold). Continued drilling in 1987-1988 ultimately tested the mineralized system to a depth of about 520m (1,700 ft) below surface in the shaft area.

The deeper holes of these programs were widely spaced and the zone dimensions and continuity below the 400 level were not well established. Also the degree of confidence in indicated assay values does not allow consideration of the resource on an economic basis that meets the standards set forth in NI 43-101.

6.4 GeoEx Limited 43-101 Technical Report April 11, 2011

On March 31, 2011, the Company announced amended inferred mineral resource and geological potential estimates for the F2 Gold System. The amended estimates replaced and superseded the estimates announced by the Company on November 29, 2010 and contained in the January 11, 2011 Technical Report. The amended estimates were prepared by P T George of GeoEx Limited, the independent Qualified Person and author of the January 11, 2011 Technical Report. Following the March 31, 2011 estimates announcement, the Company announced the filing of the associated NI 43-101 compliant technical report on April 11, 2011. The amended polygonal model inferred mineral resource estimates use a 5 g/t gold cut-off and 10 grams x metre product (core length). Block model estimates were also presented in the report. The estimates relate to an area bounded by what AMC understands is a 25 m thick bedrock crown pillar and a horizontal plane 1200 m below surface.

Table 6.3 Inferred Resource for the F2 Gold System April 11, 2011

Estimation Method	Inferred Tonnes	Capping	Cut-off grade (g/t)	Average grade (g/t)	Contained ounces**
Polygonal	5,500,000	Uncapped	5	20.34	3,597,000
Polygonal	5,500,000	10-5-2 *	5	17.29	3,057,000
Block model	6,017,000	Uncapped	5	16.49	3,190,000
Block model	6,017,000	10-5-2 *	5	15.69	3,035,000

Note: *10-5-2 capping strategy refers to an empirical capping strategy that caps gold values greater than 10 oz/ton to 10 oz/ton; those between 5 and 10 oz/ton to 5 oz/ton and those between 2 and 5 oz/ton to 2 oz/ton. Values less than 2oz/ton remain uncapped.
**The stated mineral resources in the GeoEx Limited resource estimate are in situ and undiluted and figures are rounded.

Inferred resources are too speculative to have economic considerations applied to them, and there is no certainty that the inferred resources will be converted to measured and indicated mineral resources.

For more details, reference should be made to GeoEx Limited April 11, 2011 National Instrument 43-101 Technical Report (GeoEx 2011).

6.5 Past Production

There is no past production on the Property. Activities on the Property were terminated in 1989, after test-milling of an estimated 2,500 tons of bulk sample material.

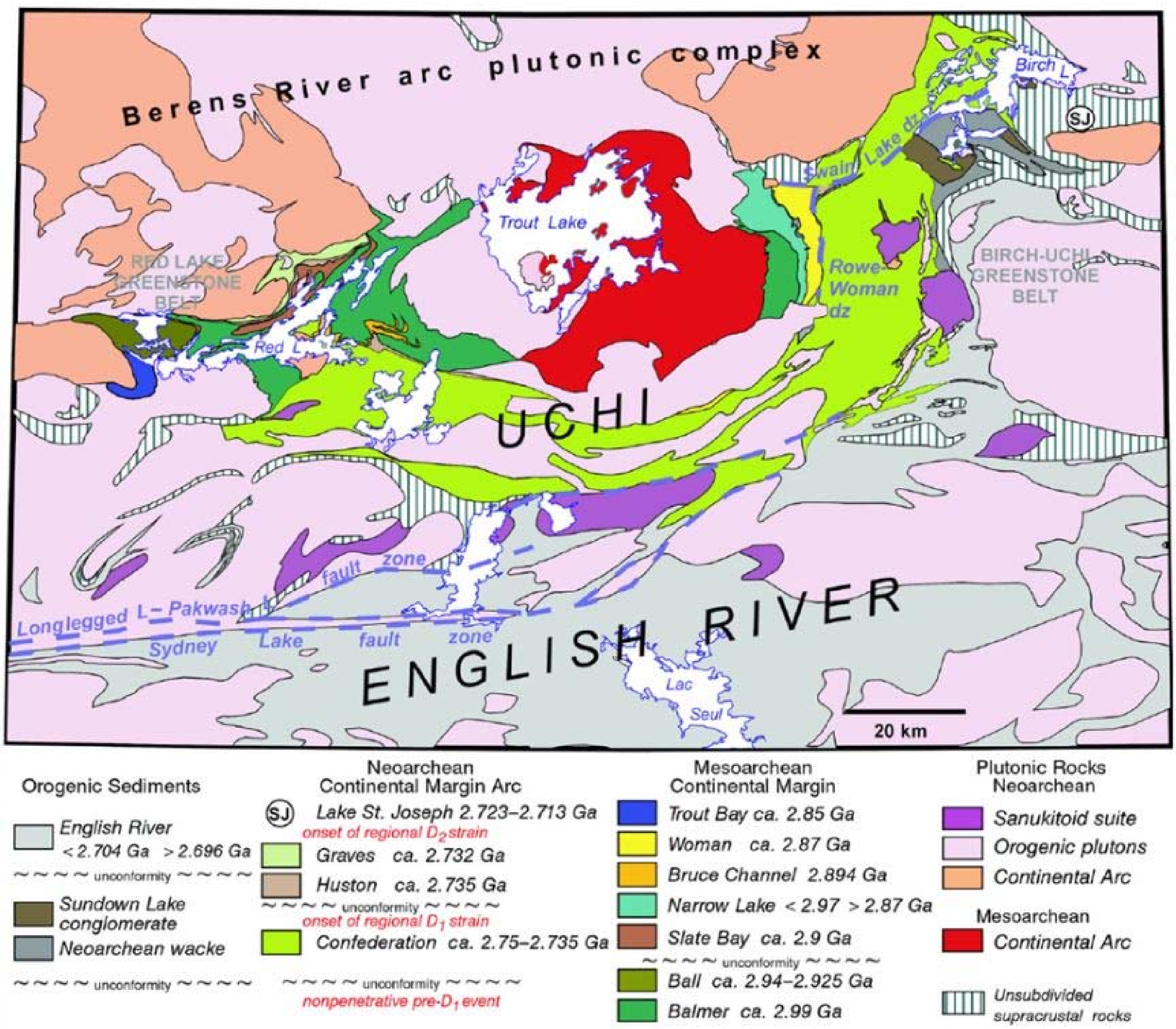
7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geology

7.1.1 Regional Geology

The Red Lake greenstone belt ("RLGB") is located in the western portion of the Uchi Subprovince of the Canadian Shield, which consists of an E-W trending sequence of volcanic and sedimentary rocks, with syn-volcanic intrusives that span a period of 300 million years. Figure 7.1 shows the regional distribution of major rock assemblages of the Uchi Subprovince (Sandborn-Barrie 2004).

Figure 7.1 Regional Geology of the Red Lake Greenstone Belt



The following review of the regional geology is derived from Sandborn-Barrie (2004).

The RLGB preserves a sequence of Achaean magmatic and sedimentary rocks that range in age from ~3.0 to 2.7 Ga (billion years). The initial period of volcanism, sedimentation, and intrusive activity, from 2,990 Ma (million years) to 2,850 Ma, is thought to have developed along a continental margin of early Achaean crust; whereas the latter periods of volcanism, sedimentation and intrusive activity developed within a subduction zone created by collision with an older fragment of Achaean continental crust (moving from a southerly direction). There are three main intrusive episodes dominated by granitic plutons dated between 2,734 Ma and 2,700 Ma.

The RLGB is subdivided into several lithological assemblages (Sanborn-Barrie et al., 2004) which include (from oldest to youngest): the Balmer Assemblage (2,990-2,980 Ma) predominantly tholeiitic and komatiitic mafic to ultramafic volcanic rocks; the Ball Assemblage (2,940-2,925 Ma) calc alkalic volcanic rocks in the northwest portion of the belt; the Slate Bay Assemblage (2,850-2,900 Ma) predominantly conglomerates, greywackes and mudstones; the Bruce Channel Assemblage (2,850 Ma) consisting of calc alkalic felsic volcanics overlain by upward fining clastic sediments, capped by chert-magnetite iron formation; the Trout Bay Assemblage (2,850 Ma) consisting of tholeiitic basalt overlain by clastic sediments and mafic to intermediate tuffs and chert-magnetite iron formation, capped by pillowed tholeiitic basalts; the Confederation Assemblage (2,748-2,742 Ma) consisting of sub aerial to shallow marine, calc alkalic intermediate to mafic volcanic rocks; the Houston Assemblage (post Confederation Assemblage) consisting of a clastic sedimentary succession; and the Graves Assemblage (2,722 Ma) consisting of calc alkalic andesite-dacite.

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 (Campbell, Red Lake Cochenour Willans, and Madsen Mines).

The RLGB is characterized by east-west trending, steeply dipping panels of volcanic and sedimentary rocks. The main stage of penetrative deformation was post 2,740 Ma. The RLGB displays evidence of several episodes of deformation, interpreted to be closely linked with extensive hydrothermal activity and gold mineralization. The first major fabric-forming event (D1) resulted in the formation of northerly trending, south plunging F1 folds and associated S1 and L1 fabrics. Superimposed on D1 structures are east to northeast trending D2 structures in the western and central Red Lake area. Northeast trending F2 folds plunge moderately to steeply to the northeast, and southeast trending folds plunge moderately (45^{o} to 65^{o}) to the southeast. A progressive change in orientation of the S2 structures occurs across the central Red Lake area, with no evidence of an overprinting relationship between the northeast striking S2 and the southeast striking ‘mine trend’ fabrics, suggesting that these fabrics formed coevally during D2. The 2,718 Ma Dome stock and supracrustal rocks adjacent to the Dome stock contain S2 fabrics, indicating that the D2 deformation probably occurred during the intrusion of the Dome stock. The onset of the penetrative D2 strain across the RLGB is interpreted to record the collisional phase of the Uchian orogeny. Post collisional (D3) strain, locally recorded in the RLGB after 2,700 Ma, displays a penetrative tectonic foliation, coplanar to the D2 fabrics throughout the central Red Lake area.

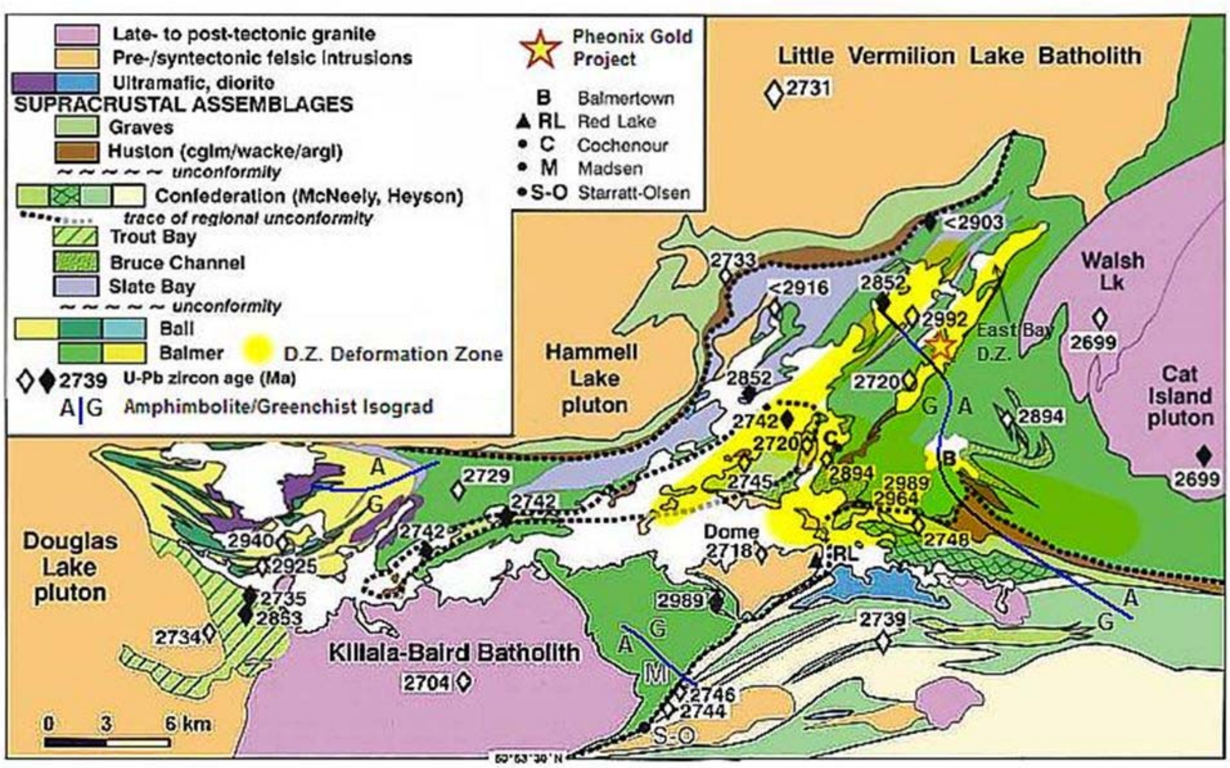
Hydrothermal alteration in the Red Lake greenstone belt is distributed in regional, zoned, alteration envelopes that show a spatial relationship to gold deposits. Distal alteration comprises calcite carbonatization and weak potassic alteration. More proximal to the gold deposits, ferroan-dolomite and potassic alteration develops. Silicification with associated gold and sulphide mineralization overprints the proximal alteration, forming extension and fault-fill quartz veins and breccias.

7.1.2 Local Geology

The local area 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; 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. The upper Balmer sequence is comprised of tholeiitic mafic volcanic rocks.

A strong north-northeast trending (UTM coordinates) structural fabric through the area is considered part of the East Bay Deformation Zone (“EBDZ”), which dominates the geology of the Project. 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. The Red Lake geology is shown in Figure 7.2, which is after Sandborn-Barrie et al, 2004. In addition, the amphibolite/greenschist isograd, which developed in association with the pluton emplacement and is commonly associated with mineralization within the Red Lake district, is shown.

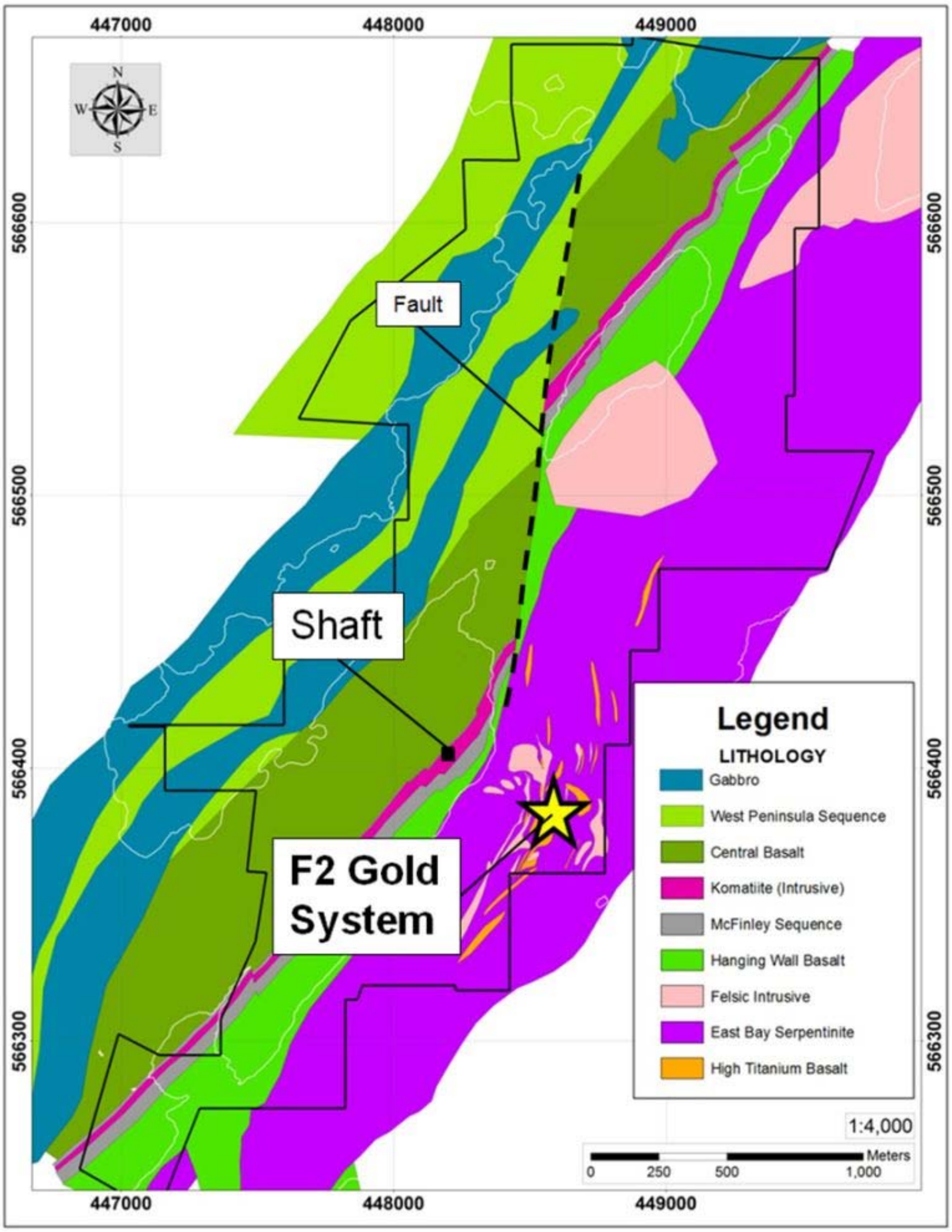
Figure 7.2 Red Lake Geology



7.1.3 Property Geology

The F2 Gold System lies within the 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 which can be correlated along the length of the property for over four kilometres (Figure 7.3).

Figure 7.3 Lithological Map of Property



A summary of the stratigraphic units found within the Project area is shown in Table 7.1.

Table 7.1 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 (syngenetic?) py/po
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 'HW 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 'HW BIF' and to the east by the FW BIF At least 5 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	Variable biotite alteration, sulphides (py, po) Silica flooding, quartz breccia and quartz veining throughout Located within the package of Basalt/Basaltic Komatiite on Figure 7.1 The High Titanium Basalt is the main host to F2 Gold System

At the Project, the EBDZ is manifested by a well-developed, northeast striking penetrative foliation (F1) 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 east, in the area of the F2 Gold System which occupies the core of the EBDZ, dips are sub vertical to steep northwest.

7.2 Mineralization

Gold deposits in the Red Lake district have been classified into three main categories: mafic volcanic-hosted; felsic intrusive-hosted; and stratabound. 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.

In the F2 Gold System, mineralization is generally conformable with the lithological boundaries and is characterized by vein and sulphide replacement mineralization. The High Titanium (HiTi) basalts are fine grained and, where fresh examples exist, comprise amphibole +/- plagioclase. The Felsic intrusives, 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 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 1500 m and horizontal distances of approximately 1200 m. Mineralized zones are associated with the contacts of these major rock types, and have been identified over vertical distances of greater than 300 m and horizontal distances of greater than 150 m.

The main zones identified to date within the F2 Gold System generally display a northeast strike (UTM Grid), although there appears to be a distinction between zones identified to the west and to the east of the Property. To the east the dip on the mineralization is steep to vertical and to the west it dips at about -55^0 to -65^0. The overall plunge of the mineralization is $-65^{0\ to}$ -75^0 degrees to the south-southwest in all areas. See Figure 7.4.

Figure 7.4 Mineralization Trend using Composite Data

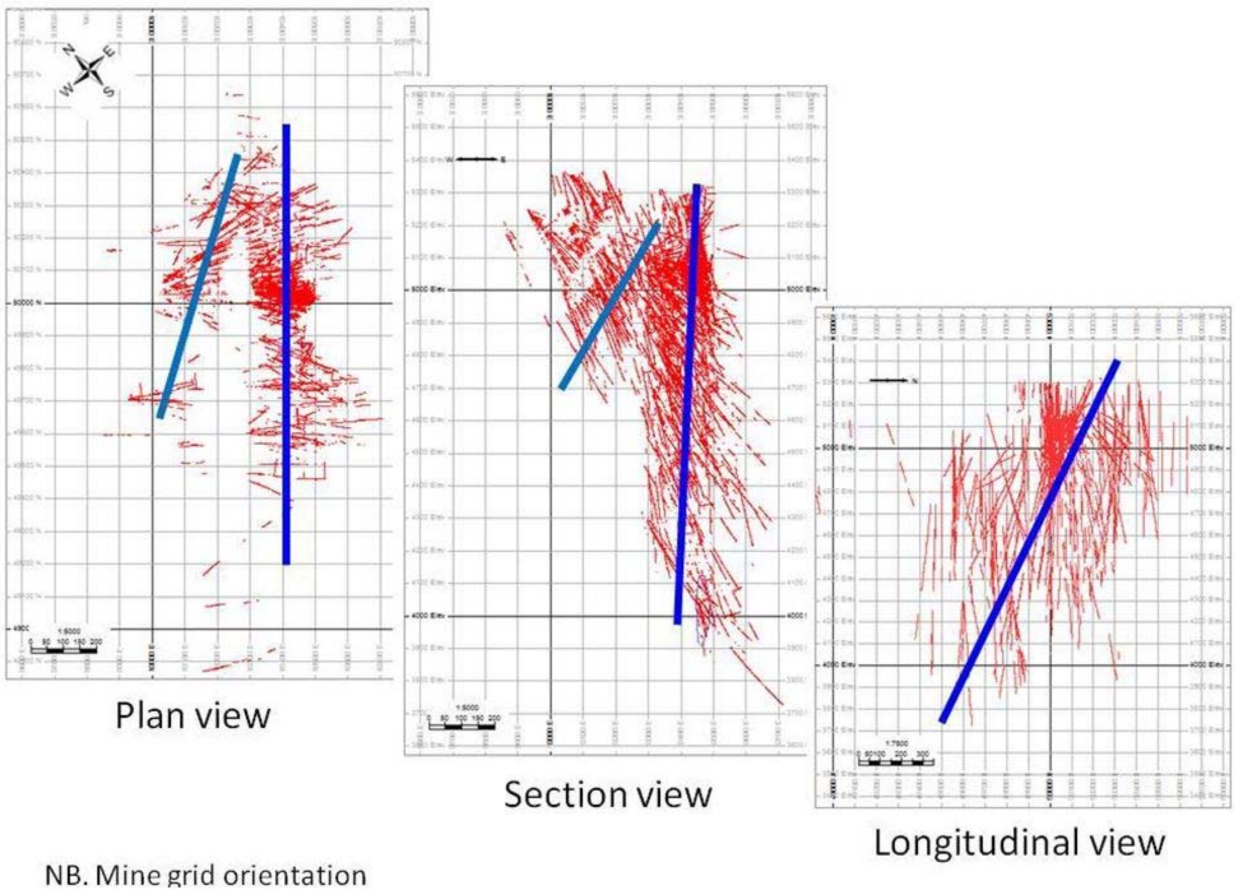
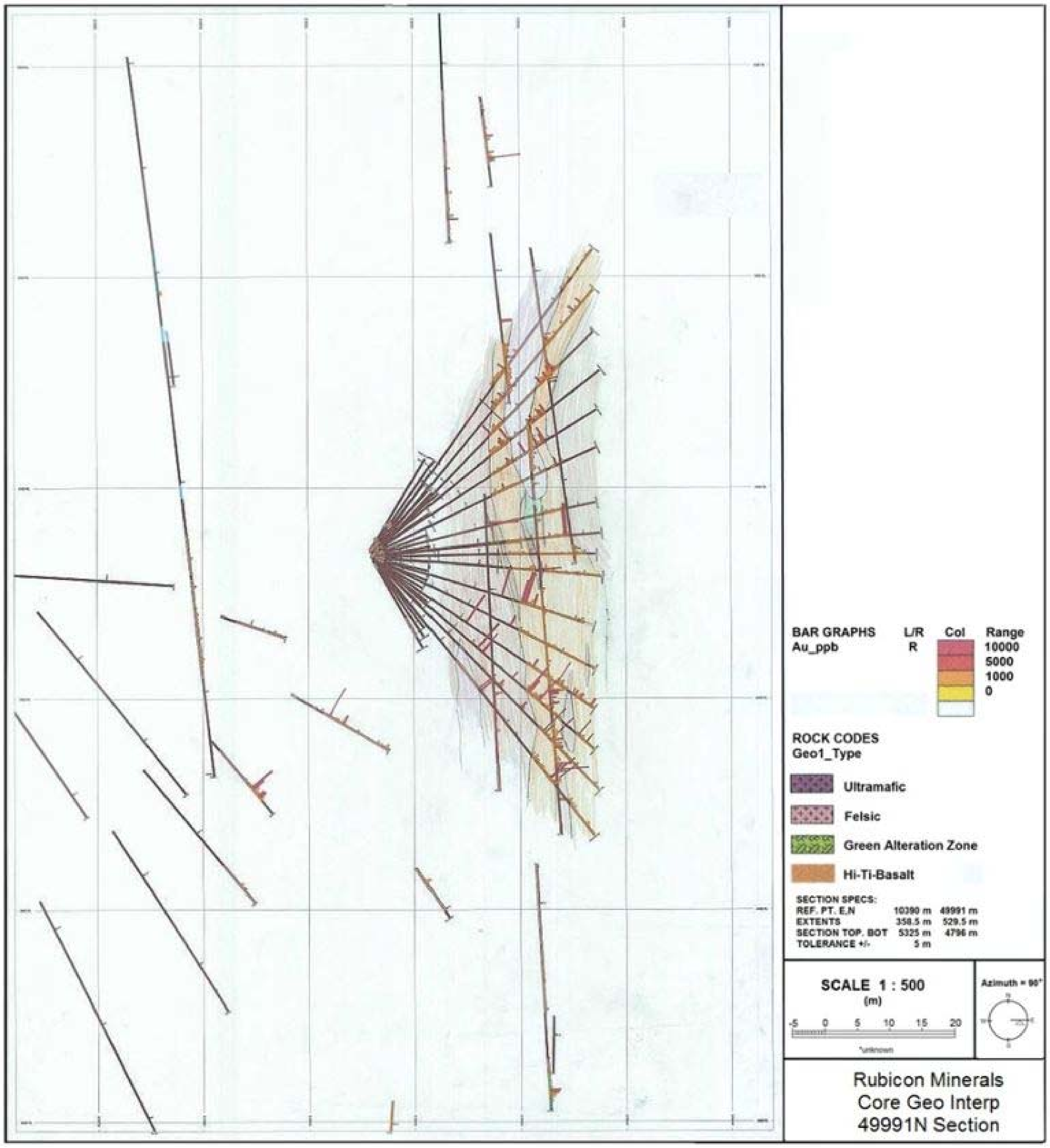


Figure 7.5 is a representative section through the well drilled section of what has been termed Domain 1, showing the geological interpretation by Rubicon of the F2 Core Zone mineralization.

Figure 7.5 Section 49991N through F2 Core



7.2.1 Main Mineralized Zones

7.2.1.1 F2 Core Zone

The F2 Core Zone represents the initial discovery zone within the F2 mineralized system. This gold zone extends to a vertical depth of greater than 500 m below surface and consists of sub-parallel lenses identified by intense biotite-amphibole-silica (+/- pyrrhotite-pyrite) alteration. Strike lengths and widths of individual zones are variable but can attain strike lengths greater than 100 m and can attain horizontal thicknesses greater than 10 m.

Gold mineralization in the F2 Gold System itself is characterized by vein and sulphide replacement mineralization which is preferentially hosted along the boundaries of two main rock types: titanium rich basalts (high iron tholeiites) and felsic intrusive rocks (bounding units); with additional mineralization associated with cross cutting structures. Gold, however, is distributed through all of the adjacent rock types, with the majority contained within the titanium rich basalt.

The broad, lower grade gold zones have grades generally between 5.0 to 10.0 g/t gold. An example of a higher grade intersection when underground drilling through the F2 Core Zone is that of 42.5 g/t Au over 6.9 m in hole 305-05 as part of a broad zone grading 20.1 g/t Au over 15.0 m. Also, hole 305-11, drilled approximately 21 m above 305-05, intersected 34.7 g/t Au over 6.7 m as part of a wider zone grading 20.1 g/t Au over 16.2 m. Both holes are sub-horizontal.

7.2.1.2 Deep Central Area

Vertically below the F2 Core Zone lies the Deep Central Area. The style of mineralization is similar to that encountered in the F2 Core Zone. Mineralization in the Deep Central Area demonstrates a vertical continuity of at least 200 m and a horizontal continuity of greater than 160m.

7.2.1.3 Southern Area

Drilling results have continued to confirm the nature of the F2 Gold System and its depth in the Southern Area. Higher grade intercepts include 754.2 g/t Au over 0.5 m at a vertical depth of 1320 m below surface in drill hole F2-100 A and 142.6 g/t Au over 0.5 m as part of the broader zone grading 9.2 g/t Au over 9.6 m at a vertical depth of 1086 metres below surface in drill hole F2-100A-W1. Mineralization in the Southern Area demonstrates a vertical continuity of at least 300 m and a horizontal continuity of over 200 m. The style of mineralization continues to be similar to that encountered in the F2 Core Zone.

7.2.1.4 Western Limb Area

This area lies between the shaft and the F2 Core zone. It typically consists of multiple orientations of mineralization, including higher grade gold mineralization associated with narrow quartz veins and occurring near the contact of felsic dykes. Mineralization in the West Limb Area demonstrates a vertical continuity of at least 500 m and a horizontal continuity of greater than 200 m.

For more information about the remainder of the individual zones, reference can be made to the April 11, 2011 Technical Report (GeoEx 2011), where individual intercepts are tabulated.

8 DEPOSIT TYPES

The Project lies within the Red Lake District and exhibits a mineralization style which can be broadly classified as an Archean greenstone belt hosted gold deposit. Primarily, the host rocks are highly-altered, supracrustal rocks; most commonly tholeiitic basalts, komatiites or their volcaniclastic or subvolcanic equivalents. Mineralization also occurs in felsic volcanic rocks, porphyries, greywackes and conglomerates. These assemblages have been regionally metamorphosed with an apparent relationship between the amphibolite-greenschist isograd and the occurrence of mineralization.

The majority of the large gold deposits of the Red Lake District are hosted by the Mesoarchean Balmer assemblage units which lie proximal to the regional angular unconformity with overlying Huston and Confederation assemblage rocks (RLGM, Madsen). Intra-belt felsic plutons and quartz porphyry dykes are also important hosts for gold mineralization, and accounted for production at the McKenzie, Gold Eagle, Gold Shore, Howey and Hasaga mines.

8.1 Red Lake Style Deposits

The gold deposits of the RLGB have been classified into three groups (Pirie 1981), according to the stratigraphic or lithologic associations described below.

8.1.1 Group 1 Deposits (Mafic Volcanic Hosted)

These occur within zones of alteration several square kilometres in extent, typified by CO_2 addition (forming Fe-carbonates) and Na_2O, CaO, and MgO depletion (Pirie 1981; Andrews et al., 1986). On a more local scale, SiO_2 and K_2O addition creates alteration assemblages consisting of quartz, biotite, fuchsite (chrome-rich muscovite), and sericite. These are commonly associated with elevated arsenic and antimony, and gold mineralization occurs in quartz-carbonate veins, quartz veins, sulphide lenses, stringers and disseminations, and in impregnations in vein wall rock. Most of the high-grade mineralization comes from quartz ± arsenopyrite replacement of early (barren), banded carbonate veins (Horwood, 1945; Dube et al. 2001 and 2002), which typically are very small targets in plan, but are relatively continuous down plunge. The High Grade Zone at the Red Lake Gold Mine, for example, occurs as several discrete ore bodies a few meters wide by a few tens of meters long that all occur within a small area (100 m x 150 m), but are known to have a vertical extent of at least 1,400 m (B. W. Dubé 2001). Tholeiitic basalt, basaltic-komatiite, and iron-formation are the dominant host rocks.

A spatial relationship exists between the ultramafic rocks and gold mineralization, with the majority of gold mineralization at Cochenour-Willans and RLGM occurring within a few hundred metres of ultramafic bodies. Dube and others (B. W. Dubé 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.

8.1.2 Group 2 Deposits (Felsic Intrusive Hosted)

The majority of Group 2 deposits occur as shallow to steeply dipping, sulphide-poor, quartz veins and lenses hosted in sheared diorite and granodiorite of the Dome and McKenzie stocks, and as quartz vein stockwork in quartz porphyry dykes and small felsic plugs. The largest of this type of deposit, the McKenzie mine, produced over 650,000 ounces of gold (Andrews et al., 1986).

8.1.3 Group 3 Deposits (Stratabound)

Group 3 deposits are only known to occur in the southern part of the RLGB and include the ore zones at the Madsen and Starratt-Olsen mines. Ore is of disseminated replacement style, 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 (B. Dubé 2000).

8.2 Deposit Types at the Phoenix Project

The Project hosts Group 1 and Group 2 type gold deposits described above.

The interpretation for the F2 Property being a Group 1 type deposit is supported by the property location along the East Bay Deformation Zone, within favourable Balmer Assemblage mafic and ultramafic rocks. Also of note is the Property's close proximity to the amphibolite-greenschist isograd. Initial mapping and sampling has demonstrated the existence of most of the associations within this group.

To the west of the F2 Property mineralization is hosted mainly at the contacts of the felsic intrusives and is contained primarily within quartz veins.

9 EXPLORATION

Rubicon has conducted an extensive exploration program on the Project since acquiring the Property in 2002. Work has included geological mapping, re-logging of selected historic drillholes, digital compilation of all historical data available, a high resolution airborne magnetic survey, a ground magnetic survey, a seismic lake bottom topographic survey, a Titan 24 geophysical survey and numerous drilling programs, all of which have been covered in earlier reports.

For details of the various earlier exploration programs please refer to the April 11, 2011 Technical Report (GeoEx 2011).

Of the approximately 81,000 m (265,000 ft) of core drilled by McFinley, approximately 23,000 m from 161 of the holes have been re-logged for the purpose of improving geological knowledge. Certain holes were re-assayed at that time but these data were not included in any resource estimates as the data is historic in nature and some elements of it could not be completely validated.

The exploration focus for 2010 and 2011 has been drilling in order to both extend and upgrade the resource, the results of which are discussed in Section 10.

10 DRILLING

10.1 Drilling Summary

Since 2002 and up to 28 February 2011, Rubicon has completed 313,030 metres of diamond drilling (182,802 metres of surface drilling and 130,228 metres of underground drilling) to February 28, 2011 on the Property. During this period, 239,000 metres were drilled on the F2 Gold System; see Table 14.2 for details. Drill program metreages are shown below in Table 10.1.

Table 10.1 Drill Programs

	Surface Holes		Underground Holes		Total
Year	**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			119	22,648	22,648
Total	**403**	**182,802**	**360**	**130,228**	**313,030**

Details of the drill programs by year, up until July 2010, can be found in the April 11, 2011 Technical Report (GeoEx 2011). Figure 10.1 illustrates how the targets within the Property have been prioritized. Sections 10.3 and 10.4 discuss the more recent drill programs.

The Property has been evaluated within the context of current knowledge of ore control systems and models of the producing mines in the Red Lake region. The majority of diamond drilling by Rubicon has targeted areas outside the confines of the historic mine site in environments perceived to have higher exploration potential, but which have limited historic work. Surface drilling has continued to infill down to a 50 m grid. Additionally, underground fans have been drilled approximately 35 m apart horizontally and with holes collared in 10^{o} vertical increments on section and crossing the F2 core mineralization. Figures 7.5, 10.2 and 10.3 illustrate the general angle of the drillholes and their relative orientation to the mineralized domains.

Figure 10.1 Key Target Areas on the Phoenix Gold Project

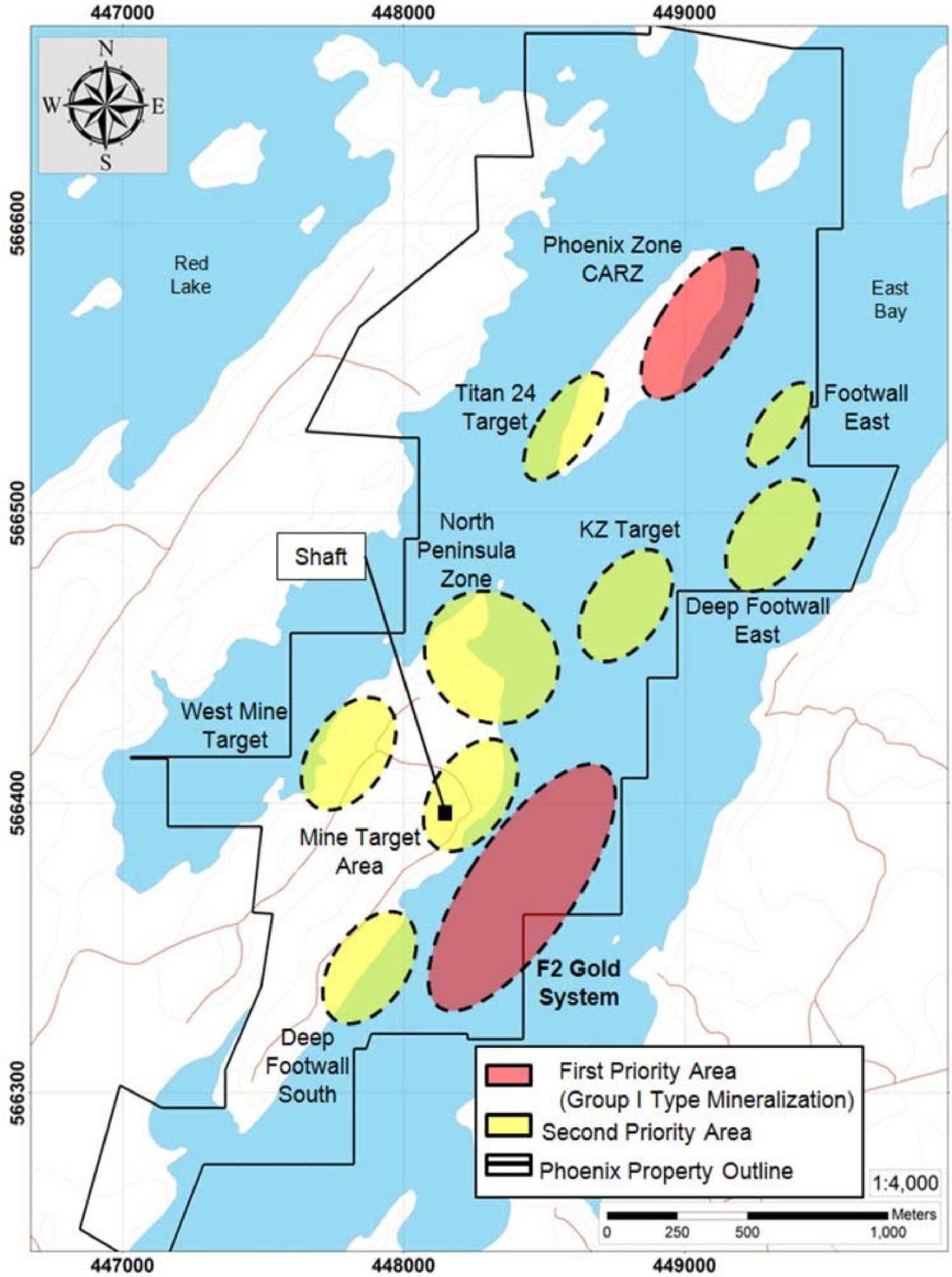


Figure 10.2 Relationship of Drillhole Orientation to F2 Zone Mineralization

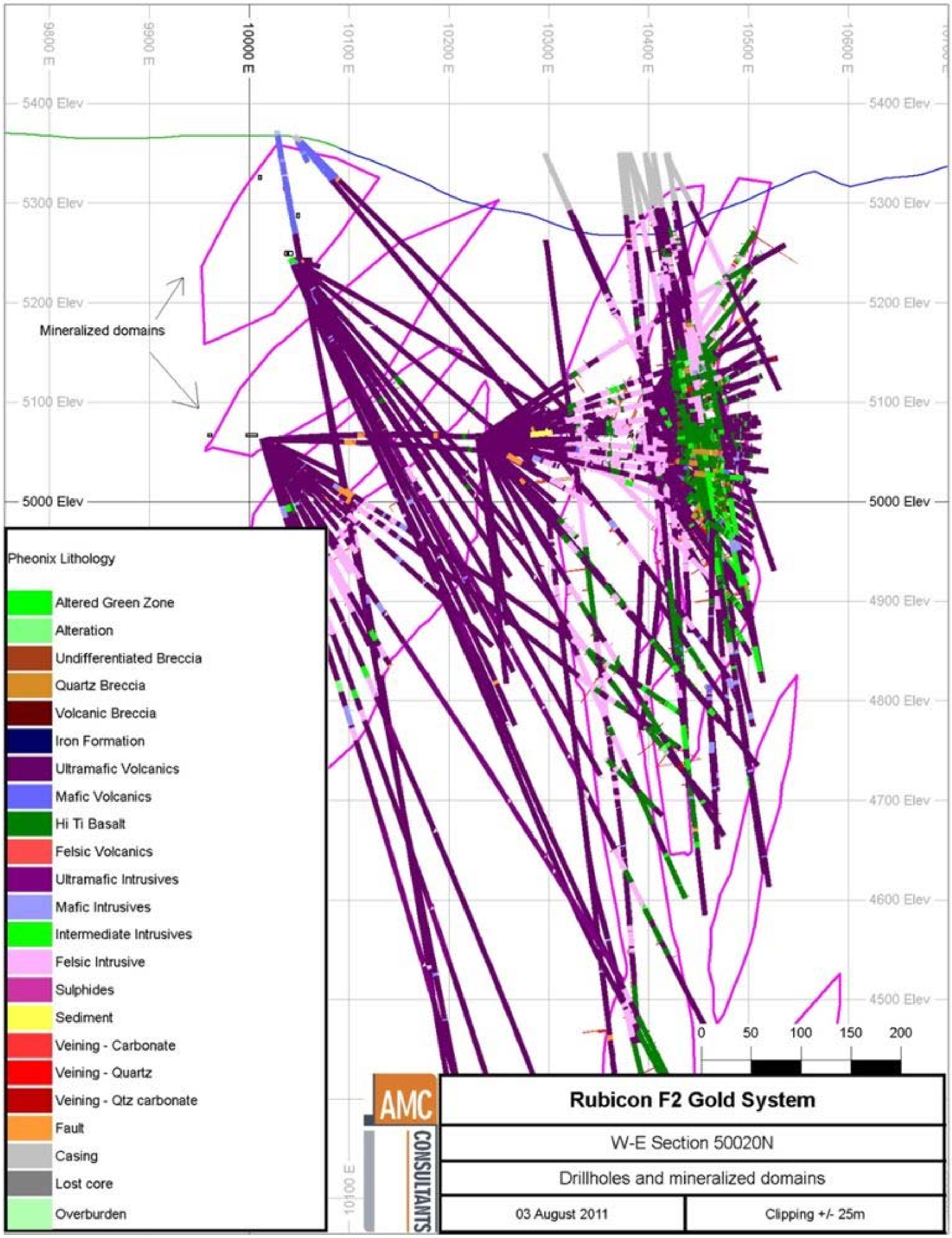
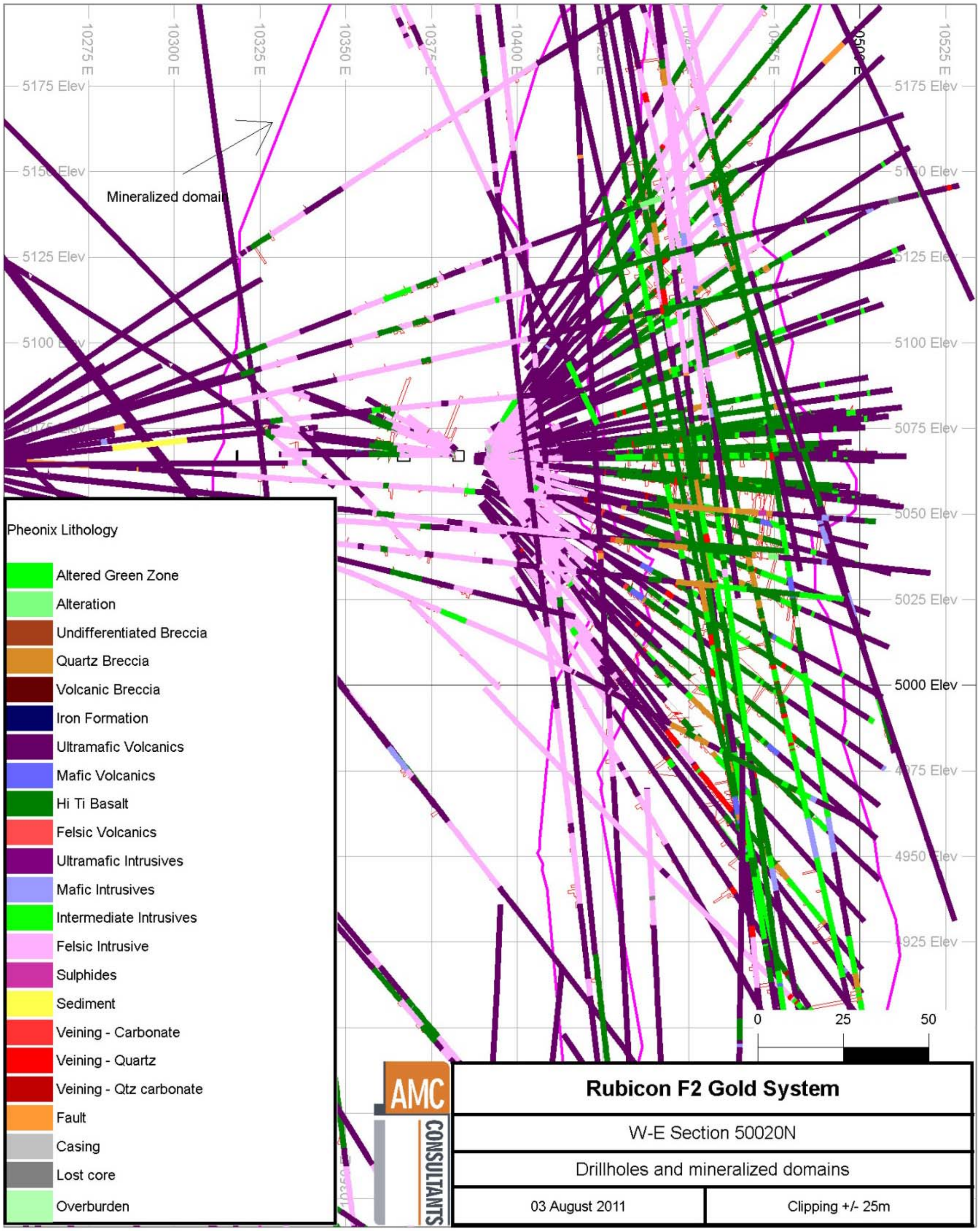


Figure 10.3 Detail of F2 Core Zone



10.2 Drilling Procedures

All proposed land and ice drill collars were surveyed with a hand held Global Positioning Survey (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 technician). Collars for barge holes were also surveyed with a hand held GPS and then marked with a buoy; the same foresight procedure was carried out. Changes in actual drill 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.

Casing for holes collared on land were left in place, plugged and cemented and covered with aluminum caps with the drillhole number etched or stamped into the cap. Holes that were drilled from the ice or barge were plugged with a Van Ruth plug at 30 m (100 ft) down hole from the base of the casing, and then cemented to the top of hole. All casing was removed from these holes.

Generally NQ2 (50 mm diameter) or NQ (46 mm diameter) core is drilled. Core is laid in wooden core boxes at the drill site, with depth markers every 3 m, sealed with a lid and strapped with plastic bindings. Boxes are delivered once a day by the drill contractor or Rubicon personnel to the on-site core logging facility. A Reflex or Ranger electronic single shot survey instrument is used to take down-hole surveys recording azimuth, inclination, magnetic tool face angle, gravity roll angle, magnetic field strength and temperature at 60 m intervals.

Core recovery during these programs has generally been excellent, and is usually in excess of 98 %. RQD measurements are completed on the core, as well as representative specific gravity and magnetic susceptibility readings as part of the procedure.

The majority of diamond 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 2500 m and from underground having a depth capacity of 1500 m. Layne Christensen Canada Limited of Sudbury, Ontario was also contracted to complete deep holes using their skid-mounted CS 4002 having 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 having a depth capacity of 1500 m. Each drill program was supervised by a Rubicon drill geologist.

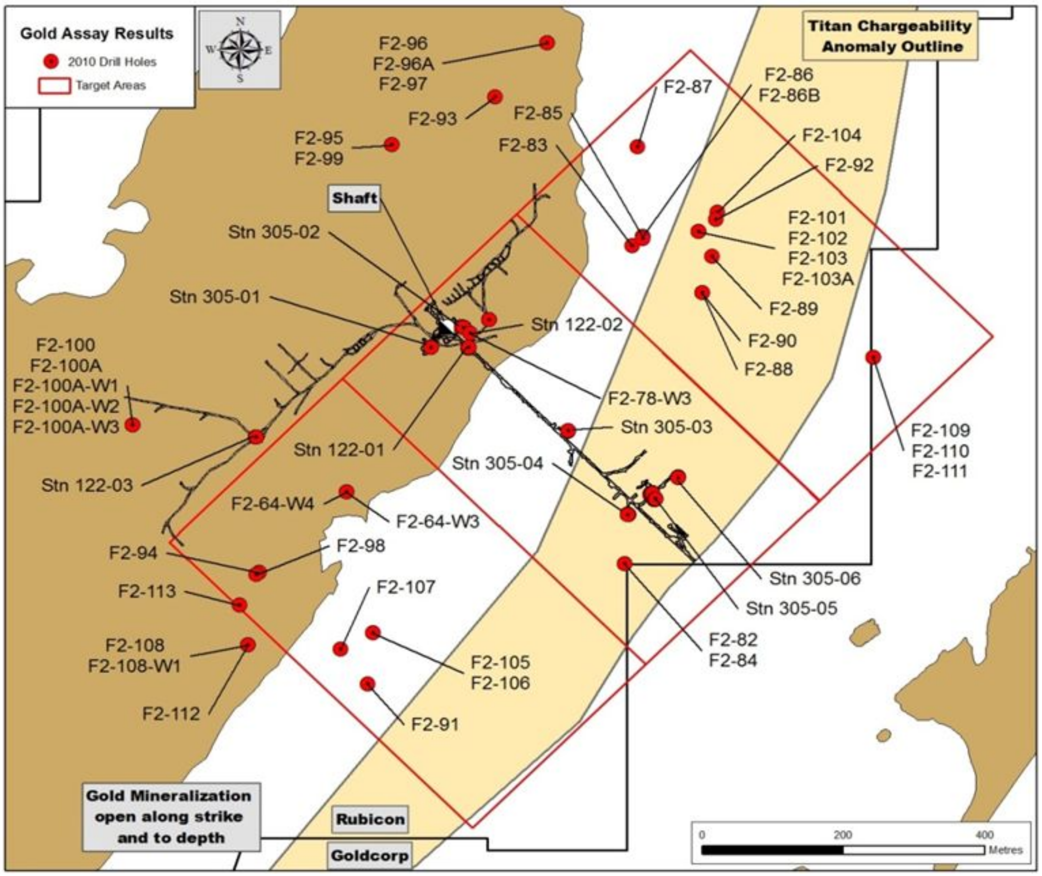
During its site visits, AMC interviewed project personnel regarding core handling, logging, sampling and storage. AMC also inspected the core storage warehouse and visited the assay sample storage depots. The field procedures are well established and understood by all field personnel. Care is taken to ensure that these procedures are closely followed. In the opinion of AMC the field procedures are of a high standard and generally meet industry best practices.

10.3 2010 Diamond Drilling Program (F2 Gold System)

Detailed information with regards to the drilling undertaken between January and July 2010 can be found in the April 11, 2011 Technical Report (GeoEx 2011). The following summary discussion combines all the 2010 information.

Diamond drilling in 2010 continued to expand the F2 Gold System and, by year-end, Rubicon had completed 119,891 m in 248 holes. A total of 37,823 m was completed from surface, while 82,068 m was completed from underground (122 m and 305 m levels). Figure 10.4 shows the location of the surface holes and the position of the Level 122 and Level 305 drill stations.

Figure 10.4 2010 Drillhole Locations



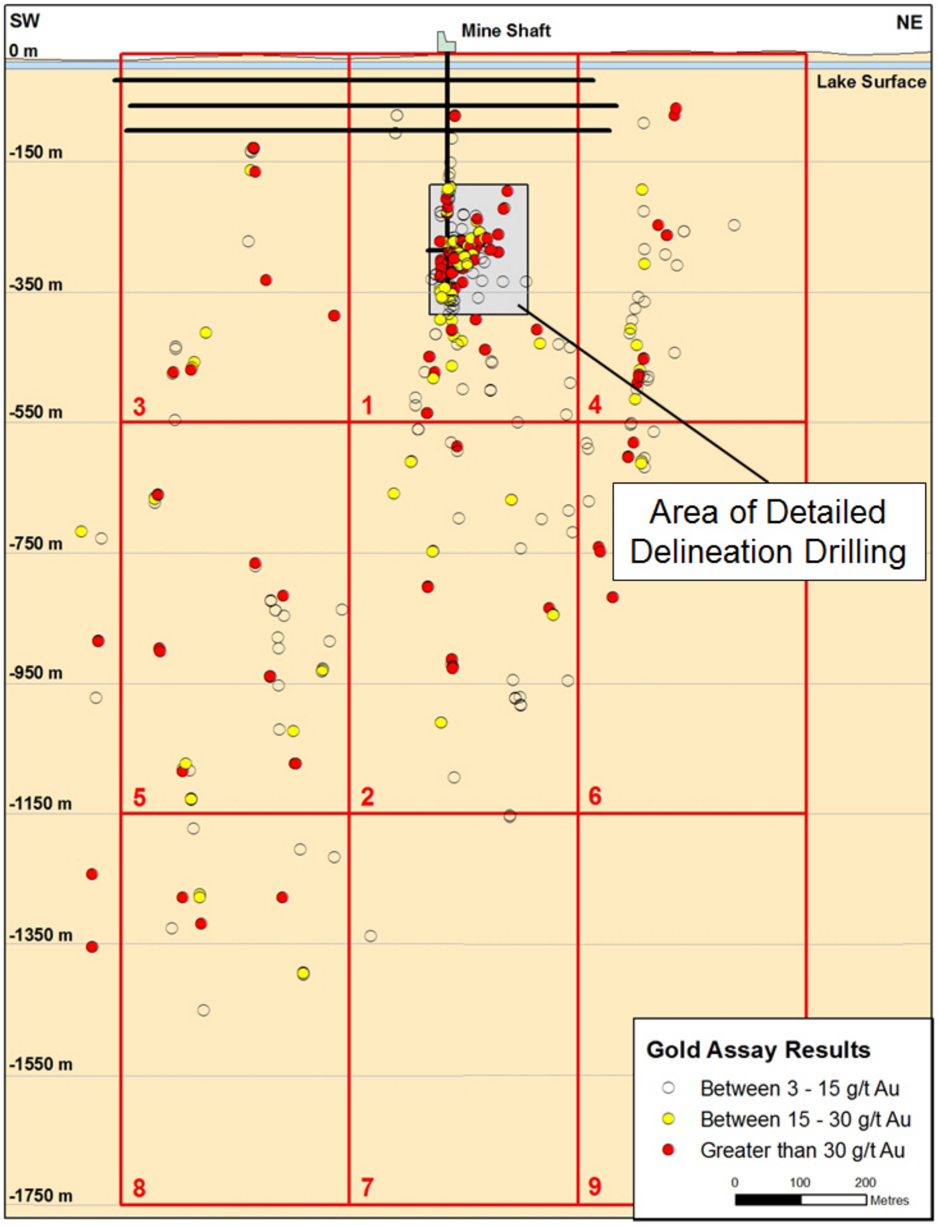
Notes. '122' prefix holes drilled from underground on the 122 m level
'305' prefix holes drilled from underground on the 305m level
'D305' prefix holes drilled from underground on the 305m level
'F2' and 'HW' prefix holes are drilled from surface

In the fourth quarter of 2010 Rubicon initiated its planned 27,000 delineation program, designed to test a 150 metre (horizontal) x 200 metre (vertical) area within the F2 Core Zone (Figure 10.4). By year-end the delineation program began to identify discrete sub-zones within the F2 Core Zone. The total F2 system has been expanded to a strike length of approximately 1,240 m and to depths of approximately 1,460 m.

The 2010 drilling program expanded the known strike length of the F2 Gold System by 165 m to the southwest, and the system remains open along strike and at depth. Delineation drilling in the F2 Core Zone also identified a previously unrecognized northeast trending subzone to the northwest of the main area of delineation drilling.

Figure 10.5 summarizes the results of the total 2010 drilling program.

Figure 10.5 2010 Significant Intercepts and Area of Delineation Drill Program



Notes: Assays are uncut. Reported results satisfy the following cut-off criteria:

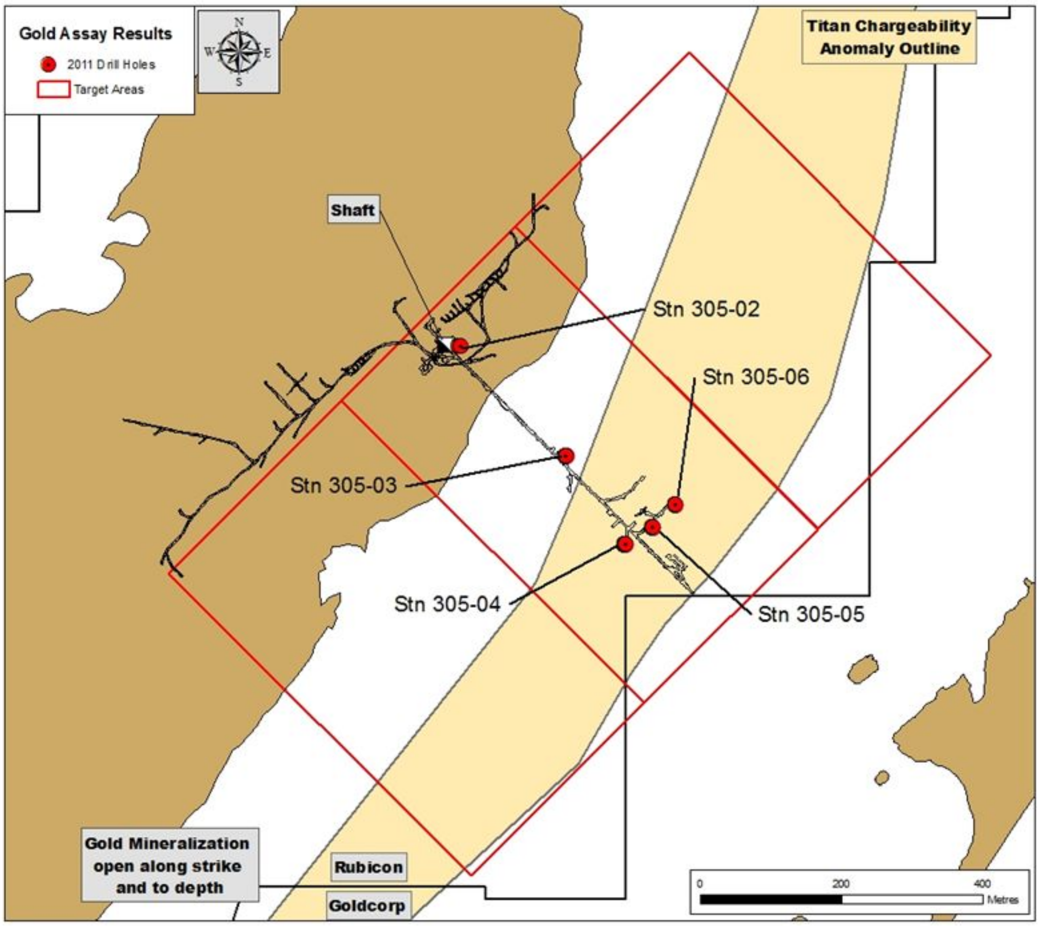
9X Exploration drilling reported results satisfy the following criteria: : An intercept equal to or greater than 10 g/t gold (gram) x (metre) product value and possessing an average grade of equal to or greater than 3.0 g/t gold.

Delineation drilling reported results satisfy the following criteria: An intercept equal to or greater than 10 g/t gold (gram) x (metre) product value and possessing an average grade of equal to or greater than 5.0 g/t gold.

10.4 2011 Diamond Drilling Program (F2 Gold System)

A total of 22,641 m were drilled in 2011 through to February 28. All drilling was conducted from underground on the 305 level, from five separate drill stations, 305-02 through 305-06. The majority of the drilling was focused on the F2 Core Zone (Figure 10.5).

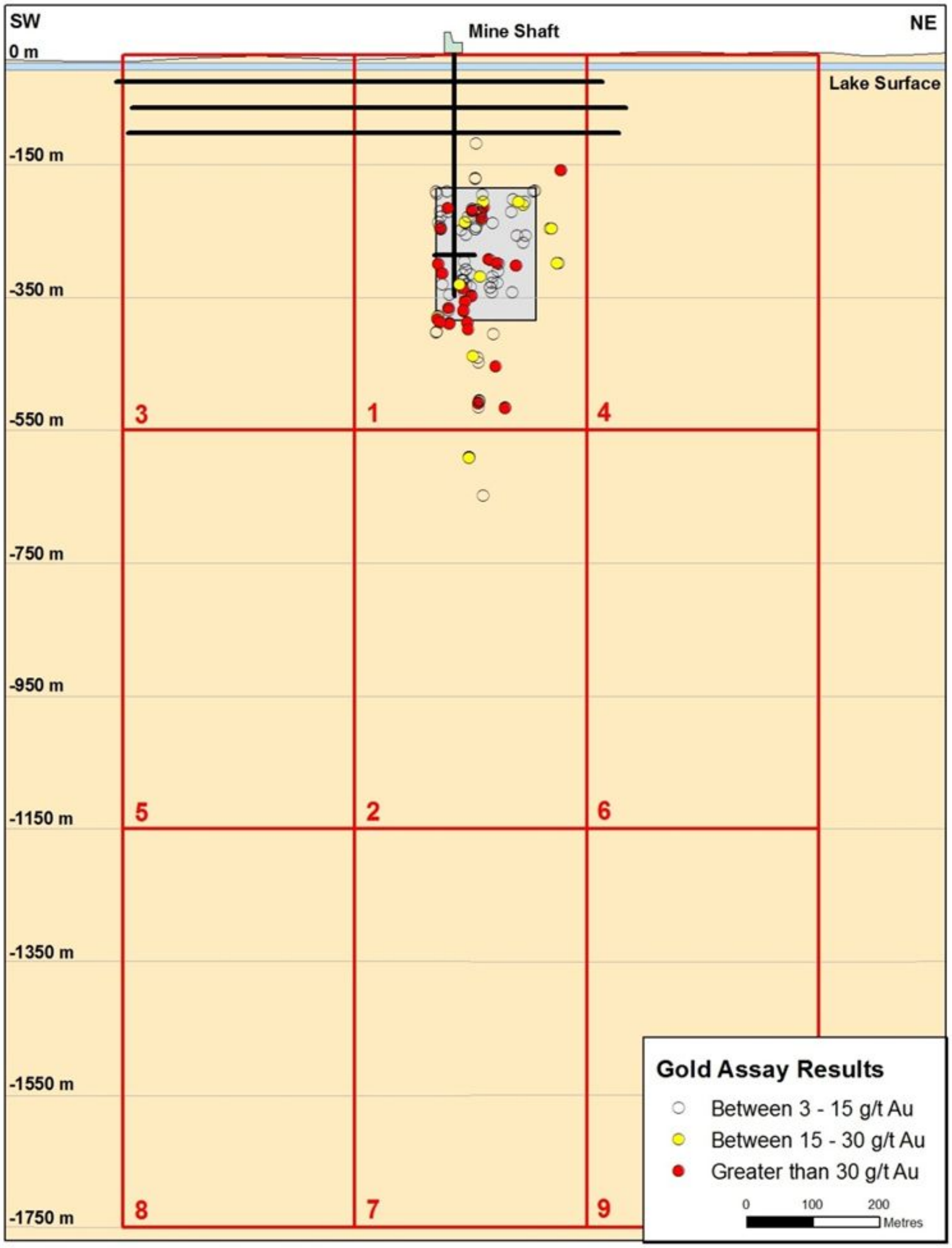
Figure 10.6 2011 Drillhole Locations



Notes: ‘305’ prefix holes drilled from underground on the 305m level
‘D305’ prefix holes drilled from underground on the 305m level and are delineation drill program holes

Figure 10.6 shows the area of the 2011 drilling program and significant intercepts.

Figure 10.7 2011 Significant Intercepts and Area of Delineation Drill Program



Notes:Assays are uncut. Reported results satisfy the following cut-off criteria:

9X Exploration drilling reported results satisfy the following criteria: An intercept equal to or greater than 10 g/t gold (gram) x (metre) product value <u>and</u> possessing an average grade of equal to or greater than 3.0 g/t gold.

Delineation drilling reported results satisfy the following criteria: An intercept equal to or greater than 10 g/t gold (gram) x (metre) product value <u>and</u> possessing an average grade of equal to or greater than 5.0 g/t gold.

The 2011 delineation drilling has continued to define the presence of 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.

The 2011 assay results to date continue to show the trend of high-grade intercepts and broad lower grade gold zones. The current results show that the interpreted individual gold zones, limited to the area of delineation drilling completed to the end of February, demonstrate a horizontal continuity of greater than 100 m in strike length and a vertical extent of over 150 m. When wider spaced drilling along strike and below the area of delineation is taken into account, the interpreted individual gold zones extend to approximately 150 m in strike length and 300 m vertical. To date the depth and lateral extent of most of these zones is open.

10.5 Bulk Density Measurements

Bulk density measurements are taken from drill core and determined by the water immersible method. The samples are weighed in air with the weight recorded, then placed in a basket suspended in water and the weight recorded again. The samples are not waxed or sealed; however, AMC does not consider natural voids to be a significant issue with respect to bulk density determination and accepts the values as presented.

The formula used is as follows:

$$\frac{\text{(Sample weight in air)}}{\text{(Sample weight in air)} - \text{(Sample weight in water)}}$$

In total there have been 6,174 samples measured through to 28 February, 2011. Section 14 has a breakdown of the samples by major lithology type.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Information regarding sample preparation, analyses and security was obtained through discussions held with Rubicon geological staff and inspection of the site facilities by AMC between March 28 and March 31, 2011. Additional information was provided from geological reports supplied by Rubicon. Information regarding sample preparation, analyses and security of samples by previous operators on the property was not readily available and, as a consequence, data deemed historic has been excluded from the resource estimate. It is AMC's opinion that the sample preparation, security and analytical procedures used for the current sampling program conform to the generally accepted Canadian mining industry best practice.

11.1 Sample Handling and Preparation

The core shack and site have 24 hour on-site security including personnel and video surveillance. Upon arrival at the core facility, the core is washed, logged and split using a diamond blade saw under the on-site supervision of a Rubicon geologist. Samples are moved directly from the core shack to the cutting shack, then are cut and shipped in individual zip tied sample bags. Approximately ten individual bagged samples are placed in a large rice bag that is sealed with a security zip tie containing a unique numbered tamper proof security seal. Samples are delivered directly from the mine site to the SGS lab in Red Lake (since 2008) by Rubicon staff. Each sample number and security seal is recorded and then verified by SGS's lab with a written acknowledgment upon receipt.

Blank and Standards assay protocols were developed in 2003 and revisited in 2009 with input from Dr. Barry Smee, Ph.D., P.Geo., Independent Geochemist, in consultation with Rubicon personnel and J.J. Watkins (Independent Q.P. 2000 - February 2003) and T. Bursey, P.Geo. (present Q.P.). Blank samples (consisting of commercially available broken tile, locally quarried quartz or barren granite boulder material) are inserted into the sample stream once in every 25 samples, to provide a check on drill core preparation in the assay laboratory. Random gold Standards are inserted into the sample stream once every 25 samples to provide a check on assay laboratory data quality. Gold Standards are prepared and certified by CDN Resources Laboratories Ltd., Delta, B.C. Rubicon has used 33 different Certified Standards, ranging in grade from 0.123 g/t to 29.21 g/t Au and currently uses the following, as per Table 11.1:

Table 11.1 Standards Currently in Use

Laboratory	Standard	Control value g/t Au	Limit +/- g/t Au	Tests exceeding control limits	% of Total tests
SGS	CDN-GS-11A	11.210	1.305	0	0.0
SGS	CDN-GS-1E	1.160	0.090	6	0.4
SGS	CDN-GS-2C	2.060	0.225	18	24.3
SGS	CDN-GS-30B	29.210	1.845	12	6.5
SGS	CDN-GS-3D	3.410	0.375	23	29.9
SGS	CDN-GS-3E	2.970	0.405	4	3.3
SGS	CDN-GS-5C	4.740	0.420	0	0.0
SGS	CDN-GS-5E	4.830	0.555	5	0.4
SGS	CDN-GS-6	9.990	0.750	0	0.0
SGS	CDN-GS-7A	7.200	0.900	3	2.2
SGS	CDN-GS-P5B	0.440	0.060	0	0.0
SGS	CDN-GS-P7A	7.200	0.900	0	0.0
SGS	CDN-GS-P8	0.780	0.090	13	6.9

Samples are reanalyzed if any anomalies in the data are observed. A more detailed description of the Standards, Blanks and Duplicates follows in Section 11.3 of the Technical Report.

Rubicon initiated an assay check sampling program in 2010 where 5% of the sample pulps are collected and sent to an independent ISO certified laboratory for assay recheck. Standards and Blanks are also inserted to provide quality control on the re-assays samples. Results from this sample check assay program are reviewed for accuracy and tracked in an action log as part of the standard QA/QC procedures. Failures are addressed and re-assayed as required.

The logged and sampled drill 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 from drillholes are stored on the mine site for long term storage and for the future.

The sample preparation procedure at the laboratory given by Rubicon is documented below:

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

11.2 Sample Analyses

All analytical or testing laboratories used by the Company are independent of the Company. Various analytical laboratories have been used by the Company over time and these are discussed below. Samples collected before 2008 were sent to either ALS Minerals (“ALS”) (preparation lab in Thunder Bay, ON., and wet lab in Vancouver, B.C.) or AccurAssay Laboratories, Thunder Bay, ON. (“AccurAssay”). Since January 2008, assays have been conducted by SGS in Red Lake, Ontario.

Dr. Barry Smee, Consultant, audited the sample preparation facilities of SGS Laboratory, Red Lake, Ontario on behalf of Rubicon. Recommendations from his audit have been implemented (Smee and Associates Consulting Ltd., 2009).

11.2.1 ALS Minerals

Gold was determined by FA fusion of a 50 g sub-sample with an AAS finish. The 'Au -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 is retained and analyzed in its entirety by FA fusion followed by cupellation and a gravimetric finish. The –150 mesh (pass) fraction is homogenized and two 50 g sub-samples are analyzed by standard FA procedures. The gold values for both +150 and –150 mesh fractions are reported together with the weight of each fraction as well as the calculated total gold content of the sample. 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 m was selected for ICP analysis. The elements Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Sc, Sr, Tl, Ti, U, V, W, and Zn were analyzed by Inductively-Coupled Plasma ("ICP") Atomic Emission Spectroscopy, following multi-acid digestion in nitric aqua regia. The elements Cu, Pb, and Zn were determined by ore grade assay for samples that returned values greater than 10,000 ppm by ICP analysis. Only a select few samples were sent for whole rock analysis where major elements (reported as oxides) and Ba, Rb, Sr, Nb, Zr, and Y were determined by X-Ray Fluorescence Spectrometry ("XRF").

Results were reported electronically to the project site in Red Lake with Assay Certificates filed and catalogued at Rubicon's Head Office in Vancouver.

ALS operates according to the guidelines set out in ISO/IEC Guide 25 – "General requirements for the competence of calibration and testing laboratories".

11.2.2 AccurAssay Laboratories

Gold was determined by FA using a 30 g fire assay charge. This procedure uses lead collection with a silver inquart. The beads are then digested and an AA or ICP finish is used. All gold assays that are greater than 10 g/t are automatically re-assayed by FA with a gravimetric finish for better accuracy and reproducibility. A Sartorius micro-balance with a sensitivity of 1 microgram (six decimal places) giving a 5 g/t (5 ppb) detection limit is used.

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

The elements Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Sc, Sr, Tl, Ti, U, V, W, and Zn are analyzed by ICP following multi-acid digestion in nitric aqua regia.

As with ALS Chemex, results are reported electronically to the project site in Red Lake with Assay Certificates filed and catalogued at Rubicon's Head Office in Vancouver.

At AccurAssay, gold, platinum, palladium, copper, nickel and cobalt analysis are accredited by the Standards Council of Canada under ISO/IEC Guideline 17025.

11.2.3 SGS Mineral Services

Samples were initially analyzed for gold using the FA process on a 30 g sample. Typically the samples are mixed with fluxing agents including lead oxide, and fused at high temperature. The lead oxide is reduced to lead, which collects the precious metals. The precious metals are then separated from the lead in a secondary procedure called cupellation. The final technique used to determine the gold and other precious metals contents of the residue is AAS. If the sample contains greater than 10 g/t Au, it is sent for a gravimetric finish. Starting in October 2009 assay sample size was increased to 50 g as a consequence of implementing the recommendations made in the Smee (2009) report.

In cases where multiple standard Au FA analyses were completed on an individual sample, gold values produced by the metallic FA are deemed to supersede FA gold values owing to the larger size of sample analysed and/or better reproducibility in samples with coarse gold.

Select sample pulps that require multi-element analysis are sent to the SGS Laboratory in Toronto, Ontario, where they undergo a multi-acid digestion. This is a combination of HCl (hydrochloric acid), HNO_3 (nitric acid), HF (hydrofluoric acid), $HClO_4$ (perchloric acid). Because hydrofluoric acid dissolves silicate minerals, these digestions are often referred to as 'near-total digestions'. However, there can be a loss of volatiles (e.g. B, As, Pb, Ge, Sb) during the digestion process. Multi-acid (four acid) digestion is a very effective dissolution procedure for a large number of minerals and is suitable for a wide range of elements.

Results are reported electronically to the project site in Red Lake with Assay Certificates filed and catalogued at Rubicon's Head Office in Vancouver and added to the master Access database stored on the Vancouver and Red Lake servers.

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

SGS operate according to the guidelines set out in ISO/IEC Guide 25.

11.3 Assay QA/QC

Since May 2010 ioGlobal Pty Ltd ("ioGlobal"), based in Vancouver, has taken over the management of the Project assay data and provided independent quality control and quality assurance reporting and database auditing. Data quality is monitored and checked on a regular basis to ensure data accuracy and lab performance.

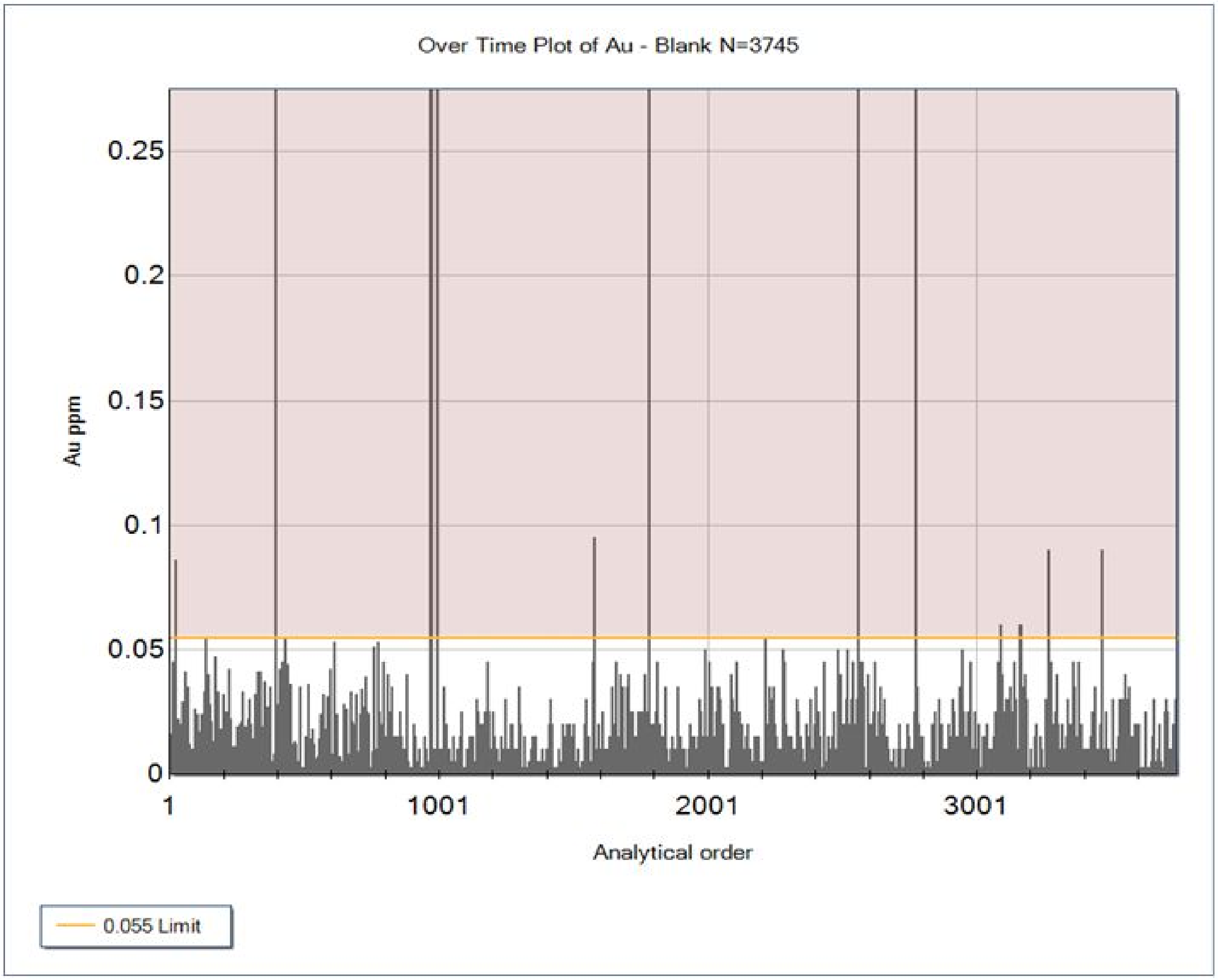
As part of the process ioGlobal reviewed all of the assay data and performed QA/QC analysis for a specific list of drillholes from the Project for the period May 2009 through to February 2011. Based on this review, ioGlobal considers that the overall QA/QC performance for the data analyzed is acceptable.

The following are excerpts from the ioGlobal report: 'Assay Quality Control Report for the Period May 2009 to Feb 2011 for the Phoenix Project', dated 28 March 2011 (ioGlobal 2011).

11.3.1 Blanks

Few blank samples (0.5%) fail the batch assessment criteria threshold of 55 ppb. Despite the lack of blank failures, there are a relatively high number of samples above 25 ppb, but below the 55 ppb threshold used (the detection limit for gold is 5 ppb). Figure 11.1 is taken from the ioGlobal report and shows the blank samples tested using FAA515 method (50 gm sample weight).

Figure 11.1 Blank Samples Tested using FAA515



Number of Tests	Count Failed	Percent Fail
3745	15	0.4

ioGlobal has suggested a further review of ongoing laboratory performance to ensure procedures are optimal to avoid contamination. Recommendations include the following:

- an audit of the sample preparation facility at the SGS Red Lake laboratory where samples are being processed focusing on sample prep area cleanliness and pulverization bowl cleaning procedures
- adjustment of blank insertion practices to ensure some blanks are being submitted directly after high grade samples. A review of results after a one-month period should determine whether any carry over is occurring
- If, after additional monitoring, performance does not improve, Rubicon should consider quartz washing between each sample to prevent carry over contamination from high grade samples

The three recommendations from ioGlobal in the bulleted list above are under consideration by Rubicon management. The lab audit will be scheduled at the earliest convenience and the need for implementation of other ioGlobal recommendations will be assessed after the audit.

11.3.2 Standards

A low bias was observed in reported values for standards CDM-GS-2C and 3D and there were a number of fails recorded for CDN-GS-P8 (prior to June 2009); this has been resolved and standards performance has been acceptable from June 2009 onwards.

A consistent, negative bias is observed in a number of standards analysed by method FAA313. Although a majority of samples still pass QA/QC criteria, this level of bias is significant. The analysis shows that the low bias was resolved in September 2009 and Rubicon has advised ioGlobal that this is consistent with the expected outcome resulting from the lab changing the flux used in the assay process (as recommended by Smee in 2009, Rubicon in-house report).

11.3.3 Duplicates

Precision data for all repeat types is considered low although acceptable for a nuggety gold deposit. The CV value for method FAA313 is greater than 15% suggesting non normal error distribution. Precision improved after November 2009 which coincides with modifications of laboratory procedures including implementation of a 50g fire assay ‘FAA515’ (increased from 30 g fire assay ‘FAA 313’).

11.3.4 Umpire Assaying

ioGlobal states “The low number of sample pairs above the detection limit for most method – lab combinations precludes assessment of bias for these instances. Comparison of FAA515 (SGS) data with Au-AA24 (ALS) data indicates no statistically significant bias. It is suggested that although no statistically significant bias can be identified, significant variation between individual sample pair results is observed. The high intra lab variability related to the nuggety nature of mineralization, means that it is difficult to assess whether this variation is a reflection of the underlying variability of the material or differences in laboratory performance. It should be noted that only a pulp umpire sample can be reasonably interpreted as relating to issues in lab performance and ioGlobal do not have information as to the nature of the umpire samples.” Since the beginning of 2010 Rubicon has been sending assay pulp duplicates to ALS Chemex in Thunder Bay for the purpose of umpire assaying.

11.4 QA/QC Results and analysis

Based on the site review and the ioGlobal QA/QC analysis, AMC has determined that the sample preparation, security, analytical procedures and application of QA/QC analysis is performed in accordance with industry best practices. Standards, Blanks and Duplicates are plotted and reviewed internally regarding a pass-fail analysis. Any failures are identified and addressed prior to data entry to the master database.

AMC understands that a full laboratory audit will be conducted in order to investigate the high intralab variability in sample pair results, and that Rubicon has also adjusted its Blank insertion protocol as follows: if the logging geologist sees visible gold the sample is sent along with bracket samples straight to the Gravimetric circuit and a blank and high grade standard is also inserted in that batch (separate manifest and batch). It is normal practice that all assays go through first pass fire assay that has a 10 g/t upper limit; any assays over this limit then go through the fire assay gravimetric finish process for >10 g/t.

12 DATA VERIFICATION

12.1 Data Verification

The diamond drilling discussed in this report was undertaken by experienced and competent Rubicon geologists under the supervision of Ian Russell, Exploration Manager for the Phoenix Gold Project and Terry Bursey, P.Geo., Regional Manager for Rubicon's Red Lake Projects. AMC staff completed a site visit between March 29 - 31, 2011 to review the Project and all relevant materials, to which it had full access. AMC believes that work completed by Rubicon was done in a professional manner and has met generally accepted industry standards for QA/QC.

Data review and verification by AMC included this site visit and review of the following: drill sections and plans with geological interpretations (1:1000 and 1:500 scale), drill core logging procedures and facilities, QA/QC procedures, independent QA/QC analysis and core cutting facilities, core storage, drill collar locations where available, drill core and related geological units, alteration and associated mineralization intersected, database and discussions with company geologists and staff.

Rubicon performs logging, surveying, sample selection and inserts QA/QC blanks & standards, etc. Data is verified and double checked by senior geologists at site (for data entry verification, error analysis, plus assay pass/fail against standards and blanks, etc.). Drillhole data is then sent to ioGlobal for independent verification and QA/QC analysis by way of external audit.

AMC has interrogated the database for overlaps, missing samples and survey aberrations using Datamine Studio™ and Excel™. One assay was found missing and 7 holes had the first survey interval missing. Given the size of the database (511 collars with 85,021 assays used to update the resource in the Technical Report), AMC has considered this to be a clean set of data.

AMC reviewed the drillhole and geological data and agrees with the Rubicon interpretation of the F2 Gold System. Wireframes were constructed by AMC to represent the bounding mineralized surface. These wireframes were constructed approximately parallel to the interpreted geology and maintain the overall geological trend of the zone.

Rubicon provided AMC with independent third party verification of the following items:

- Independent verification of QA/QC procedures (Smee and Associates Consulting Ltd 2009)
- Independent QA/QC analysis and verification (ioGlobal 2011)
- Previous independent resource model and related materials (GeoEx 2011)

12.2 Limitations on Verification

Data verification was completed by AMC. Assay data QA/QC is undertaken by ioGlobal on a continuing basis. There were no limitations regarding the verification process as Rubicon provided AMC with full access to all technical data available for the Project.

12.3 Adequacy of Data for Purpose

It is AMC's opinion that the data collected and provided to date by Rubicon is sufficient for the purpose of resource estimation. The drilling and sampling procedures are well developed and are efficiently carried out. Continued input for the assaying protocols and continued review of the assay laboratories has helped to monitor and control natural grade variance in an appropriate fashion and good database management means that the data is readily accessible and clean.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Background

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

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

In June 2011, Rubicon completed a further study (the “2011 study”) supervised by Soutex with the collaboration of Boge & Boge (1980 Ltd.). The testwork program was done on sub-samples (composites) extracted from two 1,000 t representing two underground areas on the 305L. The metallurgical testwork was conducted at G&T under the supervision of Soutex.

13.2 The Nature and Extent of the Testing

Two sets of samples were available for characterisation and testwork at G&T:

- 2010 study samples: five drill cores and two assay rejects
- 2011 study samples: two sub-samples (composites) from the two 1,000 t bulk samples

13.2.1 Elemental Characteristics

In the 2010 study, of the seven samples treated, five samples (RL-01-01 to RL-01-05) originated from drill core samples and two (RL-02-01 and RL-02-02) originated from assay rejects.

The main chemical elements were assayed using specific methods. Gold assays were conducted using a gold metallic method to reduce assay variability. Table 13.1 presents the head grade results for all samples.

Table 13.1 Sample Head Grades

Sample	Grade				
	Au (g/t)	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 of the tested samples are the following:

- On average, the samples contain 7.82 g/t Au (ranging from 4.12 g/t to 12.8 g/t).
- On average, the samples contain 2.33% S (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 contain somewhat increased levels of arsenopyrite.
- Arsenic levels in the samples were relatively low, averaging 0.04%.
- The specific gravity for the samples averages 2.78 (ranging from 2.67 to 2.84).

13.2.2 Grindability

Grindability testing based on the Bond work indexes was done first on the 2010 study samples (drill cores) and results are presented in Table 13.2.

Table 13.2 Grindability Results on Drill Core Samples

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

Additional grindability testing based on Bond work indexes was done on the 2011 study samples and results are presented in Table 13.3.

Table 13.3 Grindability Results on Composite Samples

Sample	Bond Rod Wi (kWh/t)	Bond Ball Wi (kWh/t)
Composite 1	17.7	13.1
Composite 2	15.3	10.3
Average	16.5	11.7

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, Drop Weight Testing (“DWT”) was performed on the 2011 study samples in order to allow the sizing of the SAG mill and the design of the grinding circuit using JKSimMet software. The results are presented in Table 13.4.

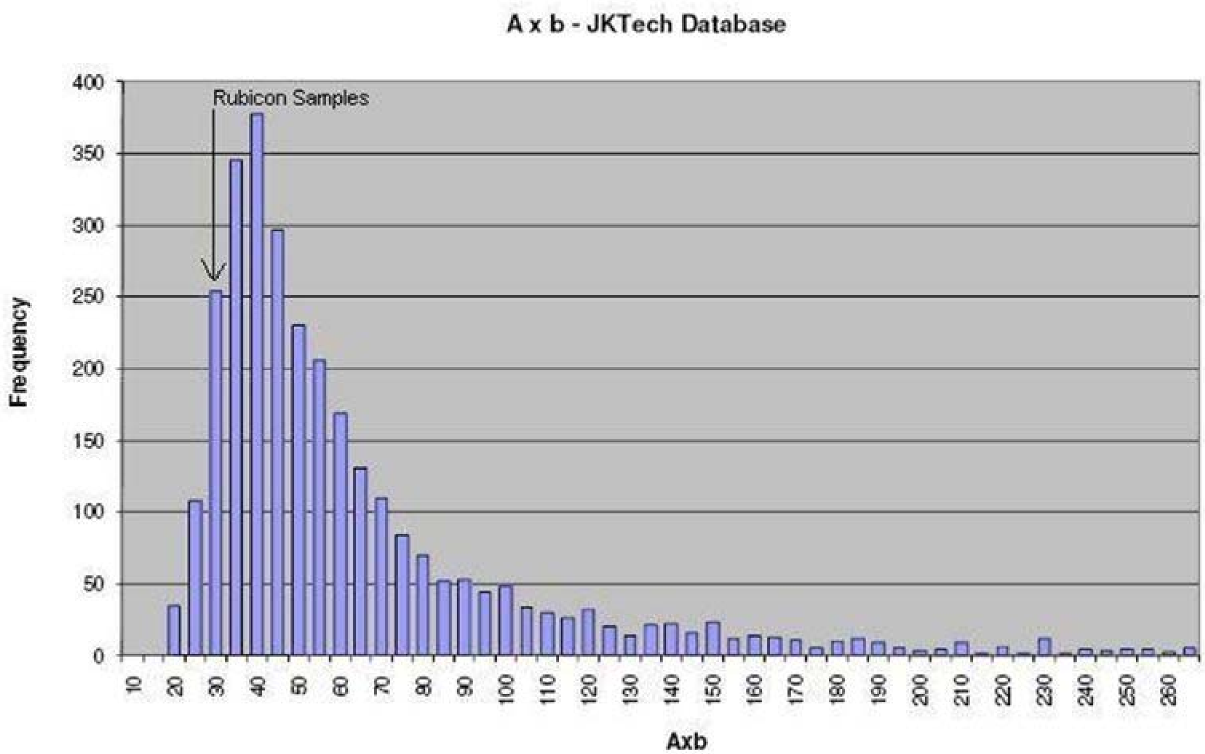
Table 13.4 DWT Results on Composite Samples

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 13.1 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 for 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 tested samples.

Figure 13.1 Frequency distribution of A*b



- Parameter ta is utilized to characterize the abrasiveness of a material. The smaller parameter ta is the more resistant is the material to abrasion.
- As part of the DWT testing, specific gravities of 30 randomly selected particles in the size range of 26.5 - 31.5 mm were determined. The samples have an average specific gravity of 3.05 (ranging from 2.87 to 3.37).

13.2.3 Implications for SAG Mill Design

Based on the DWT parameters presented in Table 13.4, SAG mill sizing can be completed (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 tonnage
- Evaluate the effect of adding a pebble crusher to achieve the 2,500 tpd tonnage by crushing a coarser SAG mill product obtained by use of larger grate openings
- Evaluate the effect of mill dimensions, grinding media charge and mill rotation

Table 13.5 presents a summary of the results obtained from the JKSimMet simulations.

Table 13.5 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 (%)	Pebbler Crusher (Y/N)	Total SAG Mill Power Requirement (HP)	Total Ball Mill Power Requirement (HP)
1 250 tpd (57 t/h)									
Option 0	22 x 12	-	-	-	-	-	-	-	-
Option 1	24 x 9	5	75	15	3.4	-	No	2,312	-
Option 2	20 x 10	10	71	12	-	0	No	1,669	453
Option 3	20 x 10	10	71	12	-	0	No	1,669	453
Option 4	20 x 12	10	71	12	-	0	No	1,642	346
Option 5	-	-	-	-	-	-	-	-	-
Option 6	-	-	-	-	-	-	-	-	-
2 500 tpd (113 t/h)									
Option 0	22 x 12	10	75	25	9.5	10	No	2,756	1,101
Option 1	24 x 9	10	75	25	9.5	10	No	2,643	1,190
Option 2	20 x 10	12	77	63	9.5	20	Yes	1,971	1,221
Option 3	20 x 10	12	77	63	6.4	24	Yes	1,976	1,150
Option 4	20 x 12	12	77	63	6.4	24	Yes	1,954	892
Option 5	20 x 12	12	77	25	6.4	18	No	2,315	1,181
Option 6	20 x 12	12	77	25	6.4	18	No	2,287	918

Results of the JKSimMet simulations allow the following conclusions:

- Option 2, consisting of one 20’ x 11.25’ flange to flange (F/F), 10’ effective grinding length (EGL) SAG mill supplied with a 2,000 HP motor and one 11’ x 16’ flange to flange (F/F) Ball mill supplied with a 800 HP motor is the preferred option.
- For the 1,250 tpd tonnage, the SAG mill will be operated at a lower rotation speed with a reduced ball charge. This will result in reduced power consumption.

13.2.4 Gold Recovery

Gold recovery testing was done on the 2010 study samples (drill cores). For that testing, two of the five drill core samples were tested using four different flowsheet arrangements; the results on flowsheet 2 are presented in Table 13.6. Tested flowsheet arrangements were the following:

- Flowsheet 1: Gravity followed by rougher flotation
- Flowsheet 2: Gravity followed by cyanide leaching for 48 hours
- Flowsheet 3: Rougher flotation only
- Flowsheet 4: Cyanide leaching only

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

- 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 of leaching. 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
- The cyanide consumptions measured during the testing program were relatively low.

Table 13.6 Gold Recovery Results on Drill Core Samples (Flowsheet 2)

Sample	Gravity Recovery (%)	Leach Feed Au (g/t)	Leach Recovery (%)	Tailings Au (g/t)	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

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. Once the plant tailings are properly treated, there are no particular environmental issues.

13.3 Basis for Assumptions Regarding Recovery Estimates

13.3.1 Range of Gold Recovery

The gold recovery results obtained from only two drill 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 drill core samples to test the selected flowsheet (flowsheet 2).

Also, although the selected flowsheet was tested on the two assay reject samples (RL-02-01 and RL-02-02), the resulting gold recoveries were not used in the calculation of the gold recovery average. The size distribution of these samples was too fine to compare to what can be obtained with an industrial grinding circuit.

Thus, a gold recovery average of 92.9% (see Table 13.6) was calculated from samples RL-01-01 and RL-01-02. An estimated soluble gold loss of 0.4% should be subtracted from that value and this leads to a final gold recovery average of 92.5%.

A sensitivity analysis of the project is usually done with a range of possible gold recovery. For the PEA, the estimated range is based on previous experience rather than results obtained during the testwork program during which the number of tested drill 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 testwork program. The range is therefore 91% to 95% and covers the uncertainties related to ore mineralogy and scale-up methodology.

To decrease the uncertainties and have a better estimation of the range of possible gold recovery, a larger number of drill core samples statistically representing the different zones to be exploited should be collected in future work.

13.3.2 Improvement in Gold Recovery

It is anticipated that a better knowledge of the ore 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.

13.4 Samples Representativeness

13.4.1 2010 Study Samples – Drill Core Samples

A protocol for selecting the drill cores needed for the 2010 study samples was designed by Soutex in collaboration with the Geology department of Rubicon. The initial requirements for the five drill core samples used in the testwork program are presented in Table 13.7.

Table 13.7 Drill Core Sample Requirements

		Samples				
		RL-01-01	RL-01-02	RL-01-03	RL-01-04	RL-01-05
Gold grade	g/t	30	10	15-30	N/A	N/A
Sample weight	kg	50	50	5	5	5
Number of holes	-	>10	>10	1	>5	>10
Ore characterization	-	Yes	Yes	Yes	Yes	Yes
Metallurgical testing	-	Yes	Yes	No	No	No

The first four samples were taken from predetermined locations according to the initially envisioned short to medium term mining scenario. The last sample 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.

Drill core samples were prepared by complying with the following elements:

- The quantity of material should have been obtained by selecting the number of material quarters from one drill core distributed equally all along the drill core sections of interest representing the zone
- The samples should have not been crushed before the shipping
- All samples should have been 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 drill core sections (hole number and depth) for material going to the ore characterization or the metallurgical testing

- Expected average gold grade
- Method used to select sections from drill cores (when applicable)
- Spatial distribution of all sampled drillholes

13.4.2 2011 Study Samples – Sub-Samples (Composites)

Two bulk samples were needed to confirm the head grades for two different underground zones. 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 the Geology department of Rubicon. From the bulk samples, several sub-samples were generated. It is important to mention that the protocol was based on Gy theory.

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

- Extraction from underground of 1,000 t of -23 mm material from a specific zone
- Crushing all material in a jaw crusher to -75 mm
- Hand sampling of a 45-gallon drum (see sample discussion in Section 13.5.2 below)
- Crushing all material in a cone crusher to -12 mm
- Tower sampling of one 10 t sample for confirming the head grade at the G&T pilot plant facility (see Chapter 17 for details)
- Tower sampling of two 1 t samples for future testwork
- Storage of the remaining ~988 t for confirming the head grade at SMC Canada Ltd. McAlpine mill (custom milling facility) (see Chapter 17 for details)

The targeted grade variability of the 10 t samples (expressed as the relative standard deviation of 1 σ) is about 7 %, meaning that there is 95 % probability that the grade will be within a range of +/-14 % (or +/- 2 σ) around the grade of the material extracted from a specific zone. The majority of the details regarding the processing of the bulk sample are given in Chapter 17.

13.5 Factors with Possible Effect on Potential Economic Extraction

13.5.1 2010 Study Samples

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

- Tested samples

  The testwork was done on drill cores originating from only two zones and the current average metallurgical performances do 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 drill core samples statistically representing the different zones to be exploited should be collected.

- Main process equipment

  At the grinding circuit for the operation at 1,250 tpd, the SAG mill will 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 will be increased and a cone crusher will be needed to crush the SAG mill recirculating load in order to reach the production target.

  At the paste plant for the 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 but it may be necessary to add a thickener.

- Plant tailings toxicity

  The characteristics of the tested samples suggest the use of a simple 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 are no significant arsenic or other deleterious elements present in the tested samples.

- Tailings 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 the ammonia has the potential to produce an effluent that is lethal for rainbow trout, it is necessary to keep it as low as possible. Further testwork is needed aimed at reducing the cyanide concentration.

13.5.2 2011 Study Samples

The following factor was identified during the treatment of the 2011 study samples:

- Gold assaying

  A series of gold grade assays were made in duplicate on about 10 samples pulverised at 95% - 100 µm. Each sample weight was 250 g and two fire assays were initially performed on a 30 g sub-sample. Table 13.8 and Table 13.9 show 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 Gy sampling theory, that the variability in the results follows a Poisson distribution. Such behaviour indicates that the 30 g 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 variability.

Table 13.8 Gold Grades for 30 g Assayed Samples (Bulk Sample #1)

Sample	Assay #1 (%)	Assay #2 (%)	Difference (%)
Head 1	10.00	4.95	5.05
Head 2	5.88	5.34	0.54
Head 3	5.49	5.82	- 0.33
Head 4	7.49	20.30	- 12.81
Head 5	5.64	12.40	- 6.76
Head 6	9.13	7.38	1.75
Head 7	7.93	5.64	2.29
Head 8	5.14	6.69	- 1.55
Head 9	8.85	8.43	0.42
Head 10	4.94	5.78	- 0.84
Head σ_{REL} (%)	26.38	57.52	-
Difference σ_{REL} (%)	-	-	45.90

Table 13.9 Gold Grades for 30 g Assayed Samples (Bulk Sample #2)

Sample	Assay #1 (%)	Assay #2 (%)	Difference (%)
Head 1	8.10	5.47	2.63
Head 2	8.09	7.17	0.92
Head 3	6.45	7.40	- 0.95
Head 4	5.92	6.33	- 0.41
Head 5	10.10	16.20	- 6.10
Head 6	8.47	7.12	1.35
Head 7	23.30	12.60	10.70
Head 8	6.96	6.42	0.54
Head 9	10.40	7.82	2.58
Head σ_{REL} (%)	54.32	41.51	-
Difference σ_{REL} (%)	-	-	33.42

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The mineral resource has been estimated by Ms D Nussipakynova, P.Geo of AMC, who takes responsibility for the estimate, under the supervision of Mr J M Shannon, P.Geo of AMC. This estimate supersedes the mineral resource estimate prepared by P T George, P.Geo., of GeoEx Limited, and reported in the associated Technical Report dated April 11 2011 (George 2011).

A block modelling approach was chosen to assist in proposed mine planning and the estimate now includes all drilling up to the end February 2011. All the modelling and the estimation were carried out using Datamine software; the completed model is named: Phoenix_AMC_0611_model.dm.

The summary results of the estimate at a cut off of 5.0 g/t Au are shown in Table 14.1 below.

Table 14.1 Mineral Resources as of 15 June 2011

Classification	M Tonnes	g/t Au	M oz Au
Indicated	1.028	14.5	0.477
Inferred	4.230	17.0	2.317

Notes: 1. CIM definitions were used for mineral resources
2. The cut off grade applied is 5.0 g/t Au
3. A capping value of 270g/t Au has been applied to the composites
4. Using drilling results to February 28, 2011

Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve. No mineral reserves have been estimated as part of the present study.

14.2 Data Used

14.2.1 Drillhole Database

The data used in the estimate consisted of drill data contained within an Access^{TM} database provided by Rubicon to AMC. The total number of drillholes provided in the database is 511, with a total logged length of 239,340 m. The average length of hole is 468.4 m with a maximum depth of 2,061 m. The total number of surface holes is 151, with 360 drilled from the underground levels. Table 14.2 gives a breakdown by year of the F2 Zone drilling used in the resource estimate.

Table 14.2 Drillholes used in Databases

Year	Number of Surface Holes	Surface Hole metres	Number of Underground Holes	Underground Hole metres	Total metres
2008	50	35,528			35,528
2009	58	37,615	42	25,512	63,127
2010	43	35,969	199	82,068	118,037
2011			119	22,648	22,648
Total	**151**	**109,112**	**360**	**130,228**	**239,340**
Totals	**511**				**239,340**

In Table 14.3 below it is seen that seven drillholes do not contain assays results and therefore a total of 504 drillholes have been used for resource estimation. Of the remaining seven, six of the holes were abandoned and one was not sampled at the time. The holes were however used for the lithological information.

Table 14.3 Summary of Data Used

Total number	Collars	Surveys	Assays	Lithology
Drillholes	511	511	504	511
Records	511	10,470	85,021	14,696

For intervals that recorded an assay value below the detection limit, AMC applied the value of 0.0025 g/t, representing half the reported default value. Also, where there were no samples a zero value was inserted.

AMC carried out standard validation procedures, has not identified any significant errors and considers the data-set fit for purpose.

14.2.2 Bulk Density

The collection of bulk density measurements is described in Section 10.5. The measurements were taken systematically throughout all rock types and there was no selectivity of samples or description in regard to mineralized as against non mineralized rock. Thus, there is likely a bias in favour of unmineralized rock for the bulk density measurements. Initial analysis was made of the densities recorded for the key rock types in the main domains by AMC and this is shown in Table 14.4.

Table 14.4 Bulk Density Values by Rock Type

Rock Code	Description	No of samples	% of Population	Mean
E1H	High Titanium basalt	998	27.71	2.96
EOT	Talc rich unit	882	24.49	2.90
I3	Felsic intrusives	790	21.93	2.67
E0	Ultramafic flow	520	14.44	2.93
E0B	Komatiitic basalt	196	5.44	2.98
E1A	Basalt	139	3.86	2.88
AGZ	Altered Green zone	76	2.11	2.94
Weighted mean of all		3601	100	2.87

JKTech Pty Ltd carried out bulk density measurements on two composite samples from the bulk samples collected in 2011, based on relative density measurements for 30 particles for each composite sample. The mean result of density in sample one is 3.05 t/m^3 and in the second sample is 2.95 t/m^3. These give an upper limit, as expected when generated by a heavily mineralized area, and therefore cannot be termed representative of the lithology types identified within the modelled area. Sample 2 in fact had a very broad range of values, from 2.65 t/m^3 to 4.10 t/m^3.

Additionally a plot of the felsic intrusive density values (Lithology type I3) against gold grade did not show an increase in the rock density against grade and so it was not possible to use this attribute to refine the density estimate.

After additional analysis, shown in Table 14.5 below, it was decided to use an average bulk density value of 2.90 t/m^3 for all rock types.

Table 14.5 Range of Bulk Density Values

Population	No of samples	Mean
12 top Rock Types	6,169	2.92
All Rock Types in Domains	4,005	2.87
7 top Rock Types in Domains	3,601	2.87
4 top Rock Types in Domains	3,190	2.87
4 top Rock Types in Domains excluding Felsics	2,400	2.93
Values from selected samples from Bulk Sample	2	2.95 – 3.05

14.3 Domain Modelling

14.3.1 Geology Model

The geology model was provided by Rubicon and consisted of wireframes in DXF format for the Hi-Ti Basalt (Code = E1H) and the Felsic Intrusive (Code = I3) units. These shapes are based on ongoing interpretation by Rubicon which has evolved as the drilling is carried out on the F2 Zone. Figure 7.5 in Section 7 is an example of one of 10 interpretive cross sections developed for the drilling fans from which the 3D model has been developed. This interpretation was reviewed by AMC at site. The base of overburden was modelled by Rubicon from the base of casing reading in the drillhole database (lake_bottom_from_casing_dtm). As this length includes casing for drilling from the ice or barges there is no estimate for the depth of overburden beneath the lake surface, although seismic evidence has estimated the accumulations of overburden/lake sediment as between 10 and 20 m on average, with accumulations up to 100 m along the structural trend underlying East Bay. On land the overburden varies from 0 to 7 m.

88 different rock codes are identified within the database table “DHGEOLOGY_2011_02_28” of which 76 are contained within the drillholes used in the model. These were grouped together into the main rock types shown in Table 14.6 below.

Table 14.6 Rock Codes Used

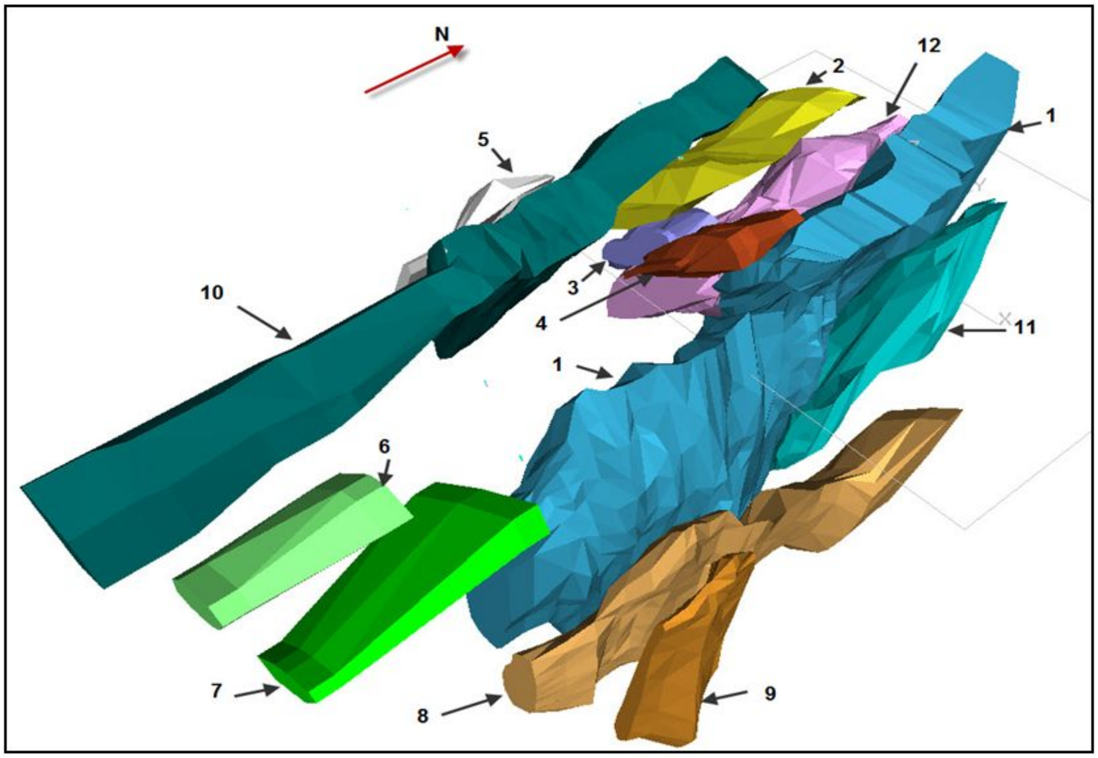
Rock Type	Rock Code
Altered Green Zone	AGZ
Alteration	ALT
Breccia	BRX, PXX, QBZ, QCB, HVB
Iron Formation	C2, C2A, C2B, C2C
Ultramafic Volcanics	E, E0, E0A, E0B, E0T, E0Y
Mafic Volcanics	E1, E1A, E1A1, E1A2, E1A3,E1F, P1V
Hi-Ti Basalt	E1H, E1H1, E1H2

Felsic Volcanics	E2, E2A, E3, E3A, E3C, P3A
Rock Type	**Rock Code**
Ultramafic Intrusives	I, I0 I0A, I0B, I0D, I0E, I0T, I0Y
Mafic Intrusives	I1, I1A, I1A1, I1A2, I1B, I1C
Intermediate Intrusives	I2, I2A, I2B
Felsic Intrusives	I3, I3A, I3C, I3E, I3P, I3Q, I3R, I3S
Sulphides	R5, V5
Sediment	S1A, S4B, ZBO
Veining	V, V1, V1A, V1S, V2, V2S, V2T, V3, V3A, V3M, V3S, V3T
Structure	FLT, SHD
Casing	ZCS
Overburden	ZOB, ZOT

14.3.2 Mineralized Domains

The definition of the mineralized domains was undertaken as an interactive process between AMC and Rubicon. Some trials with regard to a reasonable boundary for the mineralized domains were carried out. The option finally selected involved creation of broad domains, using the geological interpretation and lithological units combined with the presence of mineralization as limits. There were a total of 12 domains created which have been allocated an individual zone (domain) number and these are illustrated in Figure 14.1 below.

Figure 14.1 Isometric View of Mineralized Domains



Note: North arrow is shown in mine grid coordinates

14.4 Statistics and Compositing

14.4.1 Statistics

The source assay data used for the estimate were taken from holes entirely drilled and logged by Rubicon, and held within the database table "DHASSAYS_2011_02_28". Basic statistical analysis was carried out on the raw data in order to estimate the appropriate composite length given the dataset and to assess the grade distribution through the different domains.

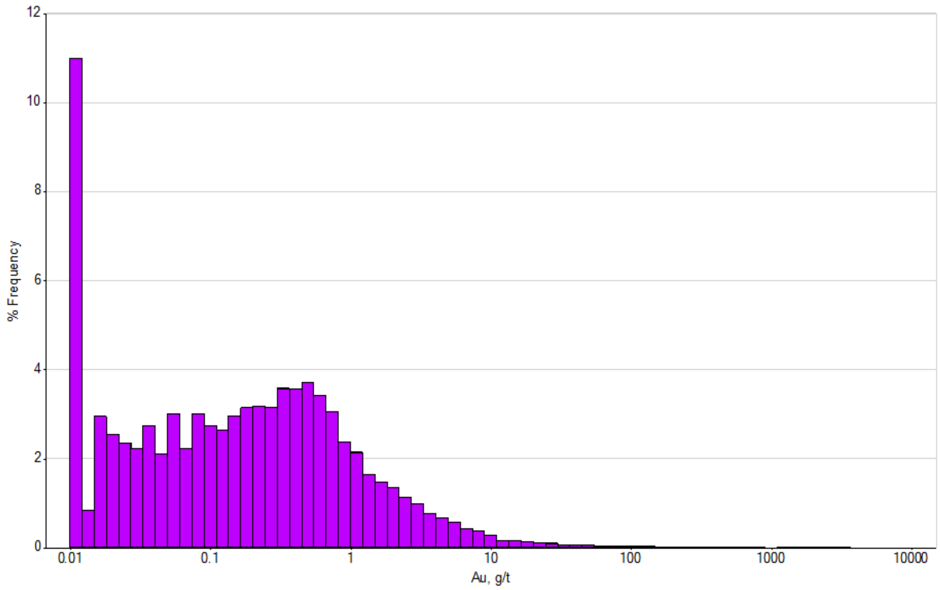
Table 14.7 shows the range of grades and number of samples for each of the mineralized domains modeled for the current resource estimate. These domains are larger and broader than the individual zones identified in previous modelling.

Table 14.7 Domain Statistics – Raw Data

Domain	Number	Minimum g/t Au	Maximum g/t Au	Mean g/t Au	Std Dev	Variance	Std Error	Coeff Var
1	55,447	0	2,898.3	1.43	22.25	494.99	0	15.58
2	1,854	0	373.8	0.39	9.28	86.15	0.01	23.64
3	908	0	4.1	0.07	0.26	0.07	0	3.55
4	330	0	42.5	1.01	3.56	12.7	0.01	3.55
5	680	0	170.9	0.81	6.93	48.08	0.01	8.55
6	159	0	7.3	0.28	0.95	0.9	0.01	3.45
7	150	0	15.7	0.5	1.43	2.04	0.01	2.84
8	3,613	0	2,287.1	1.45	40.51	1,640.91	0.01	27.88
9	733	0	17.5	0.12	0.8	0.64	0	6.59
10	4,802	0	3,151.1	0.9	45.5	2,070.13	0.01	50.56
11	3,149	0	54.8	0.24	1.92	3.67	0	8.02
12	9,794	0	2,617.8	0.58	26.74	714.81	0	46.27

Figure 14.2 shows a histogram of the raw Au grades selected from within the 12 domains. The grade distribution is log normal with a long tail, indicative of a nugget effect. The majority of the grades are below 1 g/t Au, typical of the broader zones of low grade mineralization.

Figure 14.2 Histogram of Selected Raw Au Values inside Mineralized Domains



14.4.2 Compositing

Based on a review of the sampled intervals as shown in Figure 14.3, a composite length of one meter was chosen. The sample length median equals 1 m and the histogram below shows that 90% of all selected samples equal 1m. After selection and compositing the assay data to 1 m lengths the number of samples contained within the mineralized domains is 134,043.

Figure 14.3 Histogram of Sample Length

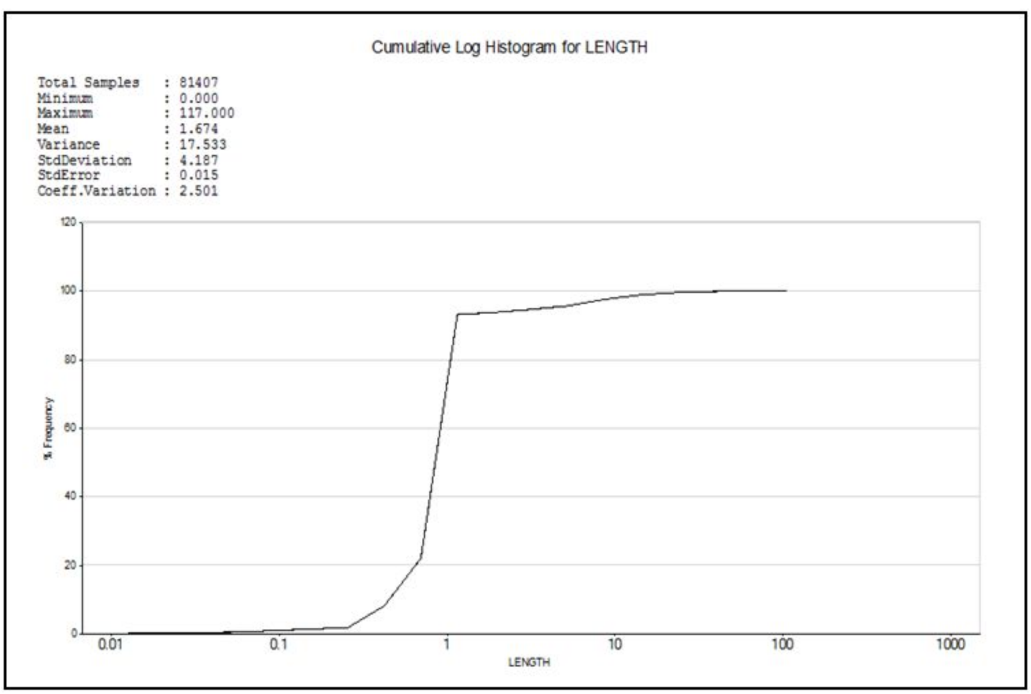


Table 14.8 Domain Statistics - Composite Data

Domain	Number	Minimum g/t Au	Maximum g/t Au	Mean g/t Au	Std Dev	Variance	Std Error	Coeff Var
1	87,814	0	1,449.3	0.65	9.23	85.14	0.00	14.19
2	5,738	0	187.5	0.08	2.59	6.71	0.00	34.22
3	3,067	0	3.0	0.02	0.13	0.02	0.00	6.29
4	611	0	37.3	0.50	2.22	4.94	0.00	4.49
5	1,667	0	170.9	0.30	4.41	19.42	0.00	14.62
6	428	0	7.3	0.09	0.51	0.26	0.00	5.48
7	244	0	8.0	0.25	0.69	0.47	0.00	2.79
8	5,299	0	1,258.3	0.64	18.16	329.92	0.00	28.46
9	958	0	15.0	0.08	0.56	0.31	0.00	7.09
10	7,763	0	1,575.7	0.34	17.92	321.02	0.00	52.89
11	5,557	0	33.1	0.12	0.99	0.99	0.00	8.62
12	14,897	0	1,308.9	0.24	10.85	117.75	0.00	45.13

Table 14.8 above shows that the mean grades for all of the domains have reduced, with a corresponding reduction in standard deviation and variance due to the smoothing effect of compositing.

14.4.3 Grade Capping

Capping of gold grades was studied in several ways. While there are undoubtedly some very high values in the dataset, consistent with the deposit being within the Red Lake camp, it was deemed important not to penalise the deposit but equally, not to overestimate the continuity or influence of the very high grades. Historically in the camp, a system termed 10-5-2 has been employed, which trims values over 10 oz per ton to 10 oz, between 5 and 10 to 5 oz, and between 2 and 5 to 2 oz. B sed on a log probability plot shown in Figure 14.4 the chosen upper capping threshold was 270g/t Au, which was applied after compositing of the selected drillhole data. Table 14.9 shows this only affected mean values for the four higher grade domains.

Figure 14.4 Log Probability Plot for g/t Au Weighted by Length

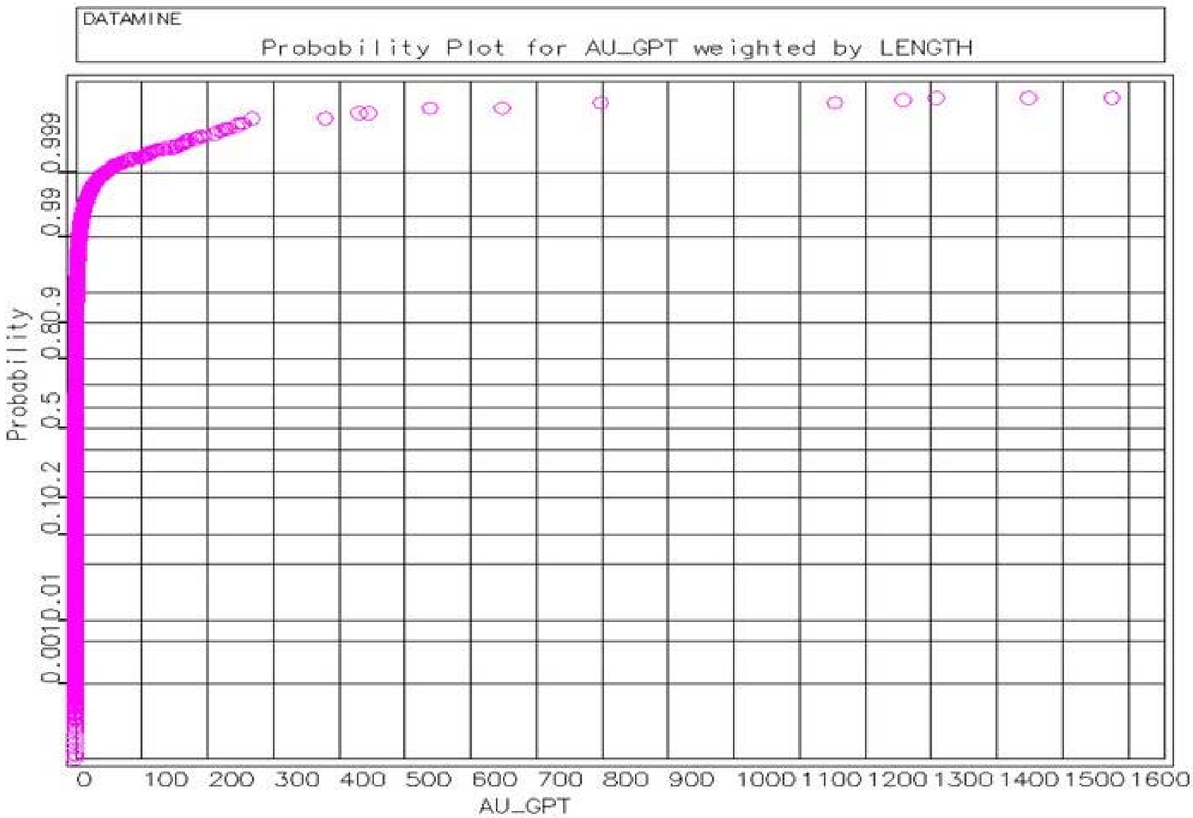


Table 14.9 Domain Statistics – Composite Data after Capping

Domain	Number	Minimum g/t Au	Maximum g/t Au	Mean g/t Au	Std Dev	Variance	Std Error	Coeff Var
1	87,814	0	270.0	0.61	5.31	28.23	0.00	8.79
2	5,738	0	187.5	0.08	2.59	6.71	0.00	34.22
3	3,067	0	3.0	0.02	0.13	0.02	0.00	6.29
4	611	0	37.3	0.50	2.22	4.94	0.00	4.49
5	1,667	0	170.9	0.30	4.41	19.42	0.00	14.62
6	428	0	7.3	0.09	0.51	0.26	0.00	5.48
7	244	0	8.0	0.25	0.69	0.47	0.00	2.79
8	5,299	0	270.0	0.45	6.71	45.06	0.00	14.86
9	958	0	15.0	0.08	0.56	0.31	0.00	7.09
10	7,763	0	270.0	0.17	3.26	10.63	0.00	19.11
11	5,557	0	33.1	0.12	0.99	0.99	0.00	8.62
12	14,897	0	270.0	0.17	2.77	7.66	0.00	16.21

14.5 Block Model

14.5.1 Block Model Parameters

Parent blocks of 2 m by 8 m by 12 m (vertical) were used in the block model, with sub blocking employed. Sub blocking used a split of x2, allowing a maximum block size of 192 m3 and a minimum block size of 0.012 m3. The parameters of the block model itself are shown in Table 14.10 and model fields used in reporting from the block model in Table 14.11. The model is un-rotated, being set up on the mine grid system, which is at +45o to the UTM system and roughly parallel to the main mineralization trend.

Table 14.10 Block Model Parameters

Item	X	Y	Z
Origin	9800	48740	3440
Parent cell size	2	8	12
Minimum cell size	0.5	2	3
Number of cells	460	242	167

Table 14.11 Block Model Fields

Field	Explanation
IJK	Identification number
XC	Centroid X coordinate
YC	Centroid Y coordinate
ZC	Centroid Z coordinate
XINC	Cell size on X
YINC	Cell size on Y
ZINC	Cell size on Z
XMORIG	X model origin
YMORIG	Y model origin
ZMORIG	Z model origin
NX	Number of cells in the X direction
NY	Number of cells in the Y direction
NZ	Number of cells in the Z direction
DOMAIN	Domain number
AU_ID3	Estimated grade of Au, g/t using Inverse Distance to the Power of 3
NSAMP	Number of samples
CLASS	Classification code : 2- Indicated; 3- Inferred
SVOL	Search Pass

14.5.2 Variography and Grade Estimation

Variography was carried out on the data selected from the principal zone (Domain 1) which incorporates the F2 Core and which has good density of data. These data were then used as an example to confirm the selected search distances. Variography was completed on the capped composite data. The variogram shown in Figure 14.5 is an across-strike variogram which demonstrates that the parameters selected are adequate to use in this instance. The variogram model shown in Figure 14.6 was used to define the search distances in orthogonal directions. Two different search directions were used for the modelling, determined from the gold grade spatial distribution. The gold grades contained within the upper part of the western side of the model have a shallower dip in the Y axis. Table 14.12 gives a breakdown of the search parameters for the two different ellipses and the domains to which they were applied. Figures 14.7 and 14.8 show the orientation of ellipses 1 and 2 in the Y plane with the drillholes clipped to within $^{+}/_{-}$ 35 m.

The chosen search parameters are shown in Table 14.12.

Figure 14.5 Omni-Directional Variogram for Zone 1

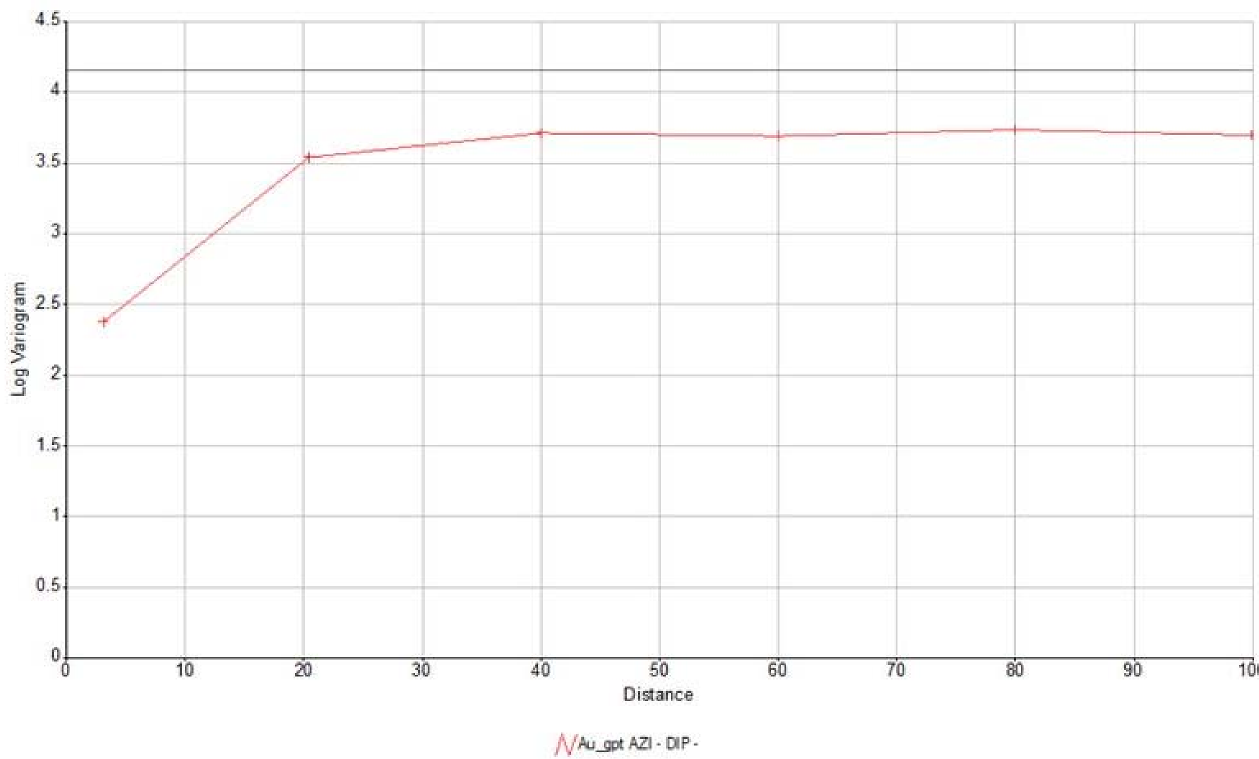


Figure 14.6 Variogram Model for Zone 1

Type	Variance	170/0	80/0	170/60
Nugget	1.7	-	-	-
Spherical	2.045	22	33	36

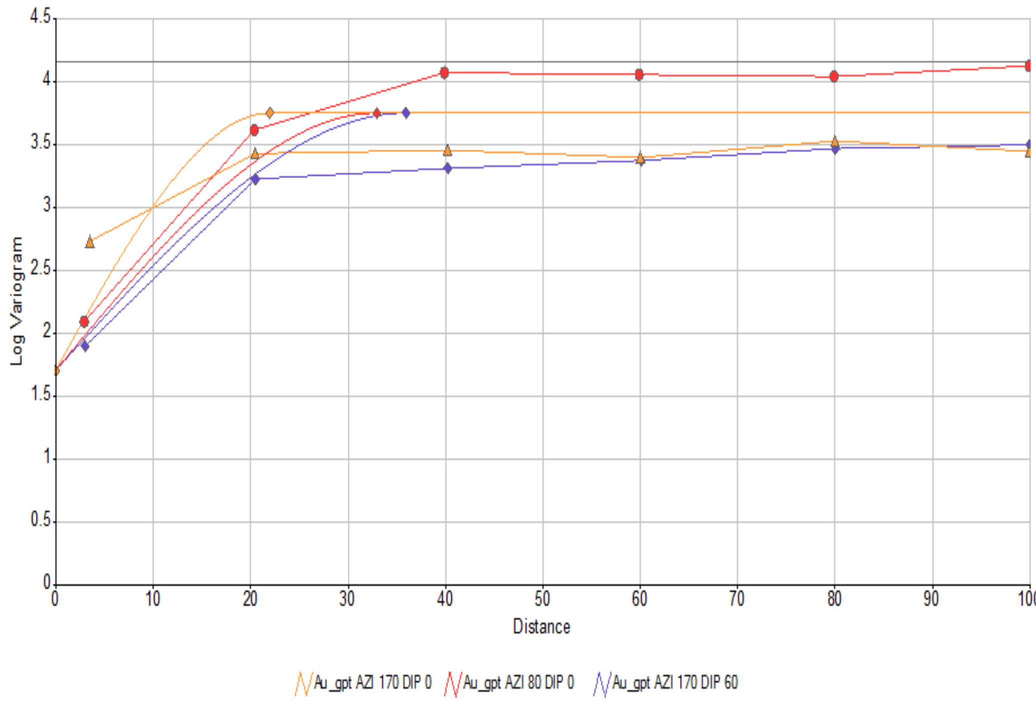
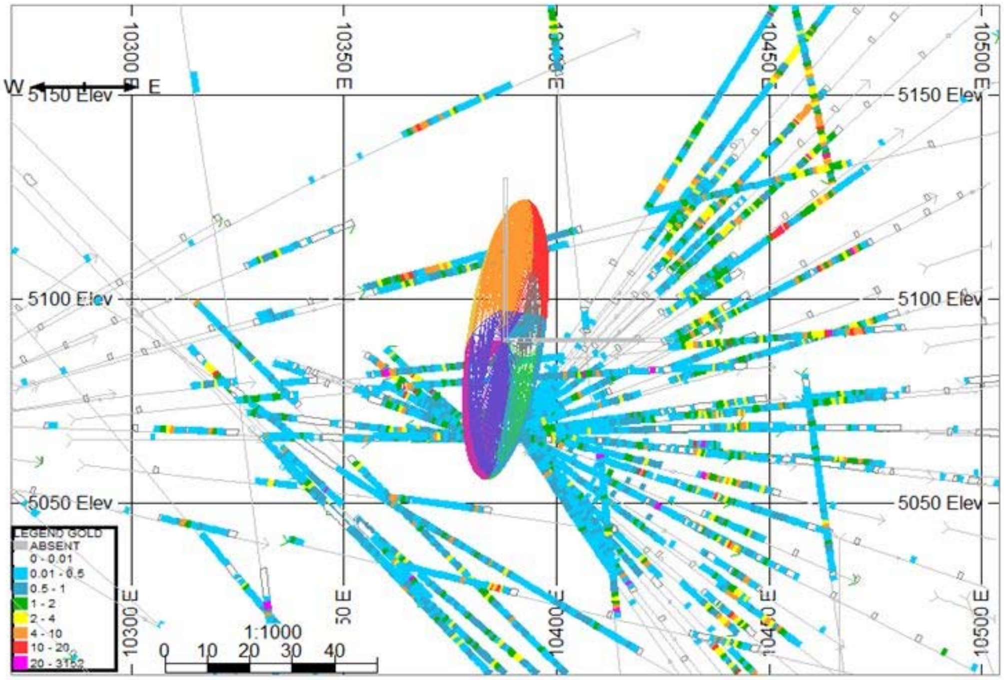


Table 14.12 Estimation Search Parameters

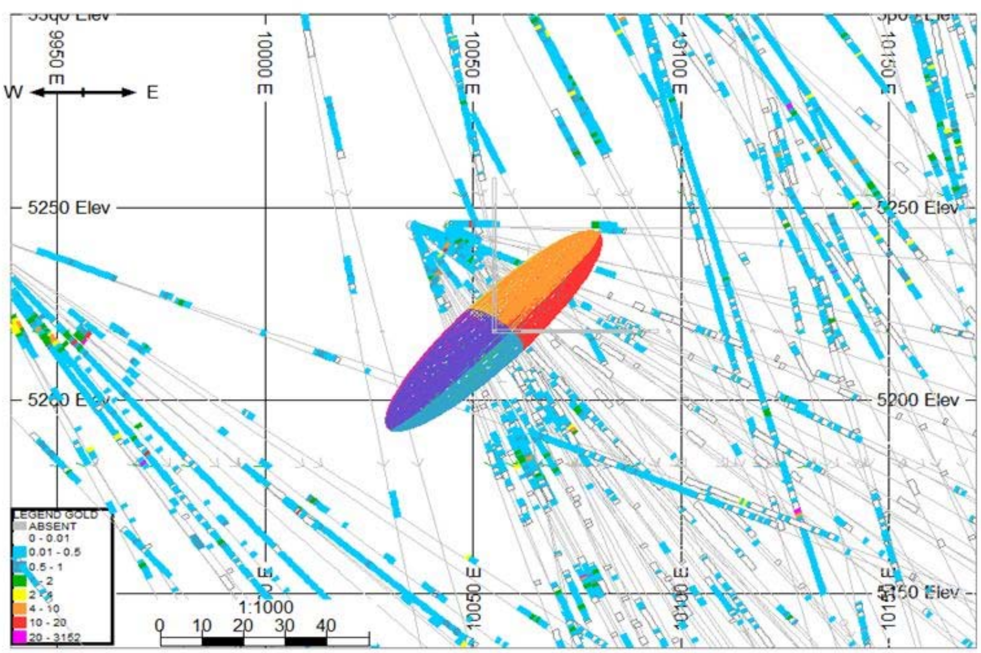
Search Ellipsoid	1	2
DOMAINS	1, 8, 9, 11, 12	2, 3, 4, 5, 6, 7, 10
Search distance on X Axis	8	8
Search distance on Y Axis	24	24
Search distance on Z Axis	36	36
Rotation angle around Axis Z	-10	-10
Rotation angle around Axis Y	-10	-45
Rotation angle around Axis X	20	0

Figure 14.7 W-E Section View of Search Ellipsoid 1



Note: North arrow is shown in mine grid coordinates
Drillholes clipped to +/- 35 m

Figure 14.8 W-E Section View of Search Ellipsoid 2



Note: North arrow is shown in mine grid coordinates
Drillholes clipped to +/- 35 m

The grade estimates reported use the inverse distance to the power of three algorithm (AU_ID3). Interpolation was performed in three passes, starting with a small ellipse and increasing the size with subsequent passes to fill blocks not already filled on a previous pass. Estimation allowed a minimum of 3 samples and a maximum of 10 samples to inform a block for the first pass and a minimum of 1 and maximum of 10 for the subsequent two passes. The model also utilized the Datamine “Max” key to limit the number of samples that inform a block to two per drillhole. Table 14.13 shows the block model configuration for each pass.

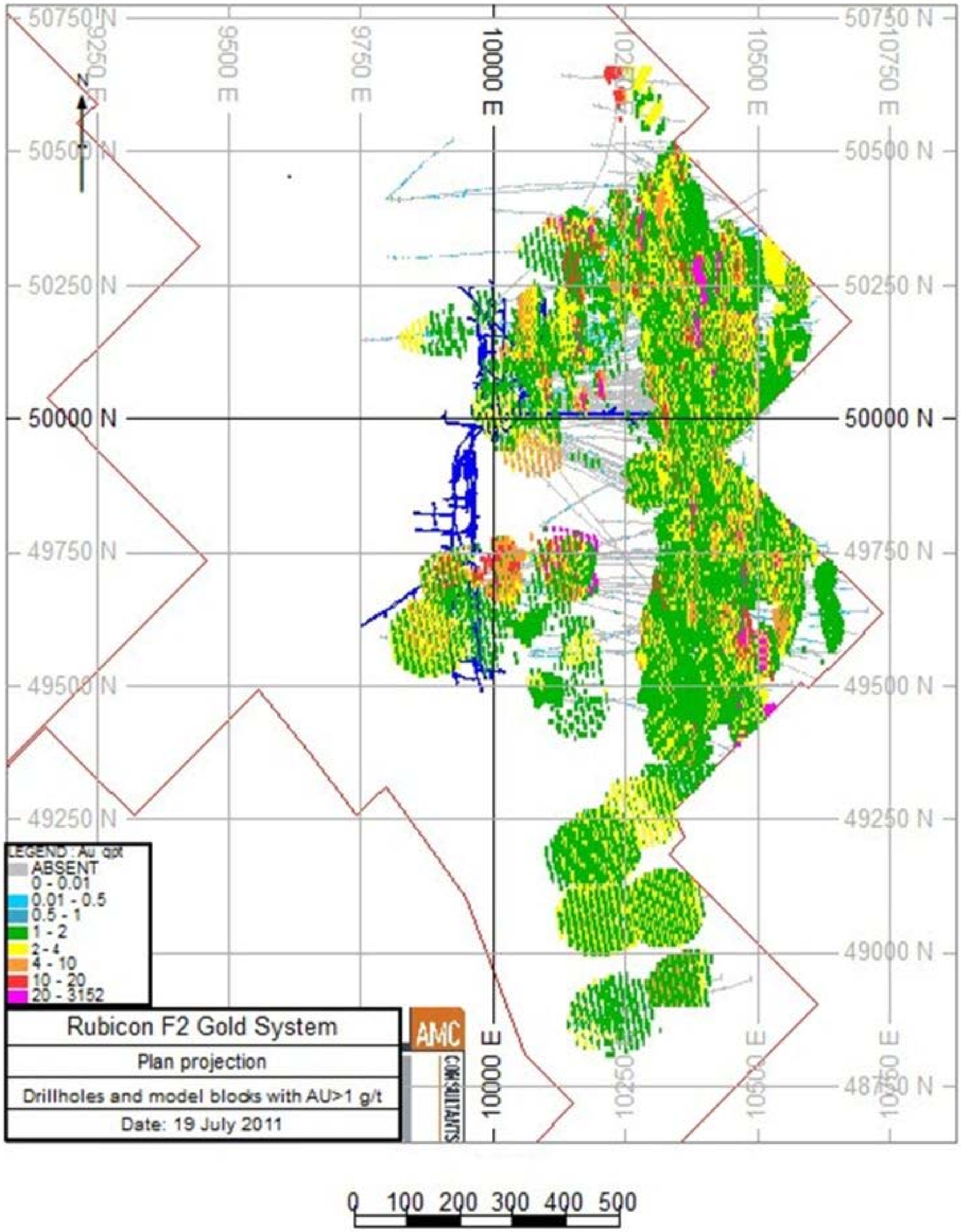
The block model was then trimmed to the bedrock surface and the property boundary.

Table 14.13 Search Parameters

Interpolation Method	Search #	ID3	Min/Max # samples per block	Max # samples per hole
ID^3	1	8 x 24 x 36 metres	3 and 10	2
ID^3	2	16 x 48 x 72 metres	1 and 10	2
ID^3	3	24 x 72x 108 metres	1 and 10	2

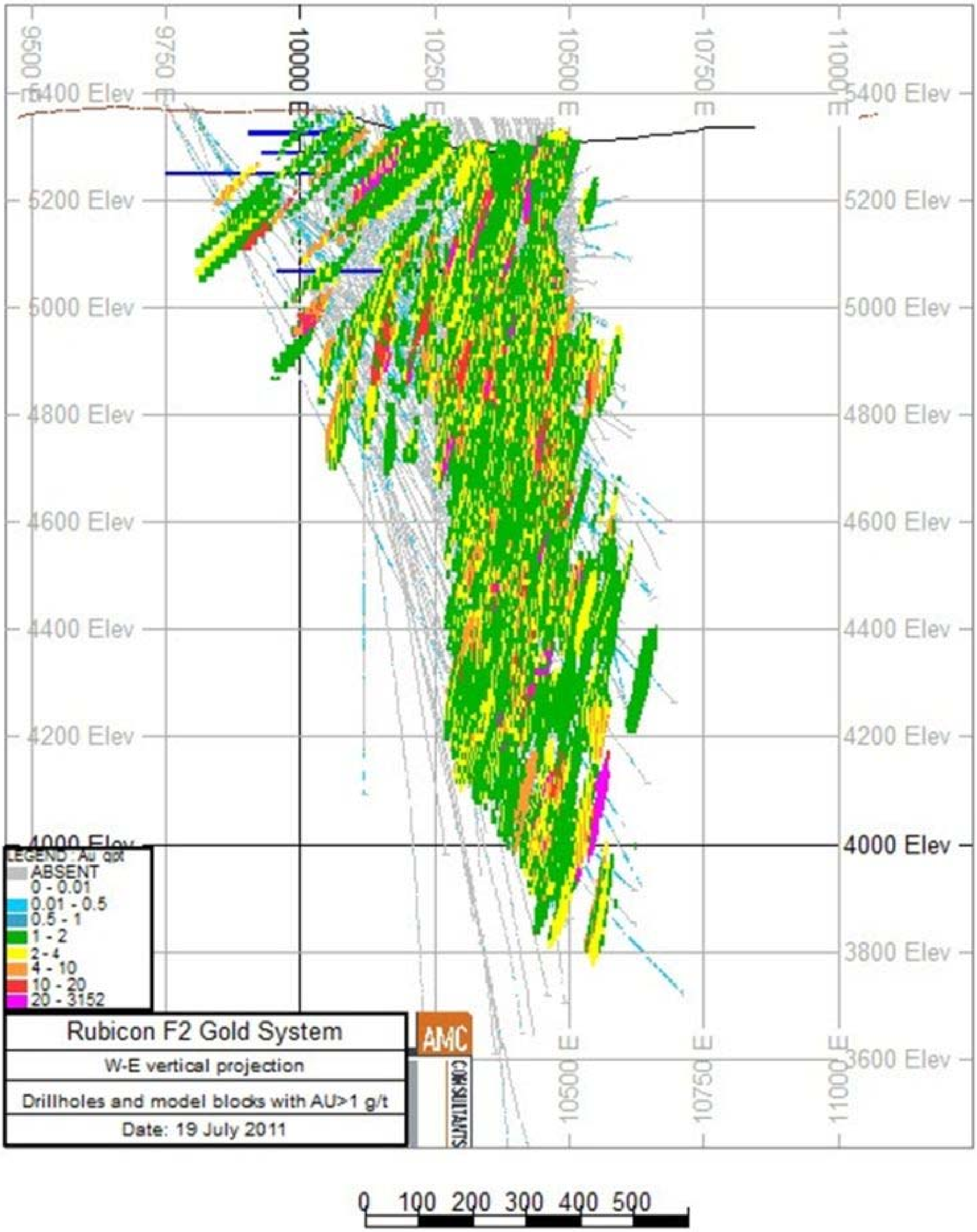
Figure 14.9 is a plan view of the complete block model filtered on grades greater than 1 g/t Au with the drillhole traces shown. The model has been clipped to the claim boundary and clearly shows the constraining effects of the wireframe boundaries of the mineralized domains. The mineralization trends within the domains are shown in a vertical projection in Figure 14.10 again filtered at 1 g/t Au. These reflect the same trends demonstrated in Figure 7.4 in Section 7.

Figure 14.9 Block Model Projected on Plan



Note: North arrow is shown in mine grid coordinates

Figure 14.10 Block Model Projected on Vertical Projection



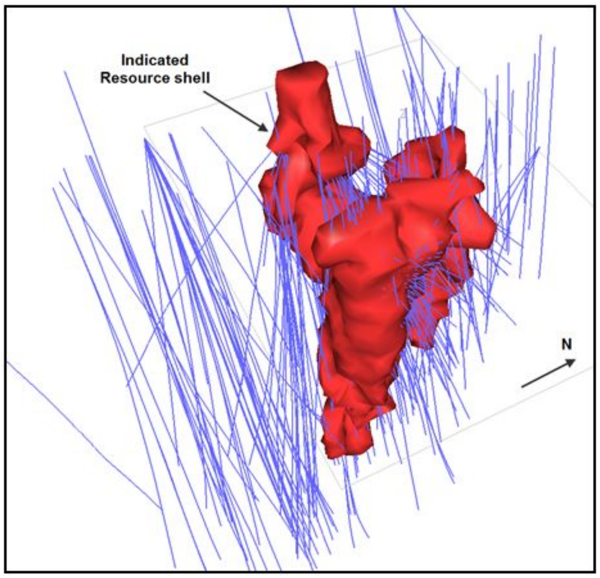
Note: North arrow is shown in mine grid coordinates

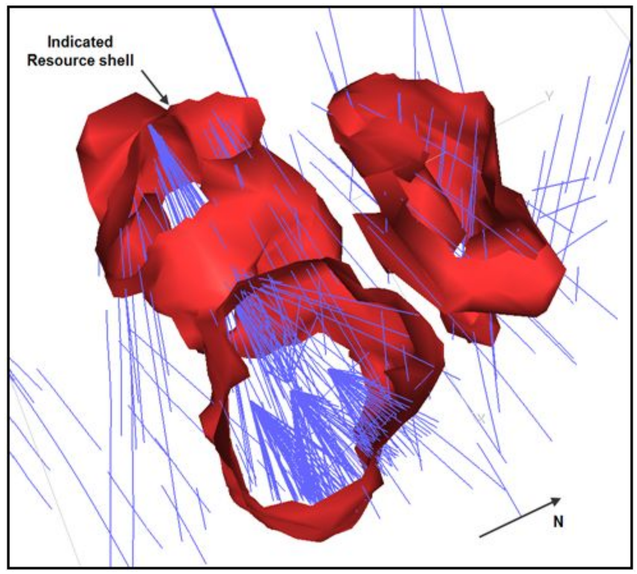
14.5.3 Resource Classification

Drillhole density was used by AMC to determine the volumes which can be considered an Indicated Mineral Resource.

A simplified search with a 30 m radius, which was based on a visual review of the drillhole density, was used to populate a 10 x 10 x 10 block model. The model was then flagged to identify blocks informed by more than 10 composites. These blocks were contoured, and the resulting bounding shells are shown below in Figure 14.11, in two isometric views.

Figure 14.11 Isometric Views of the Indicated Shell

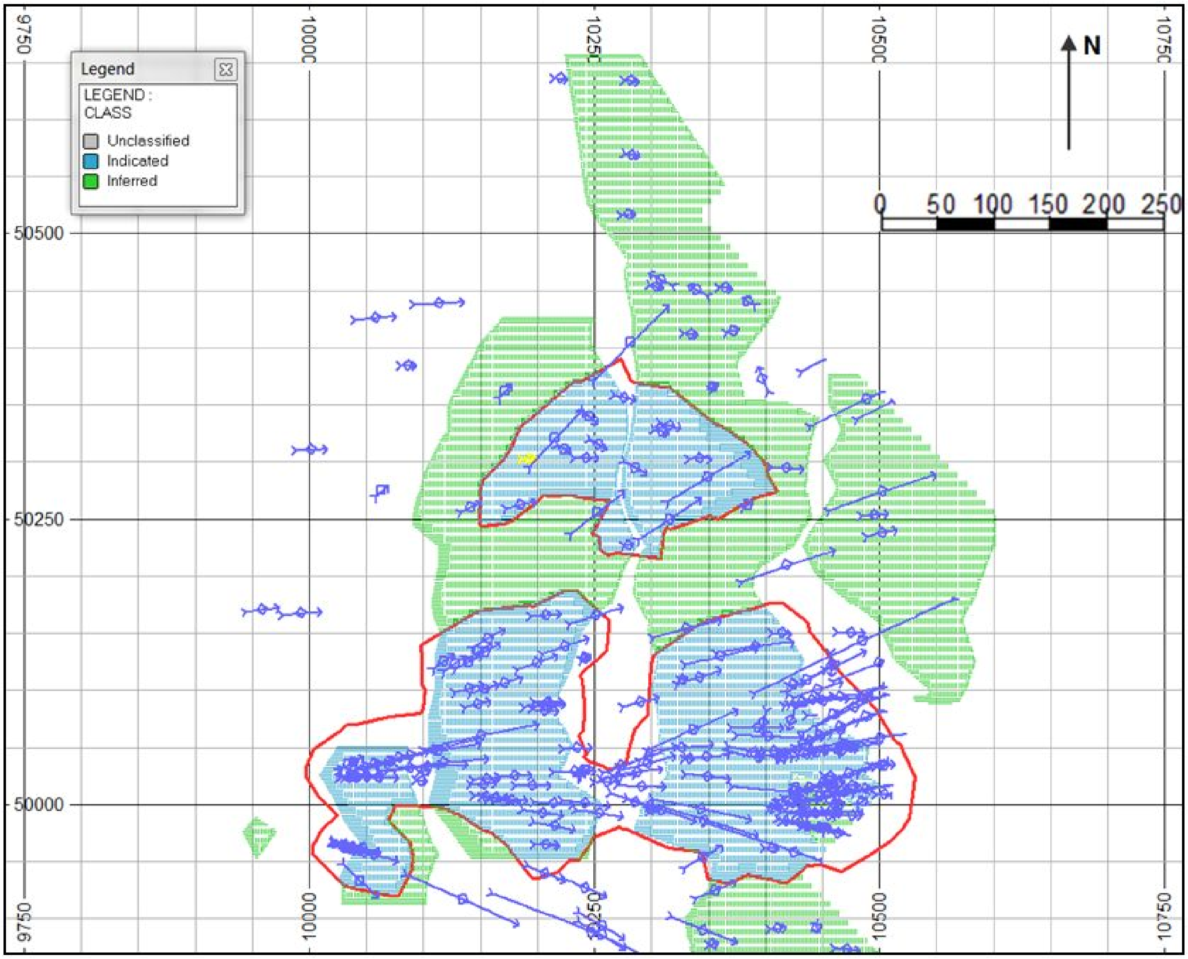




Note: North arrow is shown in mine grid coordinates

The blocks within the shell were reported as Indicated resources, with informed blocks outside the shell classified as Inferred resources. There are also uninformed blocks that have been removed from the representation in Figure 14.12 below, which is a plan view at 5000 m elevation.

Figure 14.12 Plan View of Classification



Note: North arrow is shown in mine grid coordinates

14.5.4 Block Model Validation

Section by section visual checks were carried out to ensure that the grades respected the raw data and also lay within the constraining wireframes. The composite drillhole data compares well with the estimated block grades, see Figures 14.13 and 14.14.

Figure 14.13 Plan View Comparing Drill Data and Model

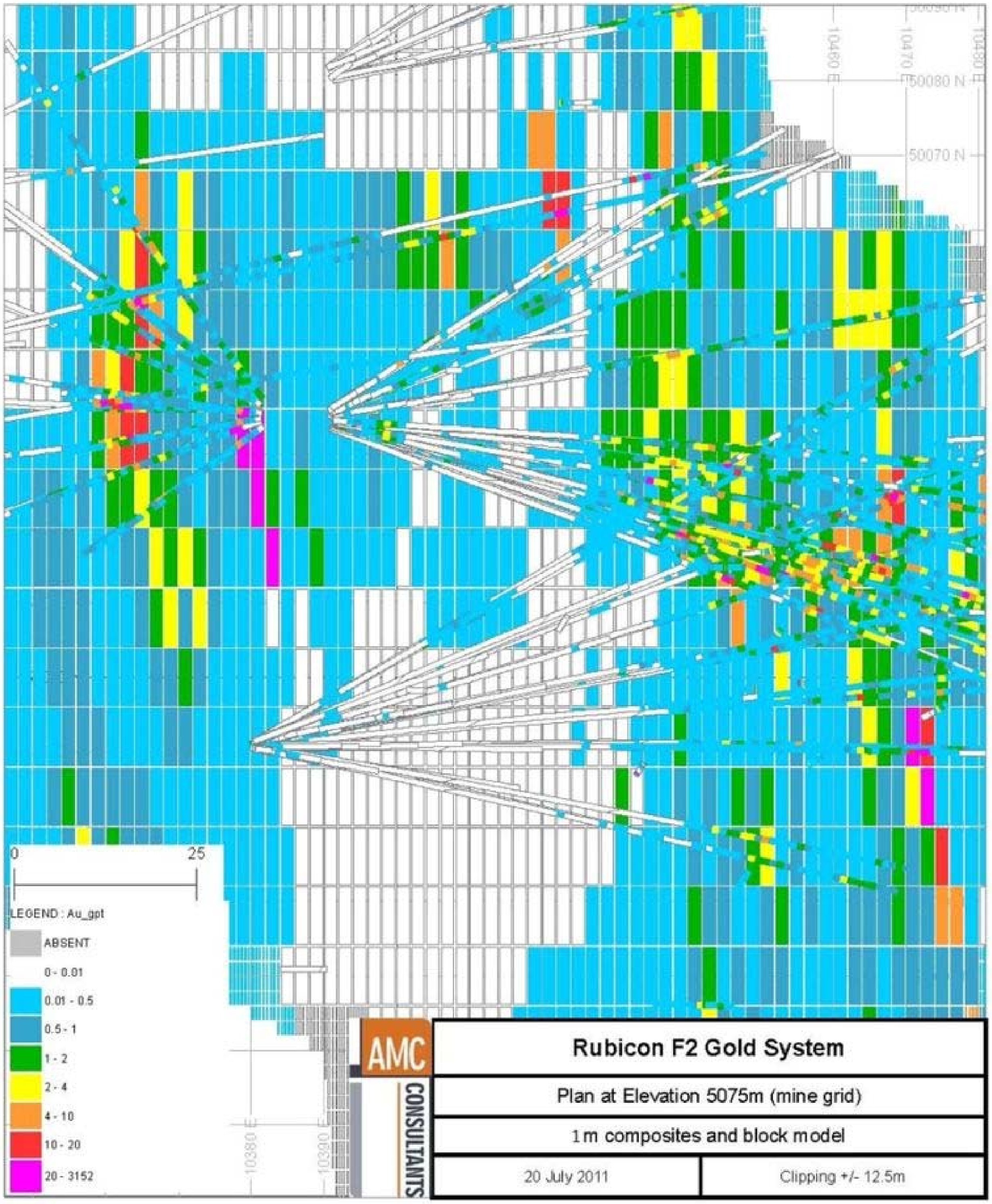
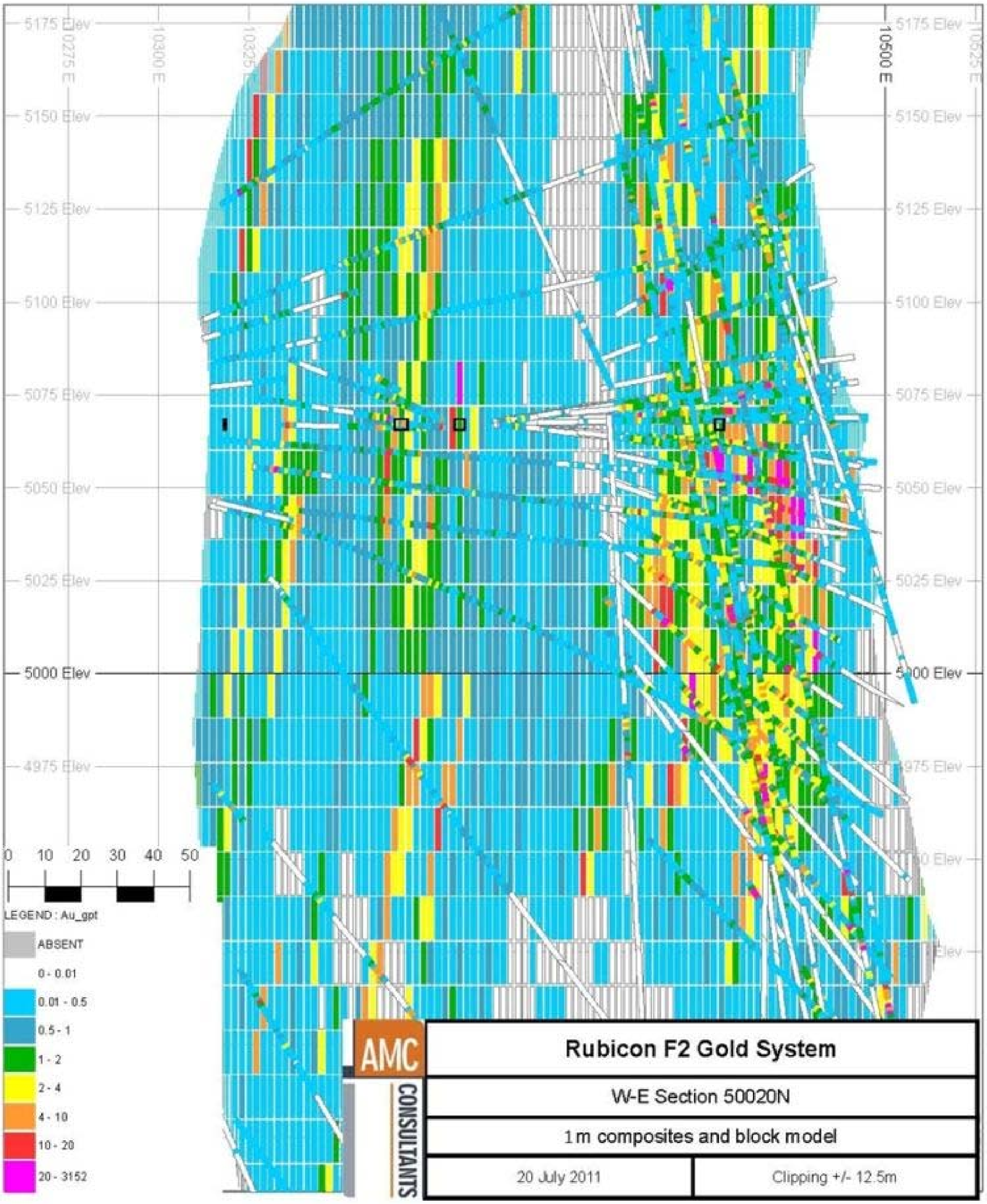


Figure 14.14 Vertical Section Comparing Drill Data and Model



In addition to the ID^3 estimate, both ID^2 and Nearest Neighbour (NN) runs were made in Datamine™. A comparison of the outputs, as well as the statistics for the raw data and composites, is shown in Table 14.13.

Table 14.14 Comparison of Composite Data to Outputs

Domain	Composite mean g/t Au	ID2 mean g/t Au	ID3 mean g/t Au	NN mean g/t Au
1	0.60	0.40	0.40	0.41
2	0.08	0.05	0.05	0.03
3	0.02	0.03	0.03	0.04
4	0.49	0.44	0.43	0.45
5	0.30	0.19	0.19	0.12
6	0.09	0.22	0.22	0.09
7	0.25	0.11	0.11	0.11
8	0.45	0.31	0.31	0.3
9	0.08	0.08	0.08	0.08
10	0.17	0.27	0.27	0.32
11	0.12	0.10	0.10	0.10
12	0.17	0.16	0.16	0.15

Note - the composites are not declustered.

An underground bulk sampling program was carried out by Rubicon in 2011 on the 305 m level. The results are shown in Table 14.15 below. The objective of the program was twofold, to confirm earlier metallurgical test work and to open up two zones and assess grade. The data in Table 14.15 have been provided by Rubicon.

Table 14.15 Comparison of Bulk Sampling to Drill Data

Item	WLB2 grade - g/t Au	F2 Core grade - g/t Au
Delineation Drilling Weighted Average	5.8	9.1
Milled Bulk Sample Testing Results	7.1	8.2

The weighted average figures from drilling are for broad zones from within which bulk samples were extracted. The bulk sample metallurgical results are from the testwork carried out by Soutex and described in Section 13.4.2.

14.6 Mineral Resource Estimate

Table 14.16 shows a summary of the mineral resource at a cut off of 5 g/t Au. This cut-off is the same as has been used previously by Rubicon, and AMC is satisfied that it is reasonable for the delineation of mineral resources, based on the grade used for mining cut-off in Section 16.

Table 14.16 Summary of Mineral Resource Estimates as of June 2011

Classification	M Tonnes	g/t Au	M oz Au
Indicated	1.028	14.5	0.477
Inferred	4.230	17.0	2.317

Notes: 1. CIM definitions were used for mineral resources
2. The cut off grade applied is 5.0 g/t Au
3. A capping value of 270g/t Au has been applied to the composites
4. Using drilling results to February 28, 2011

Inferred resources are too speculative to have economic considerations applied to them and there is no certainty that the inferred resources will be converted to measured and indicated resources.

If the data are not capped, the totals are 1.135 M tonnes at 17.2 g/t Au for 0.634 M oz for the Indicated category and 4.129 M tonnes at 21.2 g/t Au for 2.842 M oz for the Inferred category.

In Table 14.17, the totals are shown at a range of cut-offs with the resource estimate at the adopted cut-off grade emboldened. The same notes apply as above, except for Note 3.

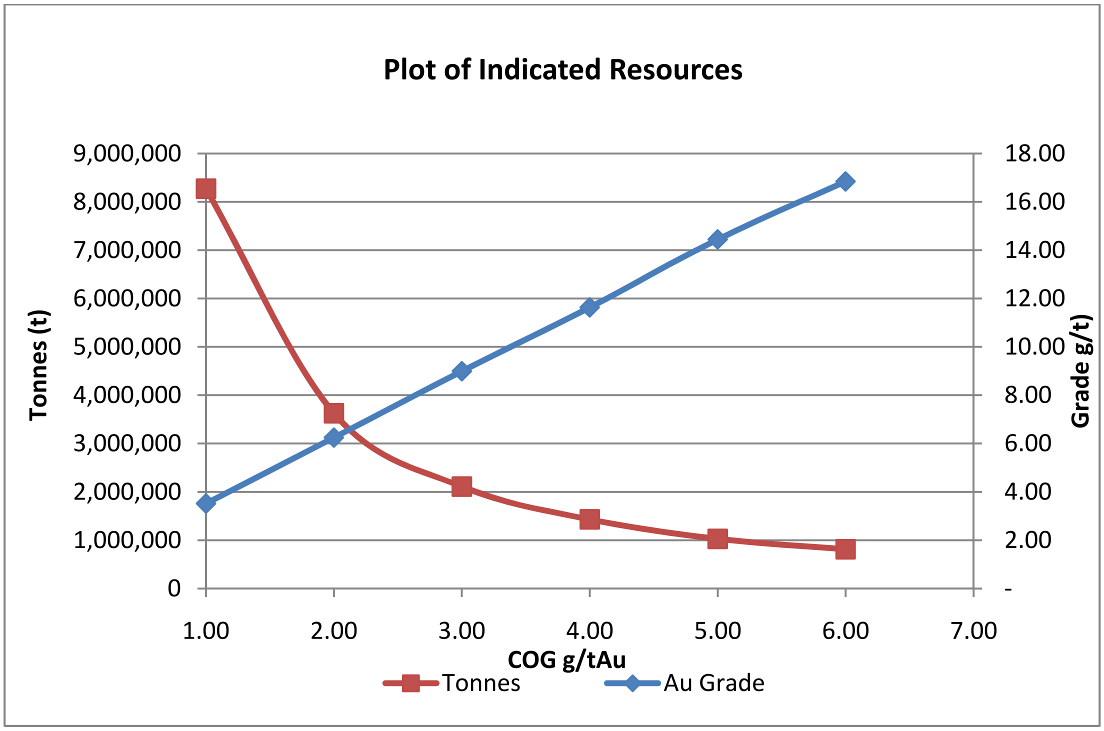
Table 14.12 Mineral Resource Estimates at a Range of Cut-off Grades

	Cut-off g/t Au	M Tonnes	g/t Au	M oz Au
Indicated	0.0	106.792	0.41	1.403
	1.0	8.274	3.52	0.936
	4.0	1.430	11.63	0.535
	4.5	1.192	13.11	0.502
	5.0	**1.028**	**14.45**	**0.477**
Inferred	0.0	727.279	0.28	6.661
	1.0	35.279	3.70	4.202
	4.0	5.674	13.83	2.523
	4.5	4.855	15.45	2.412
	5.0	**4.230**	**17.04**	**2.317**

In Section 16 the resource estimate is shown factored for conceptual mining and sliced in 61m vertical increments for the purpose of scheduling.

Grade-tonnage curves have been constructed for both the Indicated resource and for the Inferred resource. The profile is similar for both and the curve for the Indicated Resource is shown in Figure 14.15. This shows that grade is almost directly proportional to the cut-off grade (COG) applied, while tonnes have diminishing proportionality as COG is reduced.

Figure 14.15 Grade Tonnage Curve



14.7 Comparison with April 2011 Resource Estimate

Table 14.18 summarizes the GeoEx and AMC 2011 resource estimates. Note all the estimates presented are capped figures, with the GeoEx estimates capped using a 10-5-2 capping protocol, and the AMC estimate capped at 270 g/t Au.

Table 14.8 Comparison of April 2011 and AMC Estimates

Estimate	Classification	M Tonnes	g/t Au	M oz Au
AMC 2011	Indicated	1.028	14.5	0.447
(Block model)	Inferred	4.230	17.0	2.317
GeoEx polygonal	Inferred	5.500	17.3	3.057
GeoEx block model	Inferred	6.017	15.7	3.035

The AMC resource is not constrained at depth and is reported to the base of overburden, with no constraint or allowance for a crown pillar. The previous estimates were truncated at 1200 m below surface and were reported with a 25 m crown pillar removed from the total. In addition, AMC used different interpretations of mineralized domains and different search and interpolation parameters.

14.8 Potential Impacts on the Mineral Resource Estimate

AMC considers that the gold mineralization of the F2 Gold System is amenable to underground extraction, which is aided by proximity of existing infrastructure. Areas of uncertainty that may materially impact the mineral resource estimate are:

- Commodity price and exchange rate
- Ground conditions
- Mineral recovery
- Ease of locating higher grade areas and ability to mine with optimum dilution
- Environmental aspects of mining the resource are anticipated to be reasonably manageable and not material in terms of affecting project viability

15 MINERAL RESERVE ESTIMATES

There are no mineral reserve estimates to report for the Property.

16 MINING METHODS

16.1 Phoenix Site

16.1.1 McFinley Mine

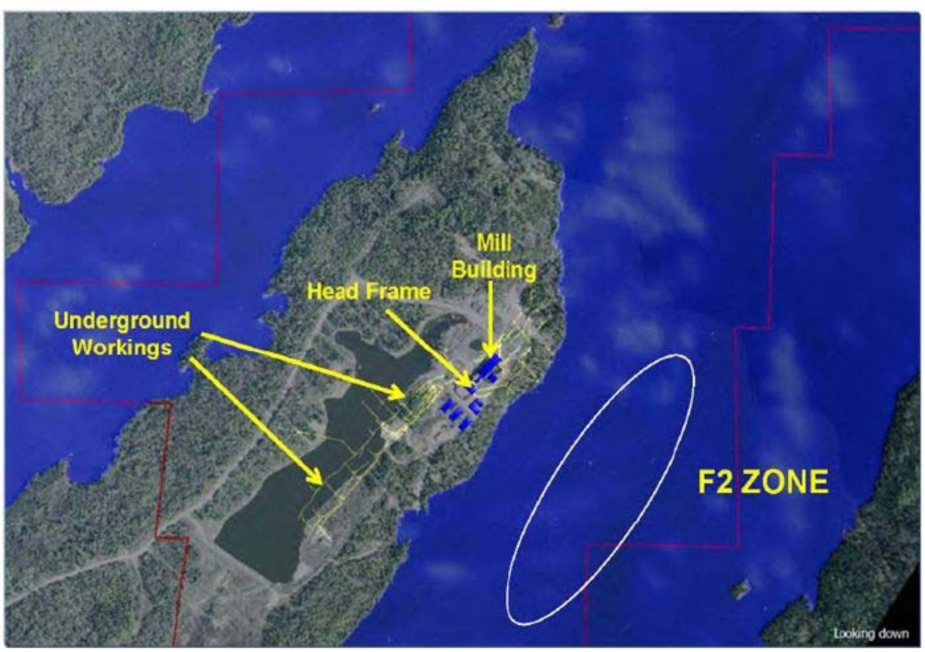
The Project site consists of the northern end of the McFinley Peninsula and adjacent areas covered by East Bay of Red Lake. The site incorporates the former McFinley Mine, which comprised underground mine workings on three levels (46L, 84L and 122L[1]), a shaft and hoist system, and other surface infrastructure. The McFinley hoist (1.5 m diameter, 147 KW, 25 mm ropes, 3 tonne payload) provided capability to hoist around 450 tpd at a depth of 305 m.

The McFinley Mine is described in more detail in the Rubicon 2009 Technical Report on the Phoenix Gold Project[2].

A 2008 AMEC Report for Rubicon entitled '*Review of Mine Workings, Former McFinley Red Lake Mines Ltd*, *2008*', presented the results of a review of the underground workings, including assessments of crown pillar stability associated with those workings.

The McFinley Mine workings are located almost entirely under the McFinley Peninsula. The F2 Zone, which is the focus of the Phoenix Project PEA, is located approximately 200 m to 400 m to the east, under East Bay of Red Lake. Figure 16.1 is a plan view illustrating the location of McFinley underground workings relative to surface infrastructure, Rubicon property boundaries and the general F2 Zone area.

Figure 16.1 Aerial View of Phoenix Project Site



[1] Mine level names refer to depth of level below shaft collar, in metres.
[2] Form 43-101F1, Technical Report, Exploration Activities of Rubicon Minerals Corporation on the Phoenix Gold Project, Red Lake, Ontario, 9 January 2009.

16.1.2 Rubicon Development

Rubicon has undertaken development work for the purpose of underground exploration. Initial work involved rehabilitating parts of the existing McFinley Mine workings, including the shaft, and establishing diamond drilling stations on 122L. Further work has included:

- deepening of the shaft to approximately 33 m below 305L
- lateral development on 305L, inclusive of a refuge station, for diamond drilling and to access F2 Zone mineralization for sampling purposes
- construction of a raise to service a shaft loading pocket below 305L
- construction of an egress raise from 305L to 122L (with access to a pre-existing raise to surface)
- construction of a waste pass raise from 305L to 244L
- more recently, development on 244L, again for diamond drilling and future access to F2 Zone mineralization

Rubicon has also recently begun construction for, and installation of, a new hoist and headframe with the capability to readily exceed the PEA envisaged steady state production hoisting rate. Further surface infrastructure is discussed in Section 18.

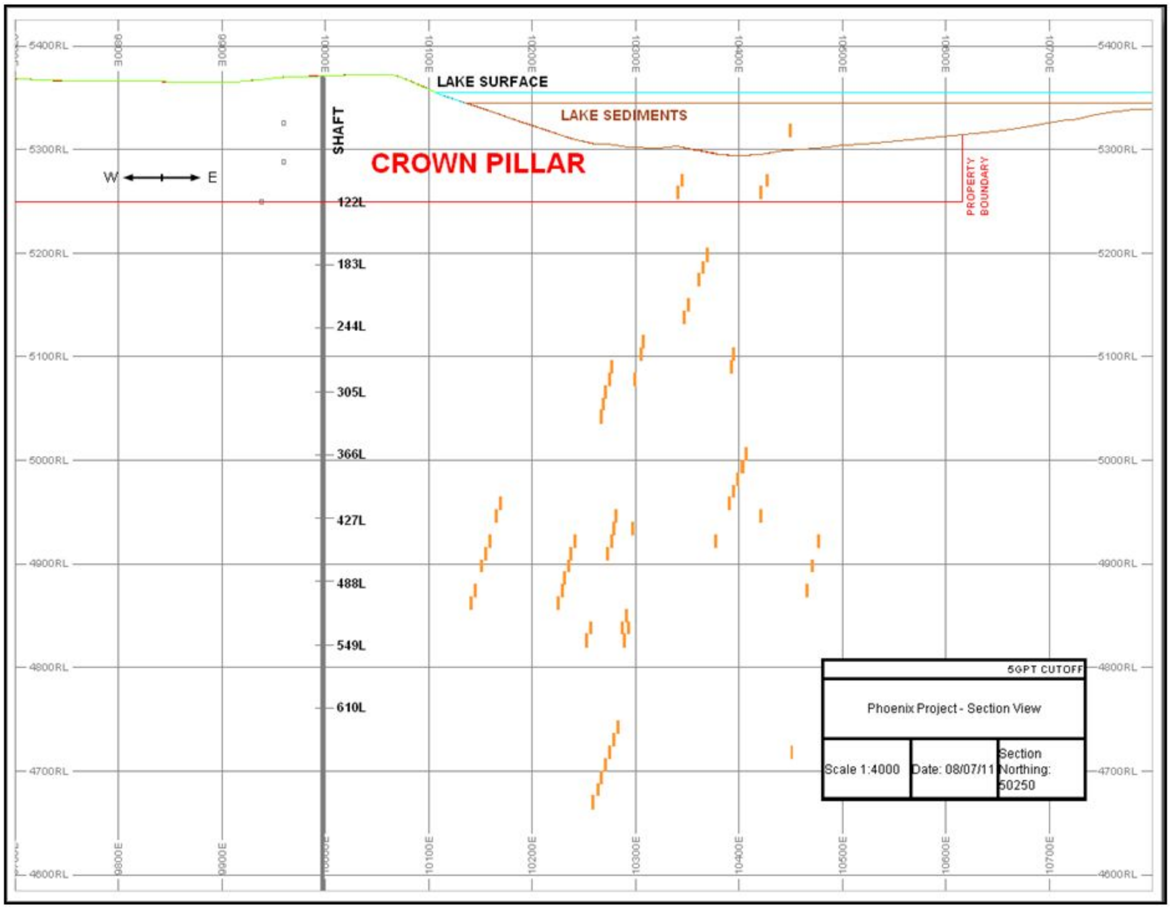
16.2 PEA Mine Design

16.2.1 Type of Mineral Resource and Initial Mining Concept

The F2 Zone is a north-trending, steeply dipping zone comprising numerous discontinuous shoots of high grade gold mineralization, hosted within ‘Hi-Ti’ Basalt and Felsic Intrusive lithologies within a larger body of ultramafic and mafic talc-rich lithologies. A further key aspect of the F2 Zone from the mining point of view is that it is largely located beneath the East Bay of Red Lake.

Consideration of the above has indicated that, not dissimilar to other mining areas in the Red Lake district, a selective, narrow excavation methodology may be applicable to the Phoenix resource, and that a stable crown pillar above the mine workings would be essential. Figure 16.2 is a cross section showing the block model interpretation of the mineralization, the lake and lake sediments, and a representative crown pillar constrained by the property boundary.

Figure 16.2 Section 5250N showing Resource Interpretation, Lake and Crown Pillar



16.2.2 Geotechnical Considerations

16.2.2.1 Underground Openings

AMC has observed ground conditions on an approximately 3-monthly basis over a period of two years at the Project. Inspections and audits have been made in some of the old workings, around shaft deepening operations, and during lateral and raise development and bulk sampling activities. Using these observations and in conjunction with Rubicon, provisional ground support standards for ongoing site activities have been developed. Direct underground assessment of the F2 Zone itself has been limited to the bulk sample areas on 305L. For the purposes of the PEA, AMC has assumed that ground conditions in and around the F2 Zone will be similar to those associated with current activities.

Mine development is expected to occur largely within three main lithology types: talc-rich ultramafic rocks, 'Hi-Ti' basalt, and felsic rocks. Stoping and stope development is expected to occur largely within 'Hi-Ti' basalt and felsic rocks.

The following comments relate to expected rock mass conditions based on observations made by AMC over the course of several visits to the project:

- Felsic and Basalt lithologies consist of generally good rock mass quality. Locally 'slabby' rock mass conditions exist in the back of drifts where shallow dipping joints are intersected. Locally 'blocky' rock mass conditions have been observed in drift intersections and pillar noses.
- Within the talc-rich ultramafics, rock mass quality ranges from fair to poor, with blocky to variably foliated rock mass conditions and, locally, zones of intense foliation possibly associated with faulting. These zones are approximately 1 m to 3 m in width and characterized by highly fractured and friable rock mass conditions, with talc and clay gouge infill minerals. Foliation is generally steeply dipping, striking approximately north-south (relative to mine grid). Significant over-break (approximately 1 m) of the backs has been experienced when drifting through these zones.

Current drift sizes at the Phoenix project are typically of the order of 2.7 m high x 2.4 m wide. In consideration of these opening sizes and those envisaged for the PEA mining scenario – namely widths typically ranging from 1.5 m to 3.0 m and cut heights of around 3 m - the AMC geotechnical work has confirmed that such openings should generally not pose any significant ground stability concerns as long as scaling and support standards (consisting of pattern bolting and screen) are followed and the paste fill envisaged for mined out areas is of specified quality and appropriately installed.

16.2.2.2 Geotechnical Investigation and Crown Pillar Design

AMC's geotechnical investigations have to date focussed on assessment of crown pillar requirements. The purpose of this work was to develop guidelines for crown pillar dimensions for input into preliminary mine design and ongoing mining studies, and to satisfy regulatory requirements stipulated under Part 3 of Ontario Regulation 240/00 – *Mine Development and Closure Plan* under Part VII of the Mining Act (the "Regulation"), for inclusion within Rubicon's Closure Plan submission for the Project. This included the following:

Compilation of relevant spatial information including:

- Surface topography
- Lake bathymetry above the envisaged mining area
- Geological interpretations including the basal contact of lake sediments, lithology contacts and faults
- Resource model
- Existing mine workings and potential mine designs
- Rock mass characterization using NGI-Q and RMR (as described by Hoek, Kaiser and Bawden, 1995) based on geotechnical logging of core from of 10 drillholes located in close proximity to the envisaged crown pillar. This includes consideration of ground water, weathering and in situ stress.
- A program of laboratory testing of rock properties, namely Unconfined Compressive Strength and Tensile Strength, of the primary rock types present within the crown pillar.
- Revision of an earlier, preliminary crown pillar assessment by AMC, which was based on an empirical method proposed by Carter and Miller (1995), using as a basis the rock mass characterization referenced above.

- Numerical modelling to estimate mining induced stresses and rock mass behaviour of the proposed crown pillar to verify the results of the empirical assessment.

The AMC geotechnical study work was specifically concerned with the crown pillar stability associated with mining undertaken and planned by Rubicon for the exploration and exploitation of the F2 Zone of mineralization. A study of crown pillar stability of the historical mine workings of the former McFinley Mine was undertaken by AMEC in 2008. To AMC's knowledge, the assessments and conclusions presented within AMEC's report remain current and valid.

AMC's assessment of crown pillar requirements considered a narrow cut and paste fill mining method and has resulted in a recommendation that a minimum crown pillar thickness of 45 m be maintained for the typical mining widths envisaged. The crown pillar thickness refers to thickness of 'bedrock' between the top mine level (projected at approximately 5249L) and the basal contact of the Lake Sediments.

Crown Pillar Risk Assessment

The purpose of the risk assessment was to assess the risk and consequences associated with crown pillar failure after mine closure. This included consideration of the following:

- Proposed mining method and stope geometries
- Proposed backfilling method using cemented fill
- Current and future land use designation
- Proximity of people and infrastructure to the site, including consideration of population density of the surrounding area and likelihood of public access to the site after mine closure
- Environmental impacts caused by a failure

AMC's risk assessment process was conducted using an industry standard approach based on ISO 31000:2009, which involved qualitative assessment of likelihood and consequences to determine the risk associated with a crown pillar failure.

It was determined that a crown pillar failure after mine closure was of low risk and consequence. This is largely due to the thickness of the crown pillar relative to the dimensions of the envisaged mine openings beneath, and the use of cemented backfill in all stope voids, which greatly reduces both the likelihood and potential consequences of crown pillar failure by eliminating the void into which the pillar can fail.

The crown pillar work described above satisfies the minimum requirements for a Geotechnical Study stipulated under Section 31 of the Ontario Regulation 240/00 for sites determined to be of low risk and consequence, and also satisfies the majority of additional requirements stipulated under Section 32 of the same Regulation for 'all other sites'.

The full rationale for the crown pillar assessment has been given in the AMC document entitled '*Geotechnical Study of Phoenix Project F2 Zone Crown Pillar*', dated February 2011.

Recommendations

AMC recommends that additional geotechnical investigation and analysis should be completed as part of any further study of the Project. This should include:

- Geotechnical interval and structural logging of drill core
- Geological mapping of underground development
- Development of a mine scale structural model
- Additional laboratory testing or rock properties
- Investigation of in-situ stress
- Assessment of appropriate mine design parameters
- Assessment of ground support requirements
- Input into mining sequencing

Reassessment of crown pillar requirements as a greater understanding is gained of the ore distribution and the impact that that may have on mining geometry and extraction sequence

AMC anticipates that the cost to complete this work would be of the order of $200,000.

16.2.3 Hydrogeological Considerations

To date, no ground water of sufficient quantity and flow to materially inhibit potential future mining has been observed underground at the Project.

AMC recommends that a hydrological assessment be undertaken as part of future project studies; the estimated cost is $50,000.

16.2.4 Mining Method and Mine Design

As described earlier, AMC has made the assumption that a narrow cut and fill mining approach would be applicable in the F2 Zone. The initial understanding of the mineralization as a lode-style deposit in a band of relatively steeply dipping structures has indicated that typical mining widths may be around 2 m. In order to minimize dilution, however, some stoping may be done at 1.5 m width or even less. Conversely, in areas where the lateral extent of the mineralization is greater, stoping widths of the order of 3 m or more may be appropriate. In consideration of the deposit nature, therefore, and also of the desire to have a simple, robust mining method for PEA analysis purposes, conventional, captive cut and fill has been adopted as the primary mining method, with paste fill being introduced after each cut in the mining sequence.

Stoping equipment envisaged is predominantly stopers, jacklegs, slushers and mucking machines, with ore dumped to an ore pass, loaded into track cars, passed to a shaft loading pocket via a grizzly/rockbreaker installation and then hoisted to surface. Waste development will also generally be done using stopers, jacklegs and mucking machines.

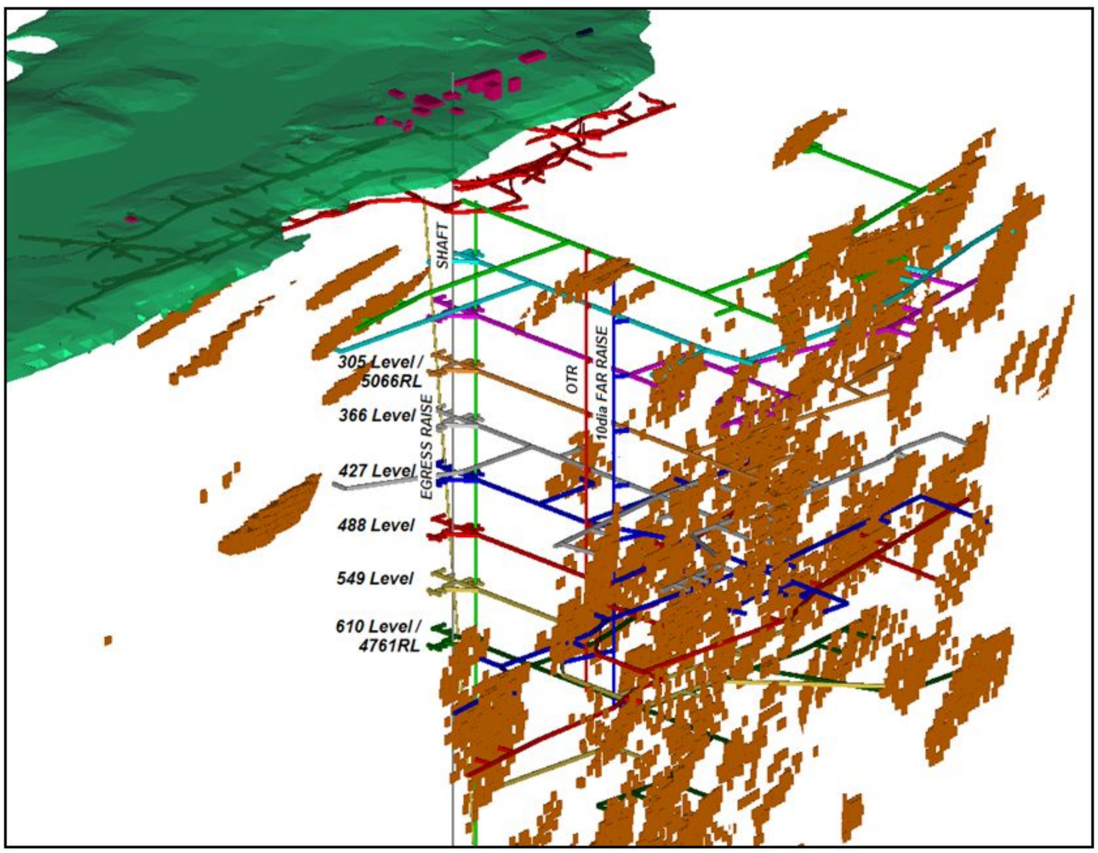
16.2.4.1 Mine Development

Cross-cut and access development from the shaft at nominal 2.4m width (W) x 2.7 m height (H) will be completed on all levels between 183L and 1403L. This development will be complementary to the present 305L development. Construction of a waste pass (1.8 m x 1.8 m), and a fresh-air raise (2.4 m x 2.4 m) equipped with a ladderway to serve also as a second egress, has been completed between 305L and the 122L. A 122L to surface airway raise was initially established as part of the McFinley operations. Similar waste pass and fresh-air raise arrangements will be carried down as additional mining horizons are developed at greater depth. Loading pockets for waste adjacent to the shaft will be established at appropriate intervals. As each level is being driven, necessary infrastructure will be established such as refuge stations, powder and cap magazines and sumps. Additional raising will consist of driving ore passes at 2.4 m x 2.4 m, these to be located at a point relatively central to the stoping areas. The required lateral and raising development is projected to continue over the first twelve years of the Project.

The shaft bottom is currently at around 30 m below 305L, with subsequent deepening of the shaft projected down to just below 1464L. For the initial mining stage and prior to completion of shaft sinking, it is anticipated that ramp development will be done to access the ore immediately below 305L, using a 2 yd scoop for mucking.

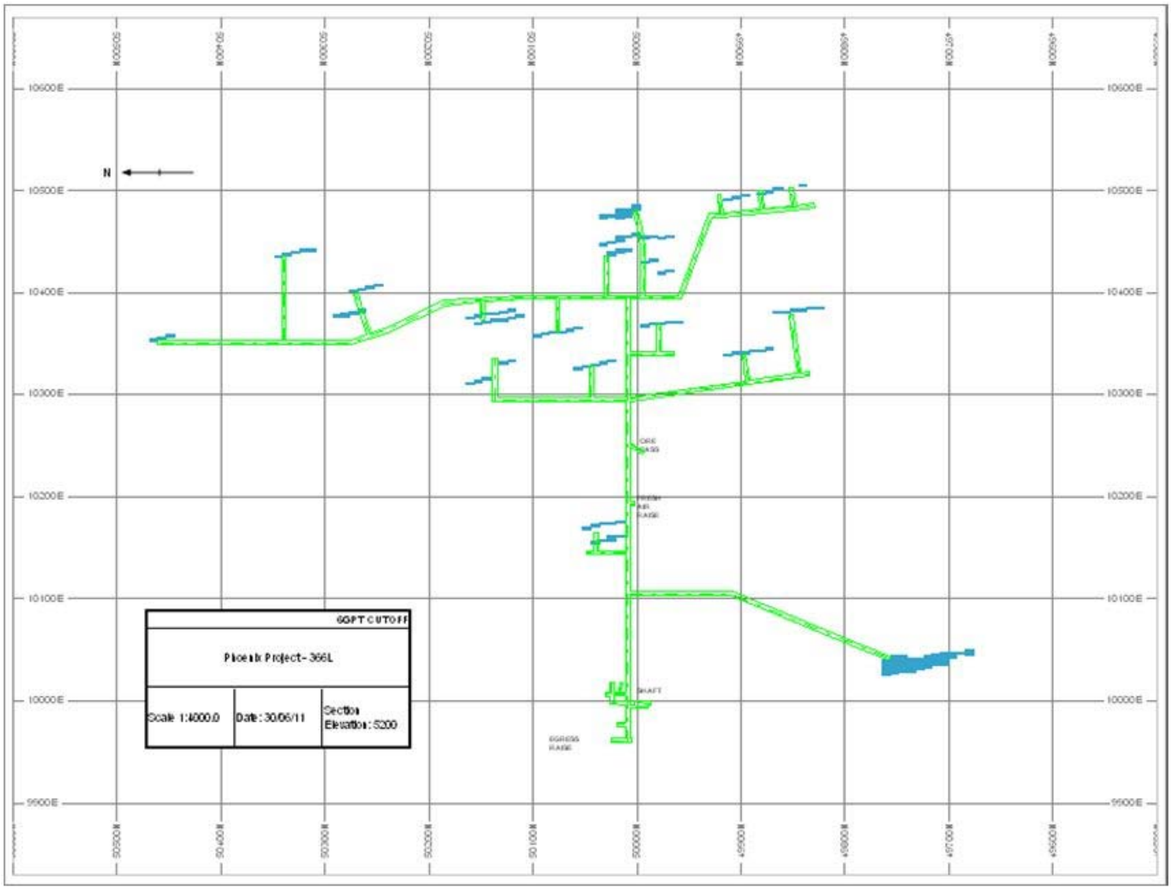
AMC has created preliminary designs for development on mining horizons at 61 m intervals between 5249 m elevation (122L) and 3907 m elevation (1464L). Figure 16.3 is an isometric view showing the block model resource interpretation and preliminary development design down to 610L.

Figure 16.3 Isometric View of Preliminary Mine Design



An example of a preliminary level layout plan is given in Figure 16.4. It shows the design for 366L, with areas coloured blue being mineralized blocks above a cut-off grade of 6 g/t at that elevation.

Figure 16.4 Preliminary 366L Layout



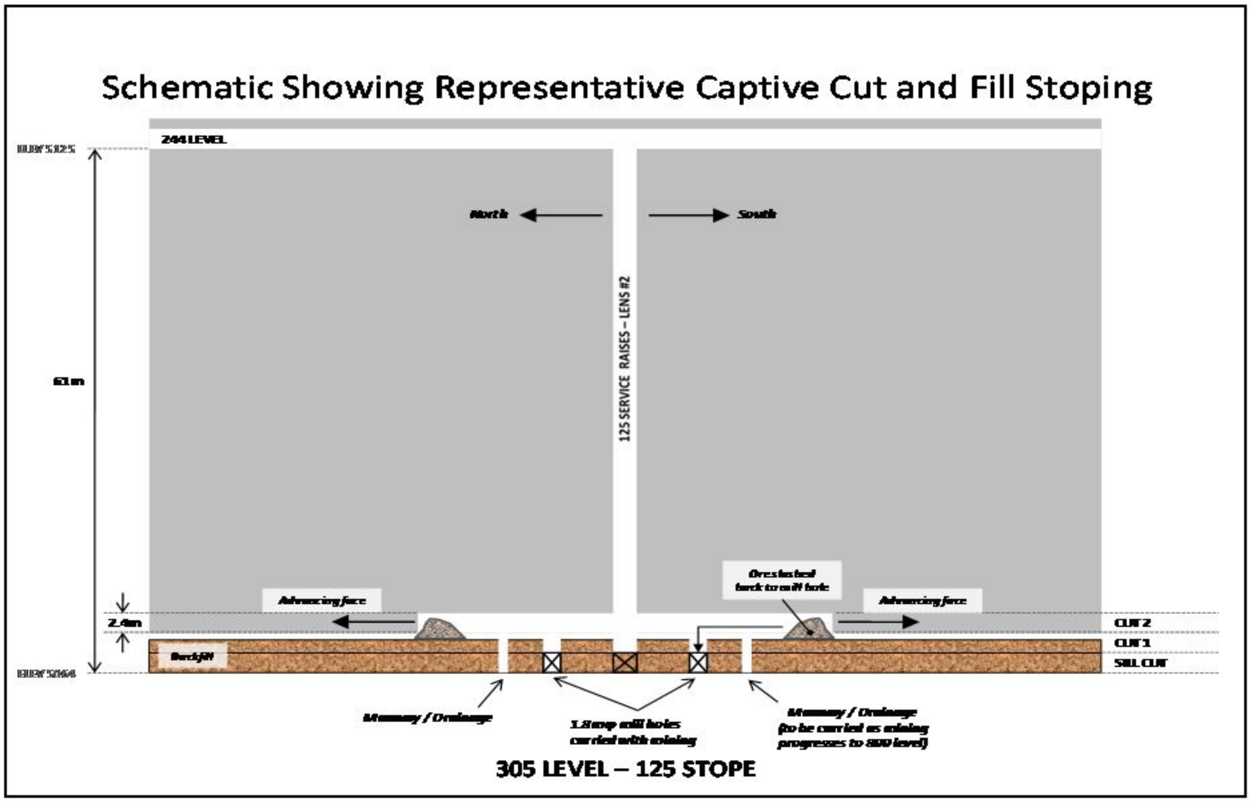
16.2.4.2 Stoping Process

Stopes mined from a particular horizon will be accessed through the access and cross-cut drifts established on that level. In the case of 366L depicted in Figure 15.4, and for other levels, some stope accesses will service a single stope whereas others may be used for two or more stopes. As indicated earlier, the PEA envisages driving mining cuts at 2.8 m high and at an average mining width of 2 m. Assuming that the ore is available for access at the level elevation, development and silling out of the first cut will take place prior to establishment of a service raise (generally one per 'lens'), which will be driven through to the level above. The second cut will then be breasted down, followed by the construction of the first lift of the mill hole and manway set-ups, the latter for both egress and drainage. Should the ore in any given stope be continuous through to the level above, then a total of 22 cuts is projected between levels, with cuts 2 through 22 being serviced from above via the service raise, while continued mill hole and manway access to the starting level below is maintained with each succeeding cut. After completion of each cut, filling with paste fill will take place after the manway and mill holes are raised to their required height.

The AMC resource modelling, together with an understanding of comparable mineralization at other sites, indicates that the resource may be discontinuous rather than fully continuous between levels, although continuity may exist for tens of metres. Variability in ore location requires flexibility in the mining approach and will necessitate a high and sustained degree of systematic effort to understand localized ore trends, including relatively closely spaced definition drilling. AMC has allowed for the mining cost and time uncertainty associated with ore location variability by assuming that approximately 50% additional working places and manpower will be required than for a situation where the ore positioning is more readily ascertained. AMC has also allowed for what it believes is an appropriate amount of definition drilling.

Figure 16.5 is a simplified depiction of the stoping process between levels.

Figure 16.5 Captive Cut and Fill Stoping Schematic



16.2.4.3 Muck Handling

As indicated previously, waste handling on the level will generally be via mucking machine into track cars, and thence via transfer raise and track cars to the appropriate loading pocket.

The same process will be used for ore removal. Ore will be slushed internally into the mill holes established for each stope, and then picked up at the bottom of the mill holes by mucking machine and moved via track cars to the ore transfer raise. These raises will be sited reasonable adjacent to the ore zone with a finger into the raise at each dump location. Loading pockets will be established adjacent to the shaft on the following levels – 305L, 485L, 732L and 1464L – with the ore on each of these horizons being again transferred to the loading pocket via mucking machine and track cars.

The headframe, hoist and skip arrangement currently being installed at the Project site is more than capable of hoisting production tonnages of the order of those envisaged in the PEA, with significant additional capacity if required.

16.2.5 Ventilation

Rubicon has provided AMC with a third-party report (‘*Final Ventilation System*’ by Fred Stockhaus, P. Eng.) that outlines ventilation requirements for shaft sinking to the 305L and reviews the main ventilation air volume capacities and fan horsepower requirements for mining rates of 750 tpd and, potentially, 1,500 tpd to an ultimate depth of 914 m. The report is based on an understanding that initial planned production will be around 750 tpd using narrow vein cut and fill mining with jackleg/stoper and mucking machine, and that 2 yd scoop trams and up to four diesel locomotives may be used for ore/waste transportation. A main fan delivery system at 100,000 cfm via a 250 HP motor is projected. AMC considers that the ventilation system proposed is appropriate for the mining methodology, equipment and production rates envisaged in the ventilation report, but that additional main and auxiliary ventilation capability would be required if significant development were to occur using diesel scoops. AMC does not consider that provision of adequate ventilation will be of material significance to the viability of the project but recommends that the ventilation study be updated during the next stage of the project. A ventilation system cost assessment is included in the PEA cost estimates.

16.2.6 Backfill

A third-party backfill study has been commissioned by Rubicon. The report of that study (‘Backfill Scoping Study, for the Phoenix Gold Project’ by Robert Currie, P. Eng) envisaged construction of a paste fill plant on surface adjacent to the existing Phoenix infrastructure. A subsequent paste plant assessment has determined that construction of a paste plant as a more integral part of the mill may be advantageous from a cost and efficiency point of view. Tailings from the mill would be used to generate a paste product that would be delivered to the underground mining horizons via borehole, and then fill pipe in the service raises to the stopes. AMC envisages that the cement proportion of the paste fill mix for a particular cut will vary as per the desired strength requirements for that cut. The conceptual mining schedule developed by AMC would necessitate mining up underneath fill on virtually all of the 22 levels between 122L and 1464L, meaning that the fill placed in the sill cuts on those horizons must have appropriate structural and stability characteristics. Above these sill cuts, little cement will be required in the fill in areas where cuts are single-pass width. In wider mineralization, cemented fill may be required if drifting is to take place alongside previously mined-out areas. An estimation of fill system capital and operating costs is included in the PEA along with mill costing.

16.2.7 Cut-off Grade

The cut-off grade (“COG”) used for the Mineral Resource Estimates was 5.0 g/t Au. AMC applied a cut-off grade of 6.0 g/t Au to the resource estimate model to generate a gross inventory for potential mining. The cut-off grade calculation was based on an initial operating cost estimate of $200 /tonne and a gold price of $1040 /oz. The operating cost estimate assumed a labour intensive, conventional cut and fill mining method in a lode-style deposit, as described above. Final cost estimation and use of a gold price of $1100 /oz for the PEA has confirmed that a mining cut-off grade of 6.0 g/t Au is reasonable at this stage of the Project.

16.2.8 Dilution

AMC has used the 2 m block dimension from the resource model as a basis for calculation of the potentially mineable resource, but has recognized that using this dimension necessarily means that there will already be 'dilution' within the block relative to any potential mining at less than the 2 m dimension. As indicated above, AMC has also recognized that mining at less than the 2 m dimension will probably be undertaken and achieved. For the PEA, however, and in recognition of the relatively high dilution rates that are generally experienced in practice in deposits similar to that of the Project, AMC has applied an additional average 'unplanned' dilution factor of 17.9 % at zero grade to the tonnages derived from the model. The unplanned dilution factor assumes average design stoping dimensions in practice at 2.0 m W x 2.8 m H with 0.1 m average circumferential dilution outside the design.

16.2.9 Calculation of Potential Mining Inventory

AMC has considered the mineral resources available for potential mining in the PEA to be those (model) blocks in the resource model above the 6.0 g/t COG and lying between 1464L and the base of the crown pillar (approximately at 122L). The available mineral resources are also constrained by the Rubicon property boundaries. To calculate the potential mining inventory tonnes and grade and to prepare for conceptual scheduling, the area between 122L and 1464L was split into 61 m thick mining blocks, the height of the block being the floor-to-floor distance between successive levels. Examination of the resource block model grades for the respective 61 m thick blocks shows the bottom three horizons having grades between two and three times that of the average model grade. The lowest grade of the bottom three blocks is also over 80% higher than any individual grade above, with one exception, that of the fourth block from the bottom, where the differential is 67%. AMC also notes that the bottom five blocks, while containing a resource that merits an Inferred category, are poorly supported in terms of drillhole data relative to the areas above. After further examining the potential impact on economic projections for the PEA, AMC has determined that a reasonably prudent approach for the bottom five horizons is to apply to them the average model grade of the 22 blocks as a whole, while maintaining model tonnages. For potential mining purposes and as described above, AMC has also applied an average unplanned dilution factor of 17.9% at zero grade to the model tonnes and, for scheduling, a 95% mining recovery factor. Table 16.1 shows the individual conceptual mining blocks with model, diluted and projected mined tonnes, diluted grade (bottom five blocks adjusted) and projected Au ounces mined. AMC recommends that a more detailed investigation and assessment of the mineralization in the bottom five horizons be part of future work on the Project.

Table 16.1 Potential Mining Blocks and Potential Mining Inventory

					Potential Mining Inventory		
Level	Block Elevations		Model Tonnes at 6 g/t COG	Tonnes @ 17.9% Unplanned Dilution	Diluted Grade g/t* Au	Tonnes @ 95% Recovery	Potential Au Oz Mined
183	5188	5249	237,566	280,090	16.05	266,086	137,284
244	5127	5188	211,612	249,491	11.54	237,016	87,901
305	5066	5127	174,678	205,945	14.44	195,648	90,859
366	5005	5066	196,542	231,723	11.82	220,137	83,682
427	4944	5005	481,044	567,151	11.44	538,793	198,204
488	4883	4944	315,786	372,312	15.52	353,696	176,506
549	4822	4883	283,051	333,717	14.00	317,031	142,734
610	4761	4822	162,313	191,367	8.47	181,799	49,526
671	4700	4761	240,127	283,110	10.87	268,954	93,952
732	4639	4700	176,843	208,498	15.72	198,073	100,087
793	4578	4639	78,612	92,684	10.13	88,049	28,669
854	4517	4578	102,872	121,286	14.43	115,222	53,446
915	4456	4517	194,666	229,511	16.50	218,036	115,644
976	4395	4456	189,712	223,670	14.44	212,487	98,679
1037	4334	4395	115,819	136,551	11.95	129,723	49,843
1098	4273	4334	103,832	122,418	16.65	116,297	62,254
1159	4212	4273	129,069	152,172	17.47	144,564	81,209
1220	4151	4212	120,499	142,068	15.85	134,965	68,793
1281	4090	4151	172,244	203,076	15.85	192,922	98,334
1342	4029	4090	131,164	154,642	15.85	146,910	74,882
1403	3968	4029	122,699	144,662	15.85	137,429	70,049
1464	3907	3968	76,560	90,264	15.85	85,751	43,708
	Totals		**4,017,310**	**4,736,408**	**13.87**	**4,499,588**	**2,006,244**

* Bottom 5 levels grade adjusted

Table 16.2 shows the process for arriving at the potential mining inventory.

Table 16.2 Process to arrive at Potential Mining Inventory

Mineral Resource Estimate at 5 g/t Au COG	**Tonnes**	**Au g/t**	**Au Oz**
Indicated	1,027,908	14.45	477,448
Inferred	4,229,936	17.04	2,316,883
Potential Mining Inventory	**Tonnes**	**Au g/t**	**Au Oz**
Gross Potential Mining Inventory @ 6 g/t Au COG	4,304,049	18.93	2,619,678
Gross Potential Mining Inventory @ 6 g/t Au COG with grade adjustment for bottom 5 levels	4,304,049	16.75	2,317,523
Crown pillar (above 5249m el)	245,112	24.45	192,679
Below conceptual mining base at 3907m el	41,627	9.72	13,009
Net Potential Mining Inventory before unplanned dilution.	4,017,310	16.35	2,111,836
Unplanned Dilution at 17.9%	4,736,408	13.87	2,111,836
Potential Mining Inventory for schedule @ 95% recovery	**4,499,588**	**13.87**	**2,006,244**

16.3 Production and Scheduling

16.3.1 Production Rate

The Project aims to have processing capability in place and sufficient access and ore development completed to mine and process 270,000 t in the third year after commencement of the project work envisaged in the PEA. This initial production rate (750 tpd) is projected to be increased to around 970 tpd in the fourth year and to reach a steady state level of 1250 tpd in the fifth year. Key elements in reaching the desired rate of production include tonnes per blast, number of blasts per crew per day, number of available faces to blast, and capability of the ore movement system.

Assuming the average stope blast advance is 2.4 m, the effective face dimensions, including dilution are 2.2 m W x 3.0 m H, and in-situ ore specific gravity is 2.9 t/m^3, an average blast will produce around 46 t. Thus, the capability to blast at least 16 rounds per day would be required for a consistent 750 tpd scenario, and at least 27 rounds per day for the 1250 tpd scenario. AMC has assumed a two shifts/day system with up to five production miners per horizon on each shift and developed a mining sequence that has up to 21 available faces at any time, again per horizon. In order to maintain a steady state production rate around 1250 tpd, concurrent mining on six horizons will be required.

The manpower profiles developed for the desired production rate are detailed below in Section 16.4. The productivity requirement is generally of the order of 1.2 linear metres per crew member per day.

16.3.2 Production and Development Schedule

A 14-year project life is envisaged for the resources scheduled in the PEA, with the first two years being devoted to key surface and underground infrastructure construction, including mill and paste plant, shaft sinking, lateral and raise development, etc. Ore production and processing would begin in Year 3 at a rate of 750 tpd, increasing to steady state production in Year 5 at 1250 tpd.

Table 16.3 shows the underground development schedule over the projected LOM.

Table 16.3 Projected Development Schedule

Waste Development	Units	Totals	Y01	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13
Drifting (C)	m	27,615	4,014	4,811	2,692	2,598	1,341	1,801	2,598	2,440	1,076	2,598	1,547	98		
	t	637,492	92,675	111,067	62,137	59,983	30,946	41,585	59,983	56,321	24,837	59,983	35,702	2,274		
Raising	m	1,464								1,037	61	245		121		
Raisebore (C)	t	18,876								13,370	786	3,159		1,560		
Raising	m	3,549	683	653		186		1,234		793						
Alimak (C)	t	58,260	11,212	10,720		3,053		20,257		13,018						
Shaft (C)	m	1,129	150	247			732									
	t	57,918	7,695	12,671			37,552									
Drifting (O)	m	9,115	1,150	1,305	1,347	858	442	595	858	805	355	858	510	33		
	t	210,420	26,536	30,120	31,103	19,799	10,214	13,726	19,799	18,590	8,198	19,799	11,784	750		
Raising (O)	m	5,221	75	852	752	397	550	275	366	488	360	457	397	252		
	t	120,515	1,731	19,668	17,348	9,165	12,697	6,348	8,449	11,265	8,311	10,550	9,165	5,817		
Total Metres		48,093	6,072	7,868	4,791	4,039	3,065	3,905	3,822	5,563	1,852	4,158	2,454	504		
Total Tonnes		1,103,481	139,849	184,247	110,589	92,000	91,409	81,917	88,231	112,564	42,132	93,490	56,651	10,402		

NB. (C) = Capital, (O) = Operating

Table 16.4 shows the mine production schedule over the projected LOM.

Production in the first three years of operation is projected to come very largely from the 183, 244, 305 and 366 levels. Initial deepening of the shaft down to 732L in the two year pre-production period to allow access development, further definition of the resource and production down to that horizon will be followed by 732 m of additional shaft sinking in the third year of production (Project Y05). The latter shaft sinking will provide access to the approximate base of the modelled resource at 1464L. Total production over the LOM is projected at 4.5 Mt, with the steady state years (Y05 to Y11) showing 450,000 tpa (1250 tpd). As indicated above in Section 16.2.8, AMC has allowed for 17.9% external dilution at zero grade beyond the average design dimensions (2.0 m W x 2.8 m H), along with a 95% mining recovery factor.

Lateral capital waste development averages about 2300m pa over the first 12 project years, with Y01 and Y02 each being, not unexpectedly, significantly higher than the average at over 4,000 m pa. Lateral operating waste development averages 760m pa over the same 12-year period, with the average for the first three years being 1267 m. Raisebore and Alimak raising for ventilation, egress and ore and waste movement averages 418 m pa through years Y01 to Y12.

It should be noted that there is no development scheduled in Years Y13 and Y14. This is a reflection of the extent of the resource as currently projected and as scheduled in the projected LOM. Should additional resource be available then additional development would be required.

Table 16.4 Projected Production Schedule

Level	Prodn.	Totals	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14
183	t	266,086			70,000	95,000	90,218	10,868								
	g/t Au	16.05			16.05	16.05	16.05	16.05								
244	t	237,016			75,000	100,000	62,016									
	g/t Au	11.54			11.54	11.54	11.54									
305	t	195,648			75,000	75,000	45,648									
	g/t Au	14.44			14.44	14.44	14.44									
366	t	220,137			50,000	50,000	85,000	35,137								
	g/t Au	11.82			11.82	11.82	11.82	11.82								
427	t	538,793				30,000	95,000	145,417	145,417	122,959						
	g/t Au	11.44				11.44	11.44	11.44	11.44	11.44						
488	t	353,696					72,118	145,415	120,000	16,164						
	g/t Au	15.52					15.52	15.52	15.52	15.52						
549	t	317,031						73,942	145,000	98,090						
	g/t Au	14.00						14.00	14.00	14.00						
610	t	181,799						39,222	35,620	75,000	31,957					
	g/t Au	8.47						8.47	8.47	8.47	8.47					
671	t	268,954							3,963	37,573	110,000	100,000	17,418			
	g/t Au	10.87							10.87	10.87	10.87	10.87	10.87			
732	t	198,073								30,214	110,000	57,859				
	g/t Au	15.72								15.72	15.72	15.72				
793	t	88,049								70,000	18,049					
	g/t Au	10.13								10.13	10.13					
854	t	115,222									90,000	25,222				
	g/t Au	14.43									14.43	14.43				
915	t	218,036									80,000	92,888	45,148			
	g/t Au	16.50									16.50	16.50	16.50			
976	t	212,487									9,994	81,890	95,000	25,603		
	g/t Au	14.44									14.44	14.44	14.44	14.44		
1037	t	129,723										70,000	59,723			
	g/t Au	11.95										11.95	11.95			
1098	t	116,297										22,141	94,156			
	g/t Au	16.65										16.65	16.65			
1159	t	144,564											53,225	90,000	1,339	
	g/t Au	17.47											17.47	17.47	17.47	
1220	t	134,965											6,798	77,087	51,080	
	g/t Au	15.85											15.85	15.85	15.85	
1281	t	192,922											78,532	77,688	36,702	
	g/t Au	15.85											15.85	15.85	15.85	
1342	t	146,910												74,622	63,714	8,574
	g/t Au	15.85												15.85	15.85	15.85
1403	t	137,429													77,949	59,480
	g/t Au	15.85													15.85	15.85
1464	t	85,751													54,216	31,535
	g/t Au	15.85													15.85	15.85
Total	t	4,499,588			270,000	350,000	450,000	450,000	450,000	450,000	450,000	450,000	450,000	345,000	285,000	99,589
Grade	g/t Au	13.87			13.57	13.42	13.41	13.06	13.12	11.69	13.64	13.96	15.27	16.17	15.86	15.85
Au	Oz	2,006,244			117,767	150,974	193,994	188,999	189,748	169,076	197,411	201,913	220,890	179,374	145,337	50,762

The PEA projection for gold mined is just over 2 M ounces. Average projected gold mined through the steady state years (Y05 through Y11) is 195 k ounces pa, while the average over the total production years, including ramp-up and tail, is 167 k ounces pa.

16.4 Manpower

16.4.1 Pre-Production Period

Table 16.5 shows the Rubicon Project team for the envisaged two pre-production years. The various contracting and consulting personnel involved with the pre-production activities are not shown but their associated costs have been captured in the Project Capital (Section 21).

Table 16.5 Pre-Production Personnel

Rubicon Project Team Pre-Production	Y01	Y02
Project Manager	1	1
Project Superintendent	1	1
HSE Environment Manager	1	1
First Nations Co-ordinator	1	1
Security/ First Aid	3	3
Loader Operators	3	3
Maintenance Superintendent	1	1
Buyer	1	1
Safety	1	1
Environment	1	1
Chief Engineer	1	1
Mine Planner	1	1
Survey	1	1
Ground Control	1	1
Mine Tech	1	1
Chief Geologist	1	1
Beat Geologist	2	2
Project Accountant	1	1
Cook	1	1
Scheduler	1	1
Leaders	4	4
Miners	13	21
Total	**42**	**50**

16.4.2 Operating Mine

Table 16.6 shows the manpower profile over the projected 12 years of the operating mine. Average total manpower over the 12 years is 251 pa; through the steady state years it is 275 pa. The labour-intensive nature of the projected mining method is reflected in the mining manpower numbers, viz. 157 pa average over the producing period and 179 pa in steady state. A 15% absenteeism factor has been included in the mining, mill and maintenance numbers.

Table 16.6 Operations Manpower

	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14
Administration			21	21	22	22	22	22	22	22	22	22	22	22
Manager - Mine			1	1	1	1	1	1	1	1	1	1	1	1
Admin. Assistant			1	1	1	1	1	1	1	1	1	1	1	1
Safety Coordinator			1	1	1	1	1	1	1	1	1	1	1	1
Envir. Coordinator			1	1	1	1	1	1	1	1	1	1	1	1
FN Co-Ordinator			1	1	1	1	1	1	1	1	1	1	1	1
HR			1	1	2	2	2	2	2	2	2	2	2	2
IT / IS			1	1	1	1	1	1	1	1	1	1	1	1
Controller			1	1	1	1	1	1	1	1	1	1	1	1
Cost Accountant			1	1	1	1	1	1	1	1	1	1	1	1
Buyer / Receiver			2	2	2	2	2	2	2	2	2	2	2	2
Security			8	8	8	8	8	8	8	8	8	8	8	8
Account Payable			2	2	2	2	2	2	2	2	2	2	2	2
Engineering			8	8	10	10	10	10	10	10	10	10	10	10
Chief Engineer			1	1	1	1	1	1	1	1	1	1	1	1
Mine Planning			2	2	3	3	3	3	3	3	3	3	3	3
Mine Technologist			1	1	2	2	2	2	2	2	2	2	2	2
Surveying			2	2	2	2	2	2	2	2	2	2	2	2
Ground Control			1	1	1	1	1	1	1	1	1	1	1	1
Ventilation			1	1	1	1	1	1	1	1	1	1	1	1
Geology			11	11	15	15	15	15	15	15	15	15	15	15
Chief Geologist			1	1	1	1	1	1	1	1	1	1	1	1
Senior Geologist			2	2	2	2	2	2	2	2	2	2	2	2
Grade Control/Beat			8	8	12	12	12	12	12	12	12	12	12	12
Mine			114	128	176	178	182	181	175	182	177	149	137	100
Superintendent			1	1	1	1	1	1	1	1	1	1	1	1
General Foreman			1	1	1	1	1	1	1	1	1	1	1	1
Supervisors			3	3	5	5	5	5	5	5	5	5	5	5
Hoist			4	4	5	5	5	5	5	5	5	5	5	5
Cage			4	4	5	5	5	5	5	5	5	5	5	5
Leaders			10	10	20	20	20	20	20	20	20	20	20	20
Miners			68	82	96	98	102	101	95	102	97	69	57	20
Nipper			3	3	6	6	6	6	6	6	6	6	6	6
Ore Flow			6	6	12	12	12	12	12	12	12	12	12	12
Backfill			3	3	6	6	6	6	6	6	6	6	6	6
Pumping			1	1	1	1	1	1	1	1	1	1	1	1
Construction/Shaft			4	4	8	8	8	8	8	8	8	8	8	8
Labour			3	3	6	6	6	6	6	6	6	6	6	6
Surface / Deck			3	3	4	4	4	4	4	4	4	4	4	4
Mill			21	21	21	21	21	21	21	21	21	21	21	21
Superintendent			1	1	1	1	1	1	1	1	1	1	1	1
General Foreman			1	1	1	1	1	1	1	1	1	1	1	1
Supervisor			3	3	3	3	3	3	3	3	3	3	3	3
Operators			10	10	10	10	10	10	10	10	10	10	10	10
Chief Assayer			1	1	1	1	1	1	1	1	1	1	1	1
Metallurgist			1	1	1	1	1	1	1	1	1	1	1	1
Technical / Lab.			4	4	4	4	4	4	4	4	4	4	4	4
Maintenance			24	24	28	28	28	28	28	28	28	28	28	28
Superintendent			1	1	1	1	1	1	1	1	1	1	1	1
General Foreman			1	1	1	1	1	1	1	1	1	1	1	1
Maint. Planner			1	1	1	1	1	1	1	1	1	1	1	1
Supervisor			3	3	3	3	3	3	3	3	3	3	3	3
Electrical			8	8	10	10	10	10	10	10	10	10	10	10
Mechanical			8	8	10	10	10	10	10	10	10	10	10	10
Instrumentation			2	2	2	2	2	2	2	2	2	2	2	2
Total Operation			199	213	272	274	278	277	271	278	273	245	233	196

16.5 Equipment

The equipment types shown in Table 16.7 below are those required to carry out and support the captive cut and fill mining as outlined in the PEA report. AMC has envisaged that, as per Item 3 below, one or two 2-yard Scooptrams will be required, particularly for ramp development that is projected below 305L and 1403L to allow production access below those horizons before necessary shaft and associated infrastructure is in place. Also required are rockbreakers to be installed at each grizzly location adjacent to the shaft. $2.75 M in each of the pre-production years has been allowed in the PEA for the mining equipment items listed.

Table 16.7 Mining Equipment

	Description
1	Jacklegs and Stopers
2	Slushers
3	2 Yard Scooptrams
4	Rail Cars
5	Motors
6	Tuggers
7	Pumps
8	Shotcrete Machine
9	Grout Pump
10	Rockbreakers

16.6 Power, Air and Water

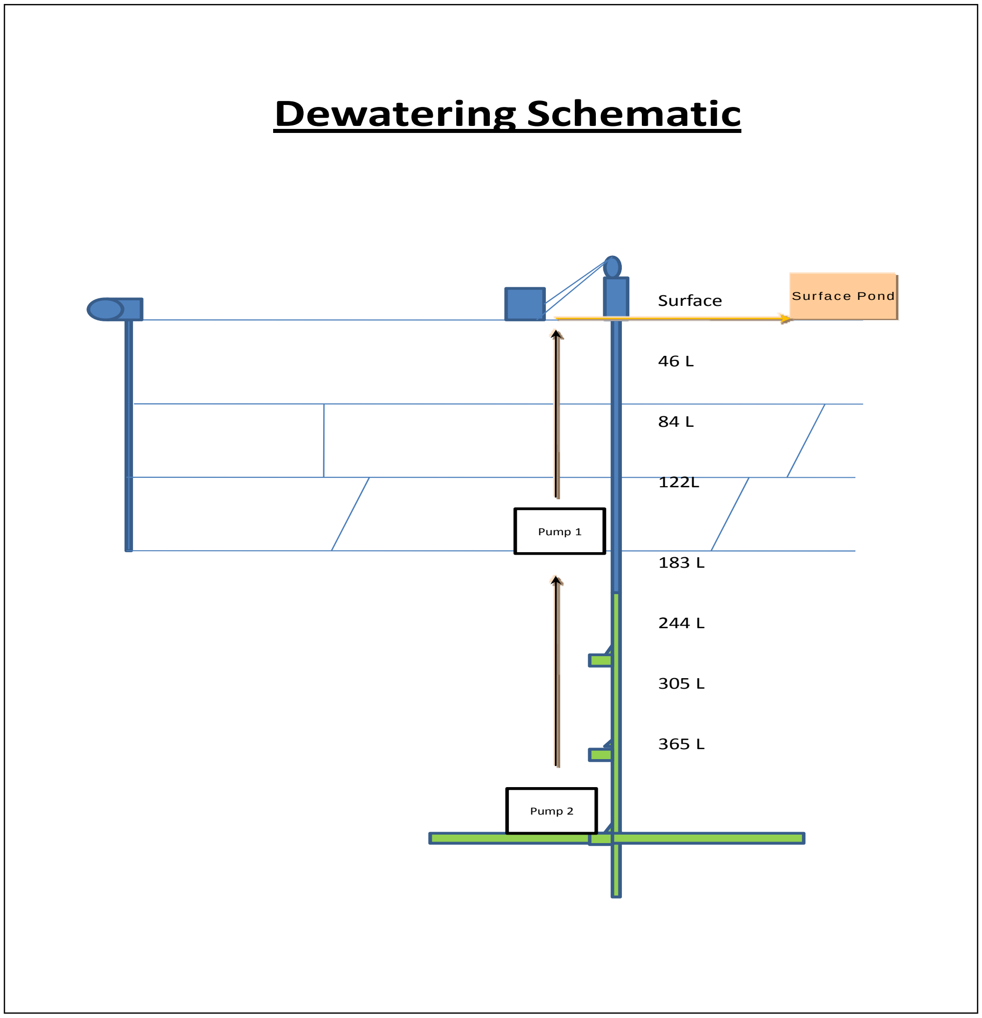
Electrical power at the Phoenix site is currently supplied by a diesel generator. Application has been made to allow construction of a 10.4 km power transmission line from the 44KV grid in the Municipality of Red Lake. This line will provide 5.3 MW of electricity and will connect to a 10MVA substation that has recently been set up on site. Power to underground will be supplied at 4160V via a shaft power cable and level sub-stations will be installed as required. Power underground will be available at 4160V, 550V and 120V.

Compressed air is currently supplied to underground via a surface compressor set-up that AMC understands will be adequate for the underground mining activity envisaged.

Mine water is pumped to a holding tank at the site from the nearby East Bay of Red Lake. The water is piped underground via a 100 mm water line for drilling use, muckpile watering, etc. Potable water for drinking is provided on surface and underground in 15L bottles.

Use of paste fill means that there should be no significant source of waste water in the mine other than ground water, which is currently handled by conventional sumps and pumping to a 100 mm shaft line. That process will continue with additional pumping capability added as required. A schematic of the pumping system in the upper part of the mine is shown below in Figure 16.6. The system shown will be carried down as the mine deepens with intermediate pumps installed approximately every 200 m.

Figure 16.6 Pumping / Dewatering Schematic

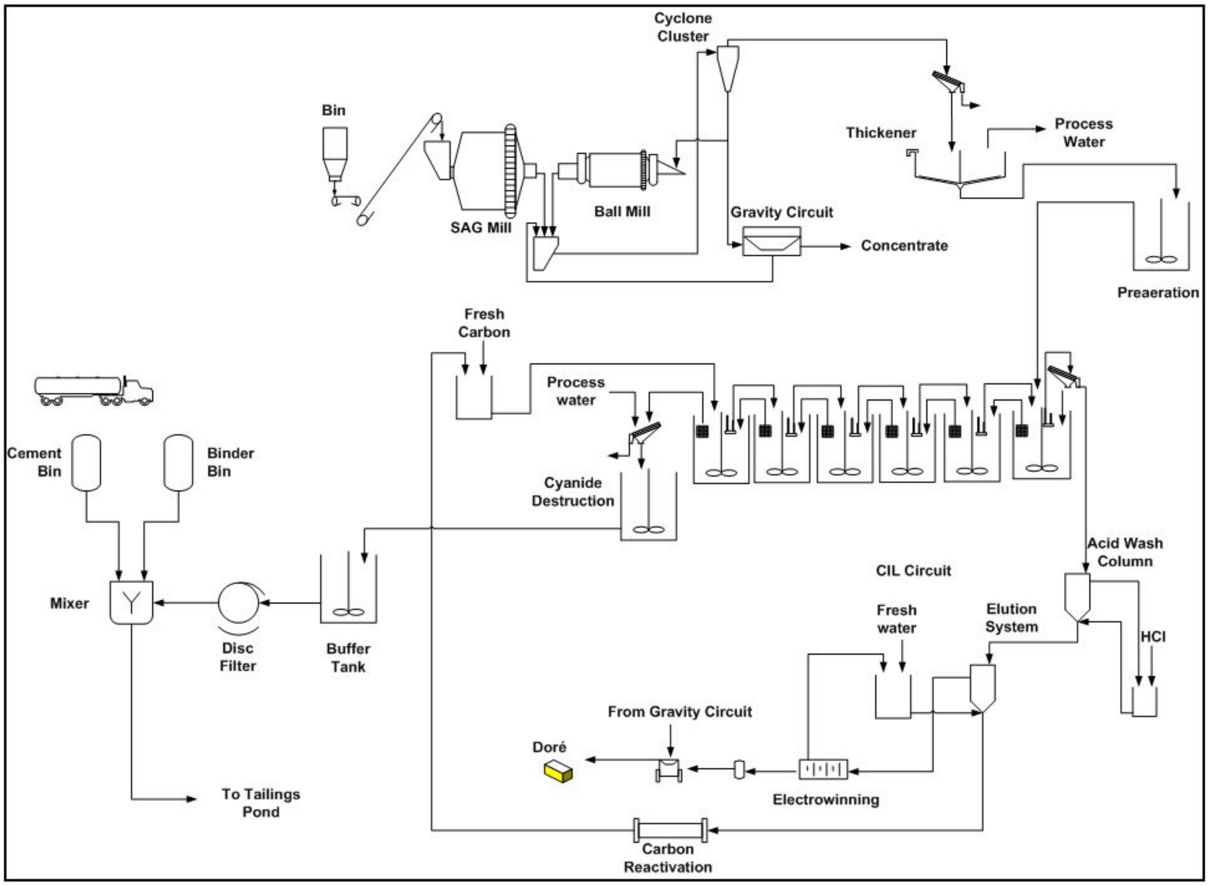


17 RECOVERY METHODS

17.1 Process Flowsheet

The simplified process flowsheet for the Project is presented in Figure 17.1.

Figure 17.1 Simplified Process Flowsheet



17.2 Process Flowsheet Summary

The process consists of a single line, starting with a semi-autogenous grinding (SAG) mill. The discharge of the SAG mill is pumped to hydrocyclones for classification. A gravity separation circuit is included in closed circuit with hydrocyclones to recover any gravity recoverable gold (GRG) prior to regrinding in a ball mill. Gold is extracted in a conventional carbon-in-leach (CIL) circuit. The loaded carbon is washed with hydrochloric acid solution to remove carbonate. Gold is then removed from the loaded carbon by stripping (elution) followed by electrowinning and smelting of doré in an electric induction furnace. The strip carbon is regenerated in a reactivation kiln before going back to the process. Fine carbon is constantly eliminated from the process to avoid gold loss in the fine carbon. However, fresh carbon is continuously added to the process.

The cyanide in the tailings from the CIL circuit is removed in a cyanide destruction tank with SO_2 and air diffuser placed at the bottom of the tank. Once the cyanide is destroyed, the tailings pass through the paste plant where the tailings are filtered to lower the water content. The filter cake is then mixed to produce paste fill. The paste produced can be sent to the TMF or used in the mine for backfill after the addition of cement and/or other binder to meet underground strength requirements.

17.3 Process Description

17.3.1 Ore Storage

A rock breaker combined with a grizzly with, typically, 23 cm openings (9”x9”), is used underground to reduce the size of the material from stoping. The material small enough to pass the grizzly is then skipped at the surface to the ore storage bin.

17.3.2 Grinding and Thickening

An apron feeder reclaims crushed material from the ore storage bin and discharges it onto the mill feed conveyor.

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 hydrocyclone underflow is directed to the ball mill feed chute for regrinding while the remaining portion goes to the gravity separation circuit.

The overflow from the hydrocyclones flows by gravity to a trash screen. The screen undersize feeds the thickener while any oversize trash is dumped into a trash bin. The thickener underflow is pumped to the pre-aeration tank.

17.3.3 Gravity Separation

The gravity separation circuit consists of a vibrating screen, a gravity concentrator and a gravity table. Undersize material from the screen flows to the gravity concentrator where GRG gold is recovered. The gold concentrate is then upgraded on a gravity table and smelted into doré in the on-site refinery.

17.3.4 Carbon-in-Leach

The underflow from the thickener is pumped to a 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 first and fourth CIL tanks for gold dissolution and pH control. Gold in the solution is absorbed into the activated carbon in the CIL circuit.

The six CIL tanks have been sized to provide 36 hours of residence time at the design flowrate and solids concentration. Each CIL tank is equipped with a single interstage screen and a carbon-forwarding pump. On a regular basis, loaded carbon is pumped counter current to the slurry flow in order to increase gold loading. The carbon-forwarding pump of the first tank transfers the slurry onto a vibrating screen to recover the loaded carbon from the slurry. Screen undersize flows by gravity back to the first CIL tank.

17.3.5 Stripping and Carbon Reactivation

Loaded carbon recovered from the slurry in the loaded carbon screen gravitates to the acid wash column. The stripping circuit should treat a 4 tonne batch in approximately 12 hours. The circuit is designed for one strip per day.

Once the acid wash is done, the spent acid is neutralized. The carbon is transferred from the acid wash column to the strip column for gold desorption. The solution strips the precious metals loaded onto the carbon which then exit through the Johnson screen from the upper side of the column. The pregnant strip 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 screened out to remove fines and then it drops by gravity to the last CIL tank.

17.3.6 Electrowinning and Refinery

The pregnant strip solution from the strip column flows by gravity into two electrowinning cells arranged in parallel, where the gold is plated on cathodes. After a certain period, the stainless steel wool cathodes are removed from the cells and cleaned with high pressure water in a dedicated cleaning box. The recovered gold sludge is mixed with suitable fluxes, usually borax, soda ash and sodium nitrate, and is fed into the crucible of the electric induction furnace. Once the gold is melted, it is poured into a mould and the doré bar is recovered for shipment.

17.3.7 Cyanide Destruction

The safety screen undersize from the last CIL tank flows by gravity into a pump box and is pumped to the cyanide destruction tank. Once cyanide destruction is done, tailings are discharged in the tailings pump box and pumped to the buffer tank in the paste fill plant.

17.3.8 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
  - Used for gold leaching and carbon stripping
- Flocculant
  - Used in the thickener to improve the settling rate
- Hydrochloric acid
  - Used for the carbon acid wash
- Lead nitrate
  - Used in the grinding circuit to improve the gold leaching kinetics in the CIL circuit
- Sulphur dioxide
  - Used for the cyanide destruction

- Lime
  - Used to control the pH throughout the process
- Copper sulphate
  - Used as a catalyst in the cyanide destruction process.
- Sodium hydroxide
  - Used for the carbon stripping and after the carbon acid wash to neutralise the residual acid
- Cement
  - Used in the paste fill plant to enhance the strength of the paste
- Binder
  - Used in the paste fill plant to enhance the strength of the paste

17.3.9 Utilities

Fresh water is used for cooling, gland sealing, reagent preparation and process water make-up. Reclaim water from the TMF is used as process water. The water from the TMF overflows into the polishing pond.

The process water distribution system consists of a single process water tank located next to the thickener to allow thickener overflow to gravitate into the process water tank. The other source of water supplying the process water tank is the water reclaimed from the polishing pond. Two process water pumps distribute the water throughout the process.

17.3.10 Air Service

Air is stored in an air receiver by two air compressors. An air receiver distributes the air for the paste plant and for instrumentation. Two air blowers are used for low pressure air distribution throughout the process.

17.3.11 Tailings Filtration

Tailings from the cyanide destruction process are pumped to a buffer tank. The tailings are then pumped to one disc filter while the other remains as spare.

17.3.12 Paste Fill Preparation

The tailings are first filtered to reduce water content and then mixed to produce paste fill. When paste is required for underground backfill, it is mixed with cement and/or other binder. The cement and binder discharged from the storage bins are controlled to achieve the proper concentration in the paste. The paste is then discharged to the distribution pumps.

17.3.13 Paste Fill Distribution

Two positive displacement pumps are used to move the paste towards the underground stopes or to the TMF for disposal. Each pump is equipped with a lube unit.

Two distribution lines are planned to both the TMF and the underground stopes. Manual valves located after the positive displacement pumps allow the possibility to move fill to either location.

17.4 Concentrator Design

17.4.1 Design Criteria

Table 17.1 presents the main design criteria used for the concentrator design.

Table 17.1 Concentrator Main Design Criteria

Parameter	Value	Units
Feed Characteristics		
Gold Head Grade (nominal)	13.87	g/t
Gold Head Grade (maximum)	20.0	g/t
Ore Moisture	5.0	%
Ore Specific Gravity	2.80	
Operating Schedule		
Scheduled Operating Days	365	d/y
Plant Availability	92.0	%
Operating Hours	24	h/d
Shifts	2	shift/d
Production Rate		
Production Target (dry)	456,250	tpy
Gold Recovery	92.5	%
Gold Production (nominal)	188,188	oz/y
Gold Production (maximum)	271,359	oz/y
Nominal Plant Feed Rate	1,250	tpd
Operation Plant Feed Rate	1,359	tpd
Future Expandable Feed Rate	2,500	tpd

17.4.2 Mass Balance

Based on a concentrator availability of 92% and a nominal feed rate of 1,250 tpd, the production target is estimated to be 456,250 tpa; the mass balance is presented in Table 17.2.

This mass balance considers a binding materials proportion of 5% for the paste plant.

Table 17.2 Concentrator Mass Balance (5 % of Binding Material)

Stream Description	Solids		Solution	Pulp		
	Solids (tph)	Solids SG	Solution (tph)	Pulp (tph)	Pulp (m3/h)	Solids (%w/w)
Grinding and Gravity Circuit						
SAG Mill						
SAG Mill Feed	56.6	2.8	3.0	59.6	23.2	95.0
SAG Mill Discharge	56.6	2.8	18.9	75.5	39.1	75.0
Ball Mill						
Cyclone Underflow to Grinding Circuit	118.9	2.8	51.0	169.8	93.4	70.0
Ball Mill Discharge	118.9	2.8	50.9	169.8	93.4	70.0
Cyclone Feed Pump Box						
SAG Mill Discharge	56.6	2.8	18.9	75.5	39.1	75.0
Ball Mill Discharge	118.9	2.8	50.9	169.8	93.4	70.0
Gravity Circuit Tailings	48.4	2.8	38.6	87.0	55.8	55.7
Screen Oversize	2.5	2.8	0.3	2.8	1.2	90.0
Cyclone Feed	226.4	2.8	177.9	404.4	258.8	56.0
Cyclone						
Cyclone Feed	226.4	2.8	177.9	404.4	258.8	56.0
Cyclone Underflow	169.8	2.8	72.8	242.6	133.4	70.0
Cyclone Underflow to Grinding Circuit	118.9	2.8	51.0	169.8	93.4	70.0
Cyclone Underflow to Gravity Circuit	51.0	2.8	21.8	72.8	40.0	70.0
Cyclone Overflow	56.6	2.8	105.1	161.7	125.4	35.0
Gravity Concentrator Circuit						
Cyclone Underflow to Gravity Circuit	51.0	2.8	21.8	72.8	40.0	70.0
Screen Water Addition	-	2.8	3.0	3.0	3.0	-
Screen Oversize	2.5	2.8	0.3	2.8	1.2	90.0
Screen Undersize	48.4	2.8	24.6	73.0	41.8	66.3
Gravity Concentrator Water Addition	-	2.8	14.0	14.0	14.0	-
Gravity Circuit Tailings	48.4	2.8	38.6	87.0	55.8	55.7
Thickening Circuit						
Cyclone Overflow	56.6	2.8	105.1	161.7	125.4	35.0
Trash Screen Water Addition	-	2.8	5.0	5.0	5.0	-
Thickener Feed	56.6	2.8	110.1	166.7	130.4	34.0
Filtrate	0.2	2.8	47.5	47.8	47.6	0.5
Thickener Feed + Filtrate	56.9	2.8	157.7	214.5	178.0	26.5
Thickener Overflow	0.2	2.8	101.0	101.2	101.1	0.2
Thickener Underflow	56.6	2.8	56.6	113.3	76.9	50.0
CIL Circuit						
Thickener Underflow	56.6	2.8	56.6	113.3	76.9	50.0
Safety Screen Water Addition	-	2.8	5.0	5.0	5.0	-
CIL Circuit Tailings	56.6	2.8	61.6	118.3	81.9	47.9
Paste Plant						
CIL Circuit Tailings	56.6	2.8	61.6	118.3	81.9	47.9
Filtrate	0.2	2.8	47.5	47.8	47.6	0.5
Cake	56.4	2.8	14.1	70.5	34.2	80.0
Cement Feed	0.3	3.2	-	0.28	0.1	100.0
Binder Feed	2.5	2.9	-	2.5	0.9	100.0
Binding Materials Feed	2.8	2.9	-	2.8	1.0	100.0
Mixer Process Water	-	2.8	5.6	5.6	5.6	-
Paste	59.2	2.8	19.7	79.0	40.9	75.0

17.4.3 Equipment List

The equipment was selected based on design criteria outlined above for a 1,250 tpd tonnage and an availability of 92%. Some major equipment items are already designed for the envisaged 2,500 tpd expansion tonnage. A major equipment list with a brief description of the equipment is presented in Table 17.3.

Table 17.3 Major Process Equipment

Equip No.	Equipment Name	Equipment Description
110-BN-01	Ore Storage Bin	25' Dia x 75' Height
210-CY-01	Cyclone Cluster	4 gMAX15-20 Krebs Cyclones
210-ML-01	SAG Mill	20' x 11.25' SAG Mill
210-ML-02	Ball Mill	10.5' x 15' Ball Mill
220-TH-01	Thickener	High Rate, 15 m Diameter
250-GC-01	Gravity Concentrator	Knelson KC-XD20
310-TK-01	Pre-Aeration Tank	8.5m Dia x 9.5m Height
320-TK-02@07	CIL Tank 1 to 6	8.5m Dia x 9.5m Height
320-PP-07@12	Transfer Pump 1 to 6	
320-SC-04@10	Interstage Screen CIL Tank 1 to 6	Kemix Pumping Screen (MPS(P)) 3 m129
320-SC-13	Loaded Carbon Screen	#
410-SC-11	Safety Screen	Vibrating 4' x 8'
420-TK-08	Cyanide Destruction Tank	7m Dia x 7.5m Height
510-CL-01	Acid Wash Column	4 t
510-CL-02	Strip Column	4 t
510-KL-01	Carbon Reactivation Kiln	
510-SC-14	Unloaded Carbon Dewatering Screen	
510-TK-09	Carbon Attrition Tank	
610-FI-01/02	Disc Filter 01 /02	
610-MX-01	Paddle Mixer	Single Mixing Main Shaft
610-TK-28	Buffer Tank	8.5m Dia x 9.5m Height
620-PP-67/68	Paste Pump 01/02	
785-BN-07	Cement Storage Bin	250 t
790-BN-06	Binder Storage Bin	250 t
810-EW-01/02	Electrowinning Cell 01/02	3.5 m^3 cell
830-FU-01	Smelting Furnace	
830-TA-01	Gravity Table	Gemini GT250 Shaking Table

17.5 Capital Costs

The evaluation was made for a 1,250 tpd tonnage concentrator and some major equipment, and provisions were also made for future expansion to 2,500 tpd.

17.5.1 Summary

Table 17.4 presents the summary of capital costs, which are divided into direct and indirect costs and are shown pre-contingency allowance.

Table 17.4 Summary of Capital Costs

Description	Cost $
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**

17.5.2 Direct Capital Costs

Table 17.5 presents the summary of the direct capital costs by concentrator sector.

Table 17.5 Summary of Direct Capital Costs

Sector	Description	Cost $
	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
Sub-Total (Material/Equipment)		35,061,141
Installation		14,026,690
Piping		4,310,412
Electricity and Control		9,231,370
Total		62,629,613

17.5.3 Indirect Capital Costs

Table 17.6 presents the summary of the indirect capital costs. Some items were not included because the study is related to the concentrator operation only.

Table 17.6 Summary of Indirect Capital Costs

Description	Cost $
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**

Operating Costs

The operating costs for a 1,250 tpd concentrator were estimated with the selected flowsheet (as per Section 17.1). The unit operating costs only apply to the nominal tonnage and head grade; any changes would have a direct effect on the unit operating costs.

The operating costs are calculated on the basis of a processing rate of 1,250 tpd. However, operating costs will have to be recalculated if there is an increase to 2,500 tpd; this would reduce the unit operating costs. Operating costs are based on the design criteria for the operating schedule.

17.5.4 Concentrator Complex Operating Cost Summary

The operating costs for the concentrator complex are estimated to be $25.60 /t of milled ore, inclusive of the paste plant other than for cement/binder costs, which are discussed below in Section 17.6.4.2. Without the paste plant operation costs, the concentrator operation costs have been estimated at around $22.00 /t of milled ore. Table 17.7 presents a summary of the operating costs for the concentrator complex.

Table 17.7 Summary of Concentrator Complex Operating Costs

Description	Cost ($)	Units
Manpower		
Plant	2 600 000	$/y
Maintenance	1 260 000	$/y
Management	900 000	$/y
Supplies		
Maintenance	660 000	$/y
Reagents	2 022 000	$/y
Consummables	1 981 000	$/y
Power		
Power	2 257 000	$/y
Total		
Complex Concentrator Operating Cost Estimate	11 680 000	$/y
Unit Cost per Tonne of Milled Ore	25.60	$/t
Unit Cost per Ounce of Gold	62.07	$/oz
Paste Plant Operating Cost Estimate		$/y
Unit Cost per Tonne of Milled Ore	3.60	$/t
Unit Cost per Ounce of Gold	8.73	$/oz
Concentrator Operating Cost Estimate		$/y
Unit Cost per Tonne of Milled Ore	22.00	$/t
Unit Cost per Ounce of Gold	53.34	$/oz

17.5.5 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 complex operating costs. The complex operating cost directly attributable to the paste plant has been estimated at $3.60 /t of milled ore. Fill system costs external to the paste plant, inclusive of underground piping, valving and instrumentation maintenance, fill barricade materials and erection, etc. have been estimated at $7.17 /t of milled ore. Total fill system operation costs have been estimated at $14.00 /t of milled ore, inclusive of cement/binder costs.

17.5.6 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 complex, including the paste plant. In addition, two people for the refinery and four people for the assay laboratory are expected to work 40 hours a week.

Nine people are projected for maintenance to operate the overall concentrator complex. 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 17.8.

Table 17.8 Manpower Costs Design Criteria

Design Criteria	Plant			Maintenance	Management
	Operation	Refinery	Assay Laboratory		
Annual Salary ($)	110,000	140,000	140,000	140,000	180,000
Hourly Rate ($)	50.37	67.31	67.31	67.31	86.54
Employees	4	2	4	9	5
Yearly Working Hours	2,184	2,080	2,080	2,080	2,080
Yearly Man-hours	34,944	4,160	8,320	18,720	10,400
Daily Working Hours	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 17.9 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 $133,000.

Table 17.9 Details of Manpower Operating Costs

Description	Total	$/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

17.5.7 Costs of Maintenance and Supplies

17.5.7.1 Maintenance Costs

The maintenance costs correspond to 2% of the estimated process equipment costs ($33 M) and are estimated at $660,000 per year. This represents around 1.5 $/t.

17.5.7.2 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. A rate of 0.15 $/kg has been added to the reagents cost to cover freight costs. Table 17.10 presents the costs for concentrator consumables and reagents.

Table 17.10 Details of Concentrator Consumables and Reagents Costs

Description	Rate	Units	Source	Annual Quantity (Including 92 % availability)	Units	Cost ($/y)
Consumables						
Grinding Ball Consumption (5")	1,560	$/t	Budget Quote	228,125	kg/y	356,000
Grinding Ball Consumption (2")	1,305	$/t	Budget Quote	547,500	kg/y	715,000
Chrome-Moly Steel Liners (SAG Mill)	683,765	$/unit	Budget Quote	1	units/y	683,765
Rubber Liners (Ball Mill)	196,031	$/unit	Budget Quote	1	units/y	196,031
Filter Cloth (Paste Plant)	30,000	$/y	Estimate	30,000	$/y	30,000
Reagents (Process)						
Flocculant	5.62	$/kg	Budget Quote	9,130	kg/y	51,400
Sodium Cyanide (NaCN)	3.20	$/kg	Budget Quote	228,125	kg/y	730,000
Carbon	2.90	$/t	Budget Quote	30	tpy	1,000
Lead Nitrate ($PbNO_3$)	5.60	$/kg	Budget Quote	114,063	kg/y	639,000
Quick Lime (CaO)	0.54	$/kg	Budget Quote	228,125	kg/y	123,000
Sodium Hydroxyde (NaOH)	0.73	$/kg	Budget Quote	21,900	kg/y	16,000
Hydrochloric Acid (HCl)	0.75	$/kg	Budget Quote	120,450	kg/y	90,000
Hydrated Copper Sulfate ($CuSO_4$*$5H_2O$)	4.93	$/kg	Budget Quote	34,932	kg/y	173,000
SO_2 Liquid	0.61	$/kg	Budget Quote	324,026	kg/y	198,000
Total						4,003,000

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 17.10. The estimated cement/binder costs are presented in Table 17.11.

Table 17.11 Details of Binder Costs for the Paste Plant

Reagents for Paste Plant	Proportion of Blend (%)	Proportion of Tonnage (%)	Cost ($/t)	Annual Quantity (Including 92 % availability and 50 % of production to backfill)	
				(t/y)	($/y)
5 % Binder Proportion					
Portland Cement	10	15	250	170	43,000
Binder	90	15	165	1,534	254,000
3.5 % Binder Proportion					
Portland Cement	10	85	250	676	170,000
Binder	90	85	165	6,086	1,005,000
Total					
Total Cost					1,472,000
Unit Cost per Tonne of Miled Ore ($/t)					3.23
Unit Cost per Ounce of Gold ($/oz)					7.82

The paste plant operation costs for binder/cement are estimated on a production rate which is equal to 50% of the projected total production. The paste plant is estimated to run with one disc filter. An average of 3.5% binder is estimated for approximately 85% of underground paste fill production, with the remainder estimated as requiring 5.0% binder. Average cement/binder costs are estimated at 3.23 $/t of milled ore and 7.82 $/t of gold produced.

17.5.8 Power Costs

The estimated required unitary power of each piece of equipment was used to calculate the total power requirement of 3,500 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 2,800 kW. Based on this number, the annual consumption is estimated to be 22,566,000 kWh for the overall concentrator at 1,250 tpd with a utilization factor of 92%. Using a unit cost of 0.10 $/kWh, the annual costs are estimated to be 2,257,000 $/y. Table 17.12 presents the details of power costs.

Table 17.12 Details of Power Costs

Description	Value	Units
Total Unitary Power	3,500	kW
Required Unitary Power	2,800	kW
Annual Consumption	22,566,000	kWh/y
Overdesign Factor	80	%
Power Cost	0.1	$/kWh
Annual Cost	2,257,000	$/y

18 PROJECT INFRASTRUCTURE

18.1 Historical Infrastructure

A three-compartment exploration shaft was developed on the McFinley Peninsula in 1955 to a depth of 428 feet and was abandoned in 1956. New facilities including head frame, hoisting facilities, 150 tpd mill complex and camp infrastructure were developed during a later program of underground development and exploration from 1983 to 1988. Underground development was focused on the 150-, 275- and 400-foot elevations. After the start of legal disputes the workings were allowed to flood in 1989. The site infrastructure was then allowed to deteriorate and buildings suffered random vandalism during the period 1990 to 2001, culminating in the total destruction of the site office by fire in 2001. The mill, hoist and head frame remained intact as vandalism largely focused on breakable items in the camp accommodation buildings.

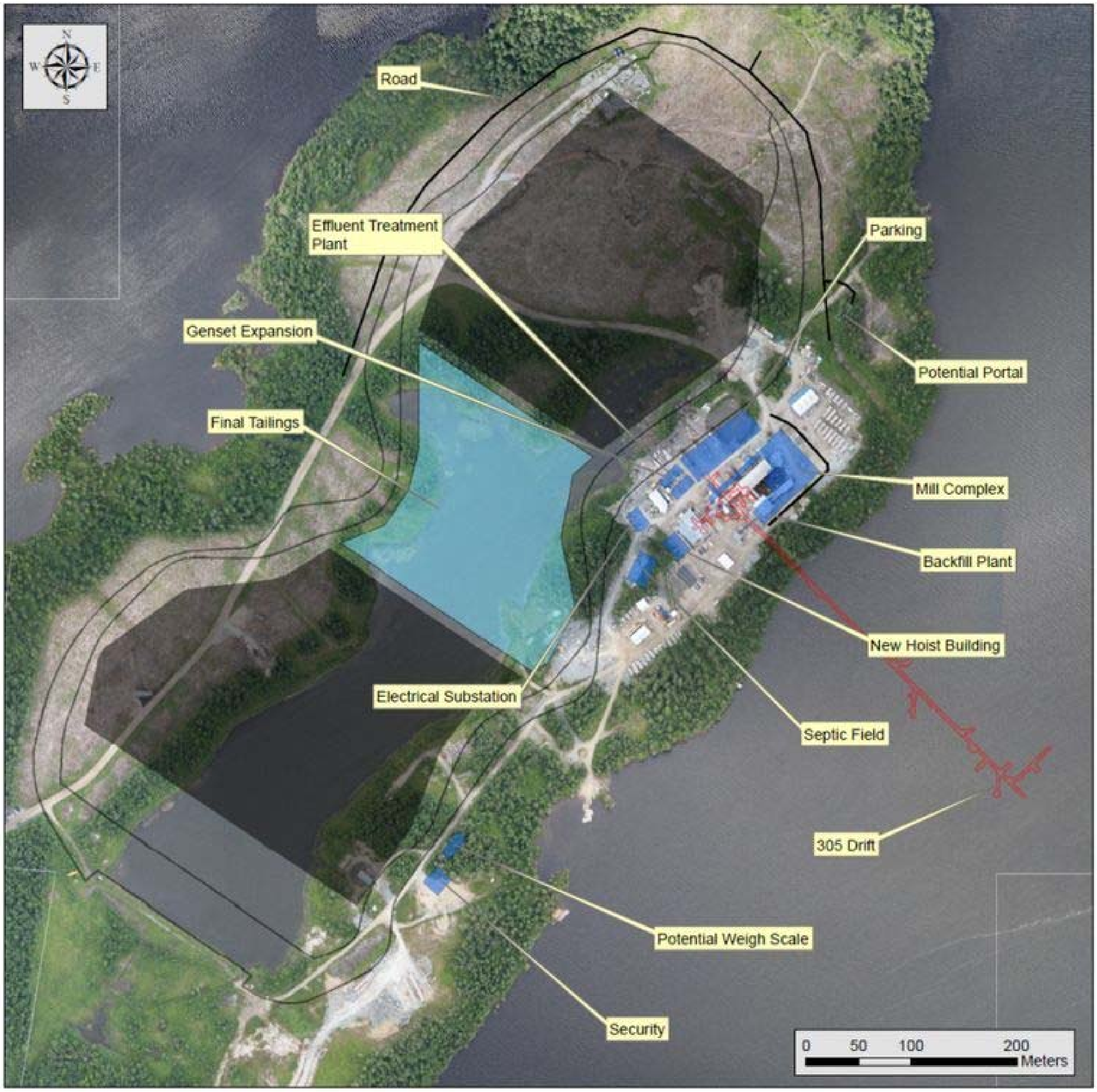
18.2 General Infrastructure for the Phoenix Project

As indicated earlier, the Project has a significant amount of infrastructure already in place. Additional infrastructure development is either underway or in the planning stage. The shaft has been rehabilitated and deepened to about 30 metres below the 305 level. In addition, a 4.3 m diameter hoist with two 932 KW motors and 41 mm diameter ropes has been purchased. It is capable of achieving 2,000 tonnes per day from a depth of 1750 m at a rate of 610 m per minute via 9 tonne skips and will readily manage the anticipated steady state production rate of 1250 tpd and the full depth of the conceived Phoenix production. Hoist and headframe construction and installation are currently in progress.

New core logging/cutting buildings, secure core storage buildings, generator building and office trailer complex have been constructed and access to the site has been restricted with a gatehouse that is staffed on a 24-hour per day basis. Other infrastructure and facilities have been rehabilitated to facilitate the on-going underground and surface exploration programs. Rubicon is currently evaluating the existing mill equipment and other existing infrastructure in preparation for the anticipated production phase.

A preliminary site plan is shown in Figure 18.1.

Figure 18.1 Preliminary Site Infrastructure Plan

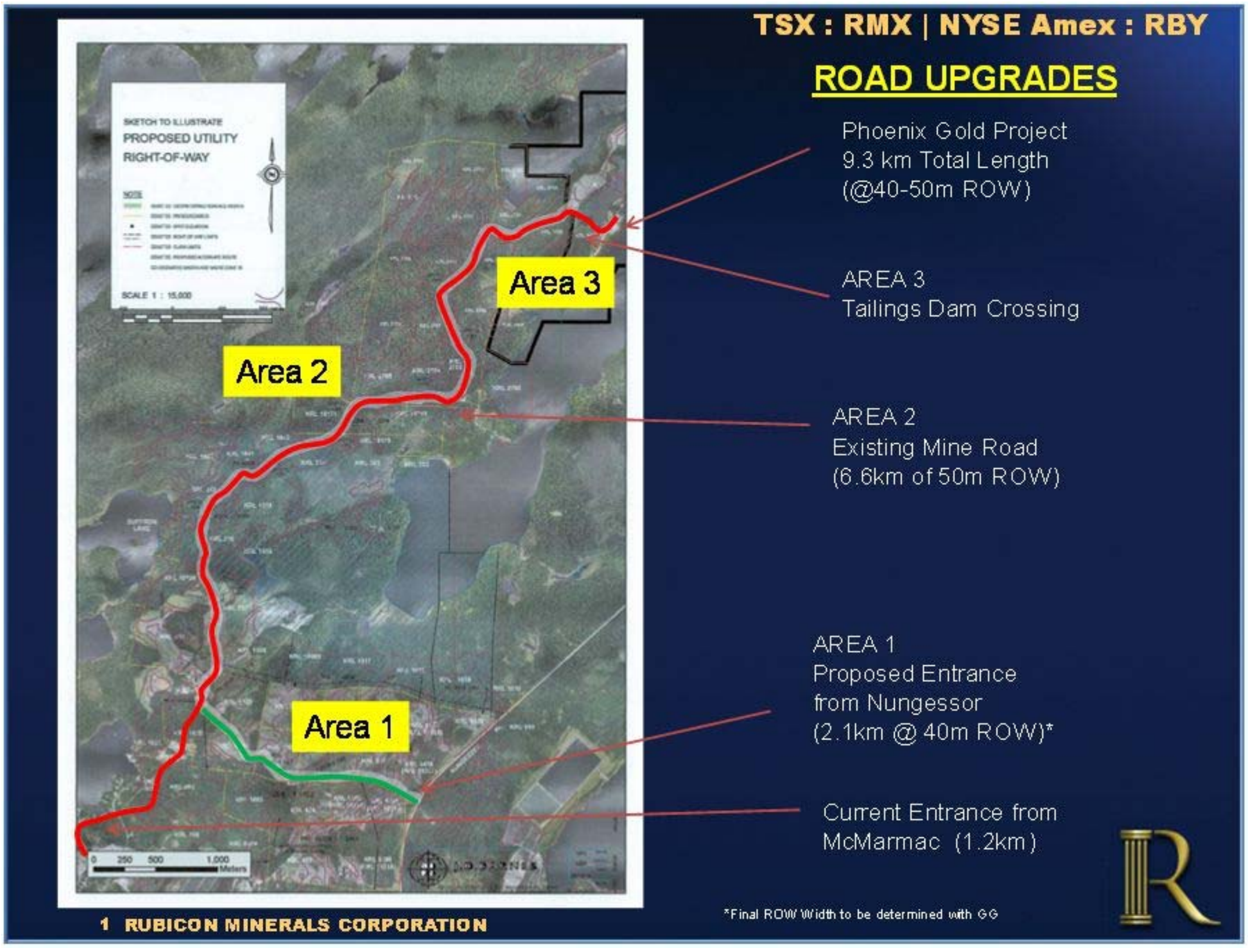


A summary of some of the key infrastructure components is given below.

18.3 Road Access

Road access to the Project site through the town of Cochenour has been established for many years. Future and long-term access will largely follow the existing road but a bypass section will be built to avoid Cochenour as shown in Figure 18.2. This new portion of the access road will be 10 m wide with a 40 m right-of-way and will connect to the existing road which has a 50 m right-of-way. The total length of the access road will be 9.3 km. Design parameters, test requirements and material quantities for the road upgrading have been established, and an associated cost estimate of $1.5 M prepared.

Figure 18.2 Phoenix Site Road Access



18.4 Power and Communications

Electrical power at the project site is currently supplied by a diesel generation set-up located adjacent to the existing hoist and mill infrastructure.

In 2011 Rubicon has accepted an Offer to Connect from Hydro One for 5.3 MW of electricity from the 44KV grid in the Municipality of Red Lake. Rubicon has constructed the required connection to the grid and is securing title to the required right-of-way, through Section 21 of the *Public Lands Act*, negotiations with landowner and leaseholders and, if required, Section 175 of the *Mining Act.* The power-line length will be 10.4 km.

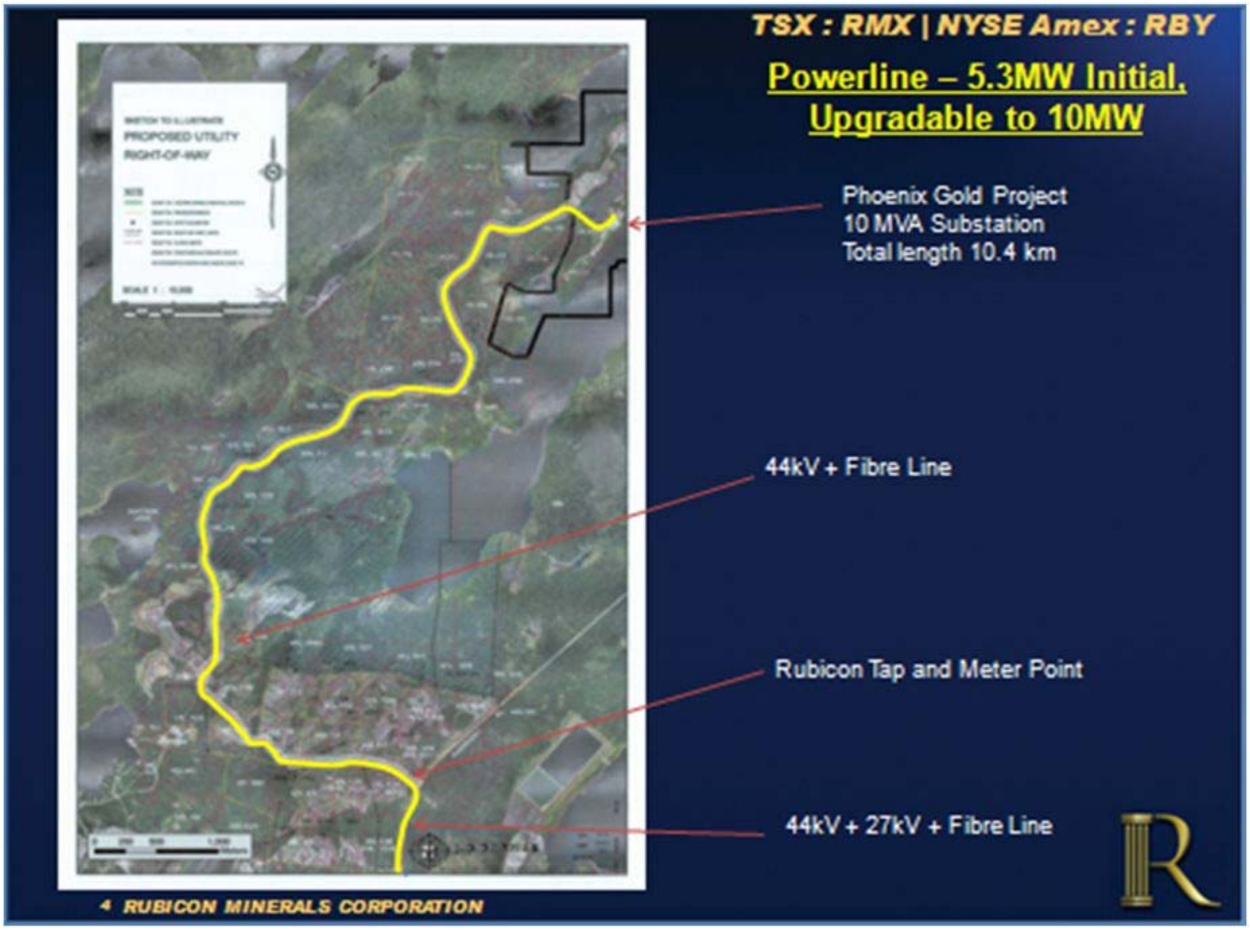
A 10MVA substation has been purchased and has been commissioned by the Electrical Safety Authority of Ontario (see Figure 18.3).

Figure 18.3 Phoenix Site 10MVA Sub-Station

A fibre-optic line will be installed along the same route as the power-line to provide communication capability for the site. Current communications are via satellite network.

Figure 18.4 shows the proposed power-line routing.

Figure 18.4 Phoenix Project Proposed Power-line Routing



18.5 Process and Potable Water

Process water is pumped from the nearby East Bay of Red Lake for use at the Project site. Potable water is currently trucked to site via tanker. Representative samples have been collected from the lake water and a design prepared for construction of a treatment plant capable of producing potable water on-site when required.

18.6 Sewage Plant

Sewage disposal is managed by Rubicon in an on-site sewage works as there is no municipal service available at the Project site. The on-site sewage works is installed, operated and maintained in accordance with a Certificate of Approval issued pursuant to the *Ontario Water Resources Act*.

18.7 Tailings Facility

A TMF 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*.

The final TMF can be clearly seen in Figure 18.1 above. Details of the TMF are given in Section 20 of the PEA.

The TMF and effluent treatment plant have been designed to withstand a 30 day duration, 1 in 100 year rain on snow event (i.e. the TMF can contain and discharge the resultant runoff via the effluent treatment plant).

18.8 Waste Dumps

There are no waste rock piles at the project site related to historic development activities; waste rock from those efforts was utilized for construction of the dam and plant site. The current Advanced Exploration Program has resulted in several minor stockpiles of waste rock (each <5,000 tonnes) at the plant site. Waste rock from future development activities will be consumed by roadbed, used to fill stopes or stored in the TMF

18.9 Stockpiles

Mineralized material will be temporarily stockpiled on surface until shipped off-site for processing or processed in the mill. There will be no significant stockpiles of mineralized material at closure.

18.10 Explosives

No explosives are, or will in future be, stored on surface. Explosive deliveries to site are moved underground immediately and stored in specifically designed magazines.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

The Phoenix mine and processing facilities are anticipated to produce high grade gold doré bars at the mine site, which are readily marketable and are expected to be sold directly to refiners such as the Royal Canadian Mint at prevailing spot prices. As of August 8, 2011, the London AM fix for gold was $1,710 /oz. There are no forward gold sales or puts anticipated in the Project cash flow model.

Relevant statistics for assessment of the Project economics are referenced in Section 22 of the Technical Report.

19.2 Contacts Relevant to Current and Future Project Activities

Rubicon has entered into various contracts under market conditions with suppliers for a range of project activities including shaft sinking and underground development, surface and underground diamond drilling, surface construction, metallurgical tests, etc. AMC considers that there is no reason to believe that similar contracts would not be readily entered into as the Project moves forward.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

The following list summarizes the environmental studies, including baseline monitoring activities, which have been completed to date either by Rubicon or by third party consultants retained by Rubicon.

- Monthly surface water monitoring since 2007 in the vicinity of the Project site
- Semi-annual sampling of ground water monitoring wells since 2009
- Archaeological assessment by Ross Associates
- Species at risk assessment by Northern Bioscience
- Background conditions study by BZ Environmental Consulting
- Aquatic biological assessment by Environmental Applications Group (“EAG”)
- Effluent mixing and plume delineation study by EAG
- Assessment of risks to the downstream environment from the Project by NovaTox Ltd
- Hydrogeological characterization by AMEC Earth and Environmental (“AMEC”)
- Phase 1 & 2 Environmental Site Assessment (“ESA”) by True Grit Consulting Limited (“TGCL”)
- Risk assessment of the ground water and soils at the Project site by Novatox
- Geochemical characterization of development rock associated with the Advanced Exploration phase by AMEC
- Geochemical characterization of development rock, ore, tailings and quarried surface rock by Chem-Dynamics
- Geotechnical assessment of underground workings by AMC

20.1.1 Discussion of Environmental Aspects

20.1.1.1 Air Quality

An emission summary and dispersion model demonstrates compliance with MOE air quality criteria during a worst-case scenario for projected mine production.

20.1.1.2 Surface Waters

Modifications to local watershed boundaries and the flow regime as a result of the envisaged production phase of the Project are summarized below.

- Watershed boundary and surface water flow directions are essentially the same for both pre-production and envisaged production periods.
- Current site runoff reports to the historic tailings impoundment and then decants to the intermittent drainage downstream of the historic containment dam. This flow sustains the downstream fish habitat.

- During the production phase of the Project, plant site runoff will be collected in the plant site sumps and recycled for use in the milling process, thereby preventing storm water discharges from the plant site to the environment. The practice of discharging treated, non-acutely lethal effluent downstream of the existing dam will again sustain the downstream fish habitat.
- Post closure, the runoff from the surface of the TMF will be routed downstream of the dam to sustain the downstream fish habitat. The plant site will be covered with an engineered dry cover that will route runoff eastward to East Bay, as described in Knight Piesold (2010). These watershed boundary modifications represent negligible changes to the intermittent drainage downstream of the TMF and to East Bay compared with the pre-development watershed areas.
- Assessment of Potential Impacts to Surface Water Quantity
- A Permit to Take Water has been issued to allow the withdrawal of a maximum of 1000 LPM from East Bay at the existing pumphouse to supply process water, fire suppression water and potable water to the Project site. A DFO-compliant fish screen is installed to prevent fish access and impingement. There is also negligible potential for the drawdown of East Bay as the requested monthly withdrawal is <5 % of the conservatively predicted low monthly flow into Red Lake, in accordance with MOE (2005).
- Ground Water
- Project hydrogeology is predominantly controlled by the exposed bedrock or the overlying cover of native clay soil, which has been estimated to have a low hydraulic conductivity of ~4×10^{-8} cm/s based on particle size distribution and initial void ratio (Rubicon comm. 2009). Shallow ground water flow is assumed to be similar to surface drainage within the Project site, i.e. primarily originating at heights of land and flowing radially downslope.
- The Permit to Take Water will be amended as required for on-going dewatering of the expanded underground workings. Since dewatering commenced in 2009, a zone of influence has existed in bedrock and this will continue until the underground workings are flooded at Close-out.
- Assessment of Potential Impacts to Ground Water Quantity
- The following measures will minimize future risk to ground water quality.
- Prior to tailings deposition in the TMF, cyanide will be destroyed using the SO2-air process to reduce the concentration to below 1 mg/L WAD cyanide.
- Tailings will be thickened to 75 to 85 % prior to deposition in the TMF, minimizing / eliminating solids-liquids separation (pore water liberation), infiltration by precipitation and the resultant seepage through the perimeter berms. The seepage collection system will be operated in accordance with the issued Industrial Sewage Certificate of Approval until the seepage poses no significant environmental risk and this permit is amended to allow the decommissioning of the seepage collection system.
- An engineered dry cover will be placed over the TMF and plant site to minimize infiltration by precipitation and resultant seepage from these areas.
- The increasingly established soil horizons over the construction rock will ensure the Close-out phase of the Project has progressively less impact than the production phase.

- Monitoring of rock and tailings will continue for the life of the Project, and the precautionary management plan in the Phoenix Project Closure Plan will be followed to prevent potential impacts from chemical stability issues.
- The management plan described in the Phoenix Project Closure Plan will be followed during the life of the Project to prevent potential impacts from ARD/ML.
- In general, ground water quality will be protected as the Project develops in accordance with MOE requirements, pursuant to the *Environmental Protection Act* and the *Ontario Water Resources Act*.

20.1.1.3 Soils

Previous environmental work (and a Phase 2 Environmental Site Assessment) identified soil metal concentrations in parts of the 1980s brownfield project site that were above Ontario MOE criteria. Subsequently, a risk assessment of the historical site footprint was conducted in accordance with Ontario Regulation 153/04 on behalf of Rubicon by Novatox. The risk assessment concluded that there was no unacceptable risk to the environment or to human health as a result of the elevated metals in soil provided the Project site is managed in accordance with applicable provisions of the Ontario *Occupational Health and Safety Act* and Part VII of the *Mining Act*. The brownfield site includes the plant site and TMF.

20.1.1.4 Terrestrial Plant and Animal Life

Rubicon undertook a search of the Natural Resources and Values Information System database in 2009 to identify biological values in the vicinity of the Project site that may pose a constraint to re-development. The search identified a moose aquatic feeding on the east side of the East Bay, approximately 700m east of McFinley Peninsula, with no impact being anticipated as a result of the Project.

A similar search of the Natural Heritage Information Centre database was undertaken in 2009. The search identified the presence of bald eagle nests within the 10 km by 10 km block in which the Project site is located. There have been no observations of nesting bald eagles at the Project site to date, although they have been observed flying over Red Lake. No nests are present within at least 1000 m of the site, and the re-development of the Project site is not anticipated to impact the bald eagle population.

Supplemental field studies in 2011 are being or will be conducted with input from the Ontario MNR to ensure adherence to the provincial *Endangered Species Act* and the Provincial Policy Statement issued pursuant to Section 3 of the *Planning Act*. This supplemental work will identify any potential biological constraints so that possible impacts may be mitigated.

Merchantable timber has been harvested within the general vicinity of the Project site in recent years during winter months. The only remaining forested area on the McFinley Peninsula is the ~60 m wide shoreline buffer along East Bay.

20.1.1.5 Aquatic Plant and Animal Life

EAG completed a baseline assessment of the benthic and fish communities in the receiving environment using an Environmental Effects Monitoring ("EEM") style control / impact approach of exposure and reference areas. This report is provided in the Phoenix Project Closure Plan.

Following a desktop review and supplemental field work in June 2009 by Northern Bioscience, no aquatic biological values were identified in the vicinity of the Project site that could pose a constraint to development. Northern Bioscience (2010) provided additional details regarding the absence of aquatic Species at Risk in the vicinity of the Project site.

Impacts to fish habitat are not anticipated due to the withdrawal of water from East Bay, in accordance with Permit to Take Water 3585-85KGHG. In addition, no negative impacts to the fish habitat provided by the local surface water features are anticipated due to the dewatering of the underground workings (AMEC, 2011).

Blasting practices will comply with minimum setback distances and detonation staggering times (Wright and Hopky (1998)) to prevent potentially harmful effects to fish and fish habitat.

Existing riparian shoreline buffers around the Project site will be maintained. Accordingly, allochthonous inputs to surrounding watercourses are not expected to be materially affected.

An EEM monitoring program will be followed for the Project life once it becomes subject to the federal *Metal Mining Effluent Regulations* ("MMER"), promulgated under the *Fisheries Act*.

20.1.1.6 Potential for Acid Rock Drainage and Metal Leaching

Rubicon has evaluated the chemical stability of the following materials:

- Historic development rock from the 1980s operation
- Development rock from the Rubicon Advanced Exploration phase
- Rockfill from the proposed on-site quarry within the TMF footprint that is associated with the envisaged production phase of the Project
- Development rock that is proposed to be excavated from the mine workings during the production phase of the Project
- Ore / tailings anticipated to be produced during mining operations

52 samples of development rock, 15 of quarry rock and eight metallurgical tailings samples have been submitted for a range of static tests. The tests included modified acid base accounting ("ABA"), shake flask extraction ("SFE") utilizing deionized water and a 3:1 liquid to solid ratio (by weight), quantitative mineralogy (Reitveld XRD) and net acid generation ("NAG"). In addition, two representative metallurgical tailings samples from testing were submitted for laboratory humidity cell testing.

Historic Development Rock and Tailings at the Plant Site

Runoff from the existing dam embankment has been monitored at the toe of the dam since 2007 at monitoring station MF-3; general observations on the dataset are provided below:

pH was observed to be consistently alkaline (8.0 average) with low sulphate concentrations (average 56 mg/L), indicating that rock on the portion of the downstream embankment of the dam that drains to the sampling location is not acid-generating.

Arsenic concentrations were observed to consistently meet the Provincial Water Quality Objectives for the protection of surface water resources in Ontario ("PWQO") of 0.1 mg/L (average 0.063 mg/L).

Lead was consistently observed to be below the PWQO (75th percentile of 0.0025 mg/L).

Average concentrations for copper, nickel and zinc were all observed to be less than respective PWQO.

Relative to background water quality in East Bay / McFinley Bay, water at MF-3 was observed to be hard (average 339 mg/L as CaCO3), with a higher conductivity (average 643 uS/cm), and generally a lower concentration of organic matter as indicated by the data for DOC, TOC and colour.

Iron was observed to be consistently present at concentrations that surpass PWQO and background (average 0.52 mg/L, 75th percentile 0.78 mg/L). Concentrations are within observed ranges at other watercourses in northern Ontario that are unaffected by anthropogenic influences

Advanced Exploration Phase - Development Rock

Development rock from the Rubicon Advanced Exploration Project predominantly comprised ultramafic host rock (>90 %), occasional felsic intrusive rocks and minor mafic volcanic rock. Samples were collected in 2008 from diamond drillhole intersections through the F2 Zone and between 25 m to 95 m of the anticipated underground workings. The samples were subjected to a range of static tests which included modified Acid Base Accounting ("ABA"), 3:1 shake flask extraction ("SFE") using deionized water, and determination of total metal content. Results of the ABA and SFE analysis are discussed below.

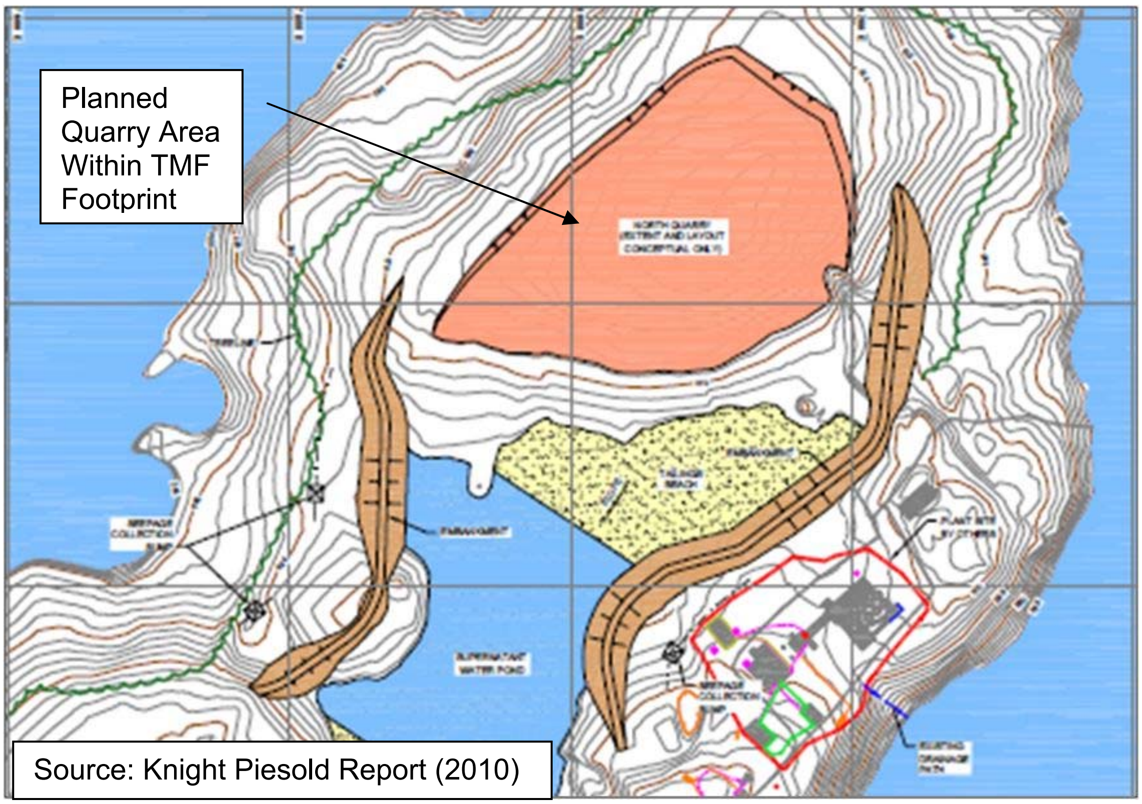
- Ultramafic rock and mafic volcanic rock from the F2 Zone were interpreted to be net acid consuming due to NP/AP ratios greater than four (ref. Price, 1997 criteria in use at the time) and very low concentrations of sulphide-sulphur in the rock samples. A single sample of felsic intrusive rock was classified as having low potential to generate ARD.
- SFE leachate data were compared to PWQO listed in MOE (1994B) as a screening tool to identify constituents that are of potential environmental concern.
- Maximum SFE results met respective metals PWQO for all samples in each lithology.
- The maximum aluminum (Al) concentration for one ultramafic sample and one mafic volcanic sample surpassed the interim PWQO. The SFE leachate sample was filtered through a 0.45 rather than a 0.2 micron filter, which is required for comparison to the interim PWQO for aluminum. Thus, the SFE result is conservatively high. The maximum SFE result for aluminum is within concentrations observed at reference locations at other areas in northern Ontario that are unaffected by anthropogenic influences.

Development rock to date has been placed within the Project industrial sewage works drainage area. The expectation that it will not pose a significant chemical stability risk has been validated by on-going water-quality monitoring and the absence of any impact to local water quality (ground and surface water). Monitoring of runoff will continue for the life of the Project.

Production Phase – Quarry Rock

The approximate area shown in Figure 20.1 below will be quarried to produce construction rock and to create containment in the TMF. During the envisaged production phase, this area will be eventually incorporated within the TMF footprint and covered with tailings.

Figure 20.1 Planned Quarry Area



From channel sampling, surface mapping and diamond drillhole data, the rock to be quarried is characterized as being mafic volcanics. The results of static testing indicated the following:

- Samples were predicted to be non-acid generating with an average NP/AP ratio of 33 and average sulphide-sulphur concentration of less than 0.3 % by weight.
- SFE results were below PWQO, with the exception of slightly elevated aluminum concentrations in five of six samples (average concentration 0.098 mg/L in comparison to the interim PWQO for aluminum of 0.075 mg/L). The SFE leachate sample was filtered through a 0.45 rather than a 0.2 micron filter, which is required for comparison to the interim PWQO for aluminum. Thus, the SFE result for aluminum is conservatively high. One sample also contained a slightly elevated copper concentration at 0.0059 mg/L in comparison to the PWQO of 0.005 mg/L (average concentration of copper in leachate from all six samples was below PWQO at 0.0023 mg/L).

For any additional bedrock excavation within the TMF footprint, the geochemical properties for each 10,000 tonne rock unit will be determined according to Section 5.7.8 of the Phoenix Project Closure Plan. If the rock unit meets construction rock criteria it will be used for construction purposes within the access road and the Project site. If sampled rock does not meet these criteria, it will be used for construction purposes within the TMF footprint and plant site, where runoff will be managed during the life of the Project.

As quarried rock from the Rubicon Property will be used for the construction of project infrastructure and will not be sold for commercial purposes, and as the applicable Letters of Patent do not reserve any sand and gravel to the Crown, Rubicon has confirmed that a license will not be required pursuant to the Ontario *Aggregate Resources Act* for the on-site quarry and use of overburden for construction purposes.

Production Phase – Development Rock

Principle development rock units relative to environmental geochemistry are listed below:

- Mafic volcanic
- Ultramafic (talc rich, komatiitic basalt, ultramafic flow)
- Felsic intrusive
- High titanium basalt

Most high titanium basalt rock is anticipated to be used for construction within the TMF and/or plant site. It is anticipated that most ultramafic, felsic intrusive and mafic volcanic development rock will meet criteria for use as construction rock for the Project site and access road.

Production Phase – Ore and Tailings

Tailings from ore that is processed on-site will report to either the TMF or the underground mine workings as backfill. Accordingly, ore will not be subjected to an on-going characterization program. Ore will only be handled within the plant site footprint during the life of the Project and an engineered dry cover will be constructed over the plant site at Close-out.

Calculated rate data have indicated that both representative tailings samples in Chem-Dynamics (2011) may ultimately turn acidic. These laboratory humidity cells are being continued. Tailings sampling and analysis will be continued to better understand the timing and extent to which ARD may potentially occur. This supplemental characterization program will determine the need to implement the management plan that is described in Section 5.7.8 *of the Phoenix Project Closure Plan* to address potential chemical stability concerns.

Surface Stability and Crown Pillar Assessment

There are no concerns with respect to instability at surface due to the historical underground workings (AMEC, 2008B). AMC has evaluated crown pillar requirements in accordance with Ontario Regulation 240/00 (as amended) to support the Phoenix Project Closure Plan. This document presents recommendations for crown pillar dimensions above the new proposed mine workings to prevent instability at surface. The recommended crown pillar thickness will be further reviewed as part of additional mining studies, and during the life of the mining operation as a greater understanding is gained of ore distribution, rock mass conditions and rock mass behaviour relative to mining geometry and extraction sequence.

20.2 Environmental Management Plans

Current environmental liabilities associated with the Project site are described in the *Phoenix Advanced Exploration Project Closure Plan*. There are no significant chemical or physical stability liabilities associated with historical development or the Advanced Exploration phase of the Project. The Closure Plan has not identified any significant chemical or physical stability issues. Potential liabilities associated with the conceived production phase of the project will be managed in a responsible manner that mitigates potential impacts and is conducive to "walk-away" closure. Specifically, Rubicon will:

- Identify and manage compliance with site-specific permits and applicable legislation via the *Phoenix Project Environmental Management System* and ancillary documents (collectively, the "PPEMS"). This will be updated to reflect new or amended permits and legislation or organizational conformance obligations; and prior to anticipated production to ensure management of development rock, tailings and water is in accordance with the Phoenix Project Closure Plan and the Industrial Sewage Certificate of Approval.

As indicated above, quarry rock and development rock lithologies are currently interpreted to be net acid consuming and to not pose a significant risk of metal leaching, with the possible exception of mineralized high titanium basalt.

20.2.1 Real Time Sampling Program

As per the Phoenix Project Closure Plan, a real-time sampling program to determine acid generation and metal leaching potential of development and quarry rock will be developed.

20.2.2 Development Rock and Ore Management

Development rock from the production phase of the Project will be temporarily stockpiled at the plant site in modest stockpiles (~5000 tonne) until it is sampled and characterized.

There will be no waste rock dumps created at the Project site resulting from production. Waste rock will be stored underground as fill or utilized and deposited within the TMF footprint.

Ore will be temporarily stockpiled on surface until shipped off-site for processing or processed in the mill. Ore will be fully utilized and there will be no significant stockpiles upon closure.

20.2.3 Site Runoff Management

During the operational phase of the Project, plant site runoff, TMF runoff and TMF seepage will be collected, treated and discharged in accordance with the MMER and provincial requirements and this will be the sole discharge from the "operations area" associated with the Project. The plant site and TMF will be covered with an engineered low-permeability cover at Close-out to prevent any significant infiltration of water and subsequent seepage from the TMF.

20.2.4 Tailings Disposal

The TMF constructed on the McFinley Peninsula in 1988 will be re-activated and expanded for the production phase of the Project. The existing dam will form the main embankment of the TMF and there will be no material development downstream to avoid potential impacts to fish habitat. Pertinent design aspects of the re-activated and expanded TMF are:

- Tailings will normally be thickened to around 75 % to 85 % solids via the thickener and disc filter at the paste plant prior to discharge within the TMF. This practice will reduce the size of the supernatant pond, reduce seepage, minimize or eliminate liquid-solids separation, improve the physical stability of the TMF and reduce both the risk and consequences associated with a potential failure. During thickened tailings deposition, discharge points will be frequently re-located to maximize the size of the wetted surface, thereby minimizing fugitive dust. In addition, tackifier and/or binder may be added to discharged tailings to bind particles together and minimize entrainment by wind.
- Late in the LOM, tailings solids may be dewatered in the mill to <15 % moisture content and mechanically placed within the TMF footprint using conventional heavy equipment.
- The TMF supernatant pond and effluent treatment plant are designed to withstand a 30 day duration, 1 in 100 year rain on snow event.
- During an average hydrologic year, ~200,000 m3 are predicted to be discharged from the TMF. With further recycling of treated water during commissioning and optimization, this estimated annual discharge volume is anticipated to be reduced.
- The TMF will be equipped with an engineered spillway to prevent a potential dam failure due to overtopping during a Probable Maximum Precipitation (“PMP”) event.
- A rockfill berm with internal drainage structure will be placed upstream of the existing dam. The existing dam will function as a secondary containment dam for seepage from the tailings that are deposited upstream of this structure.

The preliminary TMF design has storage capacity of 4.7 million tonnes of tailings solids, with the possibility to increase in the future by raising the height, concurrent with any requirements for amendments to necessary approvals. The TMF surface footprint is not anticipated to be materially changed as a result of any future raises. The TMF will be constructed, operated, maintained and monitored in accordance with *Guide to the Management of Tailings Facilities* (MAC, 1999) and Environment Canada (2009).

20.2.5 Management Plan to Prevent Impacts from Acid Generation and/or Metal Leaching

- The management plan to prevent impacts from acid generation and/or metal leaching will comply with regulatory requirements and will address the following:
- Ore and tailings will only be handled within the TMF and plant site footprints. Runoff from the plant site will be collected. At closure, the TMF and plant sites will be covered with an engineered, low-permeability dry cover that will minimize infiltration of water. Seepage collection around the TMF will continue post closure until the seepage decreases and no longer poses a risk to the environment, with input from MOE.
- Development and quarry rock that do not meet construction rock criteria will only be handled within the TMF and plant site footprint.
- Track-out of potentially acid-generating ore fines from the ramp portal that could pose a chemical stability risk will be controlled during the LOM. Controls will be employed in the upper portion of the ramp to passively wash the floor and equipment tires as mobile equipment travels up the ramp. Furthermore, the north sump will collect runoff from this area and direct it to the TMF where fugitive dust suppression measures will be in place.

20.2.6 Water Management

Plans for water management during operations are summarized below:

- Process water will be a combination of water from East Bay, clarified water recycled from the TMF, water from the plant site sump and treated water from the mine and mill. The volume from East Bay will be minimized along with the effluent discharge volume.
- Mine water will be used in the mill or directed to the TMF, and will be treated to destroy ammonia and hydrocarbons using ozone (as per the Industrial Sewage Certificate of Approval). Excess water from the TMF will be treated using an Actiflo treatment system and discharged as per regulatory and certificate of approval requirements.
- Preliminary details have been determined for the seepage collection system that will be constructed to collect and pump back runoff from the downstream TMF embankments and potential seepage through perimeter berms. The seepage collection system will be in accordance with the issued Industrial Sewage Certificate of Approval.
- Project site runoff will report to a centralized plant site sump system. Collected water will report to the mill and/or mine process during the production phase. The plant site sump will be in accordance with the issued Industrial Sewage Certificate of Approval.
- For potable water, raw water supplied to the tank at the plant site from the existing pump house on East Bay would be treated as per provincial regulations prior to use at the site. Black water would continue to be managed using the approved tertiary treatment subsurface disposal system. In addition, portable (chemical or composting) toilets would be used at selected areas of the site to dispose of black water. Grey water (showers and sinks) will be disinfected as appropriate and directed to the TMF or for re-use in the mill process, in accordance with the Industrial Sewage Certificate of Approval.
- The Storm Water Control Study required by Section 33 of O. Regulation 560/94 would be conducted once the Project becomes subject to this regulation.
- At Close-out, the TMF, the TMF seepage collection system and plant site sump will be dewatered by treating and discharging water to the underground workings and/or the environment in accordance with the issued Industrial Sewage Certificate of Approval and relevant regulatory requirements. This practice will be continued until the low-permeability cover is constructed over the TMF and plant site to minimize infiltration of precipitation and resultant flushing of porewater from the plant site and TMF. Following placement and vegetation of the low-permeability cover, clean surface runoff from the TMF will be routed to the toe of the existing dam via an engineered spillway channel, so as to return the area to the pre-development drainage pattern and sustain the downstream fish habitat. The low-permeability cover over the plant site will route clean surface runoff eastward to East Bay.

20.3 Project Permitting

20.3.1 Current Approvals

A Form 1 Notice of Project Status was submitted to MNDMF in Q4 2009 to move the Project from Preliminary Exploration to Advanced Exploration status, in accordance with Section 140 of the *Mining Act.* Current approvals for the on-going Advanced Exploration program work are listed in Table 20.1.

Table 20.1 Summary of 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	11 December 2008	Withdrawal of water from shaft
Permit to Take Water 6020-7LHPX9 (amended to 3585-85KGHG)	Ministry of Environment	Ontario Water Resources Act	19 November 2008	Withdrawal of water from East Bay of Red Lake
Certificate of Approval - Sewage 4192-7JRJ3L (amended to 9305-8C5S6Z)	Ministry of Environment	Ontario Water Resources Act	27 January 2009	Approve sewage works to manage industrial waste water
Certificate of Approval – Sewage 1384-86HQR8	Ministry of Environment	Ontario Water Resources Act	13 July 2010	Approve domestic sewage disposal system.
Certificate of Approval - Air 9500-7NGTTC	Ministry of Environment	Environmental Protection Act	27 January 2009	Approve air emissions from site
Class Environmental Assessment pursuant to Ontario Regulation 116/01	Ministry of Environment	Environmental Protection Act	14 April 2011	Allow 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 Project site.
LRIA Approval No. RL-2009-01	Ministry of Natural Resources	Lakes and Rivers Improvement Act	23 January 2009	Approve existing containment dams associated with historic TMF
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	Application submitted March 2010 and April 2010, respectively. Letter of Authority issued to approve work in 17 January 2011.	Approve easement over Crown owned surface rights; tree harvesting, power line construction and access road upgrade / extension
Phoenix Advanced Exploration Project Closure Plan	Ministry of Northern Development and Mines & Forests	Mining Act	27 February 2009	Approve development and closure of the Advanced Exploration phase. Financial assurance of ~$493,000 provided with this closure plan.
Amendment to the Zoning By-Law 1277-10	Municipality of Red Lake	Municipal By-Law 1277-10	Process completed.	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.

20.3.2 Recent Approval Application

A Form 1 Notice of Project Status was submitted to MNDMF in Q1 2011 to move the Project from Advanced Exploration to Production status, in accordance with Section 141 of the *Mining Act.* The submission of this Notice did not give indication of commencement of commercial production, but rather the intention to continue development of the Project in accordance with a Project (production) Closure Plan that would be a precursor to such production.

The list of approvals sought for the production phase of the Project, as well as the status of those applications, is shown in Table 20.2 below. The approvals that relate to the Advanced Exploration phase of the Project generally apply to the production phase of the Project also, resulting in a short list of required, additional approvals.

Table 20.2 Summary of Production Phase Approvals

Permit	Regulatory Agency	Relevant Legislation	Date of Application	Rationale
New Certificate of Approval to replace Certificate of Approval - Sewage 4192-7JRJ3L	Ministry of Environment	Ontario Water Resources Act	14 January 2011	Approve the modifications to the sewage works to manage tailings from the on-site mill and industrial waste water
Phoenix Project Closure Plan	Ministry of Northern Development and Mines & Forests	Mining Act	Q3 2011	Approve development and closure of the Production phase
Amendment to LRIA Approval No. RL-2009-01	Ministry of Natural Resources	Lakes and Rivers Improvement Act	~Q4 2011	Structural "approval" for dams associated with TMF

20.3.3 Applicable Environmental Assessment Processes

The postings on the provincial Environmental Registry for the permits listed below have not resulted in negative comments or requests for an individual environmental assessment, pursuant to Ontario's *Environmental Assessment Act.*

- Air Certificate of Approval 9500-7NGTTC and subsequent amendment applications
- Industrial Sewage Certificate of Approval 4192-7JRJ3L, subsequent amendment applications and new application for the TMF
- Permit to Take Water 6020-7LHPX6 and subsequent amendments (draw fresh water from East Bay)
- Permit to Take Water 2342-7LWRQU and subsequent amendments (dewater underground workings)
- Phoenix Advanced Exploration Project Closure Plan
- Phoenix Project (production) Closure Plan

A Class Environmental Assessment ("EA") has been completed in response to the requested easement for the access road and power line right of way to connect the Project site to Nungesser Road and work associated therein. No negative comments were received during this process, which was conducted in accordance with MNR (2003).

The Canadian Environmental Assessment Agency ("CEAA") has confirmed that the production phase of the Project will not trigger an EA pursuant to the *Canadian Environmental Assessment Act.*

As the Project progresses to an anticipated production scenario it is planned to initiate sustainability performance reporting in general accordance with the requirements of the Global Reporting Initiative (http://www.globalreporting.org/Home).

20.4 Social and Community Aspects

20.4.1 Public Information and Comments

Annual public information sessions were held in the Red Lake community in December 2008, 2009 and 2010. No negative comments were, or have been received to date.

A Notice of Commencement of Screening was published and circulated from September 2010 to November 2010, pursuant to Ontario Regulation 116/01 (Electricity Projects Regulation), related to the proposed additional use of diesel generators at the Project (<5MW cumulative capacity). The public information session held on 15 December 2010 was prepared to satisfy the requirements of Ontario Regulation 116/01 and Section 141 of the *Mining Act*. No comments or questions have been received by Rubicon regarding the Project or any other aspect of the Project since the enhanced public consultation commenced in September 2010.

The Class EA that was required pursuant to Ontario Regulation 116/01 was completed and circulated for public comment in March 2011. No comments were received and the process was completed in April 2011.

To date, there have been no complaints received by Rubicon regarding activities at the Project site. One comment has been 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 remains committed to minimizing nuisance noise associated with its activities in the Municipality of Red Lake.

One comment was received by MNR as a result of the Class EA process related to the requested easement for the access road and power line right of way to connect the Project site to Nungesser Road. It is understood that the comment was positive as per communications with staff from the Red Lake District MNR Office.

Rubicon maintains an open-door policy to proactively identify and address stakeholder concerns regarding the Project.

Table 20.3 below summarizes the Phoenix Project public consultation to date:

Table 20.3 Summary of Public Consultation

Date	Summary of Public Consultation that was Undertaken	Summary of Information Provided	Summary of Comments that were Received (if any)
Dec 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 Phoenix Project, including the diesel generator aspect.	No comments received in relation to any aspect of the Phoenix Project. There was a general discussion regarding the modernization of the *Mining Act*.
Dec 2009	Voluntary Annual Public Information Session. Notice was in general accordance with Section 8 of O. Regulation 240/00.	Overview PowerPoint presentation of the Phoenix Project, including the diesel generator aspect.	No comments received in relation to any aspect of the Phoenix 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 Phoenix Project.
Sep 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 Phoenix Project.
Dec 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.

20.4.2 Aboriginal Consultation

To date, Rubicon has undertaken consultation with aboriginal communities under the guidance of the MNDMF.

Rubicon commissioned an independent traditional use study that concluded that the Project site is within the traditional territory of Lac Seul First Nation and Wabauskang First Nation (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 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 the aboriginal communities described herein to further discuss and identify cultural heritage values within the development footprint that may warrant protection.

For the envisaged future operational and closure phases of the Project, Rubicon remains committed to continuing consultation with aboriginal communities that may be affected by the Project under the guidance of MNDMF. The strength of this commitment is evidenced by the consultation logs with Lac Seul First Nation, Wabauskang First Nation and the Métis Nation of Ontario that date back to 2007 / 2008.

In addition to satisfying consultation requirements in accordance with Ontario Regulation 240/00 (as amended) and Section 35 of the *Constitution Act*, Rubicon will seek agreements with the above noted aboriginal communities.

20.4.3 Ecosystem and Human Habitation Risk

Rubicon commissioned a conservative risk assessment to quantify the potential risks to valued ecosystem components (“VECs”) and 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).

20.5 Mine Closure (Remediation and Reclamation) Requirements and Costs.

20.5.1 Mine Closure Requirements

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. Chemical and physical stability requirements will be satisfied and monitored in accordance with regulatory requirements pursuant to Part VII of the *Mining Act*.

Close-out rehabilitation activities will be completed within approximately 36 months of project closure; major activities are summarized 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.
- The ramp portal will be filled and barricaded as per regulatory requirements. Based on the observed static water level in the underground workings from 2002 to 2009, there is potential for overflow from the ramp, therefore the barricade will be designed and constructed as a concrete bulkhead to prevent water outflow.
- The shaft and ventilation raise will be partially backfilled and sealed with an engineered concrete cap to prevent access. The site sump will be operated on a continuous basis to direct overburden ground water into the underground workings until demobilization is completed and the dry cover is being placed over the plant site and vegetated.
- 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 reduce work that will be required at Close-out. Post closure, the spillway channel will be lowered to prevent ponding of runoff and 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&F post Close-out, once the seepage rate decreases and is demonstrated to not pose an environmental risk and the issued Industrial Certificate of Approval is amended.
- 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 drainage structure upstream of the existing TMF dam will be operated on a continuous basis to pump the small volume of residual pore water to the underground workings until demobilization is completed. The drainage structure will be backfilled with development rock when it is decommissioned and a French drain will be installed over the existing dam to route potential pore water from this vicinity to the downstream toe of the existing dam. Non-vegetated areas of the downstream embankment will be covered with soil and re-vegetated.

- Site roads will be rehabilitated in general accordance with MNR (1995).
- Pipelines (water, compressed air) on the site will be flushed and left in place. Fuel pipelines (propane / natural gas) will be decommissioned as per legislative requirements and Technical Standards and Safety Association ("TSSA") 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.
- The long-term chemical and physical stability monitoring program will be continued to completion, in accordance with the Phoenix Project Closure Plan.

The access road and utility corridor from Nungesser Road to the Property are outside the scope of the Project 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 Red Lake Municipality.

20.5.2 Closure Costs

Rubicon has provided financial assurance in the amount of $493,000 as per the *Phoenix Advanced Exploration Project Closure Plan.* This amount is adequate to rehabilitate the features of the advanced exploration phase of the Project in accordance with Part VII of the *Mining Act* and other applicable regulatory requirements.

Pursuant to Section 141 of the *Mining Act* and in preparation for potential mine production, Rubicon has prepared the certified Phoenix Project Closure Plan and negotiated the payment schedule with MNDMF that is summarized below:

- $955,360 upon filing of the Phoenix Project Closure Plan (Phase 1)
- $1,421,998 before March 31 2012 (Phase 2)
- $40,279 prior to initiating development of the ramp portal
- Rubicon proposes to submit a Form 2 (Notice of Material Change) to refine the certified rehabilitation measures for the on-site TMF, the required financial assurance amount and to provide the financial assurance prior to the generation of tailings. The Phoenix Project Closure Plan provides a certified preliminary design for a dry cover (low-permeability cover) over the TMF and plant site as referenced earlier. Rubicon intends to optimize the design and estimate the cost of the dry cover prior to the generation of tailings, and has also committed to progressively place selected portions of the dry cover during the life of the TMF to minimize the reclamation work that is required at Close-out.

The above costs have been prepared by an independent engineer, as documented in the Phoenix Project Closure Plan, and are certified to be adequate for the rehabilitation of the Project in accordance with Part VII of the *Mining Act* and regulatory requirements.

In parallel with the optimization of the dry cover design and associated independent cost estimation, Rubicon is evaluating the salvage value of assets associated with the Project. Pending this evaluation, it is planned that appropriate Project assets will be pledged to a third party in exchange for a surety bond or other financial instrument to provide financial assurance, as required by the Phoenix Project Closure Plan. The salvage value of these Project assets is expected to be equivalent to the costs for the remaining rehabilitation work associated with the dry cover at Close-out.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

Capital costs have been estimated and profiled for both pre-production and sustaining timeframes, with a 30% contingency applied. AMC acknowledges the extent of existing infrastructure at the Phoenix site and that the degree of detail with which further infrastructure has been planned is at a higher level than for many PEA projects. It has deemed, however, that a 'normal' PEA level of contingency (30%) is in the best interests of the Project. This may be particularly so if a relatively significant degree of mechanized equipment is ultimately introduced to the Project. Total pre-production capital is estimated at $214 M. Sustaining capital, inclusive of permanent development, is estimated at $125 M.

21.1.1 Pre-Production Capital

Capital costs for the envisaged two-year pre-production period are summarized in Table 21.1.

Table 21.1 Pre-Production Capital Summary

Pre-Production Capital ($)	Total	Y01	Y02
Surface Infrastructure	98,000,000	32,500,000	65,500,000
U/G Infrastructure	51,462,000	25,485,000	25,977,000
Power & Utilities & Mine Supplies	4,140,000	2,070,000	2,070,000
Rubicon Project Team	8,138,000	4,069,000	4,069,000
Engineering (civil/mech/elec/geotech/mill)	2,000,000	1,000,000	1,000,000
Project Administration	1,000,000	500,000	500,000
Environmental Disbursements - sampling	180,000	90,000	90,000
Sub-Total Pre-Production Capital	164,920,000	65,714,000	99,206,000
Contingency at 30%	49,476,000	19,714,000	29,762,000
Total Pre-Production Capital	**214,396,000**	**85,428,000**	**128,968,000**

90% of the envisaged pre-production capital is shared between two areas, namely Surface Infrastructure including the concentrator complex (59%), and Underground Infrastructure including underground development (31%). A pre-contingency breakdown for each of these areas is given in Tables 21.2 and 21.3 below.

The major component (75 %) of the surface infrastructure cost is the concentrator complex, which includes the paste plant (see Section 17).

Excavation items account for 76% of the underground infrastructure capital: shaft and immediate access (30%), main lateral development (22%), stope accesses (6%), and raising (18%).

Table 21.2 Surface Infrastructure Capital Cost

Surface Infrastructure ($M)	Total	Y01	Y02
Security / Fencing	250,000	250,000	
Dry & Admin. Building	9,000,000	2,000,000	7,000,000
Parking - Lighting - Winter plug-in	100,000		100,000
Potable Water	250,000	125,000	125,000
Sewage Treatment Plant			purchased
Shop/warehouse	750,000	500,000	250,000
Compressors (2)			purchased
Hoisting Plant / Headframe			purchased
Actiflo & Ozone Treatment Plant	2,100,000	1,000,000	1,100,000
Change Over (Sinking to Production)	250,000	250,000	
Mill, including Paste Plant	73,541,000	20,000,000	53,541,000
Tailings (Phase 1)	5,000,000	4,000,000	1,000,000
Power Line & Sub-Stations			purchased
Water Treatment Plant	500,000	500,000	
Ventilation - Intake	300,000	300,000	
Ventilation - Exhaust	200,000	200,000	
Road Improvement	2,000,000	1,500,000	500,000
Equipment (loader/forklilft/trucks)	770,000	385,000	385,000
Camp	3,000,000	1,500,000	1,500,000
Sub-total	**98,011,000**	**32,510,000**	**65,501,000**

Table 21.3 Underground Infrastructure Capital Cost

U/G Infrastructure ($M)	Total	2011	2012
Shaft / Access	15,400,000	10,000,000	5,400,000
Internal Development / Ramp	11,317,000	4,871,000	6,446,000
Stope Access	3,143,000	1,395,000	1,748,000
Raising (ventilation/orepass/waste pass)	9,052,000	3,032,000	6,020,000
Room & Board & Travelling	3,000,000	1,500,000	1,500,000
Mining Equipment (narrow vein)	2,000,000	1,000,000	1,000,000
Mining Equipment (Development)	3,000,000	1,500,000	1,500,000
Electrical Sub-stations & cables	1,000,000	500,000	500,000
Loading Pocket	250,000	187,500	62,500
Haulage System, Loading, Rockbreaker	500,000	250,000	250,000
Chutes / Loading / Unloading	300,000	100,000	200,000
Pumping Station	250,000		250,000
U/G Ventilation	250,000	150,000	100,000
Phone / Leaky feeder / Radio Cap Lamps	1,000,000	600,000	400,000
Fill Line Distribution	1,000,000	400,000	600,000
Sub-Total	**51,462,000**	**25,485,500**	**25,976,500**

Note the figures in both Table 21.2 and 21.3 are before contingency.

21.1.2 Sustaining Capital

Sustaining capital costs over the PEA 12-year production life are summarized in Table 21.4.

Table 21.4 Sustaining Capital Cost

Sustaining Capital ($M)	Total	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14
Surface Infrastructure	7,713,000	4,913,000		2,800,000									
U/G Infrastructure	29,686,000	3,481,000	1,375,000	12,830,000	3,100,000	1,975,000	1,000,000	1,200,000	1,575,000	1,575,000	1,575,000		
Rubicon Project Team	2,074,000	1,292,000	195,000	196,000	195,000	196,000							
Engineering	250,000	250,000											
Project Administration	125,000	125,000											
Sub-Total Sustaining Capital	39,848,000	10,061,000	1,570,000	15,826,000	3,295,000	2,171,000	1,000,000	1,200,000	1,575,000	1,575,000	1,575,000		
Contingency @ 30%	11,954,000	3,018,000	471,000	4,748,000	989,000	651,000	300,000	360,000	472,000	473,000	472,000		
Sustaining Capital Total	51,802,000	13,079,000	2,041,000	20,574,000	4,284,000	2,822,000	1,300,000	1,560,000	2,047,000	2,048,000	2,047,000		

The major component of the sustaining capital is underground infrastructure, which makes up almost 75 % of the total estimate over the projected LOM. A breakdown of the underground infrastructure capital is shown in Table 21.5. No sustaining capital is envisaged in the final two years of the projected LOM.

Table 21.5 Underground Infrastructure Sustaining Capital (pre-contingency)

U/G Infrastructure	Total	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10
Shaft / Access	10,980,000			10,980,000							
Room & Board & Travelling	156,000	156,000									
Mining Equipment (narrow vein)	5,000,000	500,000	500,000	500,000	500,000	500,000	500,000	500,000	500,000	500,000	500,000
Mining Equipment (Development)	5,062,000	562,000	500,000	500,000	500,000	500,000	500,000	500,000	500,000	500,000	500,000
Electrical Sub-stations & cables	1,013,000	213,000		200,000		200,000			200,000		200,000
Loading Pocket (3)	500,000	250,000			250,000						
Rockbreaker / Grizzly Installation	900,000	450,000			450,000						
Haulage System & Loading	1,000,000	500,000			500,000						
Chutes / Loading / Unloading	550,000	275,000			275,000						
Pumping Station & Excavation	500,000	250,000			250,000						
U/G Ventilation	1,000,000	200,000		200,000		200,000		200,000		200,000	
Phone/Commn./Microseismic	1,212,500	62,500	125,000	200,000	125,000	325,000			125,000	125,000	125,000
Fill Line Distribution	1,812,500	62,500	250,000	250,000	250,000	250,000			250,000	250,000	250,000
Sub-Total	29,686,000	3,481,000	1,375,000	12,830,000	3,100,000	1,975,000	1,000,000	1,200,000	1,575,000	1,575,000	1,575,000

21.2 Operating Costs

An average total operating cost of $214 /tonne has been estimated. Per recovered Au ounce, the average operating cost is estimated at $519. A category breakdown of operating costs is shown in Table 21.6.

Table 21.6 Operating Costs Summary

Operating Costs ($'000)	Total	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14
Labour	398,889	25,298	27,536	36,645	36,636	36,627	36,628	36,652	36,627	36,641	33,215	31,244	25,139
Contractor	35,164	4,149	3,010	3,984	2,894	3,276	3,760	3,218	3,640	3,377	2,356	1,112	390
Material	254,057	16,279	20,534	25,151	25,248	25,417	25,383	25,095	25,417	25,194	19,086	15,750	5,503
Milling	98,991	5,940	7,700	9,900	9,900	9,900	9,900	9,900	9,900	9,900	7,590	6,270	2,191
Fill Plant/System	62,994	3,780	4,900	6,300	6,300	6,300	6,300	6,300	6,300	6,300	4,830	3,990	1,394
Reclamation	4,500	270	350	450	450	450	450	450	450	450	345	285	100
Delineation Drilling	54,588	5,000	6,481	5,000	5,000	5,000	5,000	5,000	5,000	5,000	3,833	3,167	1,107
G&A	8,203	751	973	752	751	752	751	752	751	752	576	476	166
Housing	44,996	2,700	3,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	3,450	2,850	996
Total	962,381	64,167	74,984	92,682	91,680	92,220	92,673	91,867	92,584	92,113	75,282	65,143	36,986
Cost $/t	213.88	237.7	214.2	206	203.7	204.9	205.9	204.1	205.7	204.7	218.2	228.6	371.4
Cost $/oz Au	518.6	589.0	536.9	516.5	524.4	525.4	592.6	503.1	495.7	450.8	453.7	484.6	787.7

In the steady state years (Y05 to Y11), when production is projected at 450,000 tpa (1250 tpd), the average operating costs per tonne and per recovered Au ounce are $205 and $516 respectively.

21.2.1 Mining Operation Costs

Mining operation costs have been developed using current contractor and labour rates for personnel and on a unit basis for power, explosives, ground support, etc. Total mining operation costs average about $150 per tonne, with the labour portion (including contractors) at around $95 per tonne reflecting the manpower-intensive nature of captive cut and fill mining in the lode-style resource. It is assumed that contractor labour will be used for all raising, including raiseboring, and that Rubicon personnel will be responsible for capital and operating development and stoping. Labour rates used in the PEA generally reflect those that AMC understands are currently typical in areas such as Red Lake.

A breakdown of mining operation costs is shown in Table 21.7 below.

Table 21.7 Mining Operation Costs

Mining Operations Cost ($'000)		Totals	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14
Labour (Direct)																
	Operating Dev	5,068			919	629	352	471	674	634	283	674	405	28		
	Stoping	206,433			10,509	14,636	20,410	20,323	20,182	20,209	20,461	20,182	20,371	16,783	14,597	7,770
Labour (Indirect)																
	Admin	24,681			1,984	1,984	2,071	2,071	2,071	2,071	2,071	2,071	2,071	2,071	2,071	2,071
	Technical	45,744			3,224	3,224	3,930	3,930	3,930	3,930	3,930	3,930	3,930	3,930	3,930	3,930
	Operations	116,963			8,662	7,063	9,883	9,841	9,770	9,784	9,907	9,770	9,864	10,404	10,647	11,369
Contact Labour																
	Ralsing	17,174			3,006	1,588	2,200	1,100	1,464	1,952	1,440	1,828	1,588	1,008		
Material	Operating Dev	4,262			862	549	283	380	549	515	227	549	327	21		
	Stoping	93,369			5,603	7,263	9,338	9,338	9,338	9,338	9,338	9,338	9,338	7,159	5,914	2,067
	Admin	2,250			135	175	225	225	225	225	225	225	225	172	142	50
Mining Supplies & Chutes		40,541			2,433	3,154	4,055	4,054	4,055	4,054	4,055	4,054	4,055	3,108	2,568	897
Underground services		15,749			945	1,225	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,207	997	349
Maintenance		25,288			1,517	1,967	2,529	2,529	2,529	2,529	2,529	2,529	2,529	1,939	1,602	560
Utilities		56,850			3,839	4,977	5,572	5,571	5,571	5,571	5,572	5,571	5,572	4,271	3,529	1,233
Ore Handling		11,249			675	875	1,125	1,125	1,125	1,125	1,125	1,125	1,125	862	712	249
Surface Services & Misc.		17,998			1,080	1,400	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,380	1,140	398
Totals		683,618			45,394	50,708	65,346	64,334	64,857	65,313	64,537	65,221	64,773	54,344	47,849	30,942
Cost/t $		152			168	145	145	143	144	145	143	145	144	158	168	311

21.2.2 Concentrator Complex Operation Costs

Milling and fill plant/fill system costs have been estimated to average $22 per tonne and $14 per tonne respectively.

Concentrator costs are further discussed in Section 17 of the Technical Report.

21.2.3 Other Operation Costs

An allowance of $12 per tonne has been made for ore delineation drilling relative to current drilling cost rates and an assessment of the likely drilling requirements.

Also notable in the operating costs is an allowance of $10 per tonne for personnel housing, as per advice from Rubicon, re an assessment of the likely extent of camp accommodation that would be required relative to total number of employees and those likely to live permanently in the local area.

$1 per tonne and $1.82 per tonne are the averages projected for reclamation provision and local G&A respectively, as per advice from Rubicon that AMC believes to be reasonable.

22 ECONOMIC ANALYSIS

22.1 Base Case Economics (Pre-Royalty and Pre-Tax)

AMC has assessed the pre-royalty and pre-tax economics of the Project using the base case parameters listed below:

- Au price: $US1100 /oz
- Exchange rate: $CAN1 = $US1
- Discount rate: 5%
- Gold recovery from mined ounces of 92.5%

Table 22.1 summarizes production, cost, revenues and economics for the base parameters.

Table 22.1 PEA Production, Cost, Revenue and Economics

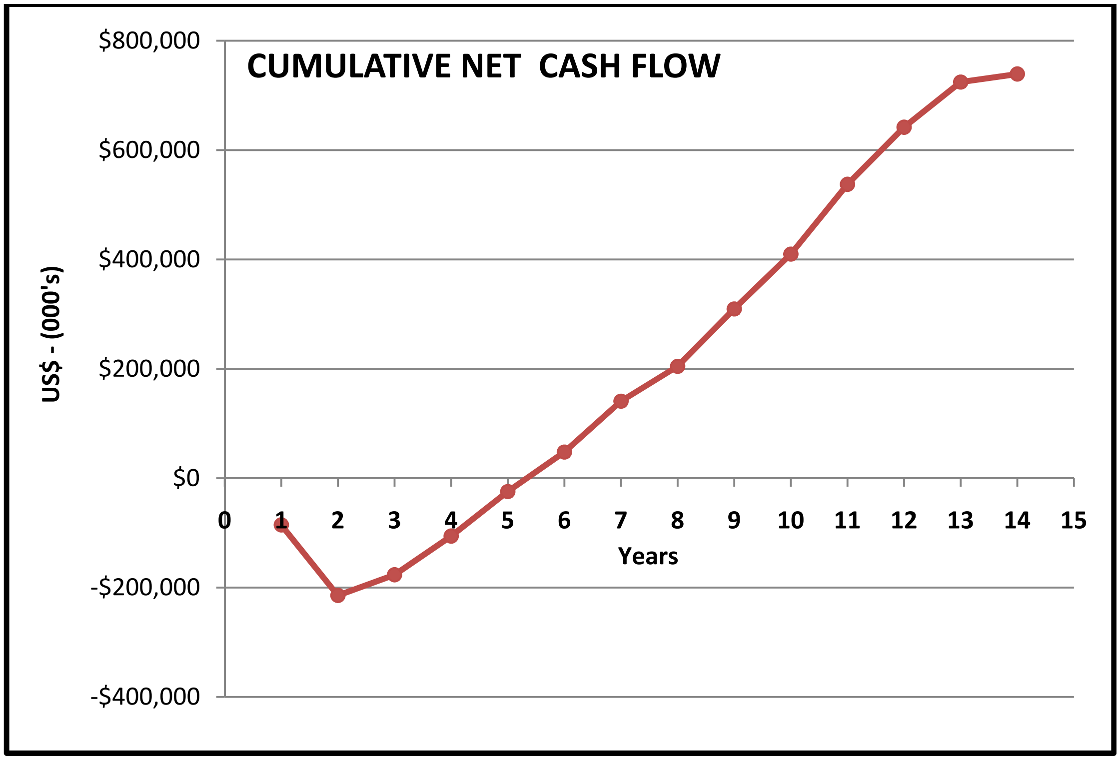
		Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14
Production Tonnes	4,499,588			270,000	350,000	450,000	450,000	450,000	450,000	450,000	450,000	450,000	345,000	285,000	99,589
Mining Rate (TPD)				750	972	1,250	1,250	1,250	1,250	1,250	1,250	1,250	958	792	277
Gold Recovery				92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%	92.5%
Diluted Grade	0.45			0.44	0.43	0.43	0.42	0.42	0.38	0.44	0.45	0.49	0.52	0.51	0.51
Net Oz/year				108,935	139,650	179,444	174,824	175,516	156,396	182,605	186,769	204,323	165,921	134,437	46,955
Cumulative Oz	1,855,774			108,935	248,585	428,029	602,853	778,370	934,765	1,117,370	1,304,139	1,508,463	1,674,383	1,808,820	1,855,774
Gold Price ($/OZ)	$1,100			$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100	$1,100
Exchange Rate	1.00			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
Revenue (CDN$ - 000's)	$2,041,352		0	119,828	153,615	197,388	192,307	193,068	172,035	200,865	205,446	224,756	182,513	147,880	51,650
Costs - (CDN$ - 000's)															
Pre-Production Capital	$214,415	85,443	128,971												
Sustaining Capital	$51,802			13,079	2,041	20,573	4,284	2,821	1,300	1,560	2,048	2,048	2,048		
Permanent Development	$73,538			4,865	5,835	2,620	24,203	5,047	14,259	2,422	10,440	3,019	829		
Operating /Stoping Cost	$962,381			64,167	74,984	92,682	91,680	92,220	92,673	91,867	92,584	92,113	75,282	65,143	36,986
Cash Flow Pre Royalties	$739,217	-85,443	-128,971	37,717	70,755	81,514	72,140	92,979	63,804	105,016	100,374	127,576	104,354	82,737	14,664
Cum. CF Pre Royalties		-85,443	-214,415	-176,697	-105,942	-24,428	47,711	140,690	204,494	309,511	409,885	537,461	641,816	724,553	739,217
NPV	$432,699														
IRR	28%														
Payback	5.3														
Total Cost/Tonne	$289.39														
Total Opex Cost/Tonne	$213.88	NB. All $ Canadian other than for Gold Price													

Key results from the assessment are:

- Gross revenue: $2.0 billion
- Total costs: $1.3 billion
- Gross revenue (pre-royalty and pre-tax): $739 million
- $NPV_{5\%}$: $433 million
- IRR: 28%
- Payback Period: 5.3 years from start of pre-production period (3.3 years from start of production).

Figure 22.1 shows the net cash flow profile over the 14-year project life (2 years pre-production and 12 years production).

Figure 22.1 Cumulative Net Cash Flow

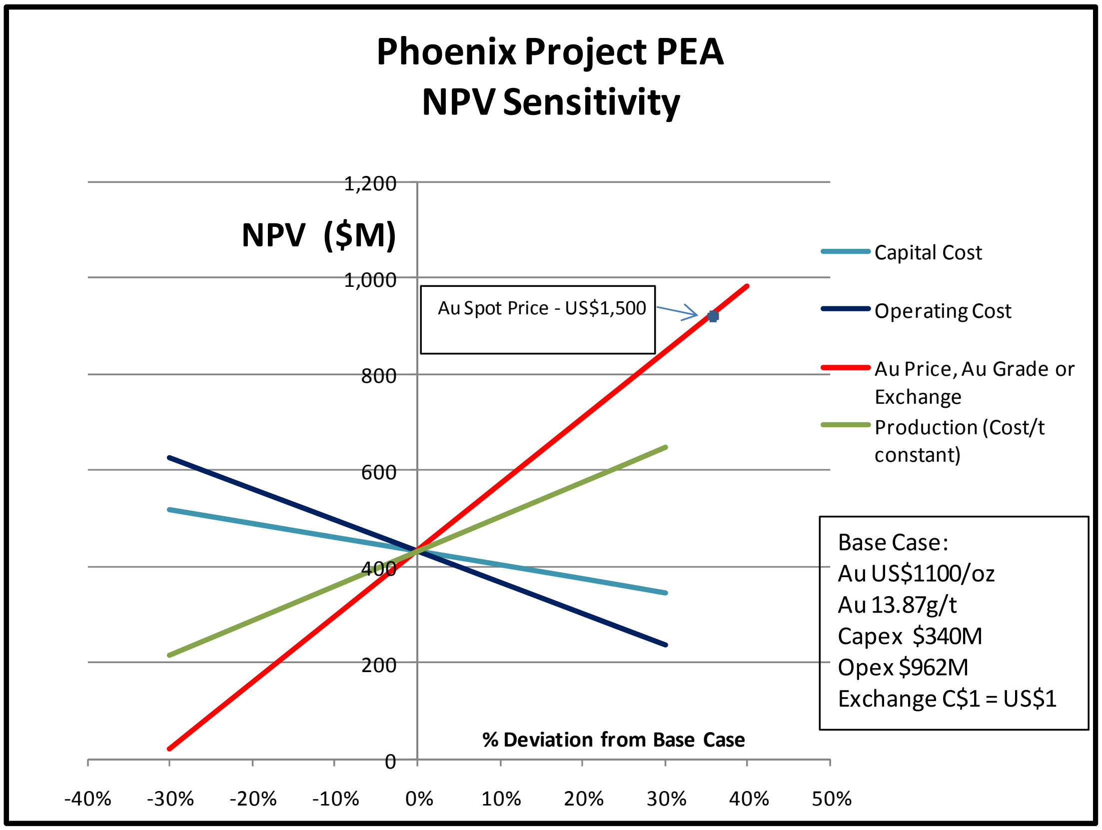


22.2 Economic Sensitivity

AMC has examined the sensitivity of the project economics (pre-royalty and pre-tax) relative to variations in gold price/grade, capital cost, operating cost and production. It should be noted that the gold price/grade assessment results are essentially the same as those that are realized by simply varying the exchange rate, e.g. a 10% drop in gold price or grade has the same economic impact as a 10% increase in the Canadian dollar against the US dollar, with all other factors remaining constant.

Figure 22.2 shows project NPV sensitivity for each of the factors noted. In examining the impact of the variation in any factor, all other factors have been assumed to remain constant. In the case of the variation in production, total operating costs have been varied proportionately with production so that the cost/tonne has remained constant.

Figure 22.2 Economic Sensitivity Chart



Gold Price or Grade Variation

Figure 22.2 shows that the Project is most sensitive to variation in gold price or gold grade (or exchange rate).

For the base case parameters, a drop in gold price or grade by 30% (gold at $770 /oz or (mined) grade at 9.7 g/t) gives a project NPV of $20 M, a 6% IRR, and a Payback Period from start of pre-production of just over 10 years. Conversely, a gold price or grade that is 3% higher than the base case (gold at US$1,430 /oz or (mined) grade at 18.0 g/t) shows NPV at US$1,352, IRR at 45% and a Payback Period from start of pre-production of just over 4 years. As also indicated in Figure 22.2, at a gold price of $1,500/oz (note 8 August 2011 London AM fix at US$1,710 /oz), NPV is US$933 M, IRR is 48% and the Payback Period is almost exactly 4 years.

Capital Cost Variation

The Project has a relatively small NPV sensitivity to changes in capital cost. A 30% increase or decrease in capital cost results in a corresponding 20% decrease or increase in NPV. There is a greater sensitivity in terms of IRR for the same variation range, with the low IRR being 21% (- 25% from base case) and the high IRR being 41% (+45% from base case), which is a reflection of the timing of capital expenditures (major capital in Y01 and Y02, much lower capital over the producing mine life). Payback Periods range from 6.4 to 4.4 years for the same +/-30% variation.

Operating Cost Variation

The Project shows a greater NPV sensitivity to variation in operating cost, with a +/-30% range in that factor giving a low-end NPV of $237 M (-45% from base case) and the high-end NPV being $628 M (+45% from base case). IRR varies from 19% to 37% for a +/- 30% change in operating cost. The corresponding Payback Periods are 6.8 and 4.6 years.

Production Variation

The NPV sensitivity to production variation is approximately the same as that for operating cost variation, with the +/-30% change resulting in respective low-end and high-end values of $216 M and $650 M. IRR variation is from 37% to 18%, while that for Payback Period is from 4.6 to 6.9 years.

22.3 Impact of Higher Gold Price on Breakeven Gold Grade

Figure 22.2 and the above discussion on gold price and grade indicate that a breakeven gold grade under otherwise constant base case parameters is around 32% lower than the average head grade in the PEA, namely about 9.5 g/t versus 13.9 g/t. For a gold price of $1500 /oz, the breakeven gold grade is around 7.0 g/t. The average mined grade at the Project will be critical to overall project viability.

22.4 Royalties and Taxes

The Rubicon Phoenix Project royalty payment obligations have been discussed in Section 4 of the Technical Report. AMC considers that the extent of the royalty obligations is such that it will not materially affect the potential viability of the Project.

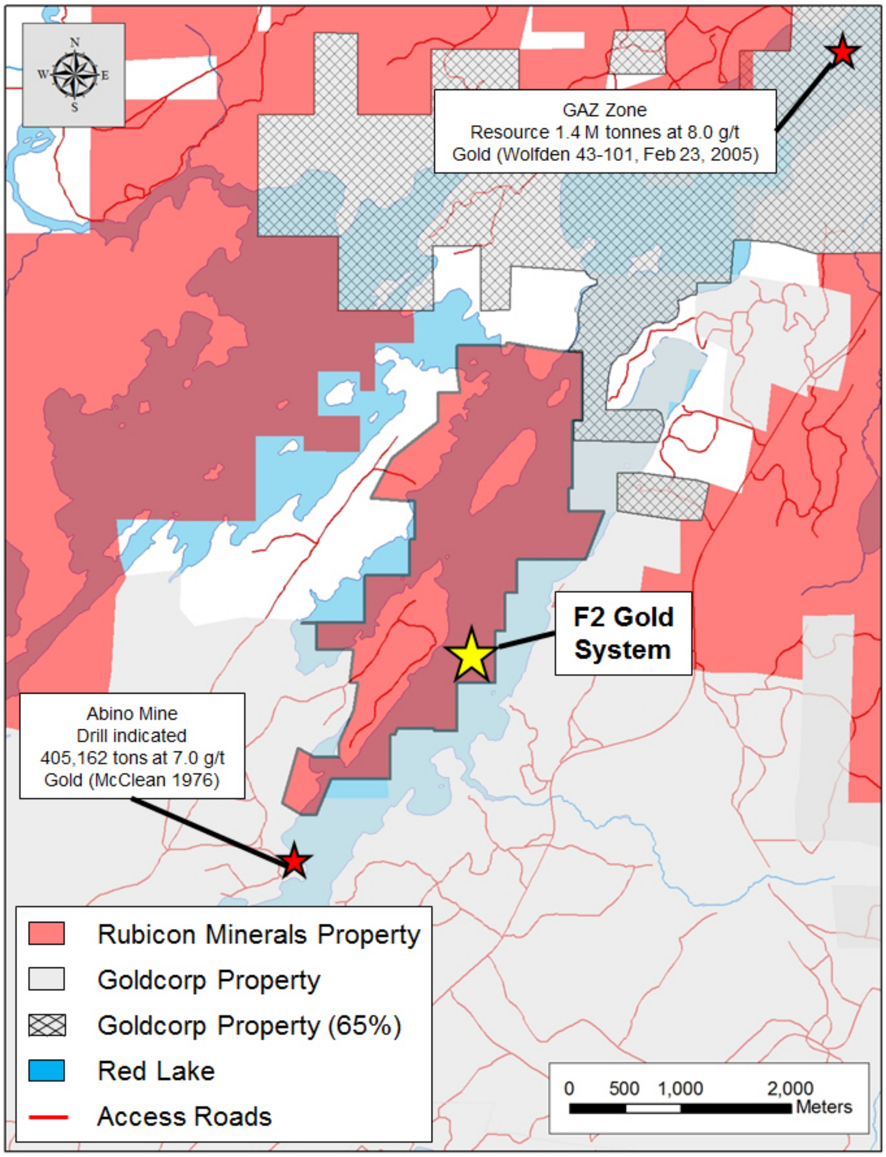
AMC does not have expertise in taxation and has not considered any tax or government levies in its economic assessment.

Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve. No mineral reserves have been estimated as part of the present study.

23 ADJACENT PROPERTIES

Figure 23.1 shows the Phoenix Property and other properties adjacent to it.

Figure 23.1 Phoenix and Adjacent Properties



23.1 East Bay

The East Bay property, hosting the 'GAZ' Gold Zone, controlled by Goldcorp Inc. (65%) and Premier Gold Mines Limited (35%) is located approximately 7.2 kilometres northeast of the F2 Gold System and 4.4 km northeast of the Phoenix claim boundary (Figure 23.1). The East Bay property is underlain by a package of Balmer assemblage rocks, dominated by tholeiitic basalts, komatiite flows, pillowed tholeiitic basalts and minor amounts of thinly bedded magnetite-chert iron formation. The gold mineralization is structurally controlled and spatially related to the northeast trending East Bay Deformation Zone and east-west cross cutting high strain zones. Individual mineralized lenses are relatively flat lying, strike northeast and dip between 25º to 45º to the northwest. The style of mineralization is characterized by visible gold in irregular quartz-carbonate stringers and silica flood zones. These zones are hosted in a dark coloured sulphidic altered rock containing conspicuous quantities of biotite, actinolite and minor fuchsite. Gold grades average 4.0 to 6.0 g/t, over widths up to 12.0 m. A 2005 Technical Report for Wolfden Resources Inc. ("Wolfden") by G.A. Harron describes an inferred resource estimate for the GAZ Gold Zone of 1,399,105 tonnes, grading 8.0 g/t gold for a total content of 326,407 ounces of gold. Wolfden was the owner of the property prior to Goldcorp Inc. and Premier Gold Mines Limited.

23.2 Abino Mine Site

The currently inactive Abino Mine Site, controlled by Goldcorp Inc., is located 2.5 kilometres to the southwest of the F2 Gold System and 0.5 km southwest of the Phoenix claim boundary (Figure 23.1). The Abino property is dominated by massive and pillowed mafic flows, minor chemical sediments and variably altered ultramafic rocks (after Sanborn-Barrie et al., 2004). These rocks are intruded by granodiorite, quartz porphyry, gabbro and diorite intrusive. The stratigraphy trends northeast, parallel to the East Bay Deformation Zone. Based on data from the Ontario Geological Survey (2004), historical production from the Abino Mine Site is stated at 2,460 tonnes, grading 17.53 g/t gold for a total historical production of 1,400 ounces of gold.

Two granodiorite-hosted auriferous zones occur on the Abino Property, located one kilometre southwest of the F2 Gold System. The northernmost of these auriferous zones extends to within a few hundred feet of the Phoenix Gold Project claim boundary. The Abino deposit is described as a stockwork of veining within granodiorite which contains erratic concentrations of native gold. Historical estimates from work reports in the 1980s suggest mineralization extends to at least 305 m (1,000 feet) below surface

The reader is cautioned that although these two properties appear to lie within the same mineralization zone as the F2 Gold System, AMC is unable to confirm estimates of tonnes and grade and therefore such information is not necessarily indicative of the mineralization on the Phoenix Gold Project that is the subject of this Technical Report.

24 OTHER RELEVANT DATA AND INFORMATION

AMC is not aware of any other data or information that is relevant to the Rubicon Phoenix Project PEA.

25 INTERPRETATION AND CONCLUSIONS

25.1 Interpretation and Conclusions

The PEA indicates the Project has significant potential to become an economically viable mining operation.

The resource modelling used as the basis for the PEA employed a cut off grade of 5.0 g/t and has resulted in estimates of 1.03 Mt at 14.5 g/t Au of Indicated Resources (477,000 ounces Au) and 4.23 Mt at 17.0 g/t Au of Inferred Resources (2,317,000 ounces Au) for the F2 Zone lode-style mineralization.

The scenario for a potential mining operation envisages a two-year pre-production period followed by a 12-year LOM using a captive cut and fill method with up to six horizons being mined simultaneously. Around 450,000 tonnes would be produced annually at steady state. Average mined grade over the LOM is projected at 13.87 g/t.

Pre-production capital expenditures of $214M have been estimated inclusive of a 30 % contingency. Total sustaining capital over the LOM is projected to be $52 M. AMC notes that some aspects of the capital estimation have been done to a much greater degree of detail than may be regarded as typical for a PEA estimate.

Average operating costs of $214 /t and $519 /oz have been estimated. Mining operation costs make up over 70 % of the envisaged total operating expenditure which, to a large degree, is a reflection of the labour-intensive mining method. The manpower numbers conceived also reflect provision for the degree of uncertainty that lode-style mining can present from an operations point of view.

Using the base case parameters of Au price US$1100 /oz, exchange rate of Can$1 = US$1, discount rate of 5%, and Au processing recovery of 92.5 %, the PEA shows the following pre-royalty and pre-tax values:

- Net Cash Flow (NCF): $739 M
- Net Present Value ($NPV_{5\%}$): $43 M
- Internal Rate of Return (IRR): 28%
- Payback Period from start of two-year pre-production period: 5.3 years

Using a gold price of US$1500 /oz (note. London AM fix at 8 August 2011 at US$1710 /oz), the economic assessment shows NCF of $1.48 Billion, $NPV_{5\%}$ of $933 M, IRR of 48 % and Payback Period of four years.

The base case economic assessment also indicates the Project NPV to have the following sensitivities to variation in some key parameters (all other factors remaining constant):

- NPV5 % ranges from $345 M to $520 M with +/- 30% variation in capital cost
- NPV5 % ranges from $237 M to $628 M with +/- 30% variation in operating cost

- NPV5 % ranges from \$650 M to \$216 M with +/- 30% variation in production
- NPV5 % ranges from \$845 M to \$20 M with +/- 30% variation in Au price or grade (or US\$ to Can\$ exchange rate).

The Project is thus seen to be most sensitive to variations in gold price, gold grade, or exchange rate.

There do not appear to be any permitting, community or environmental issues that would be a major constraint to the project.

Mine design parameters have included a 45 m thick bedrock crown pillar between the uppermost workings and the sediment base in the East Bay of Red Lake. A stable geotechnical environment with little or no major faulting, structure or stress issues has been assumed. AMC believes that these are reasonable assumptions for the rock types and conditions observed in the ongoing advanced exploration operations to-date, but further geotechnical and hydrological assessment will be required in the next phase of the project.

Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve. No mineral reserves have been estimated as part of the present study.

25.2 Project Risks

As can be typical in lode-style gold deposits, the average grades estimated for the resources (and, therefore, for the mining scenario also) have a significant dependency on higher grade drillhole intercepts. AMC considers that a key challenge for the Project, from a prospective mining and economic viability viewpoint, will be to thoroughly understand the character of the mineralization and, from this, to develop the ability to readily locate and mine, with optimum dilution, such high grade areas. AMC believes that this aspect of the Project probably presents both its greatest risk and greatest reward potential.

A further feature of lode-style deposits is that the generation of significant quantities of reserves may require a much greater degree of delineation drilling and, therefore, drilling expense than for other more uniformly distributed mineralization.

Relative to the location uncertainty associated with lode-style deposits, a variety of mining approaches, with possibly significant equipment capital and maintenance expense, may be necessary. This aspect can present both a risk and an opportunity.

Achievement of an average production rate of 1250 tpd will be dependent on having development sufficiently advanced and ore location sufficiently understood to provide a large number of available and viable stoping areas. This can be a significant challenge in a lode-style deposit.

Consequent with the above aspects of lode-style mining, high operating costs can be a significant risk.

Gold recovery estimates to date are dependent on a limited number of samples. Additional testing over a much broader sample range will be required in the next project phase. Again, gold recovery is both a potential risk and a potential opportunity for the Project.

25.3 Project Opportunities

In general, a greater degree of understanding of the zone and mineralization characteristics will facilitate the interpretation of exploration drilling data in terms of quantification of mining potential. This can both mitigate some of the risks described above and provide significant opportunity for project enhancement.

As indicated above, opportunities may exist for application of other mining methods as a greater degree of understanding of the resource is achieved. Alternatives may include mechanized cut and fill or, possibly, some form of bulk mining. Ramp development, both internal and, possibly, from surface, may also warrant consideration and provide opportunity for earlier, and possibly, higher-grade mining.

The historical McFinley Mine area may provide opportunity for additional resources and early mining.

Drilling in targeted areas below 305L should allow upgrading of resources that are currently in the Inferred category.

Further exploration down dip and along strike may identify additional resource potential. In the case of the area immediately above the 1464L, more drilling is warranted to provide greater support for the higher grade Inferred resources currently identified. This may allow a higher-grade mining scenario to be envisaged in those areas.

Efforts made by Rubicon to develop site infrastructure have resulted in an excellent platform on which to move the project towards a potential mining operation.

As referenced above in the discussion on ‘risks’, further metallurgical testing and process refinement may offer potential for enhance gold recovery.

In the later stages of the conceived mining program, and subject to gaining additional geotechnical knowledge and understanding of the crown pillar area in the context of an operating mine, and a very rigorous analysis and risk assessment, it is possible that some of the lower portion of the crown pillar area could be mined.

26 RECOMMENDATIONS

AMC recommends that the Project work continue on several levels that will serve to better understand the deposit and its practical implications for conceived future mining. Several specific recommendations are made by AMC as a result of the PEA study, These are outlined below, together with others that are based on Rubicon's advice that it considers it important at this stage of the project to test opportunities for increasing the tonnage of mineral resources and to continue its progress towards a position where it can readily move to a production status,

1 Engage a specialist to undertake a structural study of the resource. The estimated cost is $250,000, which would include significant oriented core drilling in a variety of locations and the specialist study work itself, including site visits.

2 To increase local understanding of the key characteristics of the resource related to potential mining:

   a) Closely examine the resource model relative to current development status and ease of access to mineralization, and design and develop specific drifts for local diamond drilling in at least two areas, with drives nominally parallel to, and around 25 metres from, the mineralization to be drilled. The drifting would be configured to serve both for definition drilling and for possible, future mining access. AMC has been advised by Rubicon that the overall cost for such drifting would be of the order of $7,000 /m, inclusive of all underground support, engineering, site overhead, G&A, etc. The direct development cost for the drifting will probably be of the order of $1,500/ m; AMC accepts, however, that for a non-producing operation where the focus is on understanding the resource and moving towards a viably economic operation, additional costs of the kind envisaged by Rubicon are probably not unreasonable. Estimated 300 m drifting in each area at approximately $7,000 /m for a total of $4.2 M.

   b) From the above drifts, conduct fan drilling in a vertical plane across the mineralization on sections at nominal 10 m spacing. AMC again accepts that the overall cost/m of $230, as advised by Rubicon, may be significantly more than the actual direct cost, but also accepts that this is probably not unreasonable for similar reasons to those described above. Estimated 400 m/fan + 50% additional for further, opposite direction drilling on 10 sections, again for 2 areas – cost at $230 /m approximately $2.76 M.

   c) Drift on and across mineralization in each area in a systematic manner and relate to above fan drilling via mapping, face sampling and subsequent assaying. Estimated 200 m of drifting in each area x $7,000 /m + $350,000 for sampling and assaying for a total (two areas) of approximately $3.15 M.

      - Note: Any gold recovered from the above drifting exercise could, to some degree, offset the costs incurred. However, it should be clearly accepted that the main focus of the exercise would not be 'mining', as such, but rather, developing a critical understanding and appreciation of the resource itself at a local level and what the implications may be for the optimum methodology for potential future mining of the resource. The current project development status indicates that 305L and 244L would initially provide the most ready access for areas to investigate. Other areas that may be appropriate in the future would be 183L and, potentially, other levels between 305L and 488L (see below).

3 On a larger scale relative to increasing knowledge of the resource as a whole:

a) Execute a delineation diamond drilling program in known zones focusing on Indicated Resources. Estimated 30,000 m of drilling x $230 /m = $6.9 M

b) Execute a diamond drilling program on Inferred Resources aimed at upgrading to Indicated Resources. Estimated 50,000 m of drilling x $230 /m = $11.5 M

c) Execute a deep diamond drilling program of the broader Phoenix area. Estimated 20,000 m of drilling x $230 /m = $4.6 M.

Each of the above programs would be carried out in two phases with, in each case, commencement and execution of the second phase being dependent on results from the first.

4 Conduct further mining studies, at an estimated cost of $250,000, with respect to:

a) Opportunities for early ore production and schedule optimization with a view to maximizing project NPV; this would include assessment of a ramp from surface, internal ramping and mining possibilities in the historical McFinley area.

b) Alternative mining methods such as mechanized cut and fill and long-hole stoping relative to improved understanding of the resource.

5 Undertake further geotechnical assessment as a greater understanding is gained of the ore distribution and the impact that may have on mining geometry, extraction sequence and crown pillar size requirements. In parallel with this work, also complete relevant hydrogeological studies for the Project. Estimated cost $250,000.

6 To decrease the metallurgical uncertainties and have a better estimation of the range of possible gold recovery, collect and analyse a larger number of drill core samples that would better statistically represent the different areas of mineralization. As part of this exercise, design and execute a metallurgical test program aimed at defining a fully optimized processing circuit that considers the anticipated ore blends that will be delivered to the mill. Estimated cost $400,000.

7 Do further study on the characteristics of likely mill tailings relative to their potential use for paste fill. Cost estimate $100,000.

8 Update the ventilation study to reflect projected production rate and increased depth. Estimated cost $50,000.

9 Continue working with Aboriginal Groups and undertake further environmental studies and permitting work as required.

10 Based on the Rubicon advice referenced above concerning the importance of testing opportunities for increasing the tonnage of mineral resources and continuing to move towards a production-ready situation:

a) Extend the shaft 200 m below the current elevation to allow better access for further advanced exploration targets from the 488L. Shaft sinking is estimated at $20,000 /m for a total of $4.0 M.

b) Equip the shaft at an estimated cost of $2.0 M

c) Develop 750 m of access drifting to the advanced exploration area on 488L and an additional 750 m for drill station accesses along the targets. Estimated cost $10.5 M.

d) Construct and install a rockbreaker station and a refuge station to support the advanced exploration program and to comply with mining regulations. Estimated cost $1.0 M.

e) Purchase long lead items for the anticipated mill at an estimated cost of $9.0 M.

f) Purchase an Ozone Treatment Plant to destroy ammonia ($0.6M), an ActiFlo Treatment Plant for effluent ($2.0 M), and a Potable Water System ($0.16 M).

The suggested phases and costs for these recommendations are shown in Table 26.1 below.

AMC further recommends that, relative to the project's PEA status, the largely Inferred nature of the resource, and the associated level of economic uncertainty, all activities should be undertaken in a considered and phased manner as shown in Table 26.1. AMC anticipates that the total program would take about 12 to 18 months to complete, with each phase being of the order of six to nine months in length.

Table 26.1 List of Recommendations

	Item	Activity	Quantity m	Unit cost $/m or item	No. of Areas	Activity Cost $	Phase 1 $	Phase 2 $
1	Structural Study					250,000	250,000	
2	Local Understanding							
	a	Access Drifting	600	7000	2	4,200,000	2,100,000	2,100,000
	b	Diamond Drilling	12000	230	2	2,760,000	1,380,000	1,380,000
	c	Drifting on/across mineralization + sampling and assaying	400	7000	2	2,800,000 + 350,000	1,575,000	1,575,000
3	Resource Drilling							
	a	Delineation	30000	230	2	6,900,000	3,450,000	3,450,000
	b	Increasing Inferred Drilling	50000	230	3	11,500,000	5,750,000	5,750,000
	c	Deep Drilling Exploration	20000	230	1	4,600,000	2,300,000	2,300,000
4	Mining Studies					250,000	125,000	125,000
5	Geotechnical Work					250,000	125,000	125,000
6	Metallurgical Testwork					400,000	200,000	200,000
7	Tailings Study for Paste Fill					100,000	100,000	
8	Ventilation Study					50,000	50,000	
9	Environmental and Social Impact work					p.o. 10 (e)	p.o. 10 (e)	p.o. 10 (e)
10	Rubicon Strategic Initiatives							
	a	Shaft to 488L	200	20,000	1	4,000,000	4,000,000	
	b	Equip Shaft	200	10,000	1	2,000,000		2,000,000
	c	488 L development	1500	7,000	1	10,500,000		10,500,000
	d	Rockbreaker, refuge stn	1	1,000,000	1	1,000,000		1,000,000
	e	Long Lead Items				9,000,000	4,500,000	4,500,000
Total Recommendations, including Strategic						60,910,000	25,905,000	35,005,000

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28 QUALIFIED PERSONS' CERTIFICATES

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1. I, Herbert A. Smith, P.Eng., do hereby certify that I am a Principal Mining Engineer for AMC Mining Consultants (Canada) Limited, Suite 1330, 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2. I graduated with a degree of B.Sc. in Mining Engineering in 1972 and a degree of M.Sc. in Rock Mechanics and Excavation Engineering in 1983, both from the University of Newcastle Upon Tyne, England.

3. I am a registered member of the Association of Professional Engineers, Geologists and Geophysicists of Alberta, Professional Engineers Ontario, Professional Engineers and Geoscientists of British Columbia and the Canadian Institute of Mining, Metallurgy and Petroleum.

4. I have worked as a Mining Engineer for a total of 34 years since my B.Sc. graduation. For most of that period I have worked in underground hard rock mining in Canada in progressively senior engineering roles and have specialized in mine design and planning, mining economic and viability assessment and mining studies at all levels.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 1 – 3, 15 - 16, 18 – 22, 24 – 27 for the F2 Gold System – Phoenix Gold Project, Bateman Township, Red Lake, Canada, Technical Report for Rubicon Minerals Corporation, dated effective 8 August 2011 (the "Technical Report"). I have visited the Phoenix Gold Project from 8 – 9 July 2009, and 9 February 2010 for periods of two days and 1 day respectively.

7. I have not had any prior involvement with the property that is the subject of the Technical Report.

8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

10. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 10 August 2011

Original signed and sealed by

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1. I, John Morton Shannon, P.Geo., do hereby certify that I am Principal Geologist and Group Manager Geology for AMC Mining Consultants (Canada) Limited, Suite 1330, 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2. I graduated with a BA Mod Nat. Sci. in Geology from Trinity College Dublin, Ireland in 1971.

3. I am a registered member of the Association of Professional Engineers and Geoscientists of British Columbia and the Association of Professional Geoscientists of Ontario, and a member of the Canadian Institute of Mining, Metallurgy and Petroleum.

4. I have practiced my profession continuously since 1971, and have been involved in mineral exploration and mine geology for a total of 39 years since my graduation from university. This has involved working in Ireland, Zambia, Canada, and Papua New Guinea. My experience is principally in base metals and gold.

5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 1, 14, 24 - 27 for the F2 Gold System – Phoenix Gold Project, Bateman Township, Red Lake, Canada, Technical Report for Rubicon Minerals Corporation, dated effective 8 August 2011 (the “Technical Report”). I have visited the Phoenix Gold Project from 29 – 31 March 2011, for a period of two and a half days.

7. I have not had any prior involvement with the property that is the subject of the Technical Report.

8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

10. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 10 August 2011

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5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

6. I am responsible for the preparation of Section 14 of the F2 Gold System – Phoenix Gold Project, Bateman Township, Red Lake, Canada, Technical Report for Rubicon Minerals Corporation, dated effective 8 August 2011 (the “Technical Report”). I have visited the Phoenix Gold Project from 29 November to 2 December 2010, and 11 and 12 January 2011, for periods of three and two days respectively.

7. I have not had any prior involvement with the property that is the subject of the Technical Report.

8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

9 I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

10. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the part of the Technical Report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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4. I have practiced my profession continuously since 1998, and have been involved in mineral exploration and mine geology for a total of 14 years since my graduation from university. This has involved working in UK and Canada. My experience is principally in database management and geological interpretation.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 4 -12, 23 and 27 of the F2 Gold System – Phoenix Gold Project, Bateman Township, Red Lake, Canada, Technical Report for Rubicon Minerals Corporation, dated effective 8 August 2011 (the "Technical Report"). I have visited the Phoenix Gold Project from 29 – 31 March 2011, for a period of two days.

7. I have not had any prior involvement with the property that is the subject of the Technical Report.

8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

10. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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3. I am a registered member of the Ordre des Ingenieurs du Quebec, and a member of the Canadian Institute of Mining, Metallurgy and Petroleum.

4. I have practiced my profession continuously since 1982 (including a master's degree), and have been involved in mineral processing for a total of 29 years since my graduation from university. This has involved working in Canada, Holland, Iran and Burkina Faso. My experience is principally in iron and gold ores.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 13 and 17 for the F2 Gold System – Phoenix Gold Project, Bateman Township, Red Lake, Canada, Technical Report for Rubicon Minerals Corporation, dated effective 8 August 2011 (the "Technical Report"). I have visited the Phoenix Gold Project from 16 February to 18 February 2011 and 18 April to 20 April, 2011, for periods of three days in each case.

7. I have not had any prior involvement with the property that is the subject of the Technical Report.

8. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

10. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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4. I have practiced my profession continuously since 1986 (including a master’s degree), and have been involved in mineral processing for a total of 23 years since my graduation from University. This has involved working in Canada. My experience is principally in iron and gold ore and in environment management.

5. I have read the definition of a “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that through my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 13 and 17 for the F2 Gold System – Phoenix Gold Project, Bateman Township, Red Lake, Canada, Technical Report for Rubicon Minerals Corporation, dated effective 8th August 2011 (the “Technical Report”). I visited the Phoenix Gold Project in October 2009. I was also there from the 16th to the 18th of February 2011 and from the 18th to the 20th of April 2011, for periods of three days in each case.

7. I have had no prior involvement with the property that is the subject of the Technical Report.

8. I am independent from the issuer applying all of the tests in Section 1.5 of NI 43-101.

9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

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Dated this 10 August 2011

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