RUBICON MINERALS

National Instrument 43-101 Technical Report for the Phoenix Gold Project Preliminary Economic Assessment Cochenour, Ontario

PREPARED FOR:
Rubicon Minerals Corporation

PREPARED BY:
T. Maunula & Associates Consulting Inc.

QUALIFIED PERSONS:

Tim Maunula, P.Geo.	Andrew MacKenzie, P.Eng.
Karlis Jansons, P.Eng.	Peter Broad, P.Eng.
Charles Tkaczuk, P.Eng.	Ian Horne, B.Sc. Biology

EFFECTIVE DATE: August 19, 2019

REPORT DATE: September 23, 2019

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report in accordance with Form 43-101F1, for Rubicon Minerals Inc. (Client), by Qualified Persons working for T. Maunula & Associates Consulting Inc. (TMAC). The quality of information, conclusions, and estimates contained in this report are based on: i) information available at the time of preparation as of data; ii) data from outside sources; and iii) the assumptions, conditions, and qualifications as put forth by the writer of the report. This report is intended to be used by the Client, subject to terms and conditions of TMAC. The relationship permits the Client to file this report as a Technical Report with applicable securities regulatory authorities pursuant to provincial securities legislation.

CAUTIONARY STATEMENT REGARDING FORWARD-LOOKING STATEMENTS AND OTHER CAUTIONARY NOTES

This technical report contains statements that constitute "forward-looking statements" and "forward looking information" (collectively, "forward-looking statements") within the meaning of applicable Canadian and United States securities legislation. Generally, these forward-looking statements can be identified by the use of forward-looking terminology such as "anticipates", "assumption", "base case", "believe", "budget", "conceptual", "consideration", "contemplate", "contingency", "conversion", "demonstrate", "driver", "estimates", "evaluate", "expect", "exploration", "factors", "focus", "forecast", "forward", "future", "goal", "indication", "intend", "IRR", "inventory", "likely", "LOM," "may", "measures", "mitigation", "model", "NPV", "objective", "opportunity", "optimize", "path", "PEA", "plan", "potential", "preliminary", "profile", "project", "projected", "prospective", "risk", "sensitivity", "strategy", "target", "upgrade", "viability", and "will", or variations of such words, and similar such words, expressions or statements that certain actions, events or results can, could, may, should, will (or not) be achieved or occur in the future. In some cases, forward-looking information may be stated in the present tense, such as in respect of current matters that may be continuing, or that may have a future impact or effect. Forward-looking statements reflect our current expectations and assumptions, and are subject to a number of known and unknown risks, uncertainties and other factors which may cause our actual results, performance, or achievements to be materially different from any anticipated future results, performance or achievements expressed or implied by the forward-looking statements. Forward-looking statements include, but are not limited to statements regarding Mineral Resource estimates including, but not limited to, the 2019 Mineral Resource estimate for the Phoenix Gold Project (the "Project"), any update and/or potential to improve the quantities and classification of such Mineral Resource estimates; mine plans and life; block models; the results of the 2019 Preliminary Economic Assessment for the Project (the "2019 PEA") including but not limited to any anticipated economics, such as return on capital (including IRR) and NPV, free cash flow, gold production (payable or otherwise), capital and operating costs, LOM, estimated tonnes and grade, construction period and other results of the 2019 PEA (all of which are estimates only), impact of infrastructure on the economics of the New PEA; anticipated future infill and expansion drilling; the potential amount of mineralized material and its grade to be realized from the Project and any Explore Target areas; the impact of the data from the 20,000 m of drilling planned in 2019 on the 2019 Mineral Resource estimate; the potential advancement of the Phoenix Gold Project to a viable commercial operation; using the results from the 2018 bulk sampling program for reconciliation and validation purposes; the evaluation of the McFinley Deposit and other close proximity targets for potential inclusion in a future Mineral Resources estimate, and follow-up drilling of the F2 Gold Deposit at depth and along strike. Forward-looking statements are based on the expectations, assumptions, opinions and estimates of management as of the date of this Technical Report and represent our best judgment based on facts and are considered reasonable. If such expectations, assumptions, opinions, and estimates prove to be incorrect, actual, and future results may be materially different than expressed or implied in the forward-looking statements.

Forward-looking statements are subject to a number of known and unknown risks, uncertainties and other factors which may cause the actual results, performance, or achievements of Rubicon to be materially different from any future results, performance or achievements expressed or implied by the forward-looking statements. Such factors include, among others: possible variations in mineralization, grade or recovery or throughput rates; uncertainty of Mineral Resource estimates, inability to realize exploration potential, mineral grades and mineral recovery estimates; actual results of current exploration activities; actual results of reclamation activities; uncertainty of future operations, delays in completion of exploration plans for any reason including insufficient capital, delays in permitting, and labour issues; conclusions of future economic or geological evaluations; changes in project parameters as plans continue to be refined; failure of equipment or processes to operate as anticipated; accidents and other risks of the mining industry; delays and other risks related to operations; timing, receipt and maintenance of permits and other required regulatory approvals; the ability of Rubicon and other relevant parties to satisfy regulatory requirements; the ability of Rubicon to comply with its obligations under material agreements including financing agreements; the availability of financing for proposed programs and working capital requirements on reasonable

terms; the ability of third-party service providers to deliver services on reasonable terms and in a timely manner; risks associated with the ability to retain key executives and key operating personnel; cost of environmental expenditures and potential environmental liabilities; dissatisfaction or disputes with local communities or First Nations or Aboriginal Communities; failure of plant, equipment or processes to operate as anticipated; market conditions and general business, economic, competitive, political and social conditions; Rubicon's ability to generate sufficient cash flow from operations or obtain adequate financing to fund its capital expenditures and working capital needs and meet its other obligations; the volatility of Rubicon's stock price, the ability of its common stock to remain listed and traded on the TSX,, and the "Risk Factors" in the Rubicon's annual information form dated March 22, 2019 ("2019 AIF"). The foregoing list of assumptions, risks, uncertainties, and other factors is not exhaustive. The forward-looking statements contained or incorporated by reference herein are expressly qualified by this cautionary statement as well as those in other continuous disclosure documents of the Company filed under its profile at www.sedar.com including, but not limited to, the 2019 AIF, which readers are advised to carefully review and consider as such factors could cause Rubicon's actual results, performance and achievements to be materially different from any anticipated future results, performance or achievements expressed or implied by the forward-looking statements.

Forward-looking statements contained herein are made as of the date of this technical report and Rubicon disclaims any obligation to update any forward-looking statements, whether as a result of new information, future events, or results or otherwise, except as required by applicable securities laws. The forward-looking statements contained herein are expressly qualified by this cautionary statement.

Cautionary Note Regarding Preliminary Economic Assessments

The 2019 PEA summarized in this Technical Report is only a conceptual study of the potential viability of the Project's Mineral Resource estimates, and the economic and technical viability of the Project and its estimated Mineral Resources has not been demonstrated. The 2019 PEA is preliminary in nature and provides only an initial, high-level review of the Project's potential and design options; there is no certainty that the 2019 PEA will be realized. The 2019 PEA conceptual LOM plan and economic model include numerous assumptions and Mineral Resource estimates including Inferred Mineral Resource estimates. Inferred Mineral Resource estimates are considered to be too speculative geologically to have any economic considerations applied to such estimates. There is no guarantee that Inferred Mineral Resource estimates will be converted to Indicated or Measured Mineral Resources, or that Indicated or Measured Mineral Resources can be converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability, and as such there is no guarantee that the Project economics described herein will be achieved. Mineral Resource estimates may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant risks, uncertainties, and other factors, as more particularly described in the above Cautionary Statements Regarding Forward-Looking Statements.

Cautionary Note to U.S. Readers Regarding Estimates of Measured, Indicated, and Inferred Resources

This technical report uses the terms "Measured", "Indicated", and "Inferred" Mineral Resources. The Company advises U.S. investors that while these terms are recognized and required by Canadian Securities Administrators, they are not recognized by the United States Securities and Exchange Commission (the "SEC")). The estimation of "Measured" and "Indicated" Mineral Resources involves greater uncertainty as to their existence and economic feasibility than the estimation of Proven and Probable Reserves. The estimation of "Inferred" Mineral Resources involves far greater uncertainty as to their existence and economic viability than the estimation of other categories of resources. It cannot be assumed that all or any part of an "Inferred", "Measured" or "Indicated" Mineral Resource will ever be upgraded to a higher category.

Under Canadian rules, estimates of "Inferred Mineral Resources" may not form the basis of feasibility studies, pre-feasibility studies or other economic studies, except in prescribed cases, such as in a preliminary economic assessment under certain circumstances. The SEC normally only permits issuers to report mineralization that does not constitute "Reserves" as in-place tonnage and grade without reference to unit measures. Under U.S. standards, mineralization may not be classified as a "Reserve" unless the determination has been made that the mineralization could be economically and legally produced or extracted at the time the Reserve determination is made. U.S. investors are cautioned not to assume that any part or all of a "Measured", "Indicated" or "Inferred" Mineral Resource exists or is economically or legally mineable. Information concerning descriptions of mineralization and Mineral Resources contained herein may not be comparable to information made public by U.S. companies subject to the reporting and disclosure requirements of the SEC.

Date and Signature Page

The undersigned prepared this Technical Report, titled "National Instrument 43-101 Technical Report for the Phoenix Gold Project Preliminary Economic Assessment, Cochenour, Ontario," and dated September 23, 2019, in support of the public disclosure for public listing. The format and content of this report conforms to the National Instrument 43-101 (NI 43-101) guidelines of the Canadian Securities Administrators.

Dated September 23, 2019.

Original Signed and Sealed

Tim Maunula, P.Geo.
Principal Geologist
T. Maunula & Associates Consulting Inc.

Original Signed and Sealed

Andrew MacKenzie, P.Eng.
Principal Mining Engineer
Associate

Original Signed and Sealed

Karlis Jansons, P.Eng.
Principal Geotechnical Engineer
GeoMin Initiatives Inc

Original Signed and Sealed

Peter Broad, P.Eng.
Principal Processing Engineer
Associate

Original Signed and Sealed

Charles Tkaczuk, P.Eng.
Principal Engineer
Tkaczuk & Associates Inc.

Original Signed and Sealed

Ian Horne, B.Sc. Biology
Principal Environmental Specialist
Ian Audley Horne Consulting

Table of Contents

Tables

Figures

Glossary

Units of Measure

American wire gauge	AWG
Amperes	A
Canadian dollars	C$
Centimetre cubic	cm^3
Centimetre	cm
Cubic feet per minute	cfm
Degree	°
Degrees Celsius	°C
Dry Tonnes	dmt
Feet	ft
Gallons per day	gpd
Gallons per minute	gpm
Gallons	gal
Gram	g
Grams per cubic centimetre	g/cm^3
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Inches	"
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per square metre	kg/m^2
Kilometre square	km^2
Kilometre	km
Kilopascals	kPa
Kilovolt	kV
Kilowatt	kW
Less than	<
Litre	L
Litres per minute	L/min
Mega-annum (1 million years)	Ma
Megavolt ampere	MVA
Megavolt	MV
Megawatt	MW

Metre cubic	m^3
Metre Level (relative metres level below surface)	m Level or mL
Metre square	m^2
Metre	m
Metres above sea level	masl
Metres cubic per hour	m^3/h
Micron	µm
Millimetre	mm
Million litres per day	ML/d
Million tonnes per annum	Mt/a
Million tonnes	Mt
Million years (annum)	Ma
Million	M
Ounce (troy ounce – 31.1035 grams)	oz
Ounce per annum	oz/a
Ounce per tonne	oz/t
Percent by Mass	%m
Percent mass fraction for percent mass	%w/w
Percent	%
Pound	lb
Pounds per square inch gage	psig
Tonnes per cubic metre	t/m^3
Tonnes per day	t/d
Tonnes per hour	t/h
United States dollar	US$
Volt	V

Abbreviations and Acronyms

Accurassay Laboratories	Accurassay
ALS Minerals	ALS
Ammonium Nitrate/Fuel Oil	ANFO
Atomic Absorption Spectroscopy	AAS
Canadian Dam Association	CDA
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Capital Expenditure	CAPEX
Carbon-in-Leach	CIL
Cavity Monitoring Systems	CMS
CDN Resource Laboratories Ltd.	CDN

Certified Reference Material	CRM
Coefficient of Variation	CV
Comma-Separate Values File (electronic file format)	csv
Commercial Production	CP
Cut-and-fill	C&F
Distribution Station	DS
Dominion Goldfields Corporation	Dominion Goldfields
EAG Inc.	EAG
East Bay Deformation Zone	EBDZ
East-West	E-W
Equivalent-length overbreak slough	ELOS
Exploratory Data Analysis	EDA
Exshaw General Use	EGU
F2 Basalt	F2
G&T Metallurgical Services Ltd.	G&T
General and Administrative	G&A
Global Positioning System	GPS
Gold	Au
Hangingwall	HW
High Grade	HG
High Titanium Basalt	High-Ti
Historical McFinley Shaft	now the Phoenix Shaft
Hydrochloric acid	HCl
Inductively-Coupled Plasma Atomic Emission Spectroscopy	ICP-AE
Internal Rate-of-Return	IRR
International Electrotechnical Commission	IEC
International Organization for Standardization	ISO
Inverse Distance Cubic	ID^3
Inverse Distance Square	ID^2
Inverse Distance Weighting	IDW
Inverse Distance	ID
ioGlobal Pty Ltd.	ioGlobal
Lead Nitrate	$PbNO_3$
Life of Mine	LOM
Light Detection and Ranging	LiDAR
Load-Haul-Dump	LHD
Longhole	LH
Low Grade	LG

Mass blast raise mining	MBRM
Mineral Resource Estimate	MRE
Ministry of Energy, Northern Development and Mines	ENDM
Ministry of Natural Resources	MNR
Ministry of Natural Resources and Forestry	MNRF
Ministry of Northern Development and Mines	MNDM
Mineable Shape Optimizer	MSO
Multiple Indicator Kriging	MIK
National Instrument	NI
Nearest Neighbour	NN
Net Present Value	NPV
Net Smelter Return	NSR
Northeast	NE
North-South	N-S
Operating Expense	OPEX
Ordinary Kriging	OK
Original Equipment Manufacturer	OEM
Parts per Billion	ppb
Parts per Million	ppm
Phoenix Gold Property	previously McFinley Property
Pre-Commercial Production	Pre-CP
Qualified Persons	QPs
Quality Assurance	QA
Quality Control	QC
Quantile–Quantile	Q-Q
Relative percentage difference	RPD
Rubicon Minerals Corporation	Rubicon
Semi-Autogenous Grinding	SAG
SGS Canada Inc.	SGS
Smallest Mining Unit	SMU
SMC (Canada) Ltd.	SMC
Sodium Metabisulphite	SMBS
Southwest	SW
Specific gravity	SG
SRK Consulting Canada Inc.	SRK
Stratigraphic	strat
Sulphur Dioxide	SO_2
T. Maunula & Associates Consulting Inc.	TMAC

Tailings Management Facility	TMF
Technical Standards & Safety Authority	TSSA
Thousands of circular mils	MCM
Total Precision Survey	TPS
United States	US
Universal Transverse Mercator	UTM
Unmineralized	unmin
Volume/volume	v/v
West Limb Basalt	WLB
West-East	W-E
X-ray fluorescence spectrometry	XRF

1 SUMMARY

1.1 Introduction

This technical report was prepared in accordance with National Instrument 43-101 of the Canadian Securities Administrators (NI 43-101) to provide a summary of the results of the Preliminary Economic Assessment (PEA) for the Phoenix Gold Project located in the Red Lake mining district of Ontario, Canada (Phoenix, or the Project)) dated September 21, 2019 (Technical Report); the Project is 100% owned by Rubicon Minerals Corporation (Rubicon or the Company).

This Technical Report, including the March 18, 2018 Mineral Resource estimate and PEA summarized herein, were prepared by T. Maunula & Associates Consulting Inc. (TMAC) in accordance with NI 43-101 and Form 43-101F1, and supersedes all prior Technical Reports and Preliminary Economic Analyses prepared for the Phoenix Gold Project.

1.2 Property Description, Location, and Ownership

The Phoenix Gold Project is located in the southwestern part of Bateman Township within the Red Lake mining district of northwestern Ontario, Canada. The Town of Red Lake is approximately 150 km northwest of Dryden, Ontario, and 265 km northeast of Winnipeg, Manitoba.

The Phoenix Gold Project is centred on the Phoenix Shaft, located at UTM coordinates 448,167E, 5,663,962N (NAD 83/Zone 15N) at an elevation of 369 metres (m) above sea level. The total area of the land tenure is 510.4 hectares (ha).

Rubicon has a 100% interest in the Phoenix Gold Project subject to a 2% net smelter return (NSR) royalty on most of the water portions of the property to Franco-Nevada Corporation and 1% on all tenure to RGLD Gold AG.

1.3 Accessibility, Local Resources, and Infrastructure

The Phoenix Gold Project is accessible via an 8-kilometre (km) gravel road from the community of Cochenour, part of the Municipality of Red Lake. Located on East Bay of Red Lake, the Project is also easily accessible by water. Rubicon has all surface rights required to conduct its potential operations.

The Project benefits significantly from proximity to Newmont Goldcorp's Red Lake and Campbell underground mining and processing complexes. Several businesses in the area provide services to the exploration and mining sector, and a long history of mining in the region contributes to a highly skilled work force. Other industries include small-scale logging and tourism focused on hunting and fishing.

The Phoenix Gold Project site is supplied by a 9 km power transmission line connected to Hydro One's 44 kilovolt (kV) grid in the Municipality of Red Lake. Currently, the site is authorized for a load of 5.3 megavolt ampere (MVA) utilizing the two 18 MVA double-ended transformers installed on site to step-down distribution voltages to 4,160 volts (V) for surface and underground. Further voltage step-downs are utilized locally as required for specific equipment installations.

Mine water is pumped to a holding tank on site from the nearby East Bay of Red Lake. The water is piped underground via a 100-millimetre (mm) diameter water line for drilling use, muck pile watering, and other uses. A potable water plant is fully commissioned and operating at the project site. Rubicon's workers camp has a separate potable water treatment and distribution system. Rubicon has access to local workers and fly-in, fly-out workers.

1.4 Physiography

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

Vegetation consists of thick boreal forest composed of black spruce, jack pine, trembling aspen, and white birch.

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. Seismic surveys indicate average accumulations of 10 to 20 m of lake sediments and overburden at the lake bottom, with the water depth less than 8.5 m within the property boundary. The location of the tailings storage area and other site infrastructure is covered in Section 17.

1.5 Geology and Mineralization

The stratigraphy in the East Bay area, where the Phoenix Gold Project is located, comprises submarine tholeiitic basalt, komatiite and komatiitic basalt with minor felsic intrusive volcanic rock, iron formation, and fine-grained clastic metasedimentary rocks, all of which constitute the Balmer Assemblage.

The local geology comprises a series of mine grid north–south (N–S) trending, steeply dipping to sub-vertical alternating panels of talc–altered komatiitic ultramafic flows (Ultramafic) and biotite and silica altered basaltic mafic volcanic flows (High-Ti Basalt) that have been boudinaged to form elongated

lenses. The Ultramafic and High-Ti Basalt units were intruded by dykes and sills of the Felsic Intrusive unit pre– to syn–mineralization.

An early phase of lower-grade mineralization, with gold grades generally less than 4 grams per tonne (g/t) gold (Au), occurs as quartz–actinolite–sulphide veins and stringers and as disseminated mineralization associated with quartz–biotite–sulphide alteration in the High-Ti Basalt and Felsic Intrusive units.

The higher-grade second mineralization event has been linked to an array of shear-related veins and minor localized shear zones. The gold mineralization occurs in association with disseminated sulphide mineralization in the High-Ti Basalt and also in gold-bearing quartz–actinolite veins in the High-Ti Basalt and Felsic Intrusive units.

The best gold grades occur in the thickest portions of the High-Ti Basalt, where the unit presented both favourable structural traps for developing gold–bearing veins and chemical traps where disseminated sulphides and associated gold mineralization are developed.

1.6 Mineral Resource Estimates

Under the direction of Principal Geologist Mr. Tim Maunula, P.Geo., TMAC completed the most current (April 2019) Mineral Resource estimate for the Phoenix Gold Project. The Mineral Resource estimates reported were classified as Measured, Indicated, and Inferred Mineral Resources. The effective date of the Mineral Resource estimate was March 18, 2019.

The Mineral Resource estimate for the Phoenix Gold Project is based on diamond drill hole data consisting of gold assays, geological descriptions, and density measurements. Underground development and bulk sample stopes were also taken into consideration.

The drill hole database received from Rubicon consisted of 1,631 drill holes totalling 551,811 m of core drilling. The database includes all drilling on the Phoenix Gold Project in proximity to the interpreted mineralization zones. The current Mineral Resource included an additional 106 holes that were drilled since the 2018 Mineral Resource (Golder, 2018b). The database included 104,308 gold assays that were used for modelling and Mineral Resource estimation.

The Phoenix Gold Project block models were estimated using Inverse Distance cubic (uncapped–AUCID3, capped–AUCID3). Ordinary Kriging (OK) (capped–AUCOK) and Nearest Neighbour (NN) (capped–AUCNN) were also run for validation purposes. The block models were estimated in four passes. The NN grade model used 2 m composites, and inverse distance weighting cubed (IDW3) and OK used 1 m composites for grade interpolation.

Outlier controls were applied during the interpolation passes. For Pass 1, the capped outliers were used in the normal grade interpolation. For Pass 2 through Pass 4, the outliers were excluded and not used for grade interpolation.

1.7 Mineral Resource Statement

The Mineral Resource Estimate, as prepared by TMAC, for the Phoenix Gold Project is reported in Table 1-1 (Effective March 18, 2019).

Table 1-1: Phoenix Gold Project 2019 Mineral Resource Estimate Reported at 3.0 g/t Au Cut-off Grade

Resource Category	Quantity (t '000s)	Grade (g/t Au)	Contained Gold Ounces
Measured (M)	442	6.99	99,000
Indicated (I)	2,485	6.13	490,000
M+I	2,927	6.26	589,000
Inferred	2,570	6.53	540,000

Notes: Effective date for this Mineral Resource is March 18, 2019.
Mineral Resource Estimate uses a break-even economic cut-off grade of 3.0 g/t Au based on assumptions of a gold price of US$1,400 per ounce (oz); an exchange rate of US$/C$0.77; mining cash costs of C$97/t; processing costs of C$24/t; G&A of C$6/t; sustaining capital C$20/t; refining, transport, and royalty costs of C$57/oz; and average gold recoverability of 95%.
Mineral Resource Estimate reported from within envelopes accounting for mineral continuity.
Mineral Resources are not Mineral Reserves and do not demonstrate economic viability.
There is no certainty that all or any part of this Mineral Resource will be converted into Mineral Reserve.
All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.

The Mineral Resource Estimate excludes mineralization within the crown pillar located between the lake bottom and a depth of 40 m below the lake bottom. In addition, all mineralization within underground development and outside of Rubicon's claim boundary was excluded from the Mineral Resource Estimate.

Mineral Resources are not Mineral Reserves and do not necessarily demonstrate economic viability. There is no certainty that all or any part of this Mineral Resource Estimate will be converted into Mineral Reserve.

1.8 Mineral Reserve Estimates

There are no Mineral Reserve estimates at the Phoenix Gold Project.

1.9 Operations Status

The mill is currently under care and maintenance. During 2015 and 2018, the mill processed approximately 100,000 tonnes of mineralized material. The mill was shut down in an orderly fashion

and steps have been taken to preserve the integrity of the installed equipment. Based on data and observations from the 2018 operation, several recommendations were made by the metallurgical consultant who ran the 2018 test mining program, to optimize and incrementally improve future mill operation. The mill could be restarted after a short period of operational readiness where equipment could be inspected, any modifications deemed necessary could be made, consumables could be purchased, and supervision and the operating and maintenance teams could be recruited and trained.

The paste backfill circuit including major equipment, such as the positive displacement pump and hydraulic drive, and two-disc filters have been installed but are not commissioned. As this equipment has never been operated and four years have passed, a thorough mechanical and electrical inspection is required. Commissioning needs to be completed to ensure that all components shown on the piping & instrument diagrams (P&IDs) have been installed and properly terminated. Programming of the control system must be completed and then tested. Commissioning can only be scheduled when the mill is operating and tailings are available.

1.10 Conceptual Mining Methods

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

Factoring in PEA assumptions on gold price, foreign exchange, mining loss, external dilution, mill recovery, and operating costs, the PEA derived a mining cut-off grade of 3.5 g/t Au. Table 1-2outlines the conversion estimates applied towards the tonnes and the diluted mill head grade in the new PEA conceptual life-of-mine (LOM) plan.

Table 1-2: Conversion to Conceptual Mineable Mineral Resources Estimates

Categories	2019 MRE (@ 3.5 g/t Cut-Off Grade)		Conceptual LOM (@ 3.5 g/t Au Mining Cut-Off Grade)		Net Conversion of Tonnes (%)
	Tonnes	Grades (g/t Au)	Mineralized Tonnes	Mill Head Grades (diluted) (g/t Au)	
Measured & Indicated	2,289,000	7.11	1,561,039	5.23	68.2
Inferred	2,038,000	7.39	1,484,159	5.39	72.8

Note: Numbers may not add due to rounding.

Currently, the Mineral Resource is accessed via a 730 m shaft and more than 13,500 m of ramp and levels. Levels are established at the 122 m, 183 m, 244 m, 305 m, 610 m, and 685 m Level horizons (Figure 1-1). Mine infrastructure includes muck handling facilities for all levels, a ventilation system, a mid-shaft loading pocket complete with spill pocket, and a shaft-bottom loading pocket. Ramp access has been established between the 305 m and 244 m Levels. Remaining ramp connections from the 244 m Level up to the 122 m Level are within 380 m of completion. A ramp from surface to the 122 m Level has been designed which would be 800 m in length.

In 2015, test mining using sublevel longhole methods was trialed with inconclusive results due to suspension of operations prior to test completion. Most recently, in 2018, Longhole and Uppers test mining resulted in a 35,000-tonne bulk sample. The methods supported the Mineral Resource models grade estimates as well as the competency of bulk mining stopes, with unplanned dilution values falling below 10%.

The mining conceptual LOM plan includes use of the two previously noted methods, adding cut-and-fill (C&F) and mass blast raise mining (MBRM) methods to provide the anticipated flexibility required to optimize extraction in stopes with intermediate dips or widths as narrow as 1.2 m. The MBRM method, a bulk mining method, has not been trialed in the Red Lake Camp, so Rubicon and TMAC personnel visited the Williams Gold operation at Hemlo, where they observed its employment and heard testimonies of its success. Rubicon has received a positive opinion from Manroc Developments Inc. (Manroc), practitioners of the method, that it can be employed at Phoenix. Mining method metrics and costs are shown in Table 1-3.

Bulk mining, Longhole, Uppers and MBRM are expected to provide 92% of the mineralized tonnes while mechanized C&F will provide the remaining 8%.

Operating costs are based on 2018 test mining and/or budgetary estimates provided by mine contractors. Where applicable (i.e., estimating 40-tonne loading and tram costs and cycle times), TMAC applied 1st principle modelling and calculations.

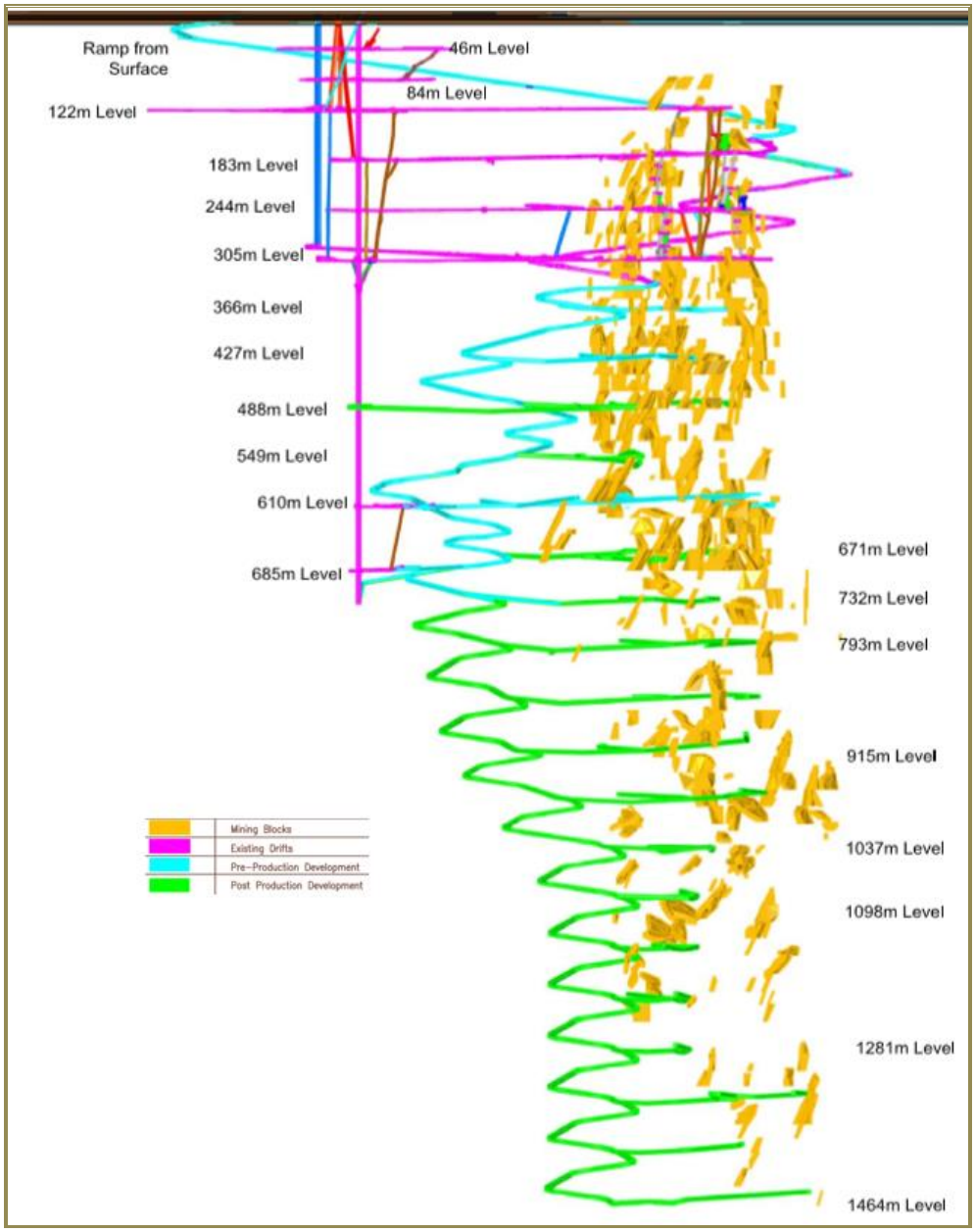


Figure 1-1: Isometric Drawing of LOM Development, Looking North

Table 1-3: Mining Methods for the PEA

Metrics	Sub-Level Longhole	Uppers	MBRM	C&F
LOM Tonnes	1,595,921	708,880	513,974	226,420
LOM Tonnes (%)	52.4	23.3	16.9	7.4
Total Stopes	66	172	43	60
Average Stope Size (Tonnes)	24,181	4,121	11,953	3,774
Average Dimensions (Height x Width x Strike)	36 m x 8 m x 25 m	16 m x 6 m x 12 m	37 m x 3 m x 22 m	12 m x 6 m x 16 m
External Dilution[1] (%)	10	15	10	3
Average Diluted Grade (After Dilution and Mining Loss)	5.34 g/t Au	5.30 g/t Au	5.31 g/t Au	5.16 g/t Au
Mining Cost per Tonne (Including Indirect Costs)	C$82.88	C$86.88	C$92.15	C$120.58
Typical Productivity Rates (t/d) for these different mining methods	400	300	600	130

Note: [1]Waste material from external dilution not carry a zero grade

1.11 Recovery Methods

The mill was designed in 2013 for an initial throughput capacity of 1,250 t/d, with provisions in the layout to increase capacity up to 2,500 t/d with modifications and additions to the existing equipment. Recent trials have demonstrated that once the original design is completed the mill can operate efficiently at 1,800 t/d with a 95% recovery.

Rubicon prepared the mill for operation in 2018 to batch process approximately 35,000 tonnes of mineralized material mined from three stopes. Rubicon successfully batch-processed this material along with 7,620 tonnes of low-grade material from various other sources. In total, 43,250 tonnes were processed, producing 5,669 oz Au. The 2018 bulk sample processing campaign yielded an improvement in overall gold recovery, with the largest gain occurring in gravity gold recovery as compared to the 2015 operation. The mill has once again been placed in care and maintenance after the 2018 test was complete.

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

kiln before being reintroduced to the process. Fine carbon is constantly eliminated (and recovered) from the process to avoid gold loss, with fresh carbon being continuously added to the process.

The cyanide contained in the tailings from the CIL circuit is eliminated in a cyanide destruction tank using the SO_2/O_2 cyanide destruction process. Either liquid SO_2 or sodium metabisulphite can be used as the SO_2 source. Once the cyanide is destroyed, the tailings are pumped to the tailings management facility (TMF) for storage.

When paste backfill is required, tailings streams can be diverted to the paste plant, where they will be filtered to lower the water content. The filter cake will then be mixed with fly ash and cement to produce a paste, which will be pumped to the underground for backfilling. The gold recovery plant, cyanide destruction process, and TMF were commissioned and operational in 2015. The pastefill plant has not yet been operated, as the Project has not yet required backfill. Major equipment for the tailings filter plant and the paste plant have been installed. However, some minor piping, electrical, and instrumentation connections remain to be completed before this equipment can be commissioned.

1.12 Project Infrastructure

Effectively all but the final infrastructure discussed in the 2013 Technical Report has been completed, except for a new office, mine dry, and warehouse. These have not been constructed but have been costed in as part of this Technical Report. The main surface infrastructure includes:

- Hoist, headframe, and hoist house
- Processing plant
- TMF
- Effluent treatment plant
- Electric power supply and substation
- Propane storage tanks
- Fibre-optic communications cable
- Compressed air supply
- Process and potable water supplies
- Sewage works
- Mine ventilation fans and heater house
- Offices, shop, warehouse, core shack, and storage buildings provide housing for related site activities.

The underground infrastructure required to support production mining includes:

- Material handling facilities
- Mine dewatering system
- Paste backfill distribution system
- Equipment service bays
- Ventilation system
- Supply lines for compressed air and process water
- Electrical power supply
- Miscellaneous facilities.

1.13 Market Studies and Contracts

The Phoenix Gold Project processing facilities are complete and able to produce high-grade gold doré bars at the site, which are readily marketable.

1.14 Environmental Studies, Permitting, and Social Impacts

Rubicon holds an approved Closure Plan for the Project, filed with the Ministry of Energy, Northern Development and Mines (ENDM), which addresses and provides financial assurance for all known environmental liabilities. The Project also holds many permits, licences, and approvals in support of Project operation. In Q2 2019, Rubicon submitted a Notice of Material Change and an amendment to the filed certified Closure Plan that when filed will support the ongoing operation of a 1,250 t/d mine.

Rubicon is currently in material compliance and has fulfilled the monitoring and reporting obligations of all permits, licences, and approvals. Rubicon is also in material compliance with obligations under federal legislation including the Canadian *Environmental Assessment Act* and Metal Mining and Effluent Regulations under the *Fisheries Act*.

An increase in production rate to 1,800 t/d, per the PEA, would require additional amendments to the Closure Plan and other permits, approvals, and licences. The key steps in the approvals processes prior to increasing production to 1,800 t/d include completing any additional required environmental assessments under the Ontario *Environmental Assessment Act*, submitting a Notice of Material Change, and filing an amendment to the certified Closure Plan under the Ontario *Mining Act*. Subsequently, other permits, approvals, and licences, under the Ontario *Water Resources Act* and *Environmental Protection Act* and others, must be obtained.

Over the past several years Rubicon has established and maintained a successful history of consultation with the local Indigenous communities and are committed to continued consultation over the life of the Project. Currently, Rubicon has Exploration Agreements in place with Lac Seul First

Nation and Wabauskang First Nation and has committed to having an Impact and Benefits Agreement (IBA) in place with both First Nations prior to production commencing.

The TMF and other sewage works were subsequently designed for expansion in 2010, and a detailed design for this expansion was completed in 2012. Rubicon received ECAs under the *Environmental Protection Act*. The TMF has been constructed (2013 to 2015) to Stage 1 design elevation in accordance with an approval issued pursuant to the Ontario *Lakes and Rivers Improvement Act*. Approximately 57,000 tonnes of tailings were deposited in the TMF in 2015 before the mine was placed into temporary suspension.

In 2018, approval was received to conduct a test mining and bulk sample processing program. The tailings from this bulk sample were also placed in the TMF. This PEA includes capital costs to improve the tailings facility by the addition of two new lifts, timed to coincide with storage requirements, to handle approximately 60% of the 3.045 Mt of tailings generated. The remaining 40% of tails are planned for deposition as pastefill underground.

Approximately C$7.7 million of financial assurance was provided to the ENDM in June 2016 in conjunction with an amended Closure Plan. The value of financial assurance was recalculated and reconfirmed for the amended Closure Plan submitted in Q2 2019.

1.15 Capital and Operating Costs

Total conceptual LOM capital expenditures, from construction start to end of conceptual mine life, is estimated to be C$255.2 million. This PEA has defined capital expenditures for estimation purposes as follows:

- Initial Capital–C$101.2 million: These are estimated capital expenditures up to the declaration of commercial production (CP), including a 15% contingency; during pre-commercial production (Pre-CP), this PEA anticipates net cash flow generation of approximately C$28.8 million from potential payable gold production of approximately 44,000 oz, which translates to a net Pre-CP capital of C$72.4 million.
- Sustaining Capital—C$154.0 million total, or C$30 million per year of CP; estimated capital expenditures incurred from the declaration of CP to the end of the LOM, inclusive of closure costs.

Most of the estimated capital costs (initial capital and sustaining) comprised expected underground development (lateral, ramp, and vertical) below the 305 m Level, and connecting the ramp from the 244 m Level to surface. On average, including indirect costs, the new PEA estimates that the average cost per underground development metre ranges between C$5,500 and C$6,500. This rate is based on actual development costs incurred by the Company during its 2018 test trial mining program. The

equipment fleet assumes equipment lease financing. Table 1-4 provides the breakdown of the estimated Pre-CP and sustaining capital.

Operating costs estimates were derived from the Company's 2018 test mining and bulk sample processing program. The Company successfully tested the sub-level Longhole and Uppers methods and utilized the Phoenix mill for processing, allowing the Company to collect actual operating cost information. The largest component of operating cost is labor (50%–60%). Table 1-5 provides a summary of the operating cost estimates and Table 1-3 the estimated cost per tonne of each conceptual mining method contemplated.

Table 1-4: Conceptual LOM Capital Cost Estimate Breakdown

Capital item:	Pre-CP (C$ millions)	Sustaining (C$ millions)
Underground Development and Infrastructure	43.2	86.7
Equipment	16.9	53.1
Surface and Mill (Including TMF, Water Treatment, Crushers, Camp Upgrades, Buildings, etc.)	22.8	6.5
Closure Costs	-	7.7
Contingency (15%)	18.4	-
Total Initial Capital	101.2	-
Total Sustaining Capital	-	154.0
Total Conceptual LOM Capital	255.2	
Capitalized Pre-CP Operating Costs	45.7	-
Proceeds from Sale of Pre-CP oz	74.5	-
Net Pre-CP Operating Cash Flow	28.8	-
Net Pre-CP Capital	72.4	-
Projected Funding Requirement[8]	80.9	-

1.15.1 Royalties and Other Production Taxes and Tax Loss Pools

The Project mineral claims that comprise the F2 Gold Zone are subject to 2.0% NSR payable to Franco-Nevada Corporation, and 1.0 NSR payable to RGLD Gold AG.

Other production taxes include estimated LOM community payments. The Company currently has approximately C$690 million of tax-deductible pools, tax losses, and tax credits (tax loss pools) available for deduction at the potential commencement of CP. Application of these tax loss pools is estimated to result in the payment of no income taxes against the LOM net income from the Project. In the LOM plan, Rubicon will utilize approximately C$169 million of federal tax loss pools. This PEA estimates that the application of tax loss pools improves conceptual LOM free cash flow by

approximately C$95.5 million. It is estimated that Rubicon would have C$521 million of unused federal tax loss pools at the end of the conceptual LOM (Table 1-5).

Table 1-5: Conceptual LOM Capital and Operating Cost Summary

Capital and Operating Cost Estimates:	Total (C$ millions)	Per Unit (C$)	Per Ounce (US$)
Capital Cost Estimates			
(A) Initial Capital (Including 15% Contingency) (Pre-CP)	101.2	-	-
(B) Capitalized Pre-CP Operating Cost	45.7	186.2/pre-CP tonne	780/pre-CP oz
(C) Royalties (3%) and Other Production Taxes[1] (Pre-CP)	3.0	12.2/pre-CP tonne	51/pre-CP oz
(D) Pre-CP Sales of 44,047 oz (Pre-CP oz)	77.5	-	-
(E) Net Pre-CP Operating Cash Flow: (D) – (B) – (C)	28.8	-	-
Net Pre-CP Capital: (A) – (E)	**72.4**	-	-
Projected Funding Requirement[2]	**80.9**	-	-
(F) Sustaining Capital (CP)	154.0	30 million/CP year	258/CP oz
(G) Total LOM Capital Expenditures: (A) + (F)	**255.2**	**41 million/LOM year**	**389/LOM oz**
Operating Cost Estimates			
Mining	268.4	88.1/LOM tonne	409/LOM oz
Processing	99.6	32.7/LOM tonne	152/LOM oz
Site G&A	23.8	7.8/LOM tonne	36/LOM oz
(H) Total Conceptual LOM Operating Costs	391.8	128.7/LOM tonne	597/LOM oz
(I) Total Commercial Operating Costs (CP): (H) – (B)	346.1	123.6/CP tonne	579/CP oz
(J) Royalties (3%) and Other Production Taxes[3] (CP)	26.9	9.6/CP tonne	45/CP oz
(K) Total Cash Costs (CP): (I) + (J)	**373.0**	**133.2/CP tonne**	**624/CP oz**
(L) All-In Sustaining Costs[4] (AISC) (CP): (F) + (K)	**527.1**	**188.2/CP tonne**	**882/CP oz**
(M) All-In Costs (AIC): (G) + (H) + (C) + (J)	**677.0**	**222.3/CP tonne**	**1,031/LOM oz**
Taxes			
Existing Tax Loss Pools	690		
Estimated Remaining Tax Loss Pools After Conceptual LOM	521		

Notes: [1]Includes community payments.
[2]Based on cumulative net cash flow analysis, factoring in timing of capital and operating costs and proceeds from gold sales. C$80.9 million represents the largest net cash flow deficit through the conceptual LOM.
[3]Includes community payments.
[4]All-in sustaining costs are presented as defined by the World Gold Council, excluding corporate G&A.

1.16 Economics, Cash Flow Model, and Valuation Sensitivities

The PEA demonstrates economics as summarized in Table 1-6.

Table 1-6: Economic Analysis Summary (base case estimates)[1]

Economic Analysis (base case estimates)	Unit	Conceptual Project LOM[2]
After-Tax Internal Rate of Return (IRR)	%	40.2
After-Tax Net Present Value[1] (NPV 5%)	C$ million	135.2
After-Tax Free Cash Flow Potential	C$ million	191.5
Exchange Rate	US dollars, "US$ or "USD"/C$	0.7519
Gold Price Assumption	US$/oz	1,325
Gold Price Assumption	C$/oz	1,762
Payback Period	years	3.9

Notes: [1]Calculated at 5% discount rate.
[2]Commercial Production defined as 70% of 1,250 t/d permitted capacity over 60 consecutive days. Cash costs, sustaining capital, operating costs, and all-in sustaining costs are calculated from the start of CP.
Based on a 3.5 grams g/t Au mining cut-off grade.

Cautionary Statement: The reader is advised that this PEA is only a conceptual study of the potential viability of the Project's Mineral Resource estimates and that the economic and technical viability of the Project and its estimated Mineral Resources have not been demonstrated. This PEA is preliminary and provides only an initial, high-level review of the Project's potential and design options; there is no certainty that this PEA will be realized. The PEA conceptual LOM plan and economic model include numerous assumptions and Mineral Resource estimates including Inferred Mineral Resource estimates. Inferred Mineral Resource estimates are considered to be too speculative geologically to have any economic considerations applied to such estimates. There is no guarantee that Inferred Mineral Resource estimates will be converted to Indicated or Measured Mineral Resources, or that Indicated or Measured Resource estimates can be converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability, and as such, there is no guarantee that the Project's economics described herein will be achieved. Mineral Resource estimates may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant risks, uncertainties, and other factors, as more particularly described in the Cautionary Statements Regarding Forward-Looking Statements.

The majority of expenditures for the Project are expected to be in C$. This PEA is supported by actual data from its 2018 test trial mining and bulk sample processing program.

Project sensitivity estimates, after-tax IRR and NPV, are provided for gold price and US$/C$ exchange rate ratio estimates in Table 1-7. Other sensitivity estimates under different scenarios for grade, throughput, capital, and operating costs are provided in Table 1-8 and Table 1-9.

Table 1-7: After-Tax IRR and NPV5% Estimated Sensitivities to Gold Price and US$/C$ Exchange Ratio

US$/C$ Exchange Ratio	Gold Price (US$/oz)				
	$1,100	$1,200	$1,325 (base case)	$1,400	$1,500
0.83	(4.4%) / ($30.1)	9.3% / $14.9	24.4% / $71.1	32.8% / $104.8	43.5% / $149.7
0.81	(0.4%) / ($17.8)	13.1% / $28.4	28.2% / $86.0	36.6% / $120.5	47.4% / $166.5
0.79	4.6% / ($1.3)	18.0% / $46.4	33.1% / $105.8	41.6% / $141.5	52.6% / $189.0
0.77	8.2% / $11.1	21.5% / $59.9	36.6% / $120.7	45.3% / $157.2	57.4% / $205.8
0.7519 (base case)	11.7% / $23.5	25.0% / $73.4	**40.2% / $135.2***	48.9% / $172.9	60.2% / $222.6
0.73	16.3% / $40.0	29.5% / $91.3	44.8% / $155.4	53.7% / $193.8	65.2% / $245.1
0.71	20.6% / $56.5	33.9% / $109.3	49.4% / $175.2	58.4% / $214.8	70.1% / $267.5

Note: * = Base Case

Table 1-8: After-tax IRR (%) / After-tax NPV5% Estimated Sensitivities to Throughput and Head Grade (C$ millions)

LOM Avg. Throughput (t/d)	LOM Diluted Head Grade (g/t Au)						
	4.50	4.75	5.00	5.31 (base case)	5.50	5.75	6.00
1,200	(5.9%) / ($34.6)	2.8% / ($7.4)	10.9% / $20.4	20.0% / $54.0	25.3% / $74.7	32.0% / $101.3	38.6% / $128.8
1,300	6.9% / $6.6	15.2% / $36.0	23.0% / $65.6	32.3% / $102.4	37.6% / $124.5	44.4% / $153.4	51.2% / $182.9
1,370 (base case)	14.9% / $34.8	23.0% / $65.4	30.9% / $96.9	**40.2% / $135.2***	45.7% / $158.8	53.0% / $190.6	59.9% / $221.4
1,500	28.7% / $87.8	37.0% / $121.9	45.1% / $156.4	54.6% / $197.5	60.5% / $223.7	68.1% / $257.9	75.5% / $291.9
1,600	38.5% / $128.2	47.1% / $165.0	55.3% / $200.5	65.6% / $246.7	71.5% / $273.6	79.4% / $309.7	87.2% / $346.4
1,700	48.2% / $169.8	57.0% / $208.1	65.6% / $246.9	75.8% / $293.4	82.3% / $323.5	90.5% / $361.7	98.7% / $397.9
1,800	57.5% / $210.4	66.4% / $250.6	75.6% / $292.5	86.5% / $342.9	93.0% / $373.2	101.4% / $408.3	109.8% / $438.8

Note: * = Base Case

Table 1-9: After-tax IRR (%) / After-tax NPV5% Estimated Sensitivities to Capital and Operating Cost (C$ millions)

LOM Capital Cost Change (%)	Total Operating Cost		
	$110.00	C$128.67 (base case)	C$150.00
-25%	80.9% / $234.9	65.6% / $189.4	48.7% / $137.3
-15.0%	67.6% / $213.2	54.0% / $167.7	38.8% / $115.6
$255.2 million (base case)	51.6% / $180.1	**40.2% / $135.2***	27.0% / $83.8
+15.0%	39.6% / $148.2	29.2% / $102.7	17.1% / $50.6
+25%	32.8% / $126.4	23.0% / $80.9	11.6% / $28.8

Note: * = Base Case

1.17 Opportunities

The following sections highlight the main opportunities to increase Project economics and reduce identified risks. Each section, while as comprehensive as is possible at this time, is not considered exhaustive as more opportunities can arise when considered from alternative perspectives.

1.17.1 Mineral Resource Estimate

In TMAC's opinion, Rubicon can potentially improve or increase the 2019 Mineral Resource estimate with the following recommendations:

- Target infill and step-out drilling in areas containing Inferred Mineral Resources (about 40 m centres drill spacing) to upgrade Mineral Resource classification and Exploration Targets (>80 m centres) to convert to Mineral Resource estimates.
- Drilling is proposed from the 244 m Level, 610 m Level, and 685 m Level; the Exploration Targets could potentially contain[1] between 0.9 to 1.2 Mt, with potential grades between 5.0 and 7.0 g/t Au.
- Evaluate McFinley Deposit and close proximity targets (specifically PEN Zone—within 500 m of current workings) which could potentially be included in a future Mineral Resource estimate.

1.17.2 Mining

- Higher development advance rates. The conceptual mine plan is based on historical, long term, economic, and sustainable development productivities, typically 3 to 4 m/d in single-heading waste development; there are many instances of higher advance rates being achieved and sustained, and any such improvements will provide flexibility in scheduling options.
- Cut-off grade optimization. The conceptual mine plan currently assumes a break-even cut-off grade based on mining, milling, and G&A costs that are all subject to improvements with technological advancements.
- Metal prices and exchange rates. Cut-off grades are heavily skewed by foreign exchange rates and precious metal prices, both of which currently reflect historical strength in the North American economy; sensitivity to a change in either gold price or exchange rate could translate to some fraction of hundreds of TMAC-defined marginal stopes being upgraded to potentially mineable Mineral Resources.

[1] According to NI 43-101 Section (2)(a), the potential quantity and grade of the Exploration Target is conceptual in nature and there has been insufficient exploration to define a Mineral Resource estimate. It is uncertain if further exploration will result in the Exploration Target material being delineated as a Mineral Resource. The Exploration Target has been defined based on blocks estimated using >80 m drill centres, lower confidence based on decreased data density, and increased cut-off grade with depth.

- The increasing availability and implementation of battery-powered equipment provides opportunity to reduce primary ventilation, which is a major consumer of power in the mine; the mine plan is based on leased diesel equipment, especially haul trucks, that could be quickly transitioned to battery-powered equipment.
- MBRM techniques form about 17% of the PEA's production schedule. MBRM does not require top sill or intermediary sill development; if proven appropriate, this method could replace some formerly classified LH stopes, which would increase the rate at which stopes could be put into production.
- The raise method is limited, in this PEA, to stope widths of 1.2 m, but with experience, could evolve to allow narrower stopes, thereby transitioning some fraction of more the more-than-300 existing marginal stopes TMAC has already identified into the minable stope category.
- Optimizing the mining sequence has the potential to reduce the amount of remote mucking, thereby providing productivity and cost benefits; the mine plan incorporates both top-down and bottom-up mining sequences depending on the mining method and nearby proximity to other stopes.
- Opportunities exist to aggregate smaller stopes into larger units that span multiple levels, and according to local geotechnical limits, to improve productivity.

1.17.3 Processing

- The mill is currently designed, constructed, and commissioned to process up to 1,250 t/d. However, the current permit limits capacity to 1200 t/d. The PEA includes changes to the retention cells so that 1,800 t/d can be produced (the mill layout allows for the addition of a second ball mill, a second hydrocyclone cluster, a pre-crushing unit, and a second stripping column if required in the future to achieve 2,500 t/d).
- The mill would benefit from having dedicated assay services available at its metallurgical lab in Balmertown, or on site, to provide timely feedback to the operators; the nominal 24-hour turnaround time for off-site assays is acceptable for the short term, but does not allow for efficient troubleshooting.
- Evaluate ore-sorting technology to potentially increase mill head grade and reduce tonnes to be milled.

1.18 Recommendations

1.18.1 Mineral Resources

TMAC recommends the following:

- Target infill and step-out drilling in areas containing Inferred Mineral Resources (about 40 m centres drill spacing) to upgrade Mineral Resource classification and convert Exploration Targets (>80 m centres) to Mineral Resources.
- Drilling is proposed from the 244 m Level, 610 m Level, and 685 m Level; the Exploration Targets could potentially contain[2] between 0.9 to 1.2 Mt, with potential grades between 5.0 and 7.0 g/t Au.
- Evaluate McFinley Deposit and close proximity targets (specifically PEN Zone—within 500 m of current workings) which could potentially be included in a future Mineral Resource estimate.
- Update the Mineral Resource estimate to incorporate drilling completed up to third quarter 2019; this would also include a re-evaluation of the geological model and confirmation of structural model (Table 26-1).

Table 26-2 summarizes the recommended exploration budget of $2,460,136 for the F2 Gold Zone opportunity to expand the Mineral Resource. The combined McFinley Deposit and PEN Zone exploration budget is included in Table 26-3.

Table 1-10: Recommended Additional Resource Update Costs (C$)

Re-assessment of Zone 2 Geological Model	$25,000
Incorporating New Drilling and Update Resource Grade Model and Classification	$75,000
McFinley Zone—exploration program and Mineral Resource estimate	$1,100,000
Close Proximity Target—Pen Zone exploration program and Mineral Resource estimate	$500,000

[2] According to NI 43-101 Section (2)(a), the potential quantity and grade of the Exploration Target is conceptual in nature and there has been insufficient exploration to define a Mineral Resource estimate. It is uncertain if further exploration will result in the Exploration Target material being delineated as a Mineral Resource. The Exploration Target has been defined based on blocks estimated using >80 m drill centres, lower confidence based on decreased data density, and increased cut-off grade with depth.

Table 1-11: 2019 Exploration Budget—F2 Gold Zone

Items	Levels	Metres	Units	Cost/Unit (C$)	Total Cost (C$)	Grand Total (C$)
Drilling	244 m Level	825	-	90	74,250	
	610 m Level	12,475	-	90	1,122,750	
	685 m Level	7,830	-	90	704,700	1,901,700
Assaying	244 m Level	-	413	40	16,500	
	610 m Level	-	6,238	40	249,500	
	685 m Level	-	3,915	40	156,600	422,600
Consumables	244 m Level	-	-	-	5,304	
	610 m Level	-	-	-	80,196	
	685 m Level	-	-	-	50,336	135,836
						2,460,136

Table 1-12: 2019 Exploration Budget—McFinley Deposit and PEN Zone

Items	Levels	Metres	Units	Cost/Unit (C$)	Total Cost (C$)	Grand Total (C$)
Drilling	685 m Level (McFinley)	1,170	-	90	105,300	-
	244 m Level (PEN Zone)	2,700	-	90	243,000	348,300
Assaying	685 m Level (McFinley)	-	585	40	23,400	-
	685 m Level					
	244 m Level (PEN Zone)	-	1,350	40	54,000	77,400
Consumables	685 m Level (McFinley)	-	-	-	7,521	
	244 m Level (PEN Zone)	-	-	-	17,357	24,879
					-	**450,579**

1.18.2 Mine and Mill

Given the positive results of the PEA and the potential for enhancement, TMAC recommends the following for the mine and mill:

- Completing a new NI 43-101 Mineral Resource model to further update the existing Mineral Resource estimate
- Confirming/updating structural geology model by incorporating 2018–2019 drilling
- Initiate feasibility level studies including further engineering and design work

- Perform tailings design, includes comparison of phase 2 and 3 dams, and costing as part of the Feasibility Study
- Hydrological study
- Geotechnical modelling
- Testing the MBRM method at Phoenix
- Remain in compliance with environmental reporting, monitoring, and auditing during future exploration and development
- Continue to advance permitting amendments including Mine Closure Plan and ECA for production, ramp, and development advancements.

Estimated costs for -non-exploration recommendations are summarized in Table 26-4.

Table 1-13: Recommended Costs (C$)

Engineering studies and design work, including feasibility studies	$1,250,000
Confirm/update pastefill strength tests	$50,000
Confirm/update structural geology model	$50,000
Tailings design, includes comparison of phase 2 and 3 dams, and costing	$300,000
Geotechnical modeling	$250,000
Procurement work to improve accuracy of CAPEX and OPEX	$200,000
Test the MBRM method at Phoenix	$2,500,000
Advance permitting amendments towards 1,800 t/d	$300,000
	$4,950,000

2 INTRODUCTION

This Technical Report was prepared by TMAC for Rubicon Minerals Corporation to summarize the results of the Preliminary Economic Assessment on the Phoenix Gold Project. This Technical Report was prepared in accordance with NI 43-101 and Form 43-101F1 and is considered effective August 9, 2019. This Technical Report and PEA supersede all prior technical reports and Preliminary Economic Analyses prepared for the Phoenix Gold Project

Rubicon is a junior mineral exploration company listed on the Toronto Stock Exchange (TSX: RMX) with their head office at:

Suite 830-121 King Street West
Toronto, Ontario, Canada
M5H 3T9

This Technical Report is intended to be used by Rubicon, subject to the terms and conditions of their contract with TMAC; this permits Rubicon to file this report on SEDAR as a NI 43-101 Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. TMAC understands that Rubicon may use the Technical Report for a variety of corporate purposes. Except for the purposes legislated under provincial securities laws, any other use of this Technical Report, by any third party, is at that party's sole risk.

2.1 Sources of Information

This Technical Report has been prepared by independent consultants who are Qualified Persons under NI 43-101. Subject to the conditions and limitations set forth herein, the independent consultants believe that the qualifications, assumptions, and the information used by them is reliable, and efforts have been made to confirm this to the extent practicable. However, none of the consultants involved in this study can guarantee the accuracy of all information in this Technical Report.

This Technical Report is based, in part, on internal company technical reports and maps, published government reports, company letters and memoranda, and public information as listed in Section 27. Several sections from reports authored by other consultants have been directly quoted or summarized in this Technical Report and are so indicated where appropriate.

A draft copy of this Technical Report has been reviewed for factual errors by Rubicon regarding the company and history of the property, and the Mineral Resource Estimate (effective date: March 18, 2019) prepared by TMAC.

TMAC has relied on Rubicon's historical and current knowledge of the Phoenix Gold Project and work performed thereon. Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Technical Report.

2.2 Qualified Persons

This Technical Report and PEA were prepared by the QPs listed in Table 2-1, and their responsibilities for each section are indicated. The following summarizes the dates of the QPs Project site visits:

- Tim Maunula, P.Geo., completed four site visits between 2017 and 2019 with the most recent on February 18, 2019
- Andrew MacKenzie, P.Eng., completed two site visits with the most recent on March 20–22, 2019
- Peter Broad, P.Eng., Karlis Jansons, P.Eng., Charles Tkaczuk, P.Eng., and Ian Horne, B.Sc., completed their site visits on March 20–22, 2019.

Table 2-1: Qualified Persons – Section Responsibility

Section and Title	Qualified Person	Company
1: Summary	Andrew MacKenzie, P.Eng.	TMAC
2: Introduction	Andrew MacKenzie, P.Eng.	TMAC
3: Reliance on Other Experts	Andrew MacKenzie, P.Eng.	TMAC
4: Property Description and Location	Tim Maunula, P.Geo.	TMAC
5: Accessibility, Climate, Local Resources, Infrastructure	Tim Maunula, P.Geo.	TMAC
6: History	Tim Maunula, P.Geo.	TMAC
7: Geological Setting and Mineralization	Tim Maunula, P.Geo.	TMAC
8: Deposit Types	Tim Maunula, P.Geo.	TMAC
9: Exploration	Tim Maunula, P.Geo.	TMAC
10: Drilling	Tim Maunula, P.Geo.	TMAC
11: Sample Preparation, Analyses, and Security	Tim Maunula, P.Geo.	TMAC
12: Data Verification	Tim Maunula, P.Geo.	TMAC
13: Mineral Processing and Metallurgical Testing	Peter Broad, P.Eng.	TMAC
14: Mineral Resource Estimates	Tim Maunula, P.Geo.	TMAC
15: Mineral Reserve Estimates	Andrew MacKenzie, P.Eng.	TMAC
16: Mining Methods	Andrew MacKenzie, P.Eng.	TMAC
17: Recovery Methods	Peter Broad, P.Eng.	TMAC
18: Project Infrastructure	Andrew MacKenzie, P.Eng. Charles Tkaczuk, P.Eng. Karlis Jansons, P.Eng	TMAC GeoMin TMAC
19: Market Studies and Contracts	Andrew MacKenzie, P.Eng.	TMAC

Section and Title	Qualified Person	Company
20: Environmental Studies, Permitting and Social or Community Impact	Ian Horne, B.Sc. Biology	TMAC
21: Capital and Operating Costs	Andrew MacKenzie, P.Eng.	TMAC
22: Economic Analysis	Andrew MacKenzie, P.Eng.	TMAC
23: Adjacent Properties	Andrew MacKenzie, P.Eng.	TMAC
24: Other Relevant Data and Information	Andrew MacKenzie, P.Eng.	TMAC
25: Interpretation and Conclusions	Andrew MacKenzie, P.Eng. Tim Maunula, P.Geo.	TMAC TMAC
26: Recommendations	Andrew MacKenzie, P.Eng. Tim Maunula, P.Geo.	TMAC TMAC
27: References	Andrew MacKenzie, P.Eng.	TMAC

2.2.1 Acknowledgments

TMAC would like to thank and acknowledge the following people who have contributed to the preparation of this report and the underlying studies under the supervision of the QPs, including the following Rubicon employees: George Ogilvie, P.Eng., President and CEO; Nick Nikolakakis, CFO; Michael Willett, P.Eng., Director of Projects; Eric Setchell, Site General Manager; Kevin Canario, CPA, CA, Corporate Controller; Isaac Oduro, P.Geo., M.Sc., Manager of Technical Services; Adrian McNutt, P.E., Metallurgical consultant; John Frostiak, P.Eng., Mining and Metallurgical consultant; Jerrett Landry, Site Controller; Denise Saunders, P.Geo.; Geologist; Michael Nerup, P.Geo., Senior Geologist; Cathy Willett, Mine Planner; Lynne Rasmussen, B.Sc. (Hons), Environmental Coordinator; and Dana Dobrescu, Land Manager.

2.3 Units of Measure and Abbreviations

Unless otherwise noted, the following units of measure, formats, and systems are used throughout this Technical Report PEA:

- Units of Measure: all references to units of measure are based on the International System of Units (SI, or metric). The primary linear distance unit, unless otherwise noted, is the metre (m).
- General Orientation: unless otherwise stated, all property-scale references to orientation and coordinates in this Technical Report PEA are presented as decimal degrees in the Rubicon mine grid; the mine grid is oriented with grid north parallel to the orientation of the East Bay Deformation Zone (EBDZ), which results in a +45.0° rotation relative to magnetic north (0.0/360.0° azimuth in mine grid equates to 045.0° azimuth magnetic north).
- Currencies outlined in the Technical Report PEA are stated in Canadian dollars (C$) unless otherwise noted.

3 RELIANCE OF OTHER EXPERTS

This Technical Report was prepared for Rubicon Minerals Corporation (TSE: RMX) for the 100% owned Phoenix Gold Project (Phoenix, or the Project) in the Red Lake mining district of Ontario, Canada.

TMAC has assumed and relied on the fact, that all the information and existing technical documents listed in the References, Section 27 of this Technical Report, are accurate and complete in all material aspects. While TMAC carefully reviewed all the available information presented, we cannot guarantee its accuracy and completeness. We reserve the right, but will not be obligated, to revise the Technical Report and conclusions if additional information becomes known subsequent to the date of this Technical Report.

Although copies of the tenure documents, operating licences, permits, and work contracts were reviewed, an independent verification of land title and tenure was not performed. TMAC did not independently verify the legality of any underlying agreement(s) that may exist concerning the licences or other agreement(s) between third parties, but has instead relied on the client's solicitor to have conducted the proper legal due diligence.

4 PROPERTY DESCRIPTION AND LOCATION

The Phoenix Gold Project is located in the southwestern part of Bateman Township within the Red Lake mining district of northwestern Ontario, Canada (Figure 4-1). The Town of Red Lake is approximately 150 km northwest of Dryden, Ontario and 265 km northeast of Winnipeg, Manitoba.

The Phoenix Gold Project is centred on the Phoenix Shaft, located at UTM coordinates 448,167E, 5,663,962N (NAD 83 / Zone 15N) at an elevation of 369 m (above sea level). The total area of the land tenure is 510.4 hectares (ha).

Rubicon has a 100% interest in the Phoenix Gold Project subject to a 2% NSR royalty on most of the water portions of the property to Franco-Nevada Corporation and 1% on all tenure to RGLD Gold AG.

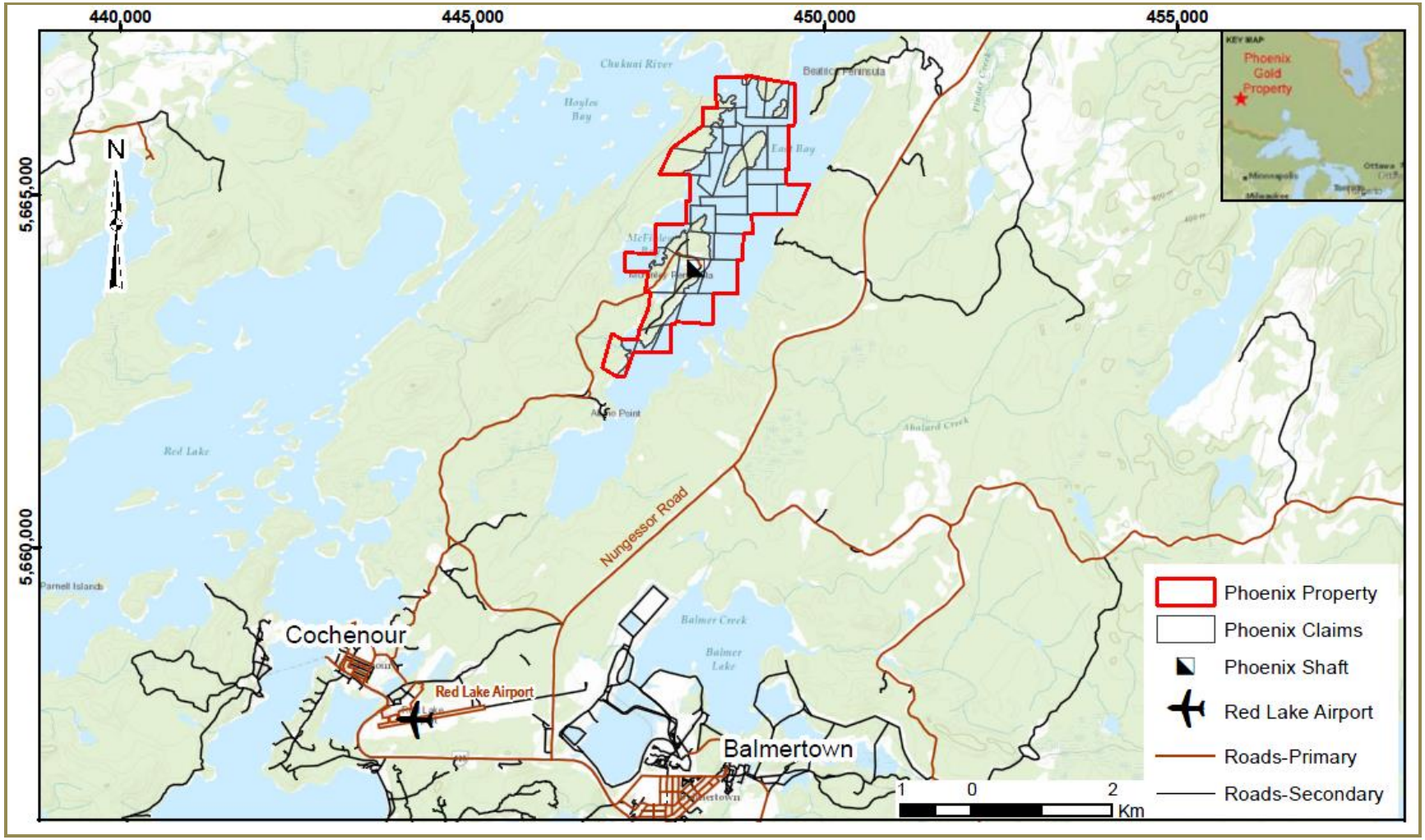


Figure 4-1: Location of the Phoenix Gold Project (Rubicon, 2019)

4.1 Property Land Tenure

The Phoenix Gold Project consists of 31 contiguous Mining Leases, Patented Claims, Mining Licences of Occupation, and a single Staked Claim (Table 4-1 and Figure 4-2) comprising:

- One Mining Lease covering four Kenora Red Lake (KRL) blocks
- Sixteen Patented Claims covering land portions of the Project
- Twenty-five Mining Licences of Occupation covering water portions of the property
- One Staked Claim.

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

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

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

The Mining Licences of Occupation are subject to payment of rents shown on the face of each licence. No application for renewal is required.

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

On June 22, 2009, Rubicon Minerals Corporation was registered with the ENDM as the 100% recorded holder for one Staked Claim. To maintain the claim in good standing Rubicon is required to carry out eligible assessment work of C$400 prior to June 22, 2022.

Table 4-1: Mineral Tenure Information

Short Legal Description	Mining Rights Number	Parcel Number	Start Date	Expiry Date	Hectares
Mining Lease					
KRL503297, KRL503298, KRL503299, KRL526262	LEA-108126	936LKP	Nov-86	31-Oct-28	56
Patented Mining Claims (Land Portion)					
K1498	PAT-7228	992DP	1-Oct-45	Not Applicable	3
K1499	PAT-7229	993DPF	1-Oct-45	Not Applicable	11.5
K1493	PAT-7224	994DPF	1-Mar-46	Not Applicable	5.1
K1494	PAT-7225	995DPF	1-Mar-46	Not Applicable	8.4
K1495	PAT-7226	996DPF	1-Mar-46	Not Applicable	10.4
KRL246	PAT-7222	997DP	1-Mar-46	Not Applicable	15
KRL247	PAT-7223	998DPF	1-Mar-46	Not Applicable	17.9
K1497	PAT-7227	999DPF	1-Mar-46	Not Applicable	13.5
KRL11481	PAT-7232	1446DPF	1-Nov-41	Not Applicable	4.2
KRL11482	PAT-7233	1447DPF	1-Nov-48	Not Applicable	6.9
KRL11483	PAT-7230	1448DPF	1-Nov-41	Not Applicable	12.2
KRL11487	PAT-7231	1452DPF	1-Nov-41	Not Applicable	15.3
K954 (recorded as KRL18152)	PAT-7234	1977DPF	1-Jan-47	Not Applicable	6.9
K955 (recorded as KRL18515)	PAT-7236	1978DPF	1-Jan-47	Not Applicable	4.1
KRL18457	PAT-7235	2449DPF	1-Jan-50	Not Applicable	7.9
KRL18735	PAT-7237	2450DPF	1-Jan-50	Not Applicable	20.9
Licences of Occupation (Water Portion)					
KRL2155	MLO-3186		1-Aug-45	Not Applicable	9.9
KRL2156	MLO-3187		1-Aug-45	Not Applicable	13.7
K1498	MLO-3289		1-Oct-45	Not Applicable	11
K1499	MLO-3290		1-Oct-45	Not Applicable	2.4
K1493	MLO-3370		1-Mar-46	Not Applicable	5
K1494	MLO-3371		1-Mar-46	Not Applicable	18.7
K1495	MLO-3372		1-Mar-46	Not Applicable	10.1
K1497	MLO-3380		1-Mar-46	Not Applicable	6.1
KRL246	MLO-3381		1-Mar-46	Not Applicable	4.3
KRL247	MLO-3382		1-Mar-46	Not Applicable	4.5
KRL11483	MLO-10495		1-Nov-41	Not Applicable	6.7
KRL11482	MLO-10496		1-Nov-48	Not Applicable	5.6
KRL11481	MLO-10497		1-Nov-41	Not Applicable	14.1
KRL11487	MLO-10499		1-Nov-41	Not Applicable	5.7
KRL11038-39 (recorded as KRL18377)	MLO-10830		1-Jan-47	Not Applicable	28.7

Short Legal Description	Mining Rights Number	Parcel Number	Start Date	Expiry Date	Hectares
KRL11031 (recorded as KRL18519	MLO-10834		1-Jan-47	Not Applicable	17.9
K954 (recorded as KRL18152)	MLO-10835		1-Jan-47	Not Applicable	9.3
K955 (recorded as KRL18515)	MLO-10836		1-Jan-47	Not Applicable	10
KRL18514	MLO-10952		1-Oct-47	Not Applicable	17.5
KRL18735	MLO-11111		1-Jan-50	Not Applicable	12.2
KRL18457	MLO-11112		1-Jan-50	Not Applicable	11
KRL18373	MLO-11114		1-Jan-50	Not Applicable	7.7
KRL18374	MLO-11115		1-Jan-50	Not Applicable	19.7
KRL18375	MLO-11116		1-Jan-50	Not Applicable	22.9
KRL18376	MLO-11117		1-Jan-50	Not Applicable	15
Staked Claim					
143643 & 312889 (single cell claims) converted from KRL4229741		N/A	22-Jun-09	21-Jun-24	16
Total Area					**525**

Note: The total hectares may not add up due rounding.

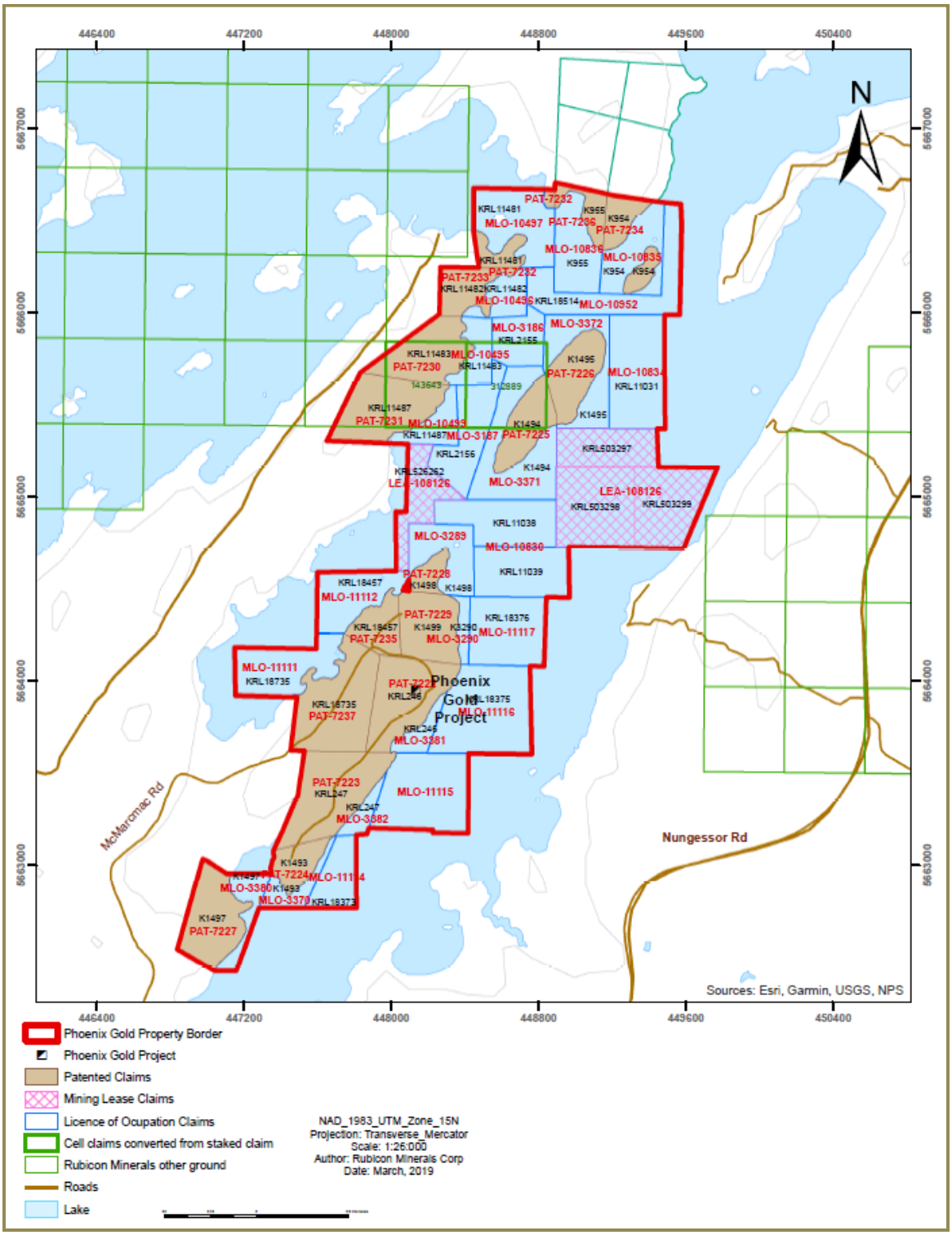


Figure 4-2: Phoenix Property Land Tenure (Rubicon, 2019)

4.2 Underlying Agreements

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

The water portions of the property, except the Staked Claim, are subject to an NSR royalty of 2% to Franco-Nevada Corporation. Franco-Nevada Corporation purchased the NSR royalty from Dominion Goldfields in August 2011. Advance royalties of US$50,000 are due annually to a maximum of US$1,000,000 prior to commercial production, of which a cumulative US$750,000 was paid by Rubicon to January 1, 2018. Rubicon has the option to acquire a 0.5% of this NSR royalty for US$675,000 at any time; however, this option is subject to a right of first refusal, whereby a third party has the initial right to exercise this option, in which case the NSR royalty to Franco-Nevada Corporation would be reduced to 1.5%. Upon a positive production decision, Rubicon would be required to make an additional advance royalty payment of US$675,000. Rubicon has confirmed that the annual payments are up to date.

The mining rights of the Patented Claims were optioned from Dominion Goldfields in June 2002 and the rights pertaining to surface portions of the same patented claims were optioned from Dominion Goldfields subsidiary 1519369 Ontario Ltd.

The surface rights of the Patented Claims are owned by 0691403 B.C. Ltd., a wholly-owned subsidiary of Rubicon. On October 25, 2011, Rubicon announced that by execution of its right of first refusal under its agreement with Dominion Goldfields, it had acquired and thereby extinguished all royalties on the blocks covering the land portions of the property. On closing the agreement, Rubicon issued a total of 1,216,071 of its common shares to Dominion Goldfields, at a deemed price per share of C$3.50, for a total consideration of C$4,256,249.

On February 10, 2014, Rubicon entered into a US$75 million gold streaming agreement (Streaming Agreement) with Royal Gold Inc. and its affiliate, RGLD Gold AG. On May 12, 2015, Rubicon entered into a US$50 million secured loan agreement (Loan Agreement) with CPPIB Credit Investments Inc., a wholly-owned subsidiary of Canada Pension Plan Investment Board.

On December 20, 2016, following the completion of the restructuring of Rubicon, the amount outstanding under the Loan Agreement was reduced to C$12 million and the Streaming Agreement was exchanged in part for a 1.0% NSR royalty on all tenure (Patented, Lease, Mining Licences of Occupation, and the staked claim) of the Phoenix Gold Project granted to RGLD Gold AG under a

royalty agreement (Royalty Agreement). On December 20, 2018, the Loan Agreement was transferred to Sprott Private Resource Lending (Collector), L.P. (Sprott).

Pursuant to the Loan Agreement (Sprott) and the Royalty Agreement (RGLD Gold AG), the Mining Lease, owned Patented Claims, Licences of Occupation, and the Staked Claim of the Phoenix property are subject to charges/mortgages in favour of Sprott and RGLD Gold AG, respectively.

4.3 Permits and Authorization

Rubicon currently holds all material permits required for it to carry out its drilling, underground exploration, and development initiatives, and is substantially permitted for potential future production on the Phoenix Gold Project at an annual average rate of 1,250 tonnes per day (t/d). The industrial sewage Environmental Compliance Approval (ECA) contains several clauses that would need to be fulfilled prior to any potential commencement of commercial production. Further amendments to some permits would be required for any potential future increases to the currently authorized production rate. Please note that the Rubicon Gold Project is in the development stage, and there is no current Mineral Reserve estimate defined for the Project. Future production is not supported by the current PEA, pre-feasibility, or feasibility study.

A full list of permits and applications, including their current statuses, is provided in Section 24.

4.4 Environmental Considerations

The current and potential production phase environmental liabilities associated with the Project site are described in the Phoenix Gold Project Closure Plan (June 2016), filed with the Ontario ENDM pursuant to Part VII of the *Mining Act*. There are no significant physical stability liabilities associated with the Project site, and chemical stability issues are limited to two areas that may require excavation and removal of contaminated soil. 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*.

4.5 Mining Rights in Ontario

The Phoenix Gold Project is located in the province of Ontario, a jurisdiction that has a well-established permitting process. This process is coordinated between the municipal, provincial, and federal regulatory agencies. As is the case for similar mine developments in Canada, the Project is subject to federal and provincial environmental assessment processes. Due to the complexity and size of such projects, various federal and provincial agencies have jurisdiction to provide authorizations or permits that enable Project construction to proceed.

Federal agencies that have significant regulatory involvement include the Canadian Environmental Assessment Agency, Environment and Climate Change Canada, Natural Resources Canada, and Fisheries and Oceans Canada.

Provincially, the Ministry of Energy Northern Development and Mines, Ministry of Environment, Conservation and Parks, Ministry of Transportation, and the Ministry of Natural Resources and Forestry each have key Project development permit responsibilities.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The Phoenix Gold Project is centred within the Red Lake mining district of northwestern Ontario, approximately 565 km by road (430 km direct) northwest of Thunder Bay and approximately 475 km by road (265 km direct) east–northeast of Winnipeg, Manitoba. Red Lake can be reached via Highway 105, which branches off the Trans-Canada Highway (Ontario Highway 17) some 170 km south of Red Lake. Red Lake is also serviced with daily flights from Thunder Bay and Winnipeg.

The Project site is accessible via 8 km of all-weather road from Nungesser Road in the community of Balmertown, in the Municipality of Red Lake (Figure 4-1).

5.2 Local Resources and Infrastructure

The Red Lake Municipality comprises six communities: Red Lake, Balmertown, Cochenour, Madsen, McKenzie Island, and Starratt-Olsen. The 2016 Canada Census indicates a population of 4,107 for the area. Mining is the primary industry and employer; other industries include small-scale logging and tourism focused on hunting, fishing and access to the Woodland Caribou Park which is now part of the Pimachiowin Aki World Heritage site. All services expected in a municipality of this size are present, including a hospital and medical clinic.

The Phoenix Gold Project site is currently supplied by a 9.0 km power transmission line connected to Hydro One's 44,000 V (44 kilovolts [kV]) M6 feeder in the Red Lake Transformer Station. There are two (parallel connected) 18 megavolt ampere (MVA) transformers in the main substation (one main, one back up) as well as 2 megawatt (MW) of diesel emergency power generation capacity. On-site distribution reduces voltage to 4,160 V for surface and underground. Further voltage step-downs are used locally as required for specific equipment installations.

Mine water supply is from the nearby East Bay of Red Lake. The water is piped underground via a 100 mm diameter water line for purposes such as drilling, muck pile watering, and others. A potable water plant is fully commissioned and operating on site. A second potable water plant is located in the camp area, although this area is not currently operational. Rubicon has all the surface rights required to conduct its potential operations at the Phoenix Gold Project, and has access to local and fly-in–fly-out workers. Workers requiring accommodations in the area are currently housed off site.

5.3 Climate

The climate in the Red Lake portion of northwestern Ontario is considered subarctic, with temperature extremes generally ranging from winter lows of approximately −45 degrees Celsius (°C) to summer highs of roughly 30°C. Average winter temperatures are in the range of −15°C to −20°C, and average summer temperatures are in the range of 15°C to 20°C. Between 1971 and 2000, annual average precipitation was measured at 686 mm, with the greatest majority being received as rainfall in the summer and early fall months (May to October). Mean annual rainfall measured 515 mm with 171 mm equivalent annual average snowfall. Average winter snow depths in the region range from 40 to 50 cm. Weather conditions have minimal impact on underground production, allowing operations to proceed all year long.

5.4 Physiography

The topography within much of the Project is mildly rugged. The elevation is commonly less than 15 m above the level of Red Lake. Glacially scoured southwest-trending ridges, swamps, marshes, small streams, and small- to moderate-sized lakes dominate the topography. Rock exposure varies locally, but rarely exceeds 15% of the surface area, and is mostly restricted to shoreline exposures. Glacial overburden depth is generally shallow, rarely exceeding 10 m, and primarily consists of ablation till, minor basal till, minor outwash sand and gravel, and silty clay glaciolacustrine sediments.

Vegetation consists of thick boreal forest composed of black spruce, jack pine, trembling aspen, and white birch. Figure 5-1 illustrates the typical landscape around the Phoenix Gold Project and the associated vegetation.

The East Bay of Red Lake with McFinley Island covers a portion of the Project directly to the north of McFinley Peninsula, representing the largest island on the property. Seismic surveys completed in the past indicated average accumulations of 10 to 20 m of lake sediments and overburden on the lake bottom, with the water depth less than 8.5 m within the property boundary. The location of the tailings' storage area and other site infrastructure is discussed in Section 16.4.

Figure 5-1: Typical Landscape in the Phoenix Gold Project Area (Rubicon, 2019)

6 HISTORY

Information in this section is summarized from a previous technical report prepared by AMC Mining Consultants (Canada) Ltd. (AMC) (2011) and references therein.

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

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

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

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

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

6.1 Historical Exploration

The extensive history of exploration activities on the Project has been described in detail in two previous reports prepared by G. M. Hogg (2002a, 2002b). One report covered the Patented Claims; the second document discussed historical work completed on the Mining Licences 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 Table 6-1. Exploration activities from 2002 through 2018 are summarized in Section 9, Table 9-1.

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 area. Trenching, sampling, and shallow drilling was undertaken by McCallum Red Lake Mines Ltd. Widespread but erratic gold mineralization was noted in cherty metasedimentary rock on both McFinley Peninsula and McFinley Island.
1941–1942	Mineral occurrences were drilled as part of the Wartime Minerals Evaluation program.
1944–1946	McFinley Red Lake Gold Mines Ltd. carried out ground magnetic surveys, a 48-drill hole drilling program consisting of 167 m (548 feet [ft]) of drilling over the McFinley Peninsula, and a 1,487 m (4,877 ft) drilling program from the ice of Red Lake.
1946–1955	Fourteen drill holes (M Series) were completed for approximately 1,585 m (5,200 ft) of diamond drilling.
1955–1956	Little Long Lac Gold Mines sank a 130 m (428 ft) vertical shaft on claim KRL 246 and completed 414 m (1,358 ft) of exploratory underground development on two levels.
1974–1975	Sabina completed 25 diamond drill holes for approximately 3,048 m (10,000 ft) of drilling on the Project; ground magnetic and electromagnetic surveys; and 10 drill holes for approximately 735 m (2,410 ft) of diamond drilling over a portion of the lake properties.
1981–1983	Sabina and McFinley Red Lake Mines completed a magnetic/electromagnetic geophysical survey over the McFinley Peninsula area, surface bulk sampling, and 3,672 m (12,046 ft) of surface diamond drilling in 33 drill holes.
1983–1984	McFinley Red Lake Mines and Sabina completed seven drill holes for approximately 646 m (2,120 ft) of diamond drilling.
1984–1985	An agreement with Phoenix Gold Mines Ltd. allowed the opening of the McFinley Shaft (now called the Phoenix Shaft) and completion of a total of 479 m (1,570 ft) of drifting and crosscutting on the 150 ft (46 m) and 400 ft (122 m) levels. Metallurgical work and mineral processing were carried out. Eighty underground drill holes totaling 1,829 m (6,000 ft) and 69 surface drill holes totaling 10,628 m (34,870 ft) of diamond drilling were completed. Funding difficulties resulted in the Project being placed on temporary standby in February 1985.
1985–1987	A total of 1,151 m (3,775 ft) of drifting and crosscutting was carried out on the 150 ft (46 m) and 400 ft (122 m) levels. A total of 7,111 m (23,333 ft) of underground drilling, 9.14 m (30 ft) of raising, and an extensive chip-sampling program were completed. A program of 12,763 m (41,874 ft) of diamond drilling was also completed in 61 surface drill holes.
1987–1989	In recognition of a nugget effect in sampling results, a decision was made to proceed with a minimum 15,000-tonne bulk sample. A 150 t/d mill and tailings management facility was constructed. Underground development (2,890 m / 9,482 ft) continued on the 150 ft (46 m) and 400 ft (122 m) levels, a new 275 Level (at 84 m) and on a ventilation raise from the 400 ft (122 m) Level to surface. Additional sampling, diamond drilling (8,730 m [28,642 ft]), and metallurgical testing were completed. Bulk sampling operations commenced in July 1988, with sampling indicating head grades in the range of 0.25 ounces per tonne gold (8.57 g/t Au) from prepared stope areas (Hogg, 2002b). Mill design problems, lack of income from bulk sampling, and lack of exploration funding forced the closure of the operation after an estimated 2,250 tonnes of material were milled. Total historical development in drifting, crosscutting, and raising is estimated to be more than 5,791 m (19,000 ft). Total historical diamond drilling focused on the McFinley Peninsula area is estimated to be 45,110 m (148,000 ft) from surface and 35,814 m (117,500 ft) from underground. An estimated 54,864 m (180,000 ft) of core is stored on the property (Hogg, 2003).
1999–2002	Dominion Goldfields foreclosed on the Mining Licences of Occupation and Mining Lease and was awarded title to the lake portion of the Phoenix Gold Project in 1999 and 2002, respectively. Dominion Goldfields and its subsidiary were subsequently awarded title to the Patented Claims of the Project in 2002.

6.2 Previous Mineral Resource Estimates

The 2019 Mineral Resource estimate discussed herein has superseded historical and past Mineral Resource estimates presented in this section. The following historical information is relevant to provide context, but is not current and should not be relied upon. The QPs responsible for the preparation of this Technical Report have not done sufficient work to classify the historical estimate as current Mineral Resources or Mineral Reserves, and Rubicon is not treating any historical estimates as current Mineral Resource or Mineral Reserve estimates.

6.2.1 McFinley Red Lake Mines (1986)

McFinley Red Lake Mines staff prepared a historical Mineral Resource estimate in 1986 (Hogg, 2002a, 2002b). The McFinley Red Lake Mines historical Mineral Resource is located approximately 450 m northwest of the F2 Gold Zone. The estimate refers to the shaft area located on the McFinley Peninsula where historical underground exploration and development, and extensive sampling, were carried out. The shaft area is in stratigraphic units separate from the current F2 Gold Zone. The 1986 historical Mineral Resource estimate was developed using underground sampling results augmented with closely spaced drill hole data. The historical Mineral Resource estimate (to a depth of 400 ft) published in 2002 was 303,006 tonnes (334,007 tons) at a grade of 6.86 g/t Au (0.20 oz of Au/ton).

6.2.2 GeoEx Limited (2010 and 2011)

GeoEx Limited (GeoEx) reported a historical Mineral Resource estimate for the F2 Gold Zone in April 2011 (GeoEx, 2011b). The historical Mineral Resource estimate was calculated for data from surface to the 1,200 m Level using the polygonal Mineral Resource estimation method, with a 5.0 g/t Au cut-off. An Inferred Mineral Resource estimate of 5,500,000 tonnes at a grade of 20.34 g/t Au was reported.

6.2.3 AMC Mining Consultants (Canada) Ltd. (2011)

AMC prepared a Mineral Resource Statement (AMC, 2011) for the F2 Gold Zone using a block modelling approach based on drilling information available to February 28, 2011 (Table 6-2). The model was not constrained by a crown pillar and was extended to incorporate all drilling data. The Mineral Resource Statement was reported at a cut-off grade of 5.0 g/t Au.

Table 6-2: Mineral Resource Statement, Phoenix Project (AMC, 2011)

Mineral Resource Category	Tonnage (Mt)	Grade (g/t Au)	Gold (Moz)
Indicated	1.028	14.5	0.477
Inferred	4.230	17.0	2.317

Notes: CIM definition standards used for Mineral Resources. Cut-off grade of 5.0 g/t Au applied. Capping value of 270 g/t Au applied to composites. Based on drilling results to February 28, 2011. The 2011 Mineral Resource estimates are not current and should not be relied upon.

In all, 511 drill holes were used in the 2011 AMC Mineral Resource Statement. AMC reviewed and accepted Rubicon's interpretations of lithologies, mineralization controls, and geology domains. Twelve mineralized domains were interpreted by AMC using a low gold threshold (0.1 g/t Au) which incorporated all significant mineralized zones.

A composite length of 1.0 m was chosen, and gold composites were capped at 270 g/t Au. The parent block size was 2 m x 8 m x 12 m, and sub blocking was used. The model blocks were assigned a gold grade using an IDW3 estimator and a three-pass search strategy with search ellipsoids adjusted to the geometry of the modelled gold mineralization. Search parameters for the first pass were 8 m x 24 m by 36 m for the second and third pass the search volumes were inflated by two and three times, respectively. An average bulk density value of 2.90 tonnes per cubic metre (t/m^3) was used for all rock types.

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

6.2.4 SRK Consulting (Canada) Inc. (2013)

SRK Consulting (Canada) Inc. (SRK) (2013b) prepared a Mineral Resource Statement for the F2 Gold Zone using a block modelling approach based on drilling information available to October 31, 2012. The database included information from 820 core drill holes (355,611 m), all drilled by Rubicon since 2008. The model was not constrained vertically by a crown pillar. The 2013 SRK Mineral Resource Statement was reported at a cut-off grade of 4.0 g/t Au (Table 6-3).

The gold mineralization wireframes were defined using an explicit wireframe interpretation constructed from a sectional interpretation of the drilling data that took into consideration structural geology investigation and modelling undertaken by SRK in collaboration with Rubicon. Mineral Resource domains were defined using a 0.5 g/t Au threshold. Within the gold mineralization domains, narrower, higher-grade sub-domains were defined using a 3.0 g/t Au threshold. SRK defined 56 gold mineralization domains (31 higher-grade and 25 lower-grade domains) that were used to constrain Mineral Resource modelling. These 56 domains were combined into three groups based on their spatial orientation: Main, Main 45, and Hanging Wall (HW). Also, the gold mineralization located outside the modelled domains was evaluated unconstrained.

Four rotated sub-celled block models were generated with block sizes and orientation specific to the mineralization domain grouping. SRK chose a primary 2.5 m x 5 m x 10 m dimension for the Main and Main 45 domains, a 10 m x 20 m x 20 m dimension for the HW domain and a 5 m x 10 m x 20 m dimension for the External domain.

Table 6-3: Mineral Resource Statement*, Phoenix Gold Project (SRK, 2013)

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

Notes: *Mineral Resources are not Mineral Reserves and do not have a demonstrated economic viability. All figures have been rounded to reflect the relative accuracy of the estimates. This Mineral Resource estimate is reported at a cut-off grade of 4.0 g/t Au and assume an underground extraction scenario, a gold price of US$1,500/oz, and metallurgical recovery of 92.5%. The Main domain includes the Main 45 domain. The 2013 Mineral Resource estimates are not current and should not be relied upon.

Sample assay data were composited to a 1.0 m length and extracted for geostatistical analysis and variography. The impact of gold outliers was examined on composites using log probability plots and cumulative statistics. SRK evaluated the spatial distributions of the gold mineralization using variograms and correlograms of original capped composited data as well as the normal score transform of the capped composited data. The block model was populated with a gold grade using OK. Three estimation runs were used, each considering increasing search neighborhoods and less-restrictive search criteria. The first estimation pass considered search neighborhoods adjusted to 80% of the modelled variogram ranges. A uniform specific gravity (SG) of 2.87 t/m^3 was applied to the lower-grade domains and a value of 2.96 t/m^3 was assigned to the higher-grade domains to convert volumes into tonnages.

6.2.5 SRK Consulting (Canada) Inc. (2016)

The 2016 Mineral Resource Statement (SRK, 2016) was based on a revised geological model that considered information from 94,575 m of new infill core drilling information acquired since October 31, 2012, the cut-off date for the previous 2013 SRK Mineral Resource Statement. The 2016

SRK Mineral Resource Statement reported included drilling information available to November 1, 2015. In addition, the Mineral Resource estimate considered information on geological continuity gained from excavated underground workings exposing the gold mineralization on several levels and in test stopes.

The Mineral Resource estimates were evaluated using a geostatistical block modelling approach constrained by 71 explicit gold mineralization wireframes interpreted using a 3.0 g/t Au cut-off grade—high grade (HG)—and enclosed in 19 explicit gold mineralization wireframes derived using a 0.5 g/t Au cut-off grade—low grade (LG). The HG domains were constructed as explicit wireframes using interval selections of assay data while the broad LG domains were constructed with polylines on vertical sections. The domains were not modelled as grade interpolants.

Assay statistics were assessed for each domain separately, and capping was applied to samples prior to compositing. Capping values were chosen based on a combination of probability plots, decile analysis, capping sensitivity plots, and three-dimensional (3-D) visualization to determine the capping values. Capping in the HG domains range from 10 to 120.0 g/t Au, and in the LG, domains range from 5.0 to 45.0 g/t Au. Gold and capped assay data were composited to a 1.0 m length and extracted for geostatistical analysis and variography.

SRK evaluated the spatial distributions of the gold mineralization using traditional semi-variograms and traditional correlograms of composited data, as well as the normal score transform of the composited data.

A block model was generated with a block size of 2.5 m by 5 m by 5 m with sub-cells at 0.5 m resolution used to honor the geometry of the modelled mineralization. The block model was populated with a gold grade using OK. Three estimation runs were used, each considering increasing search neighborhoods and less restrictive search criteria. A spatial restriction was applied to high-grade composites to further restrict their influence during estimation.

In the F2 Gold Zone, higher-grade gold mineralization was associated with crosscutting, east-west trending D2 structures, while the plunge of the gold mineralization within a given domain is controlled by the line of intersection between the domain and the crosscutting structure. Using the dynamic anisotropy function in Datamine Studio 3, polylines were used to assign an estimated dip and dip direction for each cell of that HG domain in the block model based on those intersections.

Based on SG measurement of core samples, a mean SG value for the domain type and lithology was assigned to blocks to convert volumes into tonnages. The SG of lithology and mineralization domains varied from 2.76 to 2.90 t/m^3.

SRK considered that blocks within the HG domains estimated during the first estimation pass, informed from composites from at least three drill holes from five octants and located within the full

range of the variogram for that domain, could be classified in the Indicated category within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (November 2010). SRK considered that for those blocks the level of confidence was sufficient to allow the appropriate application of technical and economic parameters to support mine planning, and to allow the evaluation of the economic viability of the deposit. Conversely, all other modelled blocks were classified in the Inferred category as the confidence in the estimates was insufficient to allow for the meaningful application of technical and economic parameters, or to enable an evaluation of economic viability.

SRK considered that the gold mineralization at the Phoenix Gold Project was amenable to underground extraction. SRK reported the Phoenix Gold Project Mineral Resource estimates at a cut-off grade of 4.0 g/t Au. The 2016 SRK Mineral Resource Statement for the Phoenix Gold Project is presented in Table 6-4. Mineralization excavated by underground development, stoping blocks, and in a 40 m crown pillar below the lake bottom has been excluded from the 2016 SRK Mineral Resource Statement.

Table 6-4: Mineral Resource Statement*, Phoenix Gold Project (SRK, 2016)

Mineral Resource Category	Quantity (t '000)	Grade Au (g/t)	Contained Gold (oz '000)
Measured	-	-	-
Indicated	492	6.73	106
Measured + Indicated	492	6.73	106
Inferred	1,519	6.28	307

Notes: *All figures are rounded to reflect the relative accuracy of the estimate. Samples have been capped where appropriate. Underground Mineral Resource estimates reported at a cut-off grade of 4.0 g/t Au assuming a metal price of US$1,125/oz of gold and a gold recovery of 92.5%. The 2016 Mineral Resource estimates are not current and should not be relied upon.

6.2.6 Golder Associates (2018)

The 2018 Mineral Resource Statement (Golder, 2018b) was based on data provided by Rubicon from surface and underground diamond drill programs, as well as chip samples and mapping from underground development completed mainly between 2002 and 2017. All data received was in the Phoenix Mine coordinate system, which is rotated 45 degrees to the east of true north. No other data translations were completed for this Mineral Resource estimate.

The Phoenix Gold Project mineralization was modelled in four zones, defined as Zones 1 to 4. A 3-D block model was constructed for estimating stratigraphy (i.e., rock type groupings) and Au grades, where stratigraphy was used as a zonal control on gold grade estimates. High-grade, outlier samples were controlled by top-cutting with a maximum distance restriction of 10 m. Resources were reported at a 3.0 g/t Au cut-off grade and classified according to CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). Density values were assigned to the model based on the default mean value of each stratigraphic unit.

The block-model validation process included visual comparisons between block estimates and composite grades in plan and section, along with global comparisons of mean grades, swath plots and smoothing ration calculations.

Table 6-5 presents the Mineral 2018b Golder Resource Statement for the Phoenix Gold Project. This Mineral Resource estimate excludes isolated blocks with little potential for mining, mineralization within the crown pillar, and all mineralized development that has been mined.

Table 6-5: Mineral Resource Statement, Phoenix Gold Project (Golder, 2018b)

Mineral Resource Category	Quantity (t '000)	Grade Au (g/t)	Contained Gold (oz '000)
Measured	188	6.80	41,000
Indicated	1,186	6.30	240,000
Measured + Indicated	1,374	6.37	281,000
Inferred	3,884	6.00	749,000

Notes: Effective date for this Mineral Resource estimate is April 30, 2018.
Mineral Resource estimate uses a break-even economic cut-off grade of 3.0 g/t Au based on assumptions of a gold price of US$1,300/oz; an exchange rate of US$/C$0.77; mining cash costs of C$97/t; processing costs of C$20/t; G&A of C$5/t; sustaining capital C$10/t; refining, transport, and royalty costs of C$53/oz; and average gold recoverability of 92%.
Mineral Resource estimate reported from within an envelope accounting for mineral continuity.
Mineral Resources are not Mineral Reserves and do not demonstrate economic viability.
There is no certainty that all or any part of this Mineral Resource estimate will be converted into Mineral Reserve.
All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.
The 2018 Mineral Resource estimates are not current and should not be relied upon.

6.2.7 Mineral Reserve Estimates

There are no historical Mineral Reserve estimates at the Phoenix Gold Project.

6.3 Past Production

There has been limited past production in the form of lateral development and trial long-hole stope mining on the property. Mining exploration activities on the property were terminated in 1989 after test-milling of an estimated 2,250 tonnes of material unrelated to the F2 Gold Zone (Hogg, 2003).

Development of the Phoenix Gold Project by Rubicon commenced in 2012 with shaft deepening and mill building foundation work, followed by the establishment of levels and associated infrastructure at the 122 m, 183 m, 244 m, 305 m, 488 m, and 610 m levels.

In 2015, Rubicon began trial stoping on the 305 m Level. Subsequent trial stoping followed on the 183 m and 244 m levels. Typical development followed mineralized material, via Alimak raising, lateral sill, and sublevel advance. Test production of three long-hole stopes was completed on the 305 m and

244 m levels. The 244-Z2-159, 244-Z1-977, and 305-Z2-030 stopes were mined, skipped to surface, and processed at the Rubicon mill facility on site.

Rubicon processed 57,793 tonnes of mineralized material, grading at 3.02 g/t Au. Rubicon achieved an average mill recovery of 91.9% and produced 5,610 oz of gold. Underground activities were suspended on November 3, 2015, and milling ceased on November 21, 2015.

In 2018, Rubicon completed a test mining and processing program that started in July and was completed in October. During that time, 43,250 tonnes of material was processed through the mill, grading at 4.08 g/t Au, for a total of 5,669 oz of gold being produced.

Within this total material run through the mill in 2018, three test stopes (244-Z1-015, 244-Z1-977, and 183-Z1-161) were developed and mined for a total of 35,629 tonnes at a grade of 4.51 g/t Au, for a total of 5,165 oz of gold being produced. A reconciliation exercise was completed on the three test stopes by comparing actual tonnes, grade, and ounces of gold to that in the 2018 Mineral Resource block model of tonnes, grade, and ounces of gold. This resulted in a positive reconciliation of 7.4% for tonnes, 5.9% for grade, and 13.8% for ounces of gold occurring (Golder, 2019).

Rubicon achieved an average mill recovery of 95.1% gold recovery, with 43.2% gold recovered from gravity. Based on operating information during the processing of the bulk sample, the processing plant was able to sustain a steady-state throughput rate of 70 tonnes per hour (t/h), which is equivalent to 1,540 t/d, on a 22-hour per day availability during the bulk-sample processing exercise.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The following description of the geology of the Red Lake Greenstone Belt was modified from Sanborn-Barrie et al. (2004) and the references therein. Orientations referenced in this section are relative to true north.

The Phoenix Gold Project is located in the Uchi Subprovince of the Superior Province of the Canadian Precambrian Shield. Within the Uchi Subprovince, the Red Lake Greenstone Belt is host to one of Canada's pre-eminent gold districts, having produced more than 29 Moz of gold since the 1930s.

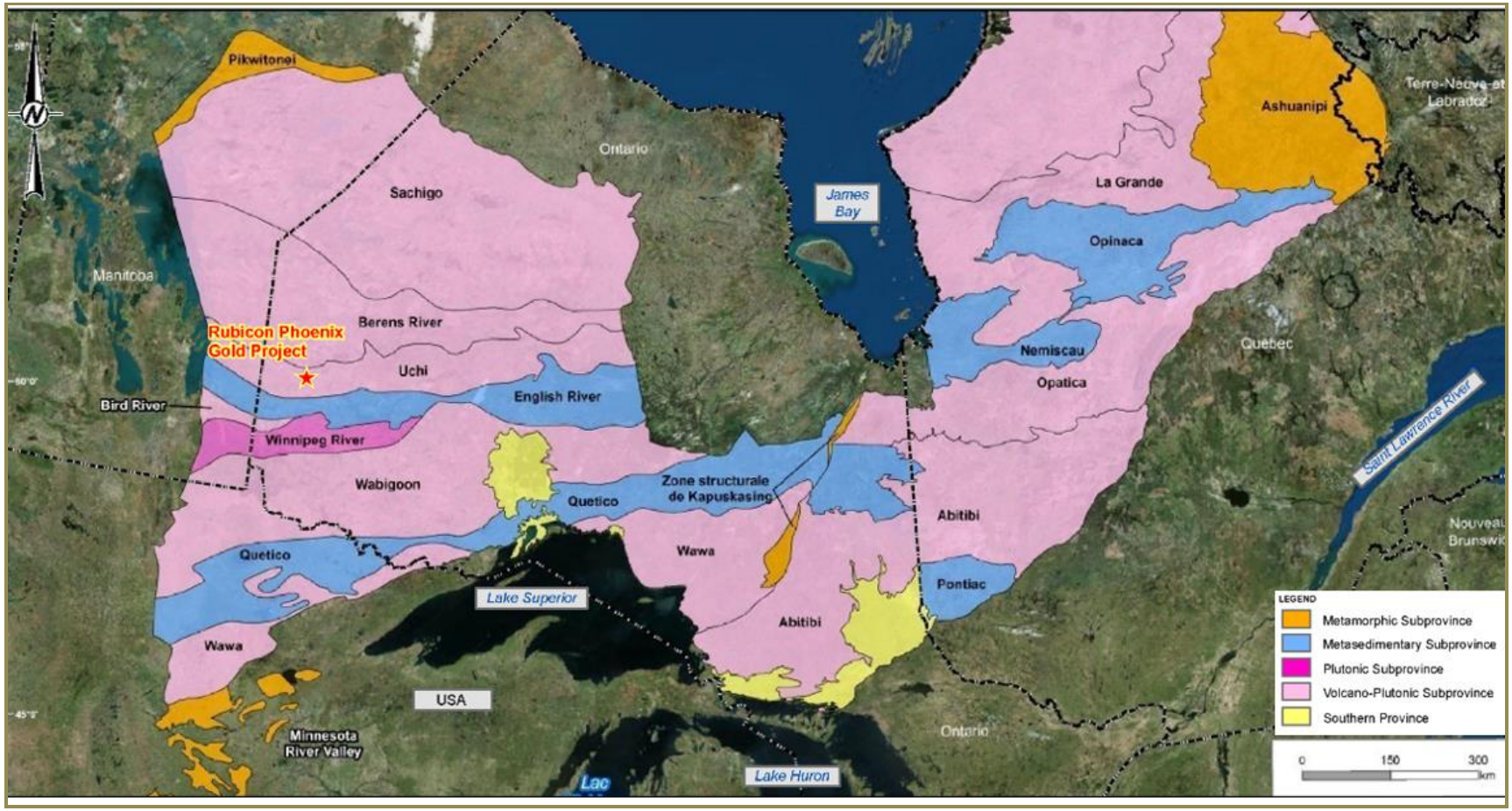
The Red Lake Greenstone Belt is interpreted to have been formed on the southern margin of the North Caribou Terrane, an ancient Mesoarchean continental block of approximately 3,000 million years before present (Ma) that makes up part of the southern Uchi Subprovince (Figure 7-1 and Figure 7-2). The Red Lake Greenstone Belt was formed and evolved as the result of extensive magmatic and sedimentary activity as well as multiple events of intense deformation, metamorphism, hydrothermal alteration, and gold mineralization that occurred between 3,000 and 2,700 Ma. Regional metamorphic assemblages indicate that peak metamorphism corresponded to greenschist and amphibolite grades.

The regional geology of the Red Lake Belt is shown in Figure 7-3, and described here in chronological order, proceeding from the oldest to the youngest stratigraphic assemblages.

Rocks of the Mesoarchean Balmer Assemblage, the oldest stratigraphic assemblage in the Red Lake Greenstone Belt, host all the major gold producers in the Red Lake District. The Balmer Assemblage is dated between circa (ca.) 3,000 and 2,988 Ma, and includes volcanic units composed of komatiite, komatiitic basalt, and tholeiitic basalt as well as lesser amounts of peridotitic and gabbroic intrusive rocks, felsic volcanic rocks, iron formation, and clastic sedimentary rocks.

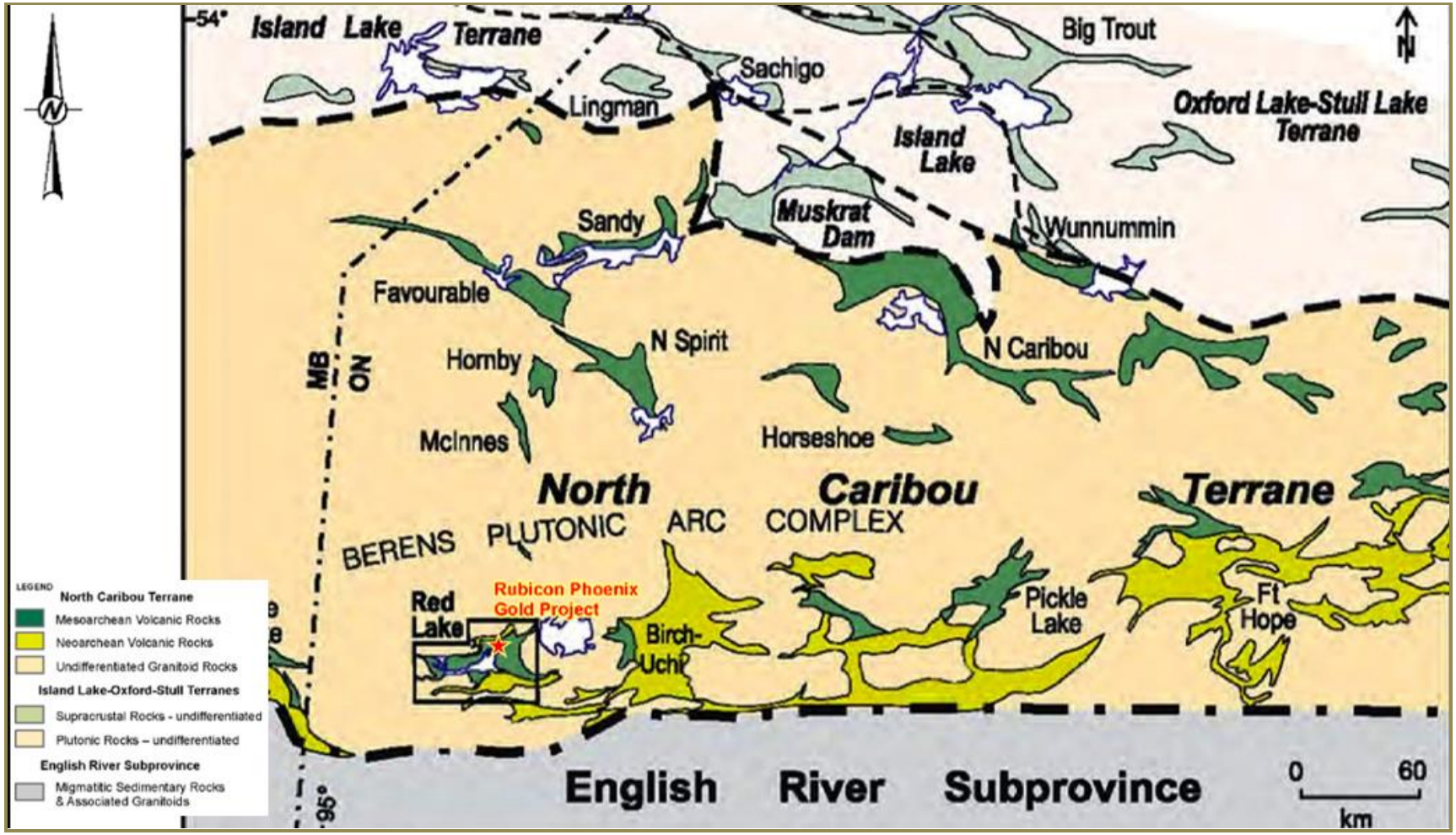
Underlying the northwestern portion of the Red Lake Greenstone Belt is the Ball Assemblage (ca. 2,940 Ma to ca. 2,925 Ma), consisting predominantly of a thick sequence of metamorphosed intermediate to felsic calc-alkaline volcanic flows and pyroclastic rocks, and lesser amounts of mafic to ultramafic volcanic rocks and peridotitic to gabbroic intrusive rocks.

The Slate Bay Assemblage (ca. 2,903 Ma to ca. 2,850 Ma) extends the length of the belt and consists of clastic sedimentary rocks including several lithological facies: conglomerates, quartzose arenites, wackes, and mudstones. The contact of the Slate Bay Assemblage with the underlying Ball and Balmer assemblages represents an unconformity (Figure 7-3).



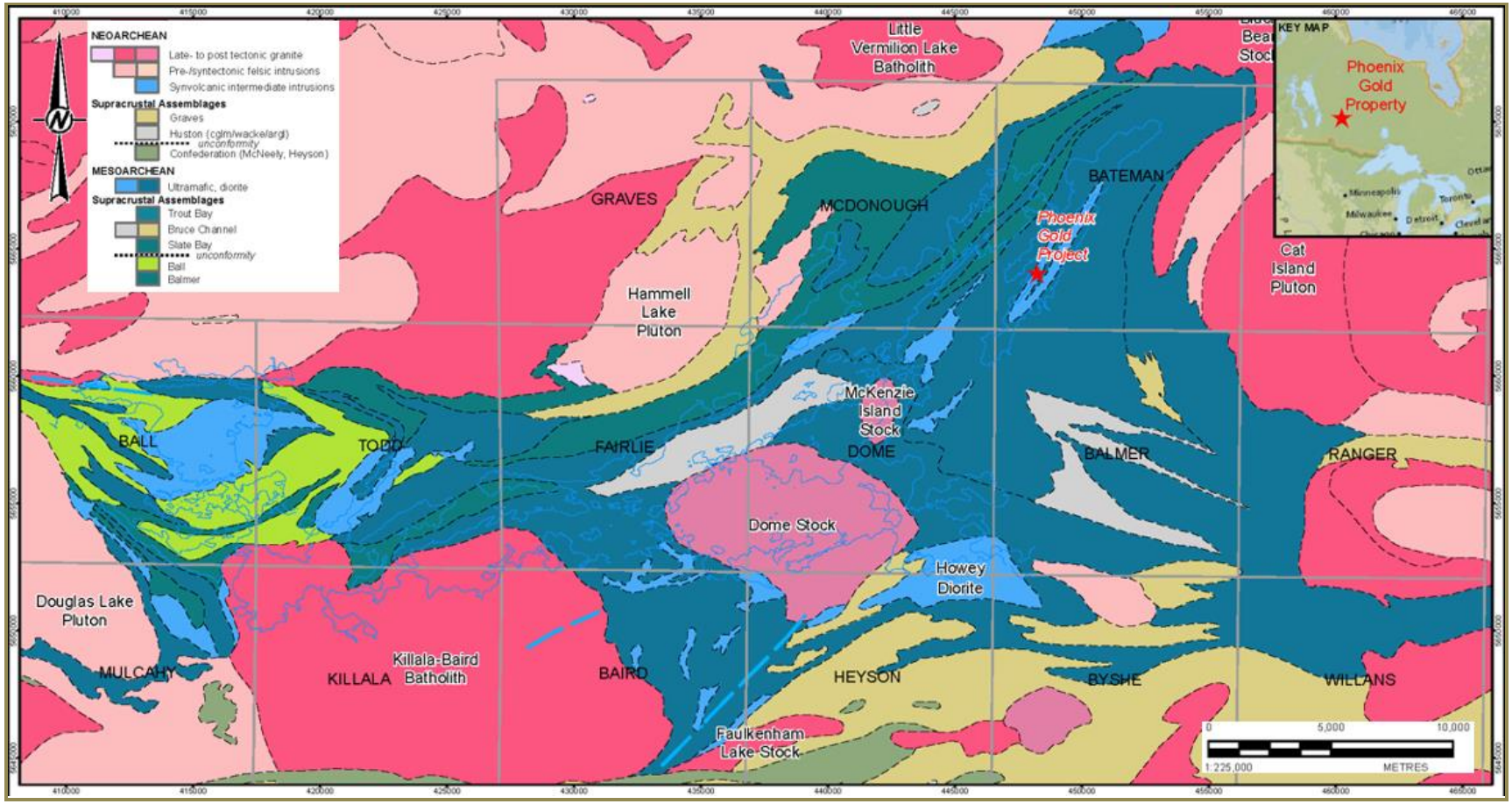
Note: Modified after Géologie Québec (GQ.MINES.GOUV.QC.CA), which is adapted from Card and Poulson (1998).

Figure 7-1: Superior Province Subdivisions (Rubicon, 2019)



Note: Modified from Sanborn-Barrie et al. (2004)

Figure 7-2: Geology of the North Caribou Terrane of the Superior Province (Rubicon, 2019)



Note: Adapted from Ontario Geological Survey (2011)

Figure 7-3: Regional Geology (Rubicon, 2019)

The Bruce Channel Assemblage (ca. 2,894 Ma) is composed of a thin sequence of calc-alkaline dacitic to rhyodacitic pyroclastic rocks overlain by an upward-fining sequence of clastic sedimentary rocks and chert-magnetite iron formation. Trace element profiles of the calc-alkaline volcanic rocks relative to the Balmer Assemblage are interpreted to indicate crustal growth at a juvenile continental margin.

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

Deposition of the Confederation Assemblage followed a pause in volcanic activity of approximately 100 Ma. The Confederation Assemblage represents a time of widespread Neoarchean calc-alkaline volcanism (ca. 2,748 to 2,739 Ma). The McNeely sequence is the oldest unit of the Confederation Assemblage; it formed during shallow marine to subaerial arc volcanism and was deposited upon the existing Mesoarchean continental margin. The McNeely sequence is overlain by and interstratified with the tholeiitic Heyson volcanic sequence that is thought to have formed during a period of intra-arc extension. In the Madsen area, an angular unconformity at the base of the Confederation Assemblage is indicated by opposing facing directions of units belonging to the Confederation and Balmer assemblages, suggesting the Balmer Assemblage was overturned prior to the deposition of the Confederation Assemblage.

The Huston Assemblage (dated between ca. 2,742 and 2,733 Ma) is represented by fine- to coarse-grained clastic sedimentary units including conglomerate, wacke, siltstone, and argillite that unconformably to conformably overlie the McNeely sequence of the Confederation Assemblage. The Huston Assemblage has been compared to the Timiskaming conglomerates commonly associated with gold mineralization in the Timmins camp of the Abitibi Greenstone Belt (Dubé et al., 2003).

The Graves Assemblage (ca. 2,733 Ma) represents a period of calc-alkaline volcanism dominated by andesitic to dacitic pyroclastic tuff. The rocks of this assemblage overlie and are locally transitional with the underlying Huston Assemblage.

Plutonic rocks in the Red Lake Greenstone Belt are temporally and, in some cases, petrologically correlated with the periods of magmatism recorded by the volcanic units belonging to the above-described assemblages. The plutonic units include mafic to ultramafic intrusions associated with the Balmer and Ball Assemblages, gabbroic sills with chemical affinities to the basalts of the Trout Bay Assemblage, small volumes of felsic dykes and diorite intrusions associated with the Confederation Assemblage, and intermediate to felsic plutons, batholiths, and stocks coeval with the Graves Assemblage. Post-volcanism plutonic activity is represented by granitoid rocks such as the McKenzie Island stock, Dome stock, and Abino granodiorite (ca. 2,720 to 2,718 Ma) that host past producing gold mines. The last magmatic event recorded in the belt occurred ca. 2,700 Ma and is represented by a series of potassium–feldspar megacrystic granodiorite batholiths, including the Killala-Baird Batholith, as well as some other granitoid plutons and dykes. Structurally, the Red Lake Greenstone

Belt underwent continental collision (the Kenoran Orogeny) ca. 2,720 to 2,710 Ma, which led to multiple episodes of intense hydrothermal alteration, deformation, metamorphism, and gold mineralization (Dubé et al., 2003). The belt records several episodes of deformation interpreted to be closely linked with intensive hydrothermal activity and gold mineralization. Current regional interpretations of the Red Lake area identify three main deformation events:

- D1: Regional NW–SE shortening, resulting in NE–SW striking folds, thrust faults, thrust related strike-slip faults, quartz veins, and penetrative regional foliation (S1) fabric.
- D2: Regional NE–SW shortening resulting in development of pre- to syn-mineralization oblique strike-slip fault systems and a fold overprint of the earlier D1 deformation. During D2 deformation in the East Bay area, oblique dextral strike slip faults reactivated D1 thrust faults, and associated D1 strike slip faults along a zone of crustal weakness inherited from earlier D1 faulting.
- D3: Regional-scale folding resulting in open folding of D1 and D2 structural features.

7.2 Phoenix Property Geology

Orientation of all property-scale geological and structural features in this report are referenced to Rubicon's local mine grid, which is oriented 045° to true north.

The stratigraphy in the East Bay area (Figure 7-4), where the Phoenix Gold Project is located, comprises submarine tholeiitic basalt, komatiite, and komatiitic basalt, with minor felsic intrusive volcanic rock, iron formation, and fine-grained clastic metasedimentary rocks, all of which constitute the Balmer Assemblage. Extensive mapping, trenching, core drilling, and geophysical surveys have defined a consistent geological sequence that can be correlated along the length of the property for over 4 km. A summary of the stratigraphic units found within the Project area is shown in Table 7-1 and Figure 7-4.

Table 7-1: Summary of Phoenix Gold Project Area Stratigraphy

Sequence	Stratigraphy
West Peninsula Sequence	Pillowed to massive basalts with banded iron formation (BIF), graphitic BIF and chert, banded silty to arenaceous sedimentary rocks and significant pyrite/pyrrhotite
Central Basalt Sequence	Pillowed and massive tholeiitic basalts with flow top breccias occasional BIF and (graphitic) argillite.
Intrusive Komatiite Sequence	Massive, spinifex, and columnar jointed basaltic komatiite bounded by 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 Footwall BIF. At least five horizons of silica/oxide (carbonate) facies BIF within pillowed and amygdaloidal basalt
HW Basalt Sequence	Pillowed to massive, amygdaloidal basalts. Variably carbonate altered, variable foliation
East Bay Serpentinite[1]	Extrusive and intrusive ultramafic rocks. Variable talcose alteration
High-Ti Basalt[2]	Variable biotite alteration, sulphides (pyrite, pyrrhotite). Silica flooding, quartz breccia, and quartz veining throughout. The High-Ti Basalt is the principal host to gold mineralization in the F2 Gold Zone.

Note: [1]Labelled as Ultra Mafic in Figure 7-4. [2]Unit is observed underground does not outcrop at surface.

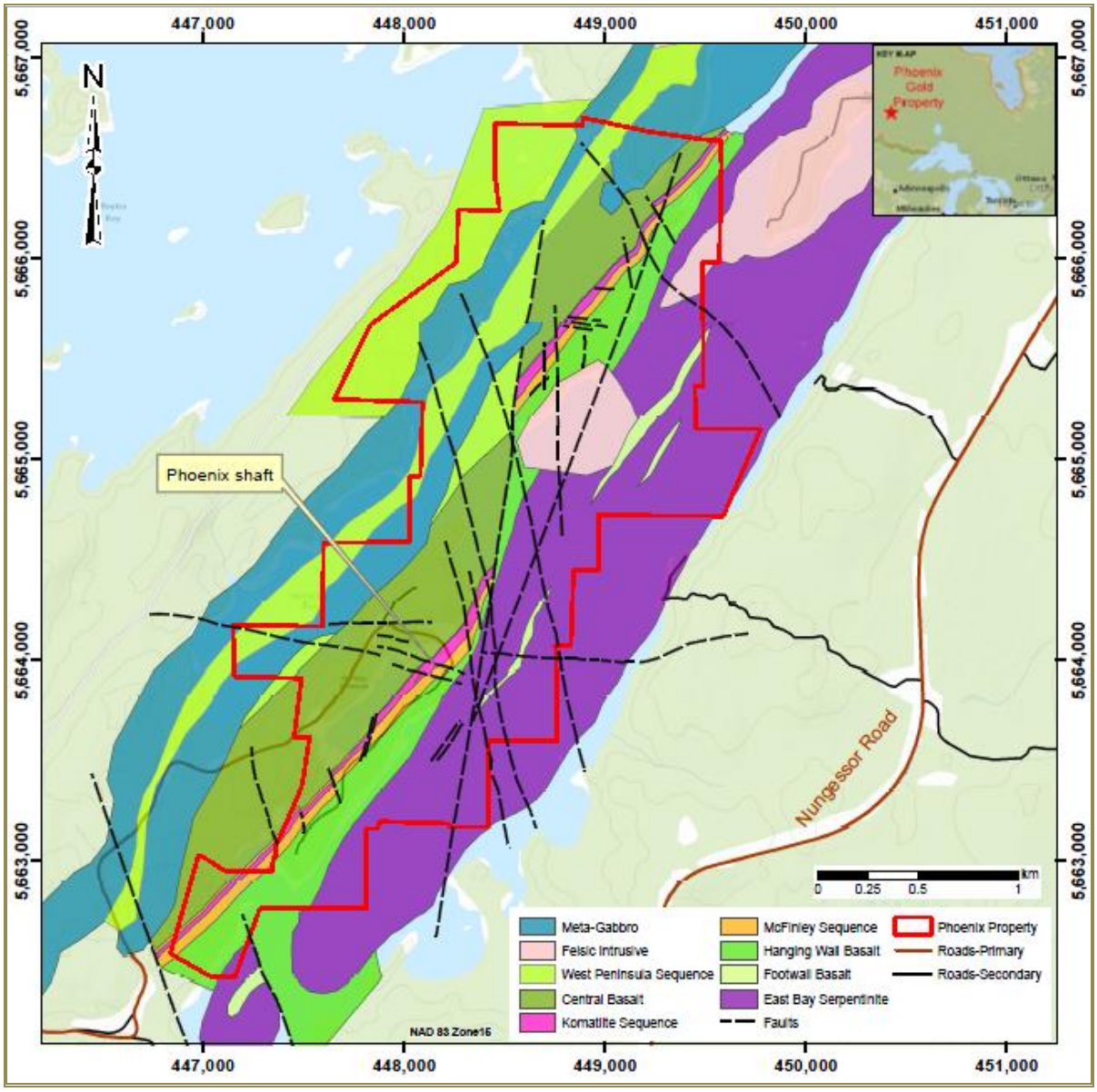


Figure 7-4: Phoenix Gold Project Surficial Geology (Rubicon, 2019)

The Balmer Assemblage basalt flows are tholeiitic and distinguished from other basaltic sequences in the Red Lake Belt by their relatively high TiO_2 contents (commonly >2 wt.%), and as a result, the unit is termed High-Ti Basalt by Rubicon.

The local geology comprises a series of mine grid north–south (N-S) trending, steeply dipping to sub-vertical alternating panels of talc-altered komatiitic ultramafic flows (Ultramafic Flows) and biotite and silica altered basaltic mafic volcanic flows (High-Ti Basalt). Three main panels of High-Ti Basalt are observed, namely the F2 Basalt Zone, West Limb Basalt Zone, and the Hanging Wall Basalt

Zone; in addition to these three main basalt panels, there are other less continuous or less well-defined panels of basalt located in the deposit area. The volcanic units are intruded by a series of quartz–feldspar porphyry felsic dykes and sills (Felsic Intrusive), as well as less abundant intermediate and mafic dykes and sills. The Felsic Intrusive dykes and sills post-date D1 deformation features and are cross-cut by mineralized D2 deformation features.

The EBDZ is located within the western portion of the deposit, where it forms a mine grid N–S orientation, steeply dipping to sub-vertical high-strain zone localized within the Ultramafic Flow unit. Within the Phoenix Gold Project area, the EBDZ forms a distinct boundary between the alternating panels of Ultramafic Flows and High-Ti Basalt units to the east of the structure, and Ultramafic Flows without interlayered High-Ti Basalt to the west of the structure.

The EBDZ may have developed as a D1 thrust fault that was subsequently steepened. Alternatively, the EBDZ may have been initiated as a steeply dipping D1 strike-slip fault. A full reinterpretation of the regional D1 tectonic history is beyond the scope of this Technical Report. The D1 EBDZ fault was later reactivated as a regional dextral shear zone during D2.

The dominant structural fabric present in the Phoenix Gold Project area is a mine grid N–S orientation that is steeply dipping to sub-vertical penetrative tectonic foliation (S1) developed during D1 deformation. The S1 foliation is well developed in the talc-rich ultramafic rocks but is generally absent or not observable in the basalt and Felsic Intrusive units.

D2 features present in the Phoenix Gold Project area are predominantly mineralized quartz–actinolite veins and discontinuous shear zones and brittle faults produced by dextral transpression along the reactivated EBDZ.

D3 regional folding resulted in gentle folding of the Phoenix Gold Project area stratigraphy along a mine grid sub-horizontal N–S-oriented fold axis.

7.3 Phoenix Gold Project Mineralization

Gold mineralization occurs primarily within High-Ti Basalt in the form of mineralized quartz–actinolite veins and in association with disseminated sulphides in the High-Ti Basalt, with lesser mineralization in felsic dykes and sills. Previous studies (SRK, 2013a) have identified an earlier low-grade gold mineralization event, with a later, overprinting, higher-grade gold mineralization event.

The early low-grade gold mineralization event appears to have formed pre- to syn-D1 as the mineralization is overprinted by the S1 foliation. The early phase of mineralization is generally low-grade, with gold grades generally less than 4.0 g/t Au, and occurs as quartz–actinolite–sulphide veins and stringers and as disseminated mineralization associated with quartz–biotite–sulphide alteration in the High-Ti Basalt and Felsic Intrusive units.

The higher-grade second mineralization event has been linked to an array of shear-related veins and minor localized shear zones interpreted to have formed as a result of D2 dextral transpression along the EBDZ. The gold mineralization occurs in association with disseminated sulphide mineralization in the High-Ti Basalt and also in gold-bearing quartz–actinolite veins in the High-Ti Basalt and Felsic Intrusive units. The mineralized veins occur in several orientations, with the east-striking, steeply-dipping vein arrays being associated with higher grade gold mineralization. Mine grid E–W-striking structures are primarily limited to the High-Ti Basalt and Felsic Intrusive; those structures are interpreted as R' shear veins associated with the regional dextral transpression. No regional or through-going deposit-scale E–W structures were identified.

7.4 Deposit Scale Structural Analysis

Golder (2018a) combined statistical and graphical orientation analysis with 3-D geological and structural modelling to evaluate the data and observations from the 2017 structural study for updating the structural interpretation and model for the Project. The 2017 structural study focused on the evaluation of structural impacts on the geometry and distribution of the host units to the mineralization, namely the High-Ti Basalt and the Felsic Intrusive, as well as evaluated controls on the distribution of gold mineralization to identify potential high-grade domains.

The underground mapping, 2017 drilling program, and structural modelling demonstrate that although mine grid E–W-oriented faults and shear zones do occur within the deposit, they are generally more localized and discontinuous in both their lateral and vertical extents than previously interpreted. They do not appear to represent deposit-scale features. The mine grid E–W oriented faults and shear zones are not necessary to explain the geometry and continuity of the N–S oriented High-Ti Basalt and Ultramafic Flow panels and the Felsic Intrusive dykes and sills.

The three main panels of basalt in the deposit, namely the F2 Basalt Zone, the West Limb Basalt Zone, and the Hanging Wall Basalt Zone are all N–S-striking, steeply dipping panels. Although they can be followed along strike and down dip, they are not single continuous panels of basalt, but rather they can be broken out into numerous segments in both the N–S and down-dip direction.

The High-Ti Basalt units have the appearance of a more or less well-developed chocolate-tablet boudinage structure. A N–S-oriented stretching, associated with deformation along the EBDZ during the D1 deformation event, and with regional dextral movement during reactivation of the EBDZ during the D2 deformation event, is interpreted to have resulted in boudinage of the High-Ti Basalt units, with the primary horizontal stretching direction parallel to the N–S orientation of the EBDZ. A component of dextral transpression, possibly relating to emplacement of large plutonic stocks to the northeast and southwest of the area, is interpreted to impart a lesser vertical component of stretching, such that the High-Ti Basalt and Felsic Intrusive units are also boudinaged in the vertical plane.

7.5 Quartz Vein Analysis and Interpretation

Quartz veins are scarce within the Ultramafic Flow units in comparison to the veins observed in the High-Ti Basalt and Felsic Intrusive units. Quartz veins occurring in the Ultramafic Flow units generally occur in isolated areas, are thin (several centimetres wide) and pinch out with lengths less than several metres. The quartz veins in the Ultramafic Flow units generally lack associated gold mineralization.

Quartz veins are common in the High-Ti Basalt, where they often occur as vein arrays comprising multiple parallel and closely spaced veins. The veins are generally present throughout most of the High-Ti Basalt, with concentrated mineralized areas where vein abundance increases significantly.

Quartz veins are present in the Felsic Intrusive units but are not as common as in the High-Ti Basalt and do not generally have the same associated elevated gold grades as observed in the High-Ti Basalt. The Felsic Intrusive unit and the High-Ti Basalt both likely underwent brittle deformation resulting in the development of structural traps controlling the emplacement of quartz–actinolite veins. The quartz–feldspar porphyry did not provide the same chemical trap as the more iron-rich (relative to the Felsic Intrusive units) High-Ti Basalt. This did not allow for significant gold mineralization to develop in the Felsic Intrusive units compared to the High-Ti Basalts.

The quartz veins in the High-Ti Basalt and the Felsic Intrusive dykes and sills are interpreted as shear and extensional veins developed during brittle deformation of the units during D2 dextral transpression. The various orientations of vein arrays are interpreted as the following dextral shear-related vein sets:

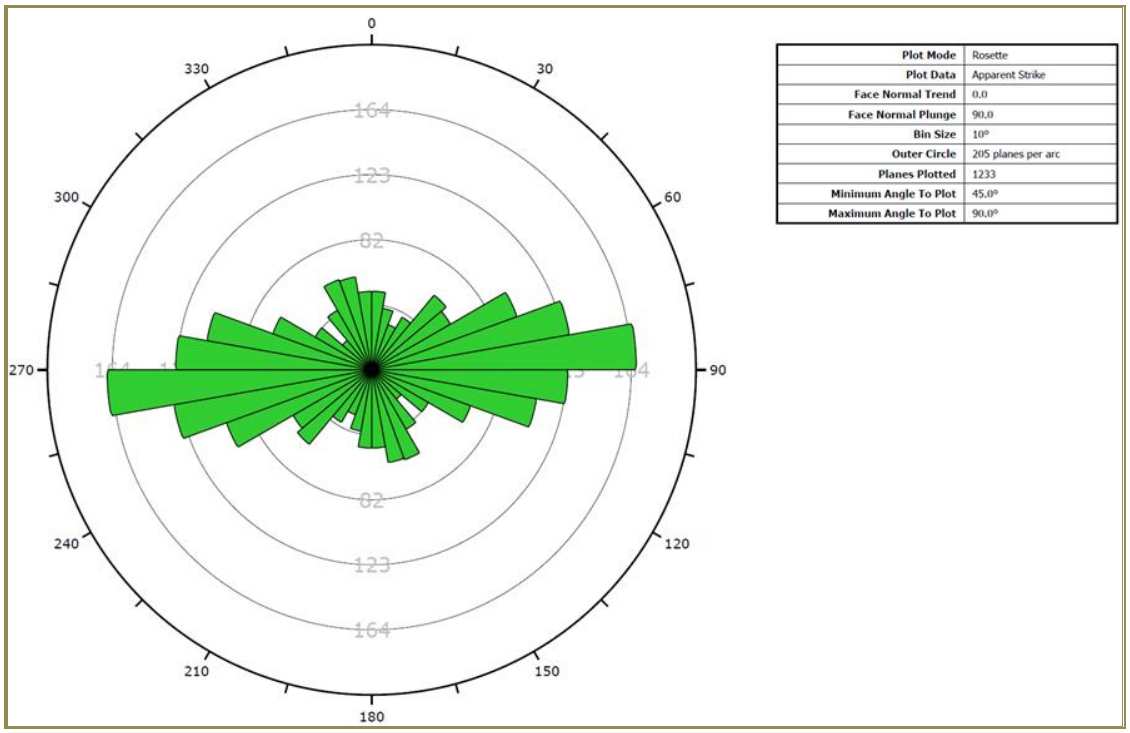
- Riedel Prime Shear Veins (R'): the most common vein orientation, striking E–W, dipping sub-vertical, oriented at a high angle to the orientation of the EBDZ, and showing sinistral shear sense indicators, antithetic to the dextral movement of the EBDZ
- Riedel Shear Veins (R): striking N–S, steeply dipping to sub-vertical, oriented at low angle clockwise to the orientation of the EBDZ, with dextral shear sense indicators synthetic to the dextral movement of the EBDZ
- P Shear Veins: striking NW–SE, steeply dipping to sub-vertical, oriented at low angle counter-clockwise to the orientation of the EBDZ, with dextral shear sense indicators synthetic to the dextral movement of the EBDZ
- Low-angle Veins: shallow-dipping to sub-horizontal extensional veins oriented approximately orthogonal to the shear veins, with vertical extensional fabrics.

The vein-set relative abundances and orientations are shown in Figure 7-5, with the E–W striking R' shear veins occurring in significantly greater numbers than the other vein types.

The R', R, and P shear veins all host gold mineralization, with the highest gold grade, generally occurring within the E-W oriented R' veins.

The higher-grade gold mineralization in the F2 Basalt Zone is observed to be spatially associated with Quartz–Breccia Zones that share the same geometry as the R′ Shear Veins. The Quartz–Breccia Zones are interpreted to have developed as multiple opening and sealing events of the E–W-striking sub-vertical R′ shear veins. A possible explanation for the development of the R′ Shear Vein–related Breccia Zones is that their sinistral sense of shear is opposed to the dextral bulk sense of shear. As a result, the R′ shear veins will not accommodate significant displacement, but they may develop into zones of intense deformation, where repeated fracturing and comminution of the vein and entrained and surrounding wall rock material results in the creation of high porosity and permeability zones for mineralizing fluids.

In areas where the Quartz–Breccia is thick and is associated with a surrounding envelope of increased abundance of mineralized quartz–actinolite vein arrays, they impart a clear E–W component to the high-grade mineralization.



Note: Data used for rose plots was limited to data with orientation confidence >5, which translates to core orientation lock angles of less than 10 degrees (Golder, 2018a)

Figure 7-5: Rose Plot of Quartz–Actinolite Veins

The Quartz–Breccia Zones have minor sinistral movement indicated by limited shear sense indicators that include shear fabrics, minor offsets, and alignment/imbrication of wall rock fragments entrained in the Quartz–Breccia Zones. A Quartz–Breccia Zone exposed in development on the 305 m Level of the mine exhibits what appears to be well-developed sinistral releasing bend geometry.

The Quartz–Breccia Zones do not appear to be thoroughgoing (cutting across all units) E-W shear zones or shear veins, but rather they are discontinuous, occurring primarily within the thickest parts of the F2 Basalt Zone. The 305 m Level Quartz–Breccia Zone cuts across the multiple panels of basalt and a thin sliver of ultramafic sandwiched between them. This is attributed to ductile strain partitioning favoured in the more plastic Ultramafic Flow units.

Quartz–Breccia Zones have been identified in the West Limb Basalt and the Hanging Wall Basalt zones but the best developed zones identified to date have been found in the F2 Basalt Zone. Evaluation for Quartz–Breccia Zones in the other panels should be a high priority in future exploration and infill drilling.

The final deformation event observed in the deposit resulted in the entire sequence of Ultramafic Flow, High-Ti Basalt, and Felsic Intrusive units having been gently folded into a broad, open fold with a N–S-oriented, sub-horizontal fold axis during the D3 deformation event. The broad open folding of the stratigraphy is apparent when viewing the deposit on a W–E (north-facing) section. A subtle change in geometry is also observed in the orientation of the quartz–actinolite veins as they undergo a slight change in orientation and their dips shallow slightly with depth below the 610 m Level.

7.6 Updated Structural Interpretation for the Phoenix Gold Project

Based on an analysis of the data and observations obtained during the 2017 structural oriented core drilling and mapping programs, Golder's conceptual model (Golder, 2018a) of the revised structural interpretation is presented in Figure 7-6. The updated structural interpretation and model include the following key elements:

- The EBDZ has been remodelled to show it as a broader zone of high strain in the Ultramafic Flow unit rather than as a discrete feature that is then offset by E–W brittle faulting per the previous model.
- Strain partitioning during D1 and D2 deformation events resulted in ductile deformation of the talc-rich Ultramafic Flow units and brittle-ductile deformation of the more resistant High-Ti Basalt and Felsic Intrusive units.
- Ductile behaviour of the Ultramafic Flow unit resulted in the generation of the pervasive N–S-oriented, steeply dipping to sub-vertical S1 penetrative foliation during D1 deformation.
- Brittle-ductile behavior of the High-Ti Basalt units resulted in the boudinage of these units with the primary stretching direction paralleling the N-S orientation, with a lesser vertical component of stretching such that the boudin necks that bound the High-Ti Basalt panels are arranged in both N–S shallowly dipping and subvertical orientations.
- The High-Ti Basalt is modelled as a series of mine grid N–S-oriented panels that have been boudinaged during D1 and D2 deformation events so that they form mine grid N–S elongated lenses that pinch out at the north and south ends. In some instances, there are

gaps of tens of metres between boudinaged basalt panels. This geometry is shown in both the N–S planar view and the vertical view (Figure 7-6).

- Ultramafic Flows and High-Ti Basalt units were intruded by dykes and sills of the Felsic Intrusive unit pre- to syn-mineralization.
- Arrays of quartz–actinolite veins with associated gold mineralization were developed in the more competent High-Ti Basalt and to a lesser degree in the Felsic Intrusive. The R', R, and P shear veins all host gold mineralization, with the highest gold grades, generally occurring within the E–W-oriented R' veins.
- The best gold grades occur in the thickest portions of the High-Ti Basalt, where the unit presented both favourable structural traps for developing gold-bearing veins and chemical traps where disseminated sulphides and associated gold mineralization are developed. Golder recommended these areas should be the focus/targets of future exploration efforts.
- The entire sequence of Ultramafic Flow, High-Ti Basalt and Felsic Intrusive units were then folded into a broad gentle fold with a mine grid N–S-oriented, sub-horizontal fold axis during D3 deformation event.
- Some deposit scale and macro scale evidence for pre-D3 folding was observed in the Ultramafic Flow units; however, at present, Golder (2018a) interpreted these features to be a result of foliation orientation variability due to dragging associated with the regional D2 dextral deformation and to warping of the foliation in boudin neck regions rather than a result of deposit-scale steeply plunging isoclinal folding.

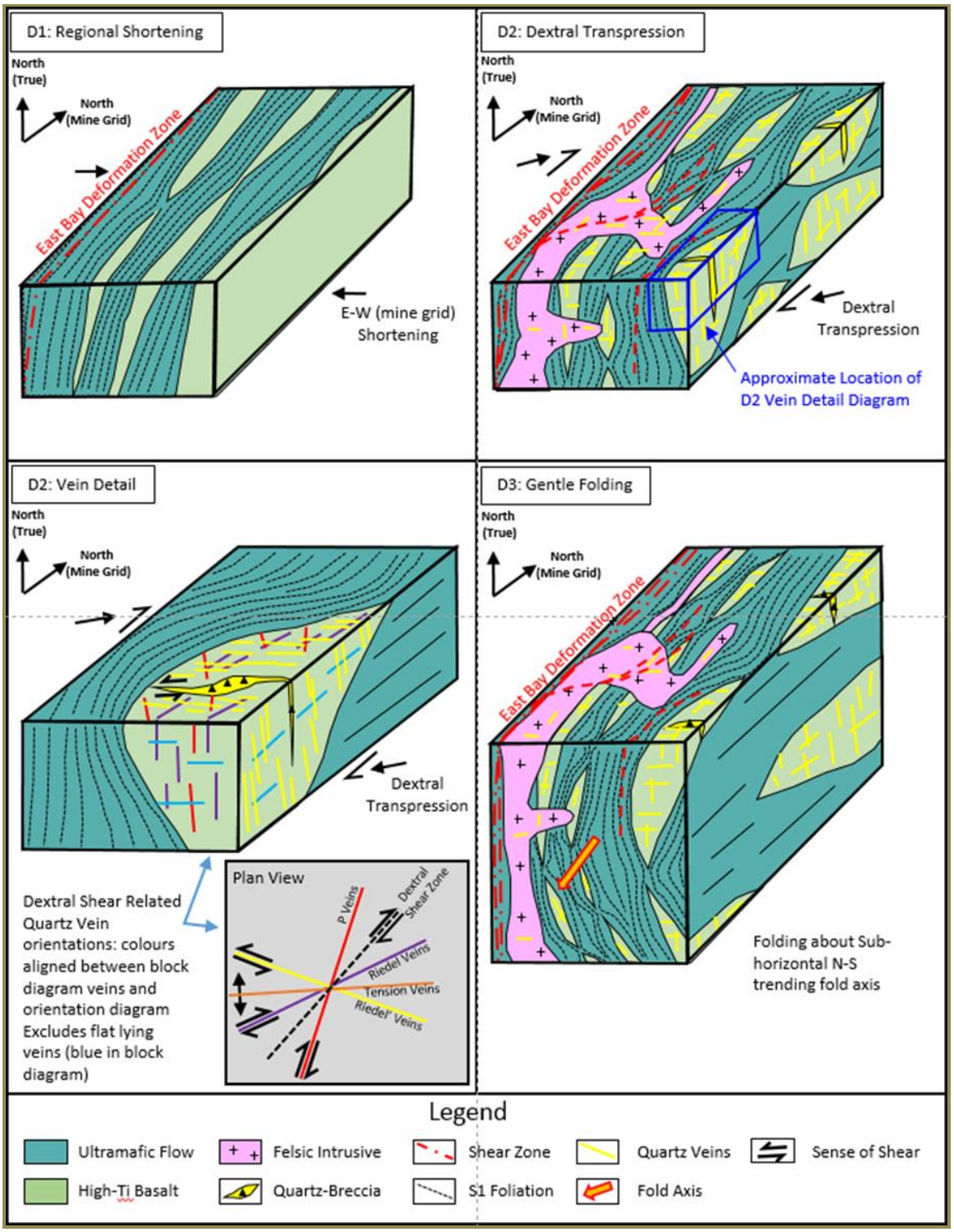


Figure 7-6: Updated Conceptual Structural Model for the Phoenix Gold Project Area (Golder, 2018a)

8 DEPOSIT TYPES

The style of veining, the lithological setting, and the structural relationship with shear zones at the F2 Gold Zone are compatible with orogenic-style gold mineralization (also referred to as mesothermal, or Archean greenstone-hosted quartz–carbonate vein gold mineralization or Archean lode gold). This style of gold deposit is typically associated with regional folding and arrays of major shear zones and is formed by circulation of gold-bearing hydrothermal fluids in structurally enhanced permeable zones. The deposits are characterized by strong lithological and structural controls and are hosted in deformed and metamorphosed volcanic, sedimentary, and granitoid rocks occurring across a range of crustal depths (Groves et al., 1998).

Orogenic gold deposits are widely distributed in the Neoarchean greenstone belts of the Superior, Churchill, and Slave provinces, and also occur in younger terranes such as the Canadian Cordillera and the Appalachian terranes. In Canada, the most important concentration of orogenic gold deposits occurs in the greenstone belts of the south-central Superior Province.

In the Red Lake mining district, most of the gold production is derived from orogenic-style high-grade quartz–carbonate veins that are associated with deformation of the Balmer Assemblage mafic and ultramafic volcanic rocks (Sanborn-Barrie et al., 2004). At the Campbell–Red Lake Mines, located to the south of the Phoenix Property, the main source of gold is within quartz–carbonate veins associated with the Campbell and Dickenson fault zones that are locally controlled by F2 folding (Dubé et al., 2001). A spatial relationship exists between the ultramafic rocks and gold mineralization, with the majority of gold mineralization at the Cochenour-Willans and Campbell–Red Lake gold mines occurring within a few hundred metres of ultramafic bodies. Dubé et al. (2001) suggested that a competency contrast between the mafic (basalt) and ultramafic (komatiitic basalt) units was important in the formation of extensional carbonate veins in fold hinge zones during deformation. The carbonate veins were then partially replaced as the result of interactions with gold-rich siliceous fluids.

The F2 Gold Zone shares attributes of other orogenic gold deposits of the Red Lake mining district. These include the association of auriferous quartz–carbonate veins with regional scale D2 deformation zones (D2 shear zones and related brittle-ductile structural features) and the favourable lithological setting of Balmer Assemblage mafic and ultramafic volcanic rocks.

9 EXPLORATION

9.1 Rubicon (2002–2018)

Since acquiring the Phoenix Gold Project in 2002, Rubicon has conducted an extensive exploration program that has included geological mapping, re-logging of selected historical drill holes, digital compilation of available historical data, ground and airborne magnetic surveys, mechanical trenching, channel sampling, a bathymetric survey, an induced polarization Titan 24 survey, petrographic studies, a topographic survey, data modelling and processing, and numerous drilling programs.

A summary of Rubicon's exploration activities undertaken at the Phoenix Gold Project between 2002 and 2018 is shown in Table 9-1. Figure 9-1 shows the gold zones and target areas for the Phoenix Gold Project and surroundings.

Table 9-1: Summary of Exploration Activities by Rubicon from 2002 to 2018

Year	Description of Work
2002	• Geological mapping • Cataloguing, numbering, and re-boxing of historical core cross-piled on property (over 60,000 m) • Digital compilation of historical data • High-resolution airborne magnetic survey • 22,000 m^2 of mechanical trenching and power washing (in 2002 and 2004) • Channel sampling (876 samples between 2002 and 2004) • Overwater bathymetric survey of Red Lake within property boundary • 1,900 m of drilling on the Phoenix Peninsula
2003	• Re-logging of selected historical drill holes (approximately 23,000 m from 161 drill holes) • Digital compilation of historical data • Phase 1 drilling program with 9,600 m of winter drilling, including ice drilling • Phase 2 drilling program consisting of 3,000 m drilled on the Phoenix Peninsula
2004	• Continued mechanical trenching, power washing, and channel sampling • Winter drilling program with 13,300 m drilled
2005	• 11,800 m of surface drilling
2006	• 1,614 m of surface drilling
2007	• 13,444 m of surface drilling
2008	• First phase of Titan 24 DCIP and MT survey • 43,800 m of surface drilling

Year	Description of Work
2009	• Second and final phase of airborne Titan 24 survey completed • Preliminary petrographic study • Surface (44,675 m) and underground (25,512 m) core drilling
2010	• Topographic survey utilizing airborne LiDAR technology (light detection and ranging) • Surface (37,823 m) and underground (82,068 m) core drilling
2011	• Surface (5,462 m) and underground (74,337 m) core drilling
2012	• Surface (40,900 m) and underground (17,627 m) core drilling (to cut-off date of Nov. 1, 2012)
2013	• Underground core drilling (876 m) to support shaft development
2014	• Underground core drilling (40,574 m), infill and step-out drilling in central portion of deposit • Surface core drilling (6,064 m) used to investigate the crown pillar
2015	• Underground core drilling (47,061 m), infill used as production support for trial stoping • Exploration surface core drilling (9,553 m) targeting the Carbonate (CARZ) Zone
2017	• Underground core drilling (28,995 m) from 244 m, 305 m, 610 m, and 685 m Levels, including: 3,500 m to evaluate E–W structures and mineralization features for the F2 Gold Zone structural model, 24,139 m of infill and step-out drilling to update Mineral Resource and structural models for the F2 deposit, and 1,356 m to test down-dip continuity of mineralization on the McFinley deposit.
2018	• Underground core drilling (20,159 m to October 15, 2018 drilling cut-off date), 17,443 m of infill drilling to upgrade inferred Mineral Resources, 2,716 m definition drilling and drilling on trial stopes as part of the 2018 Test Mining program.

9.1.1 Core Re-logging Program (2002)

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

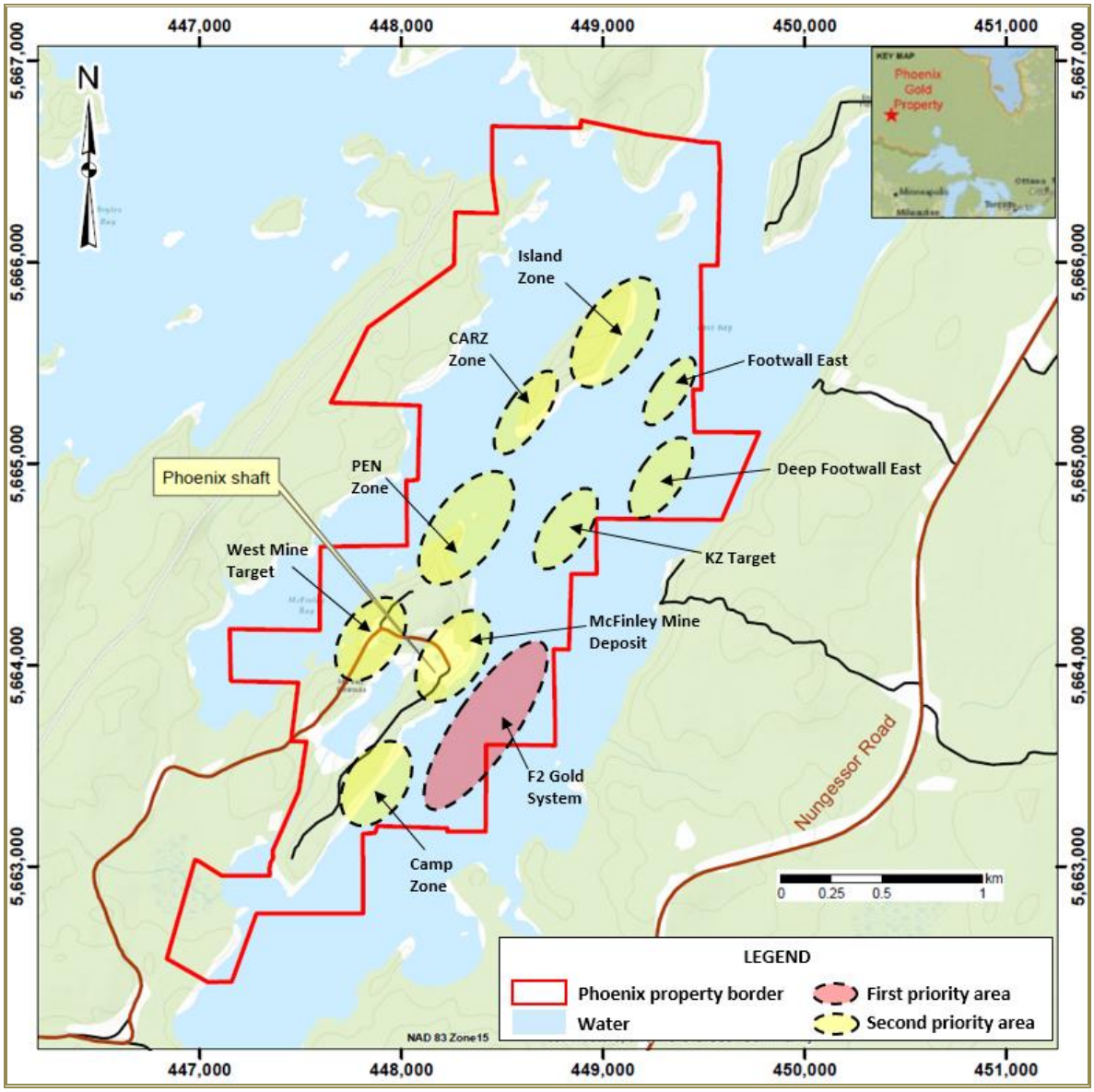


Figure 9-1: Key Target Areas (Rubicon, 2019)

9.1.2 Geophysical Surveys (2002, 2008)

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

The 2008 Titan 24 DCIP survey by Quantec Geoscience was completed after the 2008 discovery of the F2 gold zone (Figure 9-1). The Titan 24 survey successfully detected several known near-surface gold zones; the survey is also interpreted to have detected alteration that is spatially associated with the F2 Gold Zone. The defined chargeability anomaly is over 1,500 m long and appears to correlate with a zone of strongly altered host rocks and sulphide minerals that are associated with gold mineralization that extends from the southern limit of the F2 Gold Zone to the Pen Zone. The F2 Titan chargeability anomaly is one of many similar anomalies defined by the same survey along 3 km of prospective stratigraphy extending to the northeast on the property. The chargeability anomalies range from vertical depths of 200 m to over 800 m and constitute high-priority regional targets.

9.1.3 Petrographic Study (2009)

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

9.1.4 Exploration (2010–2016)

The exploration programs between 2010 and 2015 were focused on expanding the F2 Gold Zone. A LiDAR survey over the property in 2010 was used to create a high-resolution topographic map. A total of 362,345 m of drilling occurred from 2010 to 2015. Of this, 262,543 m was drilled underground to enlarge and upgrade the F2 Mineral Resource estimate. The remaining 99,802 m was drilled from the surface. This drilling targeted the F2 Mineral Resource area and other close exploration targets. A small portion of the surface drilling, 6,064 m, was for geotechnical purposes investigating the crown pillar.

There was a hiatus of drilling and exploration during 2016.

9.1.5 Diamond Drilling, Structural Geology, and Core Re-logging Program (2017–2018)

Exploration during 2018 was focused on drilling underground targets. The majority of targets were designed to upgrade the Mineral Resource for the F2 deposit from inferred to indicated or better classification. Rubicon drilled a total of 20,159 m prior to the report cut-off date. An additional 2,716 m of short hole drilling was completed for the 2018 test mining and processing program.

The 2018 Mineral Resource estimate by Golder included an update to the deposit scale structural model for the Phoenix Gold Project (Golder, 2018a). As part of this study, in 2017 Rubicon geologists re-logged 46 historical drill holes in the F2 Gold Zone, totalling 10,899 m. The primary focus of the re-logging exercise was to identify and evaluate drill holes for structural geological features that may have been overlooked or misinterpreted in earlier core-logging programs. Rubicon geologists selected

targeted drill holes with Golder' support. Formal structural point-data measurements could not be obtained because the historical drill holes were not oriented core; however, observations and measurements relative to the core axis were captured to provide a more complete picture of the structural association with mineralization. Existing assay results were cross-referenced against lithological, alteration, and structural features when revisiting the core, and proved to be informative to planning subsequent drilling programs.

10 DRILLING

10.1 Historical Drilling

The history of exploration from 1922 to 2002 is discussed in Section 6. Drilling conducted by previous owners is summarized in Table 6-1. The historical core drill holes are mainly located outside the main Mineral Resource area. However, some core drill holes targeted the Hanging Wall Basalt Zone (Part of F2 Gold Zone) between 1984 and 1987 and have been used for geology and Mineral Resource modelling.

10.2 Rubicon Drilling

Between 2002 and December 31, 2018, Rubicon completed 602,901 m of core drilling (265,224 m of surface drilling and 337,677 m of underground drilling) on the Phoenix Gold Project. Included in the total is 25,257 m of directional drilling, wedged from pilot holes to conserve drilling on deep targets, as well as 6,530 m of geotechnical drilling to inform mine development plans or provide mine services. This drilling is tabulated in Table 10-1, by location and year completed. Of the total drilling, 533,859 m were drilled to target the F2 Gold Zone.

Since the 2013 Mineral Resource Statement (SRK, 2013b), infill and step-out drilling focused on the Mineral Resource areas, testing the northern and southern extensions of the gold mineralization to assist with preparing trial stoping development in the core of the F2 Gold Zone, and to investigate the crown pillar. Between November 1, 2012, and November 1, 2015, Rubicon drilled 429 drill holes (94,575 m) (SRK, 2016).

There was no diamond drilling in 2016. With the 2017 restart of the Phoenix Gold Project, Rubicon undertook an ambitious underground exploration drilling campaign with 28,995 m of NQ oriented core drilled primarily on 244 m Level, 305 m Level, 610 m Level, and 685 m Level. Of the total metres drilled in 2017, approximately 3,500 m was structural core drilling, and this was to provide information to update the structural interpretation of the mineralized zones. The remainder of the metres was used to provide additional information to update the Mineral Resource which was completed in early 2018.

In 2018, Rubicon continued its underground drilling program, comprising a total of 20,159 m of NQ oriented core from the 244 m Level, 305 m Level, 610 m Level, and 685 m Level (Figure 10-1), to October 15, 2018 (drilling cut-off date for this report). This infill drilling is being used to update the current Mineral Resource estimate reported herein.

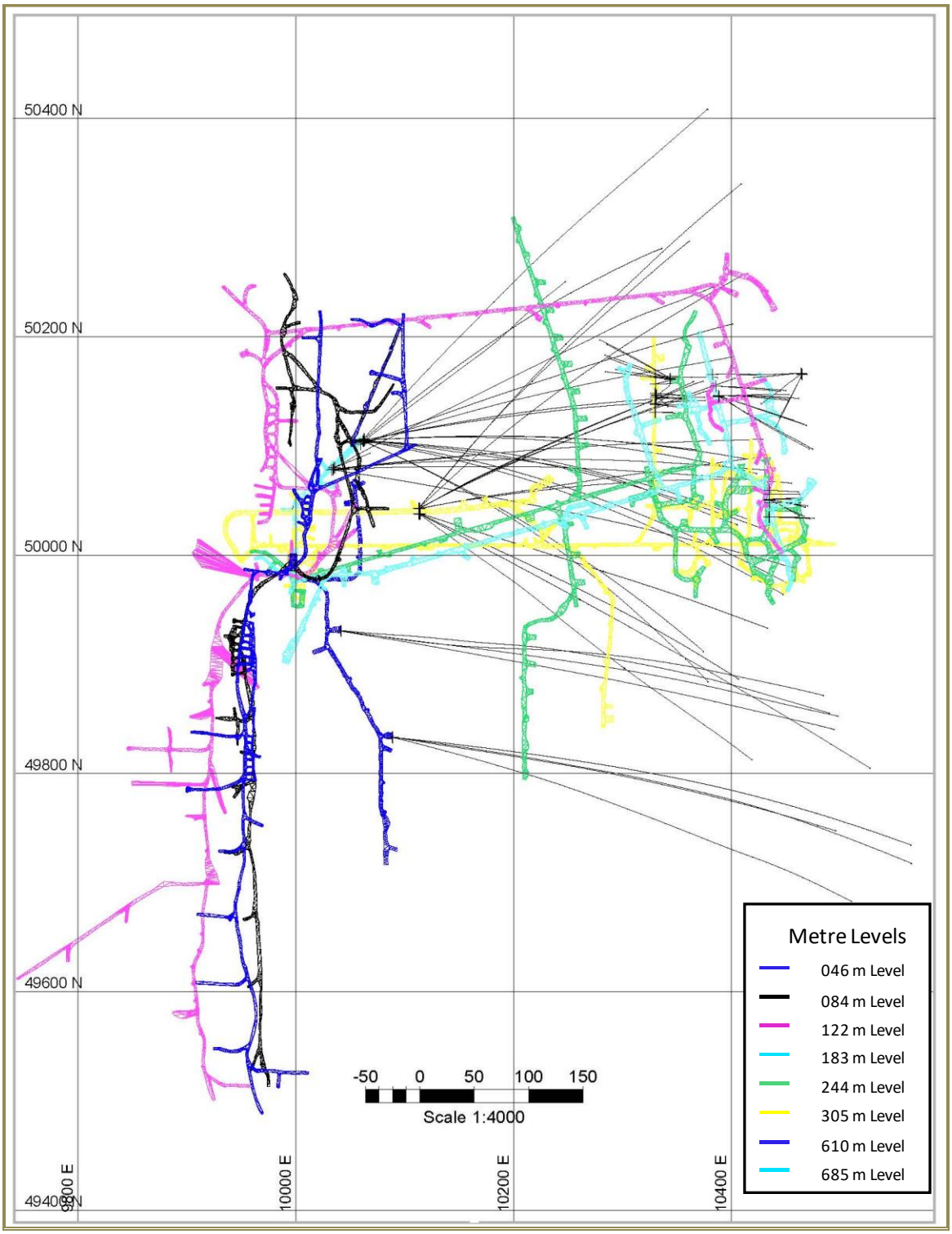


Figure 10-1: 2018 Diamond Drilling, Plan View (Rubicon, 2019)

Table 10-1 summarizes the surface and underground drilling on the Phoenix Gold Project.

Table 10-1: Summary of All Drilling on the Phoenix Gold Project to December 31, 2018

Year	Surface Holes		Underground Holes		Geotechnical Holes[1]		Total Holes	
	Count	Metres	Count	Metres	Count	Metres	Count	Metres
2002 – 2005	188	42,189	-	-	-	-	188	42,189
2006	11	1,614	-	-	-	-	11	1,614
2007	23	13,004	-	-	-	-	23	13,004
2008	65	46,199	-	-	1	593	65	46,199
2009	68	43,500	42	25,511	-	-	110	69,011
2010	73	56,334	209	83,748	-	-	282	140,082
2011	7	6,746	333	73,752	-	-	340	80,498
2012	94	39,688	42	18,314	7	1,129	136	58,002
2013	-	-	6	916	6	916	6	916
2014	39	5,966	214	42,329	28	2,410	253	48,295
2015	23	9,984	285	40,809	2	431	308	50,793
2016	-	-	-	-	-	-	-	-
2017	-	-	83	28,995	1	83	83	28,995
2018	-	-	108	23,303	-	-	108	23,303
Total	**591**	**265,224**	**1,322**	**337,677**			**1,913**	**602,901**

Note: [1]Count and metres for geotechnical drill holes are included in the statistics under the applicable surface or underground category. Meterage reported for wedged drill holes does not include the overlapping portion from the parent hole.

The majority of core drilling by Rubicon has targeted areas outside of the historical McFinley Red Lake Mines area that was historically perceived to have exploration potential. Key target areas on the Phoenix Gold Project are presented in Figure 9-1**.**

The distribution of the surface drilling targeting the F2 Gold Zone is shown in Figure 10-2. Surface drilling was completed generally along east–west sections. However, drill hole azimuth and plunge varied widely because much of the drilling was completed on the lake using a barge or on winter drill platforms. Surface drilling completed to November 1, 2012, improved the definition of the gold mineralization at a drill hole spacing of approximately 50 m or better, locally. Underground drilling targeted the gold mineralization from the 122 m, 183 m, 244 m, and 305 m Levels along east-west sections (normal to interpreted trace of the gold mineralization). Given the limited underground drilling stations available, fan drilling was necessary to target north, south, and depth extensions of the interpreted gold mineralization. The additional underground drilling reduced the spacing between drill holes in the core of the F2 Gold Zone to approximately 10 m or less (Figure 10-3).

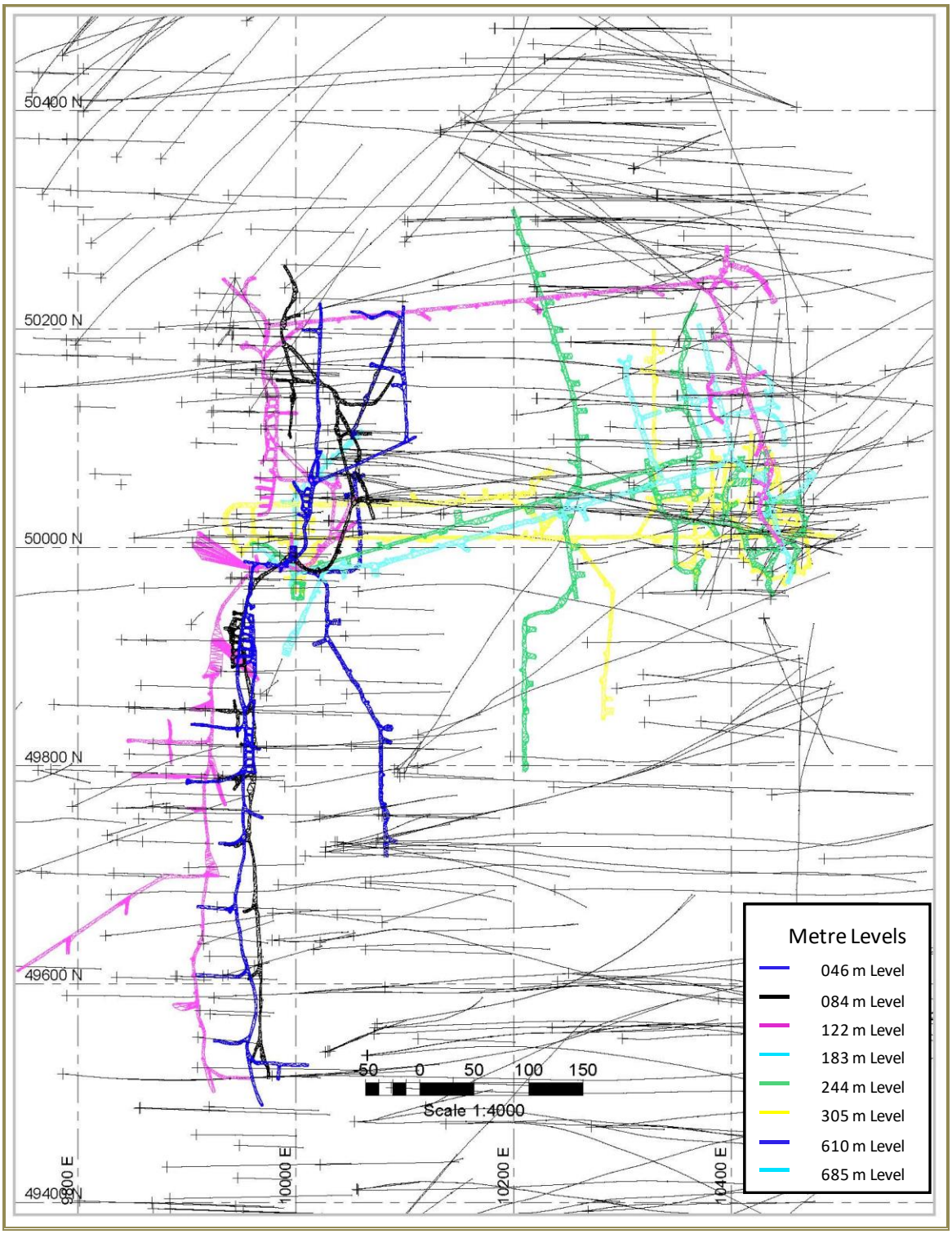


Figure 10-2: Diamond Drilling – Holes Drilled from Surface Locations, Plan View (Rubicon, 2019)

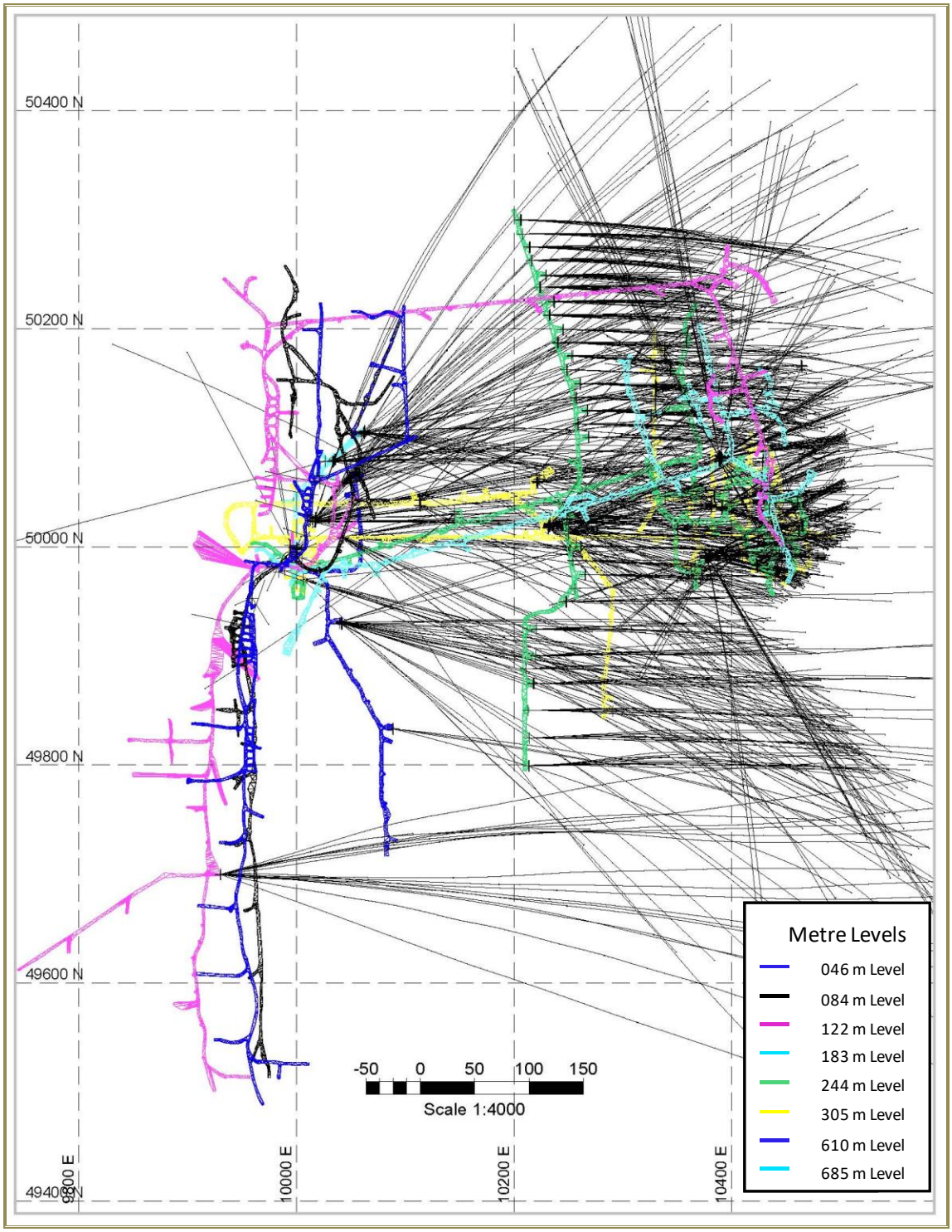


Figure 10-3: Diamond Drilling – Holes Drilled from Underground, Plan View (Rubicon, 2019)

In 2011, 340 core drill holes were drilled (80,498 m), including 6,746 m from surface and 73,752 m from underground. Underground core drilling was conducted on the 305 m Level, from seven separate drill stations, 305-02 through 305-08. Most of the drilling was focused on the F2 Gold Zone with some drill holes testing the extension of the zone along strike (Figure 10-3).

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

In 2012, a total of 136 drill holes (58,002 m) were drilled up to November 1. Underground core drilling was conducted from the 305 m Level, 244 m Level, and 122 m Level, from four separate drill stations (305-02, 305-03, 244-09, and 122-03). Surface drilling was carried out on the ice during the winter months, as well as from land. The drilling was focused on the up-plunge extension of the F2 core zone as well as a series of deep targets. Although the main focus of the 2012 drilling campaign was infill, it also expanded the known strike length of the system by 71 m and the depth by 105 m.

In 2013, six underground geotechnical core drill holes were completed (916 m) to test the lower area of the shaft.

The 2014 to 2015 drilling program on the F2 Gold Zone focused on testing the gold mineralization along strike, north, and south of the core area of drilling, and to assist with planning the test mining areas (Section 13, Figure 13-6). An exploration drift was developed on the 244 m Level parallel to the main zone of gold mineralization. The program was completed with 25 m-spaced pierce points both vertically and horizontally throughout. The program was designed to test between 5,248 m elevation to 4,943 m elevation (122 m Level to 427 m Level), targeting the High-Ti Basalt units. Phase two of the program was designed to infill, where needed, to 12.5 m spacing. Drilling along the northern portion of the deposit identified several higher-grade targets. Drilling in the far southern portion of the F2 Gold Zone confirmed the extension of the High-Ti Basalt with gold mineralization showing that the gold system is open to the south.

In 2015, Rubicon also drilled 21 surface core drill holes (9,553 m) targeting historical high-grade drilling results on the Carbonate Zone (CARZ), North Peninsula Zone (PEN) and Far East Zones (Deep Footwall East) (refer to Figure 9-1 for location). An additional 2 drill holes were completed to provide mine services and mine design information. In 2016, no diamond drilling was conducted.

For the 2017 drilling program, 27,465 m of underground NQ-sized core was drilled on the 244 m Level, 305 m Level, 610 m Level, and 685 m Level, as shown in Figure 10-1. The exploration focused on the down-dip/down-plunge extensions of the known F2 Basalt Zone, West Limb Basalt Zone, and Hanging Wall Basalt Zone units. Of the drilling completed in 2017, approximately 3,500 m of the drilling

program targeted E—W oriented structures on the 244 m Level and 305 m Level by drilling generally N–S oriented holes, and another 1,530 m were drilled on the McFinley zone, to capture structural geology information.

All of the core in 2017 was drilled as "oriented core" using the Boart Longyear (Boart) TruCore tool, to obtain true alpha and beta angles on structures and veining. In addition, approximately 11,000 m of previously logged core was re-logged to verify previous lithological, mineralization, and structural interpretations.

The 2018 drilling program continued many of the efforts begun in 2017. A total of 20,159 m was drilled underground in 101 drill holes with NQ-sized core to the report cut-off date. Drilling was from the 183 m Level, 244 m Level, 305 m Level, 610 m Level, and 685 m Level. The targets for the 2018 underground drilling were the F2 Basalt Zone, West Limb Basalt Zone, and Hanging Wall Basalt Zone units to upgrade the Mineral Resource classification.

All of the core in 2018 was drilled as "oriented core" using the Boart TruCore tool, to obtain true alpha and beta angles on structures and veining.

10.2.1 Drilling Procedures

All proposed land and ice drill hole collars were surveyed with a handheld global positioning system (GPS) instrument with an accuracy of ± 3 m. Two foresight pickets were also surveyed, and drills were set up under the direct supervision of a Rubicon geologist or geological technician. Collars for barge drill holes were also surveyed with a handheld GPS instrument and then marked with a buoy; the same foresight procedure was carried out. Changes in the actual drill hole 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 were surveyed with a differential GPS unit (sub-metre accuracy) and recorded in the database. All surveys currently use the mine grid.

The majority of the core drilling performed prior to 2013 was carried out by Hy-Tech Drilling of Smithers, British Columbia, using Tech-4000 diamond core drills both from surface (on land, ice, or barge) having a depth capacity of 2,500 m, and from underground, having a depth capacity of 1,500 m. Layne Christensen Canada Limited of Sudbury, Ontario, was also contracted to complete deep drill holes using their skid-mounted CS 4002, which has a depth capacity of 2,500 m. Orbit Garant Drilling of Val-d'Or, Quebec, was contracted to complete underground drilling using either a B-20 or Orbit 1500, which has a depth capacity of 1,500 m. Each drilling program was supervised by a Rubicon geologist. In general, NQ (50.8 mm diameter) or NQ (47.6 mm diameter) core was drilled.

From 2013 to 2015, Boart was the drilling contractor. Boart used LM 75 electric drill rigs that can drill a 1,000 m hole at various core sizes. Boart also had several air-powered drills, used for close-proximity definition drill holes. All drilling was supervised by a Rubicon geologist. Drilling was completed with NQ (47.6 mm core diameter), BQTK (40.7 mm core diameter), or AQTK (35.5 mm core diameter) size core.

For the 2017 and 2018 drill program and continuing into 2019, Rubicon contracted with Boart for underground exploration drilling, using two LM90 electric drills to core NQ (50.6 mm core diameter) core. The majority of the core was oriented using Boart's True-Core tool to provide true alpha and beta readings. Rubicon geologists supervised and logged all drilling, core was logged on site and samples were sent for assay at SGS Labs in Red Lake.

Casings for drill holes collared on land were left in place, plugged, cemented, and covered with aluminum caps, with the drill hole number etched or stamped into the cap. Prior to 2012, drill holes that were drilled from the ice or barge were plugged with a Van Ruth plug at 30 m down the drill hole from the base of the casing and then cemented to the top of the drill hole. All casing was removed from these drill holes. Since January 2012, all drill holes drilled from the ice or barges were cemented from the bottom of the hole to the base of the casing. All drill holes drilled from underground were purposely left un-grouted if the drill hole produced water at a rate of less than 5 L/min. If the drill hole produced water at a rate greater than 5 L/min, the hole was pressure grouted from the bottom to top and sealed with a Van Ruth grout plug.

10.2.2 Collar and Down-Hole Survey

For 2017 to 2019 drilling programs, Rubicon used the Boart Devi-Shot down-hole survey instrument measuring azimuth, inclination, magnetic field strength, and temperature at 30 m intervals. All collars were surveyed by Rubicon surveyors, and a select set of holes were Gyro surveyed by Reflex Instruments contractors until February 2018 to verify the down-hole survey results. In October 2018, SurveyTECH of Timmins, Ontario was contracted to train Boart personnel in proper procedures to conduct gyro surveys for all remaining holes drilled in 2018. Subsequently, Boart has been conducting a gyro survey immediately upon completion of each diamond-drill hole.

Rubicon discovered an error with underground core drill hole collar locations. In April 2013 and in January 2015, Total Precision Survey (TPS), using a gyro and plumb bob, corrected the vertical reference line (survey control points at the shaft) resulting in both a translation and rotational shift to the underground excavations from the old survey to the new survey. The collars for many underground holes required correction due to an adjustment of the underground survey control points. The TPS work in 2013 and 2015 resulted in a shift/rotation of the 84 m, 122 m, 244 m, and 305 m Levels. The result was that all drill holes surveyed after April 2013 had the "corrected" mine grid coordinates, while holes surveyed prior to April 2013 (mostly on 305 m Level) had "uncorrected" mine grid coordinates. The shift in the corrected collar coordinates ranges from approximately 0.25 m to 3.0 m.

Rubicon performed a check "closed-loop" survey on the 122 m, 244 m, and 305 m Levels, to confirm the accuracy and correct the location of the underground excavations. The closed-loop survey data were verified by TPS and an Ontario Land Surveyor to be within first- and second-order accuracy in November 2015.

11 SAMPLING PREPARATION, ANALYSIS, AND SECURITY

11.1 Sample Preparation and Security

Since 2002, upon arrival at the core storage facility, the core was washed, core orientation and measurements were performed (when applicable on oriented core), it was visually logged, and it was marked up and tagged for sampling. Down-hole depths, geological and structural features, and sample locations were marked on the core using china markers. Since March 2017, detailed structural data from oriented drill core was collected to enhance geological modelling. The logging of oriented drill core involved collecting alpha and beta angles for each structural feature, relative to an orientation line scribed along the bottom of the drill core. The location of the orientation line was placed based on Boart's TruCore drill-core orientation system and was only scribed on sections of core where there was high confidence in both the initial orientation mark and the interlocking quality of the core segments within and between sequential coring runs, per procedures described in Phoenix Gold Project Supplemental Core Logging Protocol (Golder, 2017).

Since 2007, digital photos have been taken of the core to preserve a digital record of all drill core on the Phoenix Gold Project. Until 2017, the digital photos were taken of the core before logging was completed, using a hand-held camera from an elevated position over the core logging table. At the edge of each photo, a small whiteboard was included recording the drill hole identification, down-hole depth range, and date of the photo. The photos typically captured 3–4 boxes of core laid out on the logging table and lightly misted with water to enhance colour contrasts in different lithological units. Since January 2017, Rubicon has used a customized camera stand designed to take the photos from a fixed 1 m height above the core table and ensured the camera angle was consistently parallel to the plane of the tabletop. The photo procedure was altered at this time such that the photos were now taken after core logging was completed, thereby preserving all notations written on the core. Detailed photos were also taken of interesting geological or structural features when warranted.

Rubicon geologists or consultants/contractors performed all sampling under the supervision of internal QPs and reviewed/monitored by external QPs. Samples were moved directly from the core shack to the cutting shack, where sampled intervals were cut in half and placed in plastic bags for submission to an assaying facility.

From 2002 to 2014, approximately 10 individually bagged samples were placed in a large rice bag that was sealed with a security zip tie containing a uniquely numbered tamper-proof security seal. From 2002 to 2007, samples were shipped by courier to either ALS Minerals (ALS) or Accurassay Laboratories (Accurassay) in Thunder Bay. Since 2008, Rubicon staff delivered samples directly from the mine site to the SGS laboratory in Red Lake. Each sample number and security seal was recorded, then verified by SGS with a written acknowledgment upon receipt.

In 2014, the core shipping procedure was streamlined. Core samples were cut and individually packaged for shipping, as before. Rubicon sampling personnel then sorted and placed the core samples in a larger shipping crate, allowing more samples to be shipped with fewer chain-of-custody forms. Generally, all samples from an individual drill hole would be placed in a crate, sealed with a tamper-proof security seal, and delivered to the lab by Rubicon personnel. Each sample number and security seal were recorded and then verified by SGS with a written acknowledgment upon receipt.

In 2017, the core shipping procedure was again modified such that individual shipments no longer comprised all samples from an individual drill hole. Instead, shipments were dispatched strictly in sequential sample tag order, in lots of 75 samples, each of which correlated to three complete quality control (QC) batches and one complete lab furnace batch. Generally, three lots of 75 samples were included in each shipment. The implementation of smaller lab batches resulted in faster turnaround times on assay results and improved tracking and correlation between QC samples and affected core samples.

Analytical protocols were developed in 2003 and revised in 2009 and 2011 in consultation with Barry Smee, PhD, P.Geo., an independent geochemist (Smee, 2009b and 2011). Individual samples received by the laboratory typically ranged from 0.5 to 2 kg. When necessary, samples were dried prior to any sample preparation in the laboratory. The entire sample was crushed to 2 mm in an oscillating steel-jaw crusher and either an approximate 250 g split or, in the case of metallic-screen fire assay, the whole sample was pulverized in a chrome-steel ring mill. The coarse reject and residual pulp materials were bagged and returned to the Phoenix site for secure storage. Prior to 2009, the samples were crushed to 90% −8 mesh, split into 250 g to 450 g subsamples using a Jones Riffle Splitter and subsequently pulverized to 90% −150 mesh in a shatter box using a steel puck. Silica cleaning between each sample was also performed to prevent any cross-contamination. All samples were sent for fire assay and the pulps remained on site.

Beginning in October 2009, new sample preparation protocols were implemented in accordance with recommendations from Smee (2009a). These included crushing the samples to 85% −2 mm before taking a 500 g split for pulverization. The subsample was then pulverized to 95% −150 mesh, from which a 50 g split was taken for fire assay analysis. Silica cleaning between each sample was also performed to prevent any cross-contamination. All samples were sent to an external lab for fire assay and the pulps remained on site.

Since 2017, sample pulps selected for umpire check assay analyses have been sorted at the Rubicon core site and shipped via courier in security-tag-sealed containers to Activation Laboratories Ltd. (Actlabs) facility in Thunder Bay, Ontario, for analysis. Sample manifests, listing the sample numbers were emailed to the lab, prior to shipping, and a receipt of the samples was received for each shipment, with confirmation that the security seals were intact upon delivery. Blank, duplicate, and Certified Reference Material (CRM) samples from the original testing were included in the suite of umpire check assay samples, as well as additional sealed packets of CRMs to ensure laboratory bias checks were unaffected by any potential preparation contamination at the original lab.

The logged and sampled core is securely stored at the Project site, and in a secured storage yard in Cochenour surrounded by a six-foot-high chain-link fence with a padlocked gate. There is only one road into the mine site, which has a gate with 24-hour security and restricted access. The pulps and rejects were returned from SGS and are securely stored on the Project site for long-term storage.

11.2 Sample Analyses

Since 2002, Rubicon has used three primary independent analytical laboratories for gold analysis on the Phoenix Gold Project. From 2002 to 2007, samples were either sent to the ALS preparation laboratory in Thunder Bay, Ontario; its analytical laboratory in Vancouver, British Columbia; or Accurassay in Thunder Bay, Ontario. From 2008 to 2018, samples were submitted to SGS Minerals in Red Lake, Ontario for preparation and analysis. From January 2010 to October 2012, and in 2014 and 2015 (no samples were taken in 2013), umpire check assays were conducted by ALS and Accurassay, respectively. Rubicon Lab and Actlabs, the latter of which is independent, were used for a small portion of assaying on the project; the former for analysis of production geology and mill-related process samples in 2015, and the latter for umpire check assays since 2017.

The four commercial laboratories are accredited to ISO/IEC Guideline 17025 by the Standards Council of Canada for conducting certain testing procedures, including all the procedures used by Rubicon to prepare and assay for gold. Although the Rubicon Lab was not accredited, the quantity of drill hole data from this lab was not considered by the QP to be material and was accepted for Mineral Resource estimation.

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

11.2.1 ALS Minerals (2002–2007)

Beginning in 2002, sample preparation was completed at ALS in Thunder Bay, and the pulps were shipped to ALS in North Vancouver, British Columbia, for analysis. Gold concentrations were determined by fire assay fusion of a 50 g subsample with an atomic absorption spectroscopy (AAS) finish, as the standard analytical procedure.

The gold metallics assay, also known as screen fire assaying, required 100% pulverization of the sample and screening of the sample through a 150 mesh (100 µm) screen. Material remaining on the screen was retained and analyzed in its entirety by fire assay fusion followed by cupellation and a gravimetric finish. The −150 mesh (pass) fraction was homogenized, and two 50 g subsamples were analyzed by standard fire assay procedures. In this way, the magnitude of the coarse gold effect can be evaluated via the levels of the +150-mesh material.

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

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

Umpire check assays completed at ALS in 2010 to 2012 used standard fire assay procedure on a 50 g subsample. If the sample contained > 10 g/t Au, it was re-assayed with a gravimetric finish.

11.2.2 Accurassay Laboratories (2002–2007, 2014–2015)

Gold was determined by fire assay using a 30 g fire assay charge. This procedure used lead collection with a silver inquart. The beads were then digested and an AAS or ICP-AES finish was used. All gold assays >10 g/t were automatically re-assayed by fire assay with a gravimetric finish. A Sartorius micro-balance was used with a sensitivity of 1 microgram (six decimal places) corresponding to 5 parts per billion (ppb) detection limit.

Screen metallics analyses included crushing the entire sample to 90% −10 mesh and using a Jones Riffle Splitter to split the sample to a 1 kg subsample. The entire subsample was then pulverized and subsequently sieved through a series of meshes (80, 150, 200, 230, and 400 mesh). Each fraction was then assayed for gold (maximum 50 g). Results were reported as a calculated weighted average of gold in the entire sample. Core samples were also assayed for a suite of 32 trace elements using a multi-acid digestion followed by ICP-AES. As with ALS, results were reported electronically to the Project site in Red Lake, with assay certificates filed and catalogued at Rubicon's head office in Vancouver.

For the umpire check assays from 2014 to 2015, gold was determined by fire assay using a 50 g fire assay charge. If the sample contained >10 g/t Au, it was re-assayed with a gravimetric finish.

11.2.3 SGS Mineral Services (2008–2018)

At SGS, prior to 2009, gold was analyzed using the fire assay process on a 30 g subsample. If the sample contained >10 g/t Au, it was re-assayed using a gravimetric finish. Starting in October 2009, the subsample size was increased to 50 g on the recommendations of Smee (2009a). All gold assays >10 g/t were automatically re-assayed with gravimetric finish.

Beginning in September 2018, samples from well-mineralized intersections of High-Ti Basalt were selected for metallic screen analysis at SGS, to allow for assaying results relative to coarse gold. This analysis allows for the determination of both coarse and fine material after the screening process and

can benefit in understanding the grade distribution locally. Samples to be analysed by this method were selected by the logging geologist based on mineralization observed during core logging. Samples generally ranged from 0.5 to 1.0 m long and weighed 1 to 2 kg. Upon receipt at the lab, samples were dried, then crushed to achieve a nominal sample size (~9 mesh). Samples were then split using a 14-slot, ¾-inch splitter that divides the sample into two portions. One portion was reserved as a "reject" portion in the event that a re-assay was required due to a QC failure. The other portion was pulverized, then screened using a Ro-tap assembly to 106 µm. The entire plus fraction was submitted to the lab for analysis to extinction. Two 50 g aliquots were riffled from the minus fraction and submitted for analysis. Final assays were weight ratioed back to the representative sample weight. All fractions were analysed by fire assay with a gravimetric finish. All analyses and weight fractions were reported.

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

Until 2014, results were reported electronically to the Project site in Red Lake and multiple recipients at the head office in Vancouver, with assay certificates filed and catalogued at Rubicon's head office in Vancouver, British Columbia, Canada and added to the master Microsoft Access database stored on the Vancouver and Red Lake servers. After the closure of the Vancouver office, all data is currently stored on servers located in Toronto and Red Lake.

In 2014, database management was moved from Vancouver to the Project site. Approved assay certificates from SGS were received at Rubicon Red Lake site in digital format since that time.

11.2.4 Rubicon Assay Laboratory (2015)

In 2015, Rubicon purchased and operated an assay laboratory located in Balmertown, Ontario, approximately 8 km from the Phoenix Gold Project. This laboratory processed all production geology and mill-related processing samples. A total of 1,894 samples from 63 production-related Bazooka drill holes and 1,566 chip samples taken from 411 sampling locations were processed at this lab. Gold concentrations were determined by fire assay fusion of a 30 g subsample, with an AAS finish as the standard analytical procedure. Currently, this assay lab is closed, and all assaying is outsourced.

11.2.5 Actlabs (2017–2018)

For the umpire check assay samples analyzed at Actlabs, gold was analyzed by fire assay with AAS finish on a 50 g charge from pulps that had previously been prepared and analyzed by SGS in Red Lake. Following the same analytical protocols as the original lab, all samples that returned a result >10 g/t were automatically repeated by fire assay with gravimetric finish.

11.2.6 Handling of Multiple Assay Values for One Sample

In cases where multiple assays were completed on an individual sample, gold values produced by the metallic fire assay were deemed to supersede fire assay gold values owing to the larger size of the sample analyzed and/or the better reproducibility in samples with coarse gold. When samples were analyzed multiple times by the same method (i.e., duplicate or umpire check assay analyses), the original assay was incorporated in the model. Replicate analyses were used only as QC checks to validate the original result.

11.2.7 Data Management

Data are verified and double-checked by senior geologists at site for data entry verification, error analysis, and adherence to strict analytical quality-control protocols. Drill hole data collected from 2009 to 2014 were managed by ioGlobal Pty. Ltd. (ioGlobal) and reviewed for quality assurance (QA) and QC. In 2014, database management was returned to the Phoenix site, under the supervision of Rubicon.

11.3 Sample Analyses of Metallurgical Testwork

11.3.1 G&T Metallurgical Services

Metallurgical testwork was completed at the G&T Metallurgical Services Ltd. (G&T) facility in Kamloops, British Columbia. Gold was measured by the fire assay method using a 30 g assay charge. When requested, the metallic sieve preparation method was also used. Although not accredited, the laboratory has a complete written procedure and participates in a Proficiency Testing Program accredited by the Standards Council of Canada. This facility also performed assays for iron and arsenic content using a multi-acid digestion and ICP-AES method, and assays for sulphur and carbon by combustion furnace.

G&T also performed different metallurgical testing for the characterization of the mineralized material. All tests performed were done using industry-recognized methods for the testwork. In 2013, the facility was visited by Soutex personnel (SRK, 2013b). Soutex noted that the facility has well-documented controlled procedures for all types of testing. The quality management includes ISO-9001 accreditation.

11.3.2 ALS

All the samples related to the treatment of the bulk samples were processed at SMC (Canada) Ltd.'s (SMC) McAlpine mill in Cobalt, Ontario during the summer and fall of 2011 and were sent to ALS-accredited laboratories for analysis. Gold assays were done with fire assay on a 30 g assay charge. All head-grade samples and tailings samples were prepared with screen metallic sieve preparation done on the whole received sample. All gold concentrate samples were assayed without screen metallic sieve preparation. The samples were expedited and received at the Val d'Or ALS facility, and the assays were

performed in ALS' laboratory in North Vancouver. A series of blank, duplicate, and CRM samples were also sent to the laboratory for QC. No further metallurgical testing has been completed since the mill start-up in 2015.

11.4 Specific Gravity Data

The SG database includes 6,666 records generated by Rubicon from measurements on core from 470 drill holes (Table 11-1). SG measurements were taken from representative core sample intervals (approximately 0.1 to 0.2 m long). SG was measured using a water dispersion method. The samples were weighed in air, then the uncoated sample was placed in a basket suspended in water and weighed again. Table 11-1 summarizes the measurements by rock type.

Table 11-1: Specific Gravity Data by Lithology Type

Rock Code	Description	Count	Specific Gravity			
			Average	Std. Dev.	Minimum	Maximum
E1H	High-Ti Basalt	1,396	2.96	0.10	2.20	3.72
E0T	Talc rich unit	1,600	2.90	0.05	2.61	3.15
I3	Felsic Intrusive rocks	847	2.67	0.07	2.36	3.08
E0	Ultramafic flow	1,264	2.92	0.08	2.50	3.76
E0B	Komatiitic basalt	370	2.98	0.07	2.61	3.24
E1A	Basalt	198	2.89	0.09	2.67	3.54
AGZ	Altered green zone	97	2.93	0.09	2.69	3.20
Other	Other	894	2.88	0.12	1.85	3.45
Total		**6,666**				

Note: Std. Dev. = standard deviation

In 2017, a suite of samples was selected for SG testing at Actlabs, to confirm the measurements taken by Rubicon. Sample pulps were shipped to Actlabs and analyzed using method RX17 for SG on pulp, which is measured using the relative volumes of solids to water and air in a given volume. The 2017 SG results and the difference from 2016 values are summarized in Table 11-2. The results for High-Ti Basalt were further subdivided into unmineralized (unmin: <2.99 g/t Au), low-grade (low: 3.00 to 9.99 g/t Au) and high-grade (high: >10 g/t Au) sources (Golder, 2018b). The difference noted in the SG values of Komatiitic Basalt from 2016 and 2017 measurements is attributed to the inclusion of samples in the 2017 dataset which were logged as Komatiitic Basalt but were mixed Komatiitic Basalt/High-Ti Basalt units.

Table 11-2: 2017 Specific Gravity from pulps by Lithology Type (Golder, 2018b)

Rock Code	Description	Count	Specific Gravity				Difference
			Average	Std. Dev.	Minimum	Maximum	
E1H	High-Ti Basalt (overall)*	24	2.95	0.09	2.80	3.18	0.01
	**High-Ti Basalt (unmin. & low)*	*(19)*	*2.93*	*0.08*	*2.80*	*3.11*	
	**High-Ti Basalt (high)*	*(5)*	*3.02*	*0.12*	*2.85*	*3.18*	
E0T	Talc rich unit	6	2.94	0.07	2.84	3.05	-0.04
I3	Felsic Intrusive rocks	1	2.62	-	2.62	2.62	0.05
E0	Ultramafic flow	0	-	-	-	-	
E0B	Komatiitic basalt	8	2.91	0.11	2.70	3.02	0.07
E1A	Basalt	0	-	-	-	-	
AGZ	Altered green zone	0	-	-	-	-	
Other	Other (V2_BX veins)	5	2.87	0.10	2.79	3.04	0.01
Total		**44**					

Notes: Std. Dev. = standard deviation. Difference = 2016 SG – 2017 SG

11.5 Quality Assurance and Quality Control Programs

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

Analytical QC measures typically involve internal and external laboratory procedures implemented to monitor the precision and accuracy of the sample preparation and assay data. They are also important to identify potential sample sequencing errors and to monitor for contamination of samples.

Sampling and analytical QA/QC protocols typically involve taking duplicate samples and inserting QC samples (CRMs and blanks) to monitor the reliability of the assay results throughout the drill program. Umpire check assays are normally performed to evaluate the primary lab for bias and involve re-assaying a set proportion of sample rejects and pulps at a secondary umpire laboratory.

11.5.1 Rubicon Sampling (2008–2015)

Rubicon monitored the internal analytical QC measures implemented by the primary laboratories it used for analysis. In addition, Rubicon implemented external analytical QC measures starting in 2008 on all sampling conducted at the Phoenix Gold Project. The analytical QA/QC program was designed and monitored by both internal and external QPs. For drill core, analytical control measures used by

Rubicon consisted of inserting control samples (blank, grade-matched CRMs, and field duplicates) in all sample batches submitted for assaying. For 2015 production-related sampling, including Bazooka drill core, chip sampling, and muck sampling, the external QC measures consisted of commercially sourced CRMs only.

No drilling took place in 2013 or 2016 with associated geochemical sampling.

From 2008 to 2010, the blank samples consisted of store-bought white garden stone (quartz or quartzite). In 2010, Rubicon used material sourced from a granite boulder located near Red Lake. From February 2011 to July 2015, Rubicon used granite slab purchased from Nelson Granite in Vermillion Bay, Ontario. Beginning in August 2015, locally sourced granite from Red Lake was used, after submitting several samples to verify that it was barren of gold.

Field duplicates consisting of half core were taken from June 2009 to November 2015. Since 2017, Rubicon is submitting coarse reject duplicates only.

Twenty-nine different commercial CRMs, with various gold grades, were sourced from CDN Resource Laboratories Ltd. (CDN) to monitor sampling accuracy between 2008 and 2015. Control samples used range from 0.121 to 29.21 g/t Au (Table 11-3).

Table 11-3: Specifications of CDN CRMs Used by Rubicon on the Phoenix Gold Project between 2008 and 2015

Gold CRM	Recommended Value (g/t Au)	Standard Deviation (g/t)	Number of Samples
CDN-GS-P1	0.121	0.011	58
CDN-GS-P5B	0.44	0.02	90
CDN-GS-P7A	0.77	0.03	93
CDN-GS-P8	0.78	0.03	178
CDN-GS-10	0.82	0.05	3
CDN-GS-1J	0.946	0.051	170
CDN-GS-1H	0.972	0.054	297
ZCDN-GS-1G	1.14	0.05	91
CDN-GS-1E	1.16	0.03	1,649
CDN-GS-1L	1.16	0.05	186
CDN-GS-1P5A	1.37	0.06	16
CDN-GS-1P5B	1.46	0.06	83
CDN-GS-1P5L	1.53	0.07	5
CDN-GS-9	1.75	0.07	123
CDN-GS-2B	2.03	0.06	77
CDN-GS-2A	2.04	0.095	5
CDN-GS-2C	2.06	0.075	243

Gold CRM	Recommended Value (g/t Au)	Standard Deviation (g/t)	Number of Samples
CDN-GS-3E	2.97	0.135	107
CDN-GS-3D	3.41	0.125	180
CDN-GS-5C	4.74	0.14	1
CDN-GS-5E	4.83	0.185	1,244
CDN-GS-5J	4.96	0.21	162
CDN-GS-5A	5.1	0.135	10
CDN-GS-5F	5.3	0.18	431
CDN-GS-6A	5.69	0.24	478
CDN-GS-7A	7.2	0.3	121
CDN-GS-6	9.99	0.25	8
CDN-GS-11A	11.21	0.435	17
CDN-GS-30B	29.21	0.615	170

Control samples (including blanks, gold CRM samples, and field duplicates) were inserted every 25 samples. In addition, umpire check assays were performed on approximately 3% to 5% of samples.

In addition to in-house monitoring, analytical QC data produced by Rubicon between 2002 and 2007 was reviewed in a report by AMC (2011). Analytical QC data collected between 2008 and October 2012 was summarized and analyzed in a 2013 Technical Report by SRK (2013b). Analytical QC data for the drilling completed between 2014 and 2015 was reviewed and summarized in a Technical Report by SRK (2016). TMAC has reviewed the work completed and relied on SRK's review. Historical drill holes drilled prior to 2002 do not have known analytical QC data.

11.5.2 Rubicon Sampling (2017–2018)

Rubicon monitored the internal analytical QC measures implemented by the primary laboratories it used for analysis. In addition, Rubicon implemented external analytical QC measures on all sampling conducted at the Phoenix Gold Project since the previous Technical Report (SRK, 2016). The analytical QA/QC program was designed and monitored by both internal and external QPs. For drill core sampling, analytical QC measures by Rubicon consisted of inserting control samples in all sample batches submitted for assaying. For most 2017–2018 production-related sampling, including chip sampling and test hole sampling, the external QC measures consisted of commercially sourced CRMs only, testing both high- and low-grade ranges. Rubicon also monitored internal laboratory blank and duplicate analyses for production-related sampling. For 2017–2018 muck sampling, Rubicon relied entirely on the laboratory's internal QC measures, except for high-grade analyses, which were covered under Rubicon's blanket policy requiring the laboratory to insert a blind external CRM with all gravimetric analyses.

Blank samples were inserted to monitor sample cross-contamination during the sample preparation process as well as to identify potential sample sequencing issues. The blank used in 2017–2018 consisted of locally sourced granite from Red Lake that had previously been tested to verify that it was barren of gold. Blanks were inserted at a minimum of 1 per 25 samples, preferentially placed after samples expected to return higher assay values, especially when visible gold had been observed in the core. Multiple blanks were inserted in batches when numerous high-grade samples were noted by the geologist. A total of 514 blank samples were used in the new data covered in this report.

CRMs were used to monitor the accuracy of the gold assays and to check for laboratory bias when samples were sent for umpire check assays. The selection of the CRM sample in each batch was made by the geologist to match the expected grade of the samples analyzed by fire assay—atomic absorption spectrometry (AAS). Additionally, the lab was required to insert a blind high-grade standard whenever a gravimetric analysis was completed on samples that assayed >10 g/t by the AAS method. Seven commercial gold CRMs sourced from CDN were used in sampling on the 2017–2018 program ranging from 1.16 to 29.21 g/t Au (Table 11-4).

Table 11-4: Specifications of CDN CRMs Used by Rubicon on the Phoenix Gold Project in 2017-2018

Gold CRM	Recommended Value (g/t Au)	Standard Deviation (g/t)	Number of Samples	Number of Failures	Percent Failures
CDN-GS-1P5L	1.53	0.07	326	4	1.2%
CDN-GS-4E	4.19	0.11	79	7	8.9%
CDN-GS-6A	5.69	0.24	16	2	12.5%
CDN-GS-7F	6.90	0.21	61	3	4.9%
CDN-GS-11A	11.21	0.47	55	2	3.6%
CDN-GS-30B	29.21	0.62	55	13	24%

Replicate samples included coarse reject and pulp duplicates as a check on laboratory precision in assaying. Two replicate samples were completed in every batch of 25, one from the pulps and one from the coarse rejects. Approximately 970 replicate analyses were completed on the new data covered in this report.

Additionally, in July 2018, seven drill holes were selected for resampling of approximately 50 m continuous intercepts from each drill hole, including both mineralized and unmineralized lithological units (effectively field duplicate analyses). The remaining half of the hole was sampled, over the same intervals as the original sampling, assigned new sample identities, and submitted to SGS for routine fire assay analysis. The program comprised 470 core samples, including the usual complement of CRMs, blanks, and replicate samples. Results of this sampling are discussed below.

Rubicon conducted umpire check assay programs throughout the program. In 2012, 5% of the sample pulps were re-assayed by ALS. In 2015, 3% of the sample pulps were re-assayed by Accurassay (2014–2015). Since 2017, 5% to 6% of the sample pulps were re-assayed at Actlabs.

Analytical results were validated by monitoring analytical results of QC samples inserted with the routine core samples submitted for assaying. The QC data was monitored concurrently with data collection, allowing immediate resolution of any issues identified. Both internal and blind QC samples were plotted on Shewhart control charts regularly to identify outliers and trends in the control samples, which would indicate potential issues with the assay data. Paired data (replicates and umpire check assays) were analyzed using bias charts, quantile-quantile, relative difference plots, and Thompson-Howarth precision plots. Examples of the QC monitoring charts are shown in Figure 11-1 through Figure 11-5.

For the 2017–2018 drill programs, the blind CRMs (or standards) were within the range of accepted values with no significant trend or bias noted. Several examples of Shewhart control charts used for monitoring the blind standards are illustrated in Figure 11-1 and Figure 11-2. These charts record all QC samples, including blind CRM failures. Outliers were noted, and explanations provided on the respective charts. All QC failures for batches having significant gold assay values were re-assayed. For batches having no significant gold values, when the CRM result failed on the low side, the core was reviewed, and if no significant results were expected for the samples in that batch, a geological override was applied and the assay values were accepted. Thirty-one of the total 592 CRM samples failed their initial assay and of these, 27 were re-assayed successfully and 4 were resolved with a geological override. In general, outliers for the blind CRMs tended to be biased toward low values. No significant cross-contamination of samples was noted in the analytical process (Figure 11-3). When the blind preparation blank assayed >0.1 g/t cut-off, the entire batch was re-assayed from the coarse reject to obtain acceptable results. If acceptable results could not be obtained from the coarse reject, the remaining half of the core was submitted for re-assay, with a request for a silica wash after every sample through the entire preparation process. Two examples of internal laboratory CRM charts (Figure 11-4 and Figure 11-5) are also presented. These charts plot all CRM results, including failed CRMs that were ultimately re-assayed by the lab.

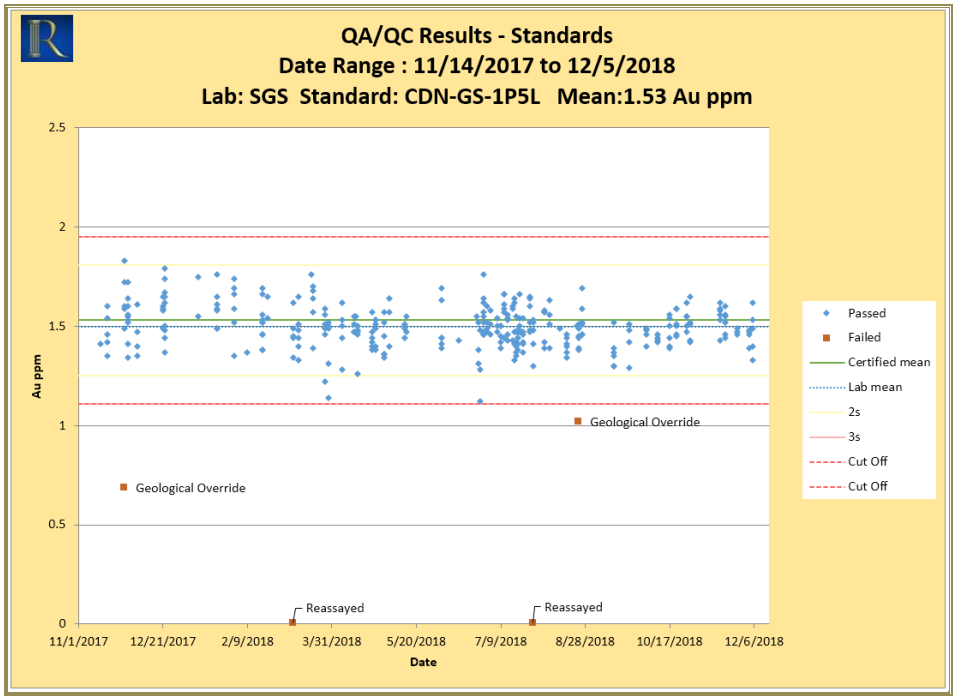


Figure 11-1: Shewhart Control Chart for CDN-GS-1P5L

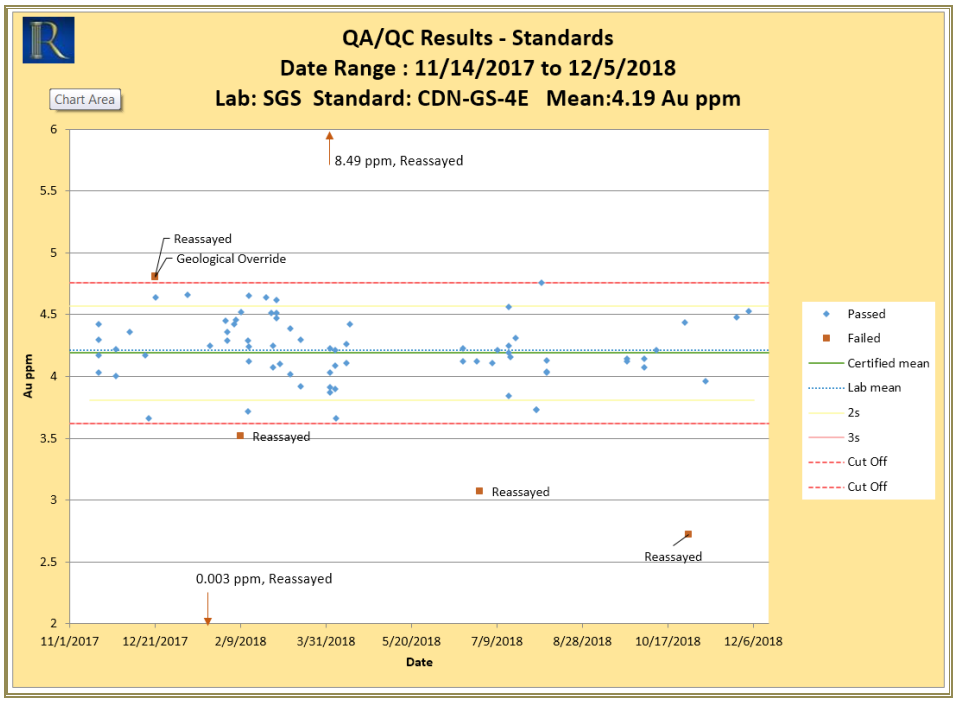


Figure 11-2: Shewhart Control Chart for CRM CDN-GS-4E

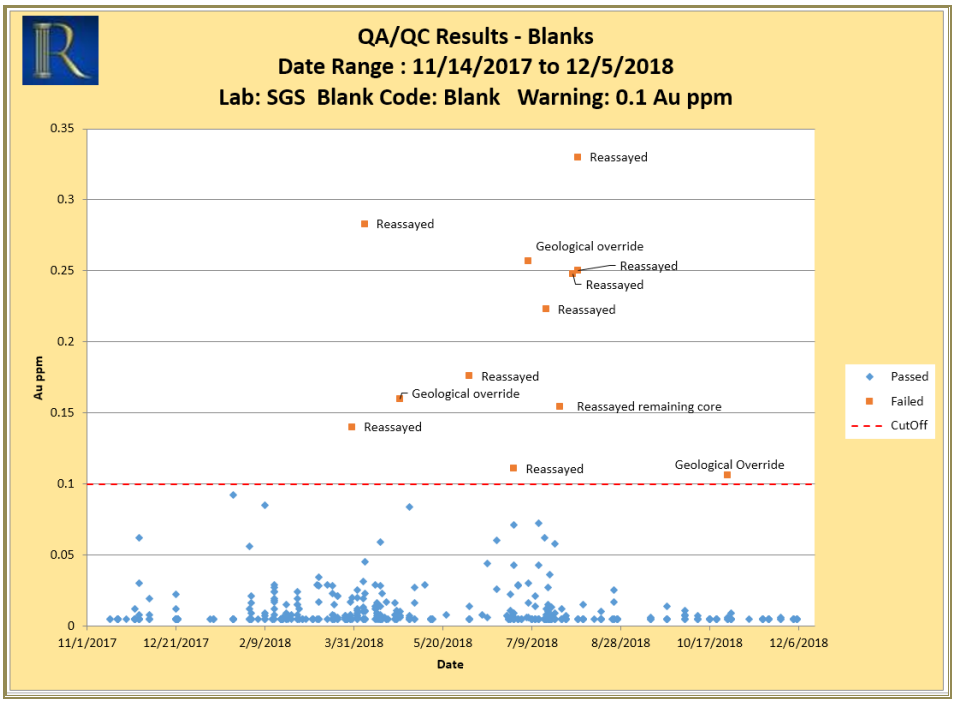


Figure 11-3: Shewhart Control Charts for Monitoring Results for Blank Samples

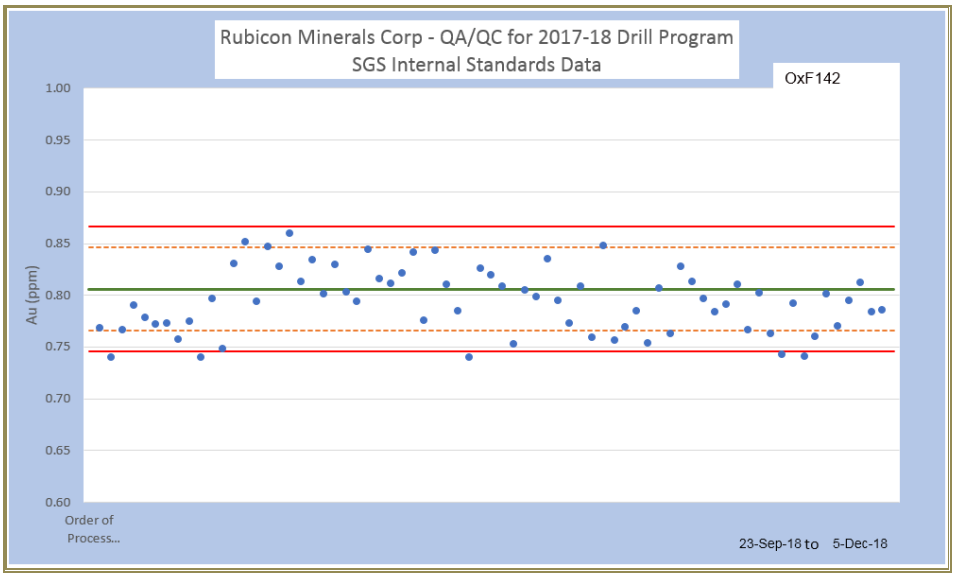


Figure 11-4: SGS Internal CRM OxF142 Processed with 2018 Drill Program Samples

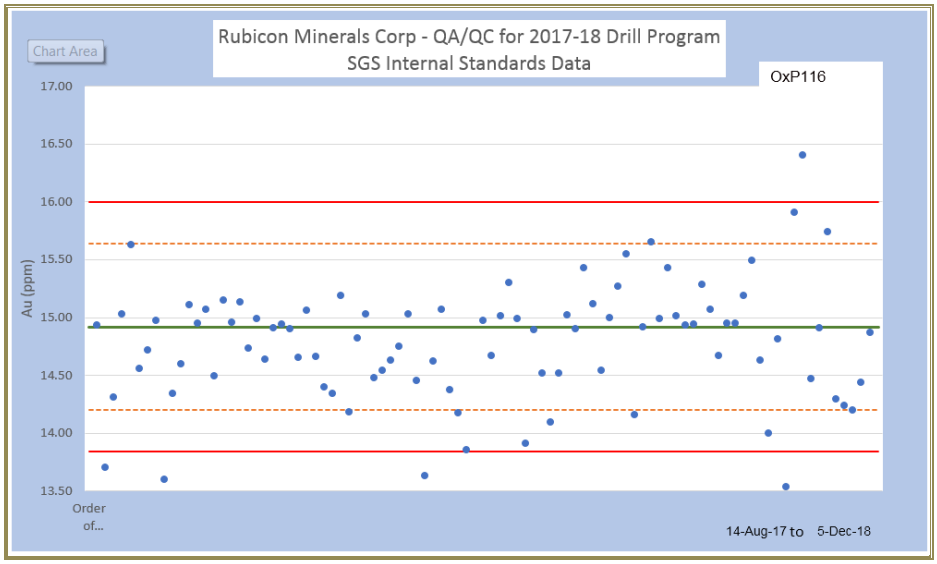


Figure 11-5: SGS Internal CRM OxP116 Processed with 2017-2018 Drill Program Samples

Analytical QC failures were identified as:

- Any blank sample that reported >0.1 g/t Au
- Any CRM result that reported with a difference >3 standard deviations from the certified mean or recommended value for the standard
- More than two sequential CRM results that reported with differences >2 standard deviations from the certified mean or recommended value, having the same positive or negative bias.

Results were tracked in an action log as part of the standard QA/QC procedures. Failures were investigated and samples were re-assayed as required.

SGS's performance on the CRMs used in the new data covered by this report is summarized Table 11-5 and Table 11-6.

Table 11-5: Internal Lab CRMs

Constituent	Certified Value	Absolute Standard Deviations					RSD (%)	# Used	Lab Mean	Bias (ppm[1])	Percent Bias (%)
		1SD	2SD Low	2SD High	3SD Low	3SD High					
Fire Assay											
OxF125	0.806	0.020	0.766	0.846	0.746	0.866	2.48	433	0.793	-0.013	-1.6
OxF142	0.805	0.019	0.767	0.843	0.748	0.862	2.36	71	0.794	-0.011	-1.4
OxN117	7.679	0.207	7.265	8.093	7.058	8.300	2.70	12	7.471	-0.208	-2.7
OxN134	7.667	0.155	7.357	7.977	7.202	8.132	2.02	1	-	-	-
OxP116	14.92	0.360	14.200	15.640	13.840	16.000	2.41	88	14.72	-0.15	-1.0

Note: The QA/QC database stores gold grades in parts per million (ppm) rather than g/t. 1ppm Au =1 g/t Au

Table 11-6: Blind CRMs

Constituent	Certified Value	Absolute Standard Deviations					RSD (%)	# Used	Lab Mean	Bias (ppm)	Percent Bias
		1SD	2SD Low	2SD High	3SD Low	3SD High					
Fire Assay											
GS-1P5L	1.53	0.07	1.25	1.81	1.11	1.95	4.58	326	1.50	-0.03	-2.1%
GS-4E	4.19	0.11	3.81	4.57	3.62	4.76	2.69	79	4.22	0.03	0.6%
GS-6A	5.69	0.24	5.21	6.17	4.97	6.41	4.22	16	5.95	0.26	4.6%
GS-7F	6.90	0.20	6.08	7.72	5.67	8.13	2.97	61	6.97	0.07	1.0%
GS-11A	11.21	0.44	10.34	12.08	9.91	12.52	3.88	55	11.01	-0.20	-1.8%
GS-30B	29.21	0.62	27.98	30.44	27.37	31.06	2.11	55	28.94	-0.27	-0.9%

Note: The QA/QC database stores gold grades in parts per million (ppm) rather than g/t. 1 ppm Au =1 g/t Au

Scatterplots and quantile–quantile (Q–Q) plots for pulp and coarse reject duplicate pairs indicate a good correlation between original and duplicate assays. The Thompson-Howarth precision for the pulp and coarse reject duplicates are 12% and 19%, respectively, which is within typical ranges for Archean lode gold deposits.

The results of the resampling (field duplicate) program indicated that the correlation of results was generally good, in that the samples that returned anomalous results in the original sampling were also anomalous when re-assayed. However, the precision on the results was quite poor, evidenced by the broad scatter of data, particularly below 1.2 ppm, and of greatest concern, in the 2 to 5 ppm range, which affects values near cut-off grade (i.e., waste to ore category boundary). Re-assay samples assaying greater than 5 ppm tended to be biased toward the re-assay result, but this was likely due to the inherent sampling bias related to the selection of samples that originally ran lower than anticipated. Because of the aforementioned bias in this dataset, precision analysis was not undertaken, as it would not be considered reliable.

Check assay results from pulps analysed at Actlabs indicated an excellent correlation with the original assay values. The linear correlation and rank correlation values of 0.99 and 0.97, respectively, confirm that the datasets are very similar. Mean values for the datasets were comparable, having a difference of only 0.07 ppm. The Thompson-Howarth precision was 10% for the entire dataset, which is within expected tolerances for pulp duplicate analyses.

11.6 Qualified Person Opinion on the Adequacy of Sample Preparation, Security, and Analytical Procedures

It is TMAC's opinion that the sample preparation, security, and analytical procedures used by Rubicon are consistent with standard industry practices and that the data are suitable for the 2019 Mineral Resource estimate. TMAC has no material concerns with the geological or analytical procedures used or the quality of the resulting data.

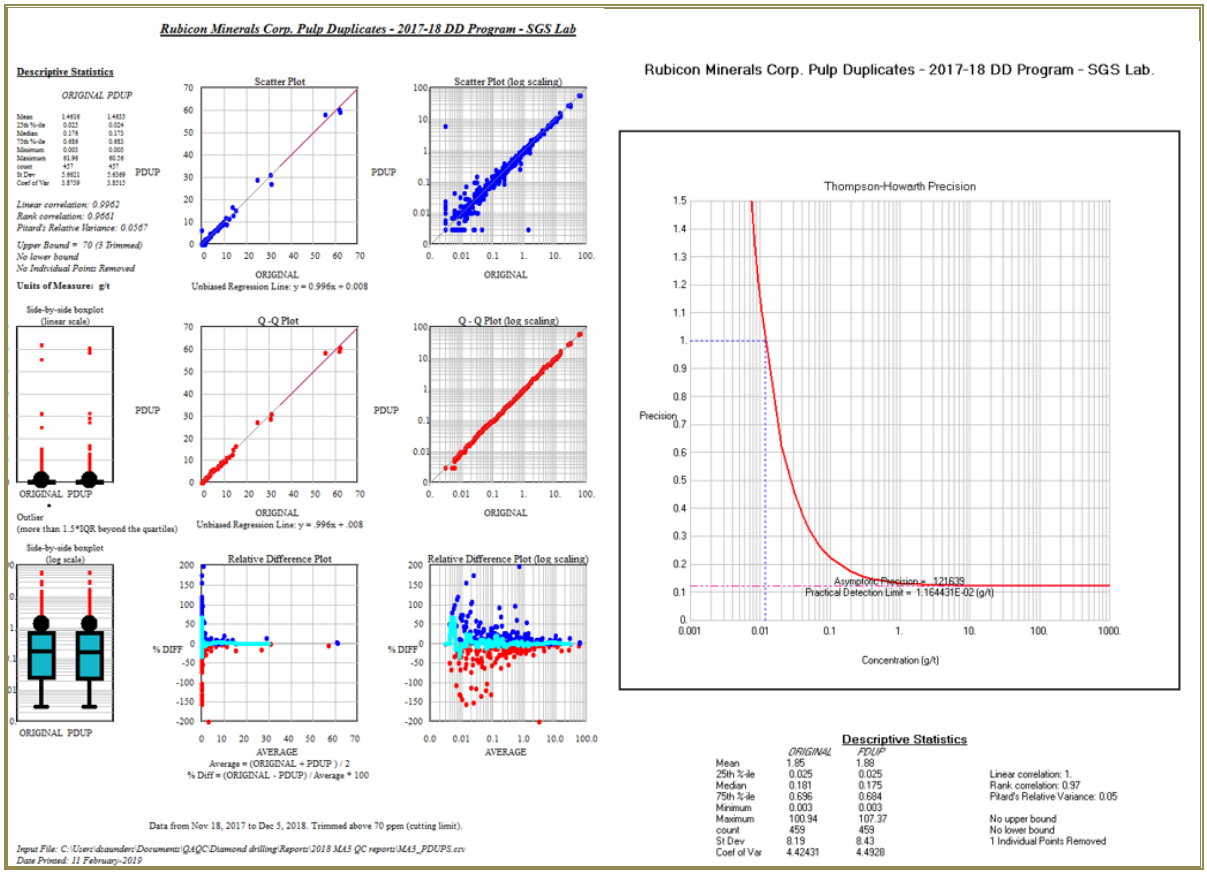


Figure 11-6: Scatter Plots, Q-Q Plots, Relative-Difference Plots, and Thompson-Howarth Plots Used to Monitor Precision on Duplicate Assay Pairs for Pulp Duplicates

12 DATA VERIFICATION

TMAC conducted data verification during the update of the Mineral Resource Estimate. This included the built-in checks associated with importing data in GEMS, random checks of database assays compared with assay certificates, and review of the QA/QC performance during the 2017–2018 drill programs (Section 11). This data verification was supported by four site visits during the past two years. Exploratory data analysis, as discussed in Section 14, was an additional component of the data verification process.

12.1 Site Visit

Tim Maunula, P.Geo., conducted four site visits to the Phoenix Gold Project:

- April 17 to 21, 2017
- June 5 to 9, 2017
- September 7, 2018
- February 18, 2019.

The site visit activities included:

- Review of surface geology, mineralization, and structures
- Underground inspection of geology, structural controls, mineralization, underground; development, and active diamond drilling stations
- Review of drill hole logging, sampling, and associated QA/QC procedures
- Review of re-logging and resampling procedures for historical drilling
- Review of chain of custody from drilling to assay lab
- Confirmation of drill logs and sampling
- Inspection of the SGS Laboratory located in Red Lake.

For the most recent site visit in 2019, TMAC reviewed logging and sampling of drill hole 610L-18-06 (Figure 12-1).

No significant issues were identified during the site visits. The data collection, sampling procedures, and chain of custody were found to be consistent with industry standards and in accordance with Rubicon internal procedure documentation.

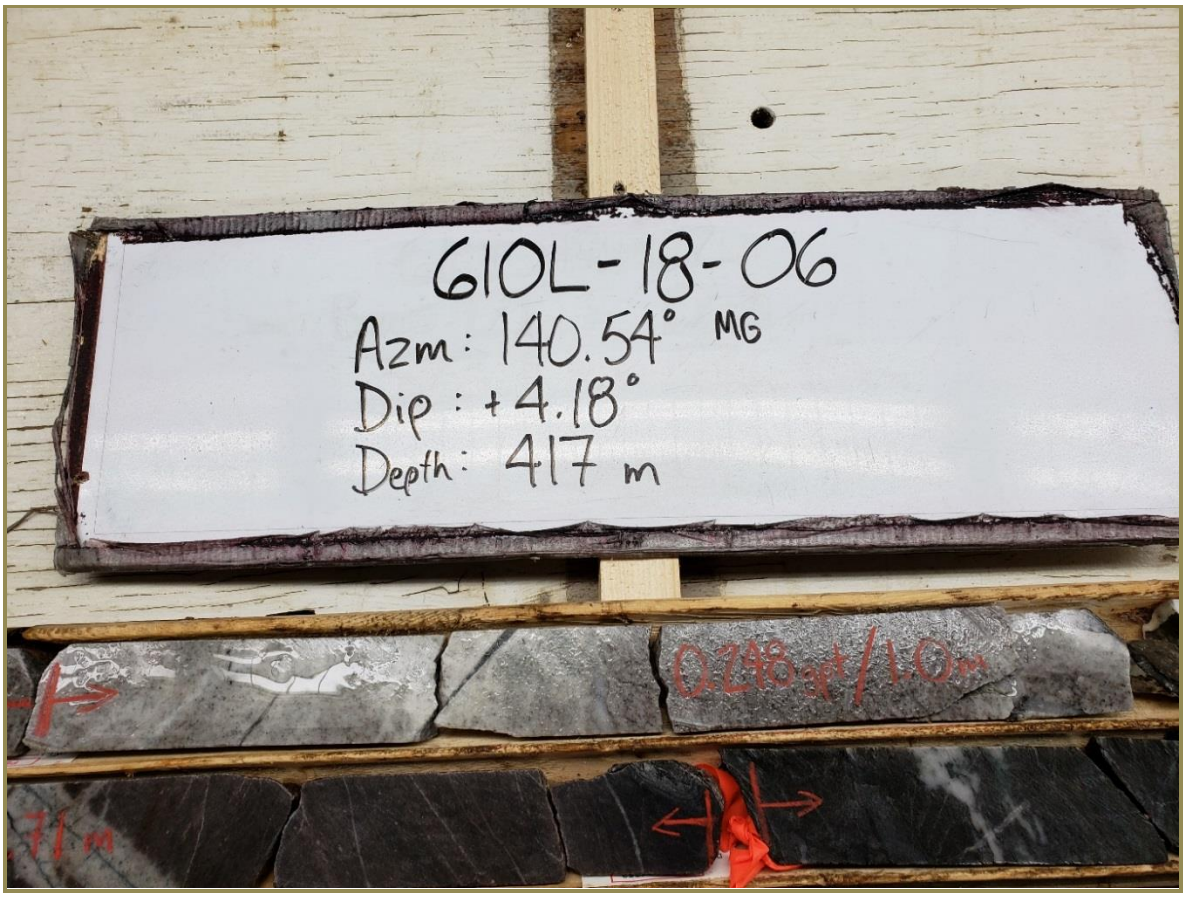


Figure 12-1: Selected Core Interval, Drill Hole 610L-18-06 (Rubicon, 2019)

12.1.1 SGS Laboratory Inspection

The TMAC QP conducted unannounced inspections of the SGS laboratory in Red Lake during each of the 2017 site visits. The manager of the laboratory led the tour through the sample receiving area, sample preparation area, fire assay area, and the wet lab. A detailed audit of the laboratory was not conducted, but no issues were identified.

12.1.2 Independent Sampling by Golder (2017)

Golder (2018b) selected intervals from five holes from the 2017 drill program for validation logging and sampling. The TMAC QP participated in the validation and sampling process.

A total of 27 core samples and 3 control samples (2 CRM standards and 1 blank) were taken from quarter sawn core. Golder elected to combine some of the shorter sample intervals, ranging from 0.3 m to 0.5 m, into longer composite samples due to the low sample volume of the quartered core.

The core was compared to the logged descriptions and found to be reasonably accurate with some minor inconsistency issues identified.

Golder samples were quarter sawn and placed into plastic sample bags, then combined into larger rice bags and secured with a security seal. All samples were then shipped to the Actlabs facility in Ancaster, Ontario, for fire assay using the same analytical procedures as used by Rubicon.

The Golder assay results were then compared to the Rubicon database and summarized in Figure 12-2. Despite some obvious sample variance, most assays compared within reasonable tolerances for the deposit type, and no material bias was evident. One Rubicon outlier assay (1,182 g/t) from a short sample interval (0.35 m) was not repeatable, and the composite grade of the entire 1.35 m interval plotted off the scale of the graph at 85.62 g/t. The Golder composite grade for the same interval was 4.38 g/t.

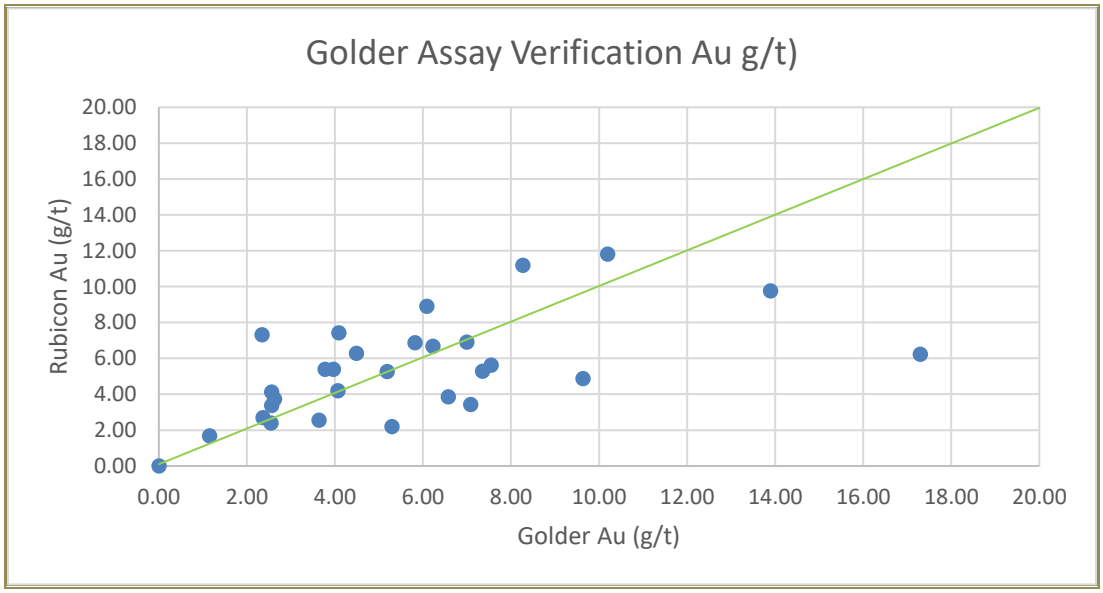


Figure 12-2: Scatterplot Comparison of Verification Samples (Golder, 2018b)

TMAC did not collect an independent sample for verification.

12.1.3 Database Verification

In 2018, Golder completed spot check verification of 281 Au assays from representative areas within the modelled mineral zones, focusing on samples having a grade >2.0 g/t. Sample intervals were selected from holes spanning date ranges from 2008 to 2017. Golder did not identify any material issues, and the data were found to match the original lab certificates. A summary of the data validation is listed in Table 12-1.

Table 12-1: Drill Hole Sample Data Validation

Year	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017
# of Samples	12	37	112	15	51	0	1	1	0	52
# of Errors	0	0	0	0	0	0	0	0	0	0

TMAC supplemented the Golder verification with verification of the post-2017 drill hole data. TMAC selected 34 assay certificates filtering by submission period (2017Q4 to 2018Q3) and by level (305 m Level, 610 m Level, and 685 m Level). A total of 831 assays from 21 drill holes were verified. TMAC did not identify any material issues.

12.2 Conclusions

On completion of the data verification process, it is the TMAC QP's opinion that the geological data collection, sampling, and QA/QC procedures used by Rubicon are consistent with accepted industry practices, and that the database is of suitable quality to support the 2019 Mineral Resource Estimate, as reported in Section 14.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

This section summarizes the metallurgical testwork completed on samples from the F2 Gold Zone between 2008 and 2012 to support the conceptual design of a processing plant for the 2013 Technical Report (SRK, 2013b). This section also summarizes the processing of material mined from selected stopes in the F2 Gold Zone in 2015 and 2018. The metallurgical testwork information for this section was extracted from the SRK 2013 Technical Report. There was no further laboratory metallurgical or process development testing conducted after 2012; however, the mill operated in 2015 and 2018 providing data to assess the mill metallurgical design parameters. The 2015 mineral processing summary has been described in the 2016 Technical Report (SRK, 2016) and the 2018 Technical Report (Golder, 2018b). Mill operation in 2015 and 2018 confirmed that the basic process gold recovery design criteria at 1,250 t/d can be achieved.

In 2018, Rubicon Minerals successfully batch-tested, in the Phoenix mill, additional material mined from three stopes in the F2 Gold Zone to validate its Mineral Resource model. Mining of the first test stope commenced in Q4 2017, while processing occurred in 2018. Prior to processing the test stopes material, approximately 1,700 tonnes of waste-rock was used to bed in the mill. An additional 7,620 tonnes of low-grade material from non-stope sources was then run through the mill. In total, 43,250 tonnes of mineralized material was processed, producing 5,669 oz Au and 1,043 oz Ag. The 2018 bulk sample processing campaign yielded an improvement in overall gold recovery, with the largest gain occurring in gravity gold recovery as compared to the 2015 operation. The 2018 campaign reinforces the conclusion that the mill at the Phoenix project is robust and capable of processing the mineralized material extracted from the F2 Gold Zone at the 1,250 t/d design level and can achieve the 1,800 t/d design level with relatively minor downstream process changes. The process facility has now treated a total of 100,543 tonnes and produced 11,279 fine oz of gold (after settlement).

13.1 Summary of Historical Testwork

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

In October 2010, Rubicon completed a metallurgical testwork program (the 2010 study) performed by Soutex. This study was done on small samples from different underground zones. The testwork program was conducted at G&T's Burnaby test facility under the supervision of Soutex (G&T, 2010). This study included running a metallurgical testwork program, developing a preliminary milling process, and designing a preliminary concentrator. The design addressed the gold recovery process, from mineralized material delivered from the mine to the process plant for gold extraction, to

producing gold doré and discharging cyanide-free tailings to the tailings management facility (TMF). Paste backfill plant considerations and the TMF were not included in this study.

In September 2011, Rubicon commissioned Soutex to perform additional metallurgical testwork. The study was done on representative subsamples (composites) extracted from two approximately 1,000-tonne bulk samples representing two underground areas on the 305 m Level. The metallurgical testwork program was also conducted at G&T under the supervision of Soutex (G&T, 2011) which subsequently confirmed that the sample preparation, security, analytical procedures, and specific testing were consistent with generally accepted industry standards and adequate for the purpose of this study. Iron and arsenic content was determined using multi-acid digestion and ICP-AES technology, while sulphur and carbon were determined in a combustion furnace. Gold assays were conducted by fire assay using 30 g charges after sieving 100 g samples to reduce assay variability and remove "metallics."

Characterization of mineralized material competency for semi-autogenous grinding (SAG) milling (Table 13-1) was performed by G&T under the supervision of JK Tech Pty Ltd. (JKTech, 2011). The grinding circuit design was validated by simulation with SGS Minerals Services (SGS, 2011).

Table 13-1: Grindability Results on Composite Samples (G&T 2011)

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

To complete this grindability testing, a drop-weight test (DWT) was also performed on the 2011 bulk sample at SGS's laboratory to permit sizing of a SAG mill using JKSimMet software, licenced by the JKTech company. These results are shown in Table 13-2.

Table 13-2: DWT Results on Composite Samples (2011 Study)

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

The "A x b" component (Figure 13-1) characterises the competency of the ore to SAG milling, or its "resistance to impact fracture." A value of 30 is in the lower 12% of ores in the JKTech database and indicates a very difficult ore for SAG milling while a small "ta" parameter (Figure 13-2) indicates a material that is resistant to abrasion; this places the Rubicon ore in the second hardness class.

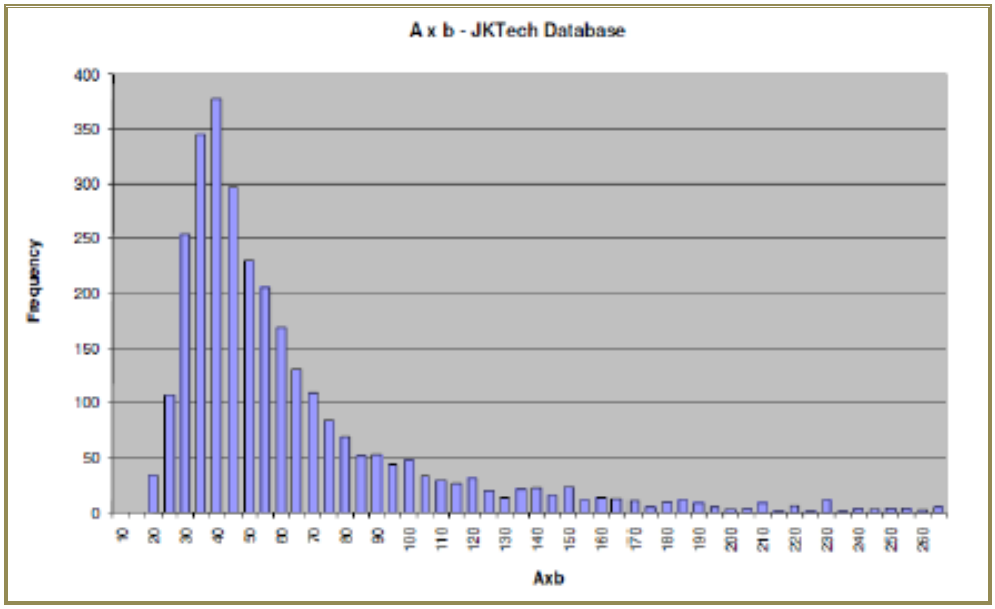


Figure 13-1: Frequency Distribution of A*b (JK Tech Database)

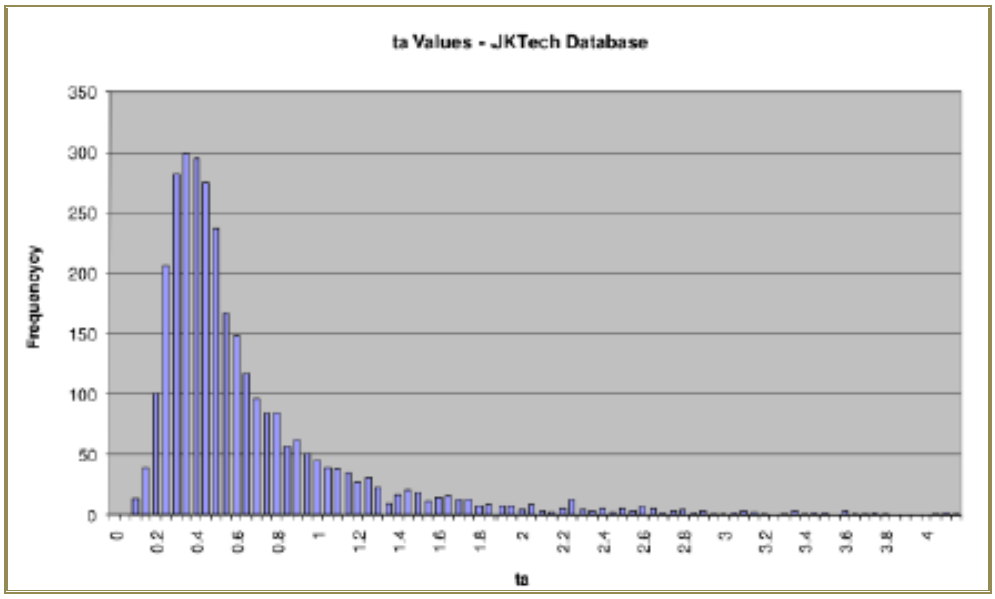


Figure 13-2: Frequency Distribution of ta (JK Tech Database)

Based on this data the preferred mill choice was for a single 20 ft x 10 ft SAG mill (6.1 m x 3.4 m), and a minimum 1,790 kW (2,400 horsepower [hp]) motor, with one 11 ft x 16 ft ball mill (3.4 m x 4.9 m) with a 597 kW (800 hp) motor. At 1,250 t/d the SAG mill can operate at a slower rotation speed and reduced ball charge, resulting in lower power consumption.

In July 2012, the processing of two approximately 1,000-tonne bulk samples was completed at Sabin Minerals Corporation (Canada) Ltd., McAlpine Mill, under the supervision of Soutex, to reconcile the bulk sample grades against the Mineral Resource Estimate (Soutex, 2013).

The process plant construction at the Phoenix site was initiated in 2013 and completed in early 2015. The process flowsheet constructed consisted of a SAG and a ball mill in closed circuit with hydrocyclones, gravity concentration, and CIL followed by carbon elution, electrowinning, cyanide destruction, paste preparation, tailing disposal, and refining. Doré is produced by smelting the gravity concentrate and electrowinning sludge. The cyanide destruction circuit using the SO_2/O_2 process was installed to reduce the cyanide levels in the tailings slurry prior to deposition in the TMF. The detoxified tailing will also feed the paste backfill preparation plant when underground operations require backfill.

The material handling systems, grinding, gold recovery, and refinery areas of the process plant were commissioned and operated between May and November 2015, processing a total of 57,793 tonnes of low-grade development material and mineralized material extracted from test stopes in the F2 Gold Zone. The average head grade to the mill in 2015 was 3.02 g/t.

The mill ceased operating on November 21, 2015 and was placed into care and maintenance. A total of 5,610 oz Au were produced. Grade and recovery values reported in the SRK Technical Report (SRK, 2016) have been adjusted to include clean-up ounces in the Mineral Processing and Metallurgical Testing section of the 2018 Technical Report (Golder, 2018b). Gold recovery achieved was 91.9%. This is consistent with the results obtained in the metallurgical testwork used by Soutex for process flowsheet development in the 2013 PEA (Figure 13-3). The cyanide destruction circuit achieved the target cyanide levels required at the TMF during operation in 2015.

In 2018, Rubicon Minerals prepared the mill to batch-process approximately 40,000 tonnes of mineralized material to be mined from three stopes within the F2 Gold Zone. The principal objective of the test was to verify the gold grades from the selected stopes relative to the updated 2018 Mineral Resource model (Golder, 2018b) This bulk test provided another opportunity to confirm the process capability of the mill. A metallurgical consultant who was experienced in commissioning new mills and restarting mills previously shut down was retained to supervise the preparation and operation of the facility for the bulk test program. Many deficiencies were found and corrected. Some deficiencies were legacy issues, either missed or left uncorrected in 2015. While the company attempted to shut down and place the plant in care and maintenance in an effective manner, there were some oversights and omissions. Care and maintenance focus was placed mainly on the grinding mills, which were rotated on a scheduled basis. As a result of the hurried 2015 shutdown, water pumps that were not drained, and sat idle for three years, experienced corrosion and required rebuilding. During the winter of 2015–2016, the mill building was not heated; some freezing damage occurred to some of the instrumentation but was corrected prior to start-up. The knowledge gained will benefit future start-ups and shut-downs.

Rubicon Minerals successfully batch-processed material mined from three stopes in the F2 Gold Zone along with 7,620 tonnes of low-grade material from various other sources. In total, 43,250 tonnes were processed, producing 5,669 oz Au. The 2018 bulk-sample processing campaign yielded an improvement in overall gold recovery, with the largest gain occurring in gravity gold recovery as compared to the 2015 operation. The 2018 campaign reinforces the conclusion that the mill at the Phoenix Project is robust and capable of processing the mineralized material extracted from the F2 Gold Zone at the 1,250 t/d design level and can achieve the 1,800 t/d design level with relatively minor downstream process changes. The batch test results also provide valuable data and information to support a conclusion that the basic design criteria produced by Soutex for an expansion to 2,500 t/d is realistic and achievable with the installation of the additional equipment.

13.2 Gold Recovery Estimates

Coarse gold is a frequent occurrence in the Red Lake area, so after processing the two 10-tonne subsamples, the gold particles in the gravity concentrate were measured and analysed. Four different recovery options were tested:

1. Gravity followed by rougher flotation
2. Gravity followed by cyanide leaching
3. Rougher flotation only
4. Cyanide leaching only.

The samples responded well to gravity separation indicating that the minerals and host rock were adequately separated at the design grind of 80% passing 105 µm. The detection of carbon in the ore presented some concerns and contributed to the decision to initially suggest a CIL circuit in case the ore became refractory, although this was later found not to be a problem.

Good recovery was achieved using flotation, but was marginally less than for cyanide leaching. The batch cyanidation leach tests were conducted over a 48-hour period and 94% of the gold was extracted; however, the extraction curve indicated that for this ore a 36-hour leach would achieve similar results at a reduced cost. Cyanide consumption was relatively low but was not optimized at this level of testing.

Given the significant effect of silver on gold recovery, it seems deficient that silver content generally has not been analysed for or reported, as it could affect which variant of gold recovery was selected post-cyanidation.

Using gravity separation ahead of cyanidation did not significantly improve overall gold recovery, but it generally reduced the required residence time in the leach tanks and reduced the risk of coarse gold particles being locked up in the circuit or being discharged to tailings by recovering the liberated gold early in the process It was thus determined that a combined gravity and cyanidation circuit was the

preferred option, and with the inclusion of a gravity circuit, it was decided that the cyanide should be added after grinding. This process is common in the industry, and thus training of operators and staff would be facilitated. Option 2 was therefore chosen, despite a slight cost increase for the two-stage process. These results are shown below in Table 13-3.

Table 13-3: Gold Recovery on Core Samples for Gravity-CIL Flowsheet (2010 Study)

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

This confirms that gold recovery is dependent on feed grade, but is within the expected accuracy of a PEA, and can be used to predict recovery within the range of feed samples that have been tested.

13.2.1 Projected Gold Recovery

The gold recovery results obtained from only two core samples (RL-01-01 and RL-01-02) were used to evaluate the average gold recovery, using gravity and cyanide leaching, for the 2013 PEA (SRK, 2013b); these results are presented in Figure 13-3. It should be noted that the gold grade used at that time for metallurgical testing was >5 g/t Au, while the mineralized material subsequently processed in the Phoenix mill in 2015 and 2018 was <5.5 g/t Au.

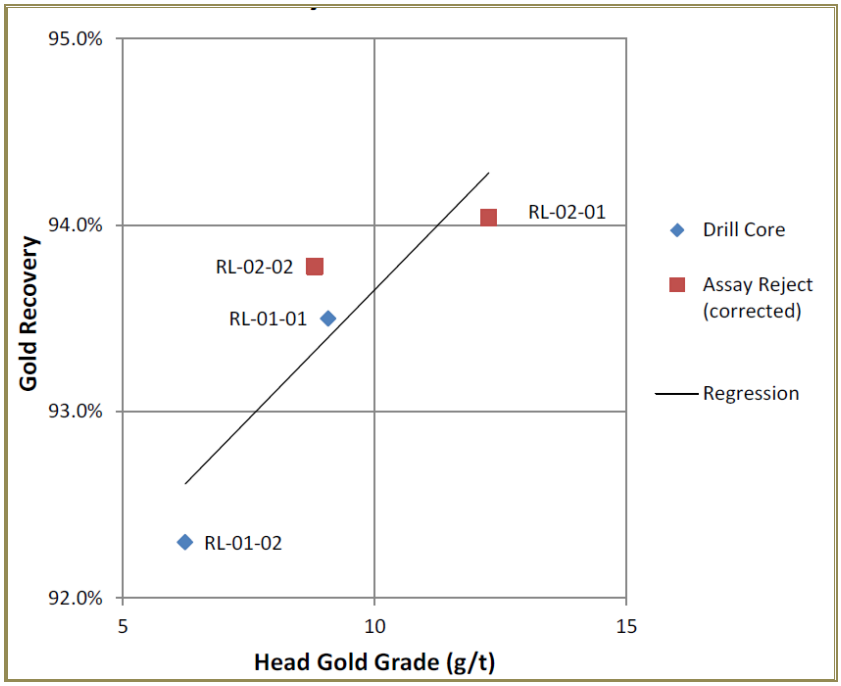


Figure 13-3: Effect of Head Gold Grade on Gold Recovery

13.2.2 Actual Gold Recovery Achieved During Operation in 2015

The mill commenced operation in May 2015 using gravity and CIL to recover gold. Operation of the mill was intermittent, as the mine could not sustain the designed daily feed rate of 1,250 t/d. Mill operation ceased on November 21, 2015, and the majority of the gold-locked inventory contained in the gravity and leach circuit was recovered during cleanup in 2016.

During commissioning and start-up of the process plant, the mill treated low-grade mineralized material mined during underground mine development. During initial operation, the ammonia levels in the TMF were greater than the allowable discharge limit. Pond levels were high. Two CIL tanks were repurposed to recover ammonia and enable effluent to be discharged from the TMF. Despite the loss of two stages of CIL, the actual gold recovery achieved from processing between May and December 2015 was 91.9% for the 57,793 tonnes of mineralized material from test stopes, grading an average of 3.02 g/t. This is consistent with the results obtained in the metallurgical testwork, which was used for the estimates in the 2013 PEA (Figure 13-3). In the 2013 PEA, the grade recovery relationship was developed from a small number of samples, with head-grades in a higher range than were delivered from the stopes mined. By extrapolating this curve into the lower head-grade range of the mineralized material milled in 2015, the expected recoveries fall into the 88% to 90% range. The recoveries achieved by the mill are relatively high, at 91.9% for the lower-grade material, when compared to the extrapolated grade recovery curve.

The final reconciled metallurgical data by month for the processing of test stopes is shown in Table 13-4. The total amount of gold recovered was 5,610 oz Au, including 741 oz Au recovered from the ball mill cleanup after operations ceased. Figure 13-4 displays the relationship between head grade and recovery, derived from actual plant data combined with metallurgical testwork data.

Table 13-4: Monthly Metallurgical Reconciliation for 2015 During Test Mining (including cleanup ounces)

Parameter	May	Jun.	Jul.	Aug.	Sep.	Oct.	Nov.	Dec.	Total
Mill Feed (dmt)	13,226	11,747	5,940	8,460	6,318	275	11,826	-	57,793
Gold Poured(oz)	0	742	448	570	738	0	1,915	-	4,412
Inventory Change (oz)	795	-227	116	319	-172	34	-370	-	740*
Change in Cathodes (oz)	0	178	94	49	100	0	-421	-	0
Gold in Tails (oz)	73	76	36	102	75	2	93	-	457
Gold in Mill Feed (oz)	868	769	693	1,041	740	35	1,217	-	5,610
Grade (g/t)	2.04	2.04	3.63	3.83	3.64	4.00	3.20	-	3.02
Recovery (%)	91.5	90.1	94.8	90.2	89.9	94.7	92.4	-	91.9

Note: *740 oz recovered in 2016 from partial mill cleanup included in gold accounting
Source: Rubicon internal document

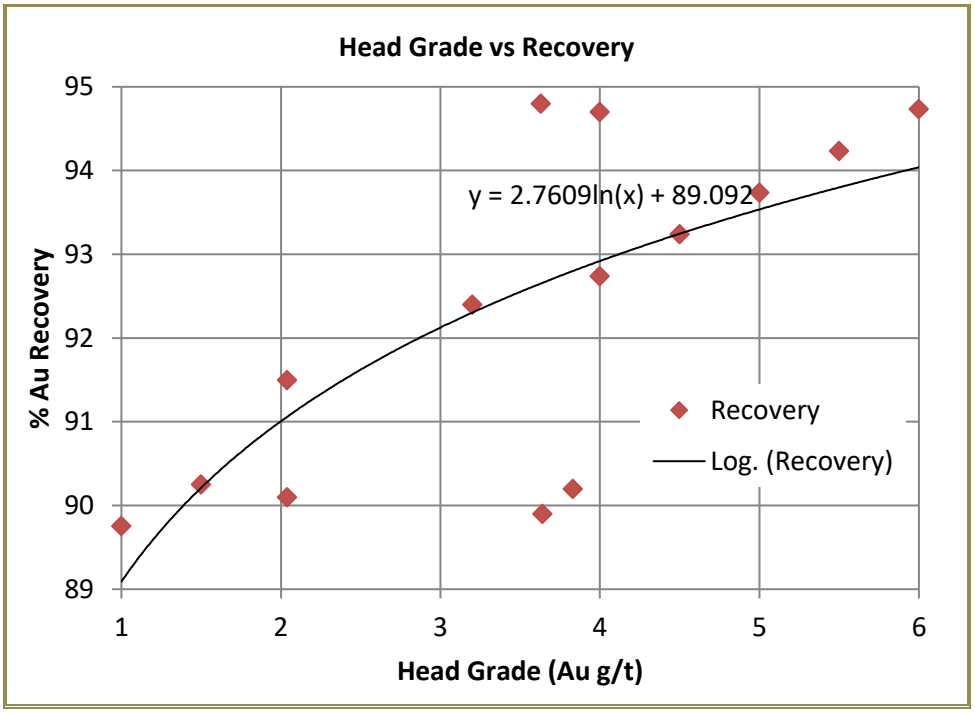


Figure 13-4: Head Grade and Recovery Derived from 2015 Actual Plant Data and Metallurgical Testwork Data

13.3 Mill Feed Sources – 2015

During commissioning and start-up of the process plant, the mill treated low-grade mineralized material mined during underground mine development. This is standard practice in commissioning a new facility. The main source of feed was from the HW and West Limb Basalts of the F2 Gold Zone. Several stoping areas (305-Z2-030, 305-Z2-489, 305-Z1-065, 305-Z1-980, 244-Z2-159FW, 244-Z1-994, 183-Z1-161, and 183-Z1-164) were developed in preparation for mining, accounting for 61% of the mineralized material milled. The balance of mill feed was mined from four test stopes (305-Z2-030, 305-Z1-065, 244-Z1-977, and 244-Z1-994). Development muck in the F2 Gold Zone accounted for 17% of the mineralized material milled. The remaining 22% of tonnes milled was waste rock that entered the system while mining low-grade mineralized material. The primary mineralized material sources are shown in Figure 13-5.

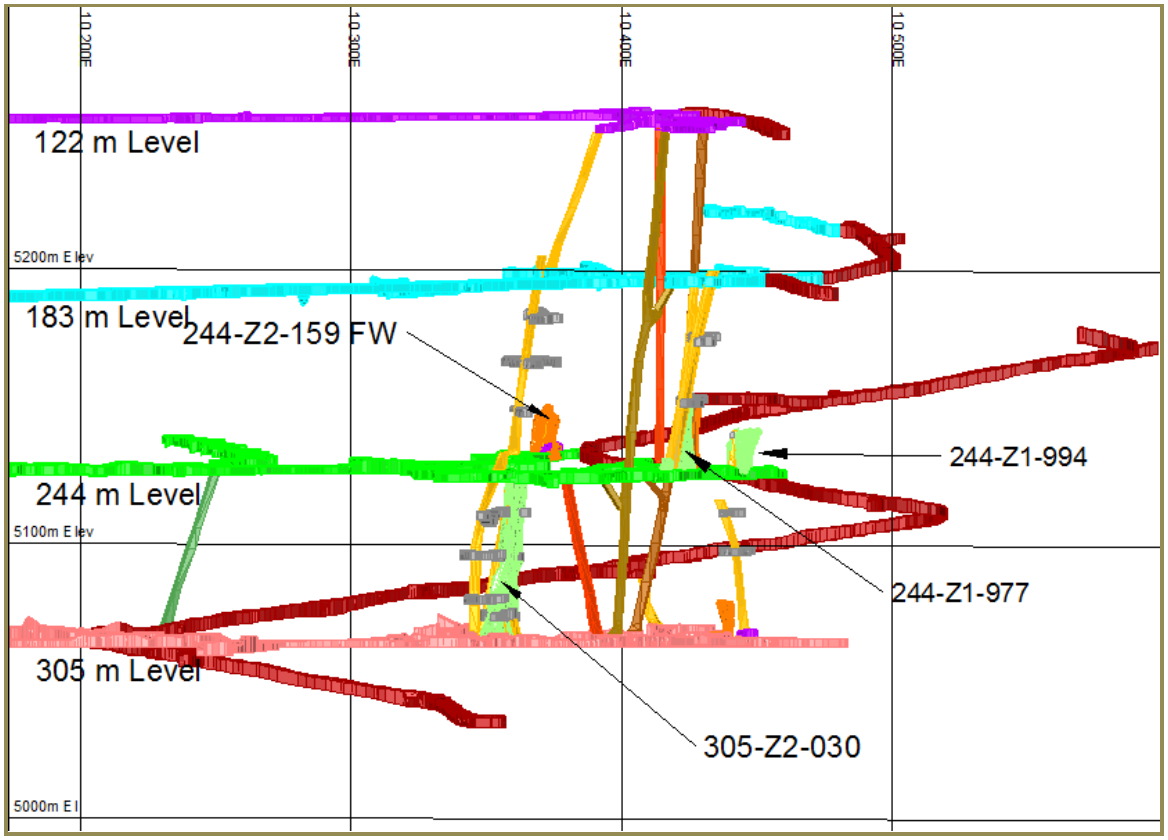


Figure 13-5: Mineralized Material Sources 2015, Looking North (Rubicon, 2019)

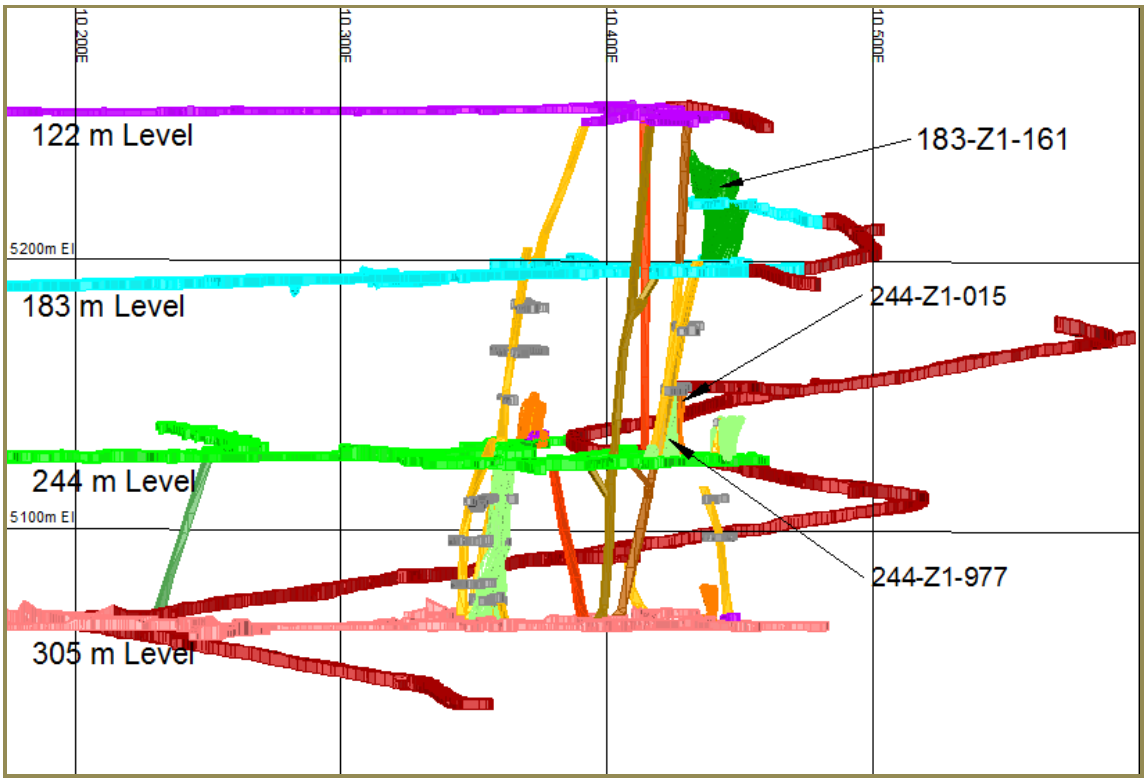


Figure 13-6: Mineralized Material Sources for 2018 Bulk Test, Looking North (Rubicon, 2019)

13.3.1 Actual Gold Recovery Achieved In 2018 Bulk Test

Rubicon continued its exploration program through 2017 and into 2018. The ultimate goal was to verify the 2018 geological model by batch processing approximately 40,000 tonnes of mineralized material from selected test stopes in the F2 Gold Zone. A rigorous plan was prepared to lay out the stopes to be mined within the F2 Gold Zone, extensively sample and survey the material mined from the stopes, and ensure that this material was kept segregated through to and during milling. Rubicon retained the services of a metallurgical consultant who was very experienced in starting up and operating mills that were newly constructed or had been shut down for extended periods. When operations ceased in 2015, Rubicon implemented a plan for placing the milling facility under care and maintenance. This included turning the heat off in the mill building. The mill building was used as a store for mine equipment during the shutdown period and some equipment and instrumentation was damaged by freezing. This was revealed by inspections in early 2018, which also showed that some remaining legacy deficiencies from the original commissioning and operation. Other issues identified were the result of lines and tanks not being completely drained. Most notably, the freshwater pumps were not drained corroded and required a rebuild. Some elements in sensors and instrumentation have limited service lives and had to be replaced. The belt scales were recalibrated and verified by dump tests, and the material handling system was modified as needed to maintain steady feed rates and control spillage.

The grinding circuit was bedded in with a small amount of waste and low-grade material prior to commencing the batch processing of mineralized material from the test stopes. The mill was operated and maintained by a combination of contract operators and maintenance personnel along with some direct hires. Basic assay services to support the operation, with a 24-hour turnaround, were provided by the SGS laboratory in Red Lake, while the more complex analyses such as loaded carbon and gravity concentrate samples were performed by SGS Lakefield. Turnaround times were not ideal for the operation, but the data received was adequate for metallurgical balancing and reconciliation. A basic cleanout of sumps and easily accessible gold traps, as well as a circuit inventory were performed after material from each stope was processed. A more comprehensive cleanup was completed after the final bulk sample was processed. The cleanup gold ounces recovered were allocated to the test stopes on a pro rata basis. The metallurgical results of the bulk test program are summarized in Table 13-5. The test stopes were identified as 244-Z1-015, 244-Z1-977, and 183-Z1-161. The processing results for each source of mill feed are shown separately, but for block model reconciliation the data for Stope 244-Z1-161 is combined. It is important to note that the gold recovery from each of the test stopes was consistently higher than the recovery from the other mineralized material.

Table 13-5: Metallurgical Results of the 2018 Test Mining Program

	Total	Low Grade (977 & 015)	244-Z1-015	244-Z1-977	183-Z1-161 Down Hole	244-159 FW LH	183-Z1-161 Sump	LG Dev (Sump)	244-Z1-161 Upper
Grade (Au g/t)	4.08	1.47	3.66	3.55	4.98	2.34	2.22	2.99	5.76
Recovery (%)	94.8	98	95.5	94.7	93.4	87.1	86.1	92.9	96.7
Tonnes	43,250	3,396	6,230	10,394	9,093	1,542	1,206	1,477	9,412
Gold (oz)	5,669.3	160.7	792.3	1,184.6	1,455.6	115.9	86.1	142.1	1,732

During the 2018 bulk test, the Phoenix mill demonstrated that it could efficiently process mineralized material mined from the F2 Gold Zone. Average gold recovery in 2018 was notably higher, at 95.1% than the 91.9% achieved in 2015. Gravity gold recoveries ranged from 40% to 50%, and gold recoveries from the test stopes ranged from 93.4% to 96.7%. In 2015, monthly gold recoveries ranged from 89.9% to 94.8%. The recovery improvement between 2015 and 2018 is significant. This overall recovery improvement is primarily attributed to the gravity gold circuit performance. Deficiencies were found in the Knelson concentrator installation that would have prevented it from working optimally in 2015. Changes were also made in primary hydrocyclone control that stabilized the slurry feed to the Knelson concentrators. The impact of reducing the gold load reporting to the leach circuit, combined with having all six CIL tanks available for leaching as originally designed increased the leach retention time. Reagent requirements were also reduced and the reagent addition controls improved. Lead nitrate was added to the leach circuit in 2018, but had not been used in 2015. Lead nitrate is generally added to leach circuits to mitigate the effect of sulphide ions on the leaching of gold. Lead nitrate was included in the reagent regime in the plant design criteria. A grade recovery trend was generated from the operating data and is shown in Figure 13-7.

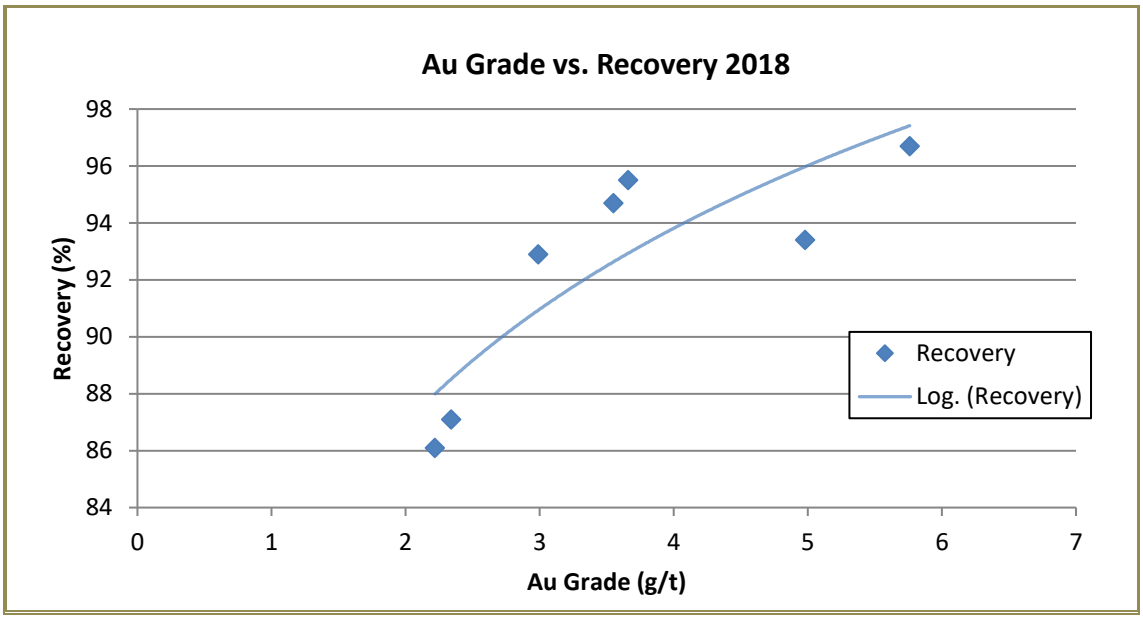


Figure 13-7: Gold Grade vs. Recovery for 2018 Test Mining

13.4 Plant Operating Data

Table 13-6 summarizes key plant operating data from 2015 and 2018. Significant improvements in metallurgical performance were achieved in 2018. These were the result of having a more stable feed to the SAG mill, finding and correcting legacy issues in the Knelson concentrator installation during the pre-start-up period, having improved control of the cyclone pack which fed the gravity concentrators and optimizing the collection and discharge periods during operation. Other metallurgical improvements were the result of having all six CIL tanks available for leaching and the addition of lead nitrate to the leach circuit. In 2015, the Knelson concentrators were operated, but did not perform as expected. Having a functioning gravity circuit in 2018 reduced the gold load reporting to CIL. That combined with having all six CIL tanks available improved CIL recovery. In 2015, two CIL tanks were dedicated to ammonia reduction and were not available for leaching.

Table 13-6: Test Mining Results

Description	Unit	2015	2018
Tonnes Milled Dry	dmt	57,793	43,250
Total Production	oz Au	5,610	5,669
Gravity Production	oz Au	0	2,421
Recovery	%	91.9	94.8
Head Grade	g/t Au	3.02	4.08
Tailings Grade	g/t Au	0.25	0.21

The QP of this section, Peter Broad, considers evaluation of actual plant operating data to be the best indicator for future recovery estimates; however, minor changes may be required to improve safety and dependability of operations.

13.5 Additional Improvements in Gold Recovery

The metallurgical results achieved in 2018 were significantly better than those achieved in 2015, and the feed ore was kept under 150 mm (6"). As a result, 95% recovery from mineralized material grading between 3.5 g/t Au and 5.0 g/t Au is achievable with the current milling circuit. Replacement of the gravity table with intensive leaching of gravity concentrates should improve security and facilitate the movement of this material. Installing the leach tanks omitted in the 2015 construction is recommended, and converting the CIL to CIP should be considered in future if carbon attrition, a major contributor to gold losses, and reagent consumptions are higher than expected. It is anticipated that with better knowledge and understanding of the gold deportment in the various zones of gold mineralization, incremental increases are possible. With a stable steady-state ore supply grading a nominal 5 g/t Au, combined with continuous operational improvements, gold recoveries >95.0% may be realized in future years of operation.

13.6 Statement of Representativeness of Samples in the 2015 Operation

The mineralized material processed in 2015 was test mined from the F2 Gold Zone. Metallurgical testing and process flowsheet development were based on a small number of samples and two 1,000 tonne bulk samples that were custom processed. Metallurgical testing was conducted on samples with gold grades higher than those delivered to the mill during the 2015 milling campaign. The grade to the mill averaged 3.02 g/t Au, which was much lower than the development testwork, which ranged from 5 g/t Au to 15 g/t Au. Grade recovery data generated during mill operation in 2015 was incorporated into the grade recovery curve developed for the Project. The head grade and recovery results obtained from 2015 mill operation have been adjusted to include actual ounces recovered from the partial mill cleanup. The mineralized material milled was lower-grade than the head grades used in the development testwork, but the relationship held and could be considered robust considering that 57,793 tonnes were milled.

In 2015, mill operations were intermittent and leach capacity was sacrificed, as two leach tanks were temporarily repurposed to reduce ammonia concentrations to acceptable discharge levels. At start up, the TMF ponds contained water with extremely high ammonia concentrations that could not be discharged to the environment.

In 2015 the Phoenix mill never achieved steady-state operation under optimal, controlled conditions. The expectation is that under optimum operating conditions, at steady state, gold recoveries should be higher than those achieved in 2015.

The metallurgical QP interviewed the senior mine geologist and relied on information supplied by Rubicon personnel that the source of the material milled in 2015 was from the F2 Gold Zone.

13.7 Statement of Representativeness of Samples in the 2018 Bulk Test

The mineralized material processed in 2018 was mined from three test stopes in the F2 Gold Zone (244-Z1-015, 244-Z1-977, and 183-Z1-161) and low-grade material from previously developed areas. The material was extensively sampled as it was mined and transported to the surface stockpiles. Additional details are found in the mining section. No laboratory metallurgical testing has been conducted on samples from this deposit since 2011. The two milling campaigns, however, have provided sufficient data to conclude that the basic design criteria of the Phoenix mill can be achieved. The initial response of the material milled in 2015 demonstrated that the mill, as configured, was capable of extracting gold from the F2 Gold Zone.

Ore hardness was typically below design and the SAG mill starting charge was 4.8% v/v (less than half the design at 72.5% of critical speed and 15,000 kPa bearing pressure). Maximum tonnage peaked at 90–95 t/h before being reduced to an average of 75 t/h (1,800 t/d). The METSO Poly-Met ½-in grates were more than adequate for this operating range. This high tonnage operation resulted in a 25% reduction of the design ball charge and a 54% reduction in connected SAG mill power and indicated there were opportunities for improving gold extraction and optimizing reagent usage.

The head grades processed in 2018 ranged from 1.47 g/t Au to 5.36 g/t Au, with 82% of the mill feed between 3.55 g/t Au and 5.36 g/t Au. The head grade and recovery results obtained from 2018 mill operation include actual gold ounces recovered from the partial mill cleanup. The gold recovery achieved from processing the material mined from the bulk test stopes was higher than the recovery attained during metallurgical testing. This was also the case in 2015 when the mill first operated. The average head grade in the 2018 bulk test was higher, at 4.08 g/t Au compared to 3.02 g/t Au in 2015, but still fell into the low end of the range of metallurgical test sample grades (5 g/t Au to 15 g/t Au). The 2018 bulk test confirms that the grade recovery relationship developed during metallurgical testing is conservative, as the mill has been able to meet or exceed the predicted recovery at lower head grades.

The metallurgical QP met with Rubicon management and relied on information regarding sources and grade of mineralized material provided during those meetings. The QP also visited the mill and discussed the work managed by Adrian McNutt, the metallurgical consultant for the bulk test.

13.8 Factors with Possible Effects on Potential Economic Extraction

13.8.1 Main Process Equipment

For the operation of the grinding circuit at 1,250 t/d, it was expected that the SAG mill would be operated at a lower speed with a reduced ball charge. This was experienced in the early stages of the

2015 test mining and milling operation. During the bulk sample testing in 2018, the SAG mill ball charge was managed in line with the design criteria. The mills were test run to determine the grinding capacity of the installed mills and identify downstream limitations. Although the mill is currently able to process up to 1,250 t/d, the grinding mills are capable of processing approximately 1,800 t/d with minor equipment changes and additions required in the downstream processes. The mill layout allows for the addition of a second ball mill, a second hydrocyclone cluster, and a second stripping column that would allow processing of up to 2,500 t/d.

Additional incremental improvements in the plant can be expected in the future. The mill would benefit from having dedicated assay services available at its metallurgical lab in Balmertown or on site, to provide timely feedback to the operators. The nominal 24-hour turnaround time is acceptable for the short-term operation but does not allow for efficient troubleshooting.

A mineralogical study of the tailings generated could provide information that could identify potential improvements in recovery or reduce costs. It is important for the metallurgist to understand the deportment of the gold lost in tailings and testing could provide further additional information. Liberated gold is reporting to the tailings suggesting an opportunity to optimize grind by improvements to the gravity circuit.

The design paste backfill requirement is 55% by weight of the mill feed. For 1,250 t/d operation the requirement is 687.5 tonnes at 80% solids. One disc filter should meet the operating requirements at 1,250 t/d, with the second unit on standby. The second disc filter could be used to increase paste production to 990 t/d to meet the underground requirements at 1,800 t/d if needed. There is provision to install a third disc filter; the decision to install this filter could be deferred until a definite need for additional filtering capacity is demonstrated by operation of the installed equipment. This additional capacity could either be used to meet increased paste backfill requirements for 2,500 t/d operation or to produce dewatered tailings. Additional paste mixing and paste pumping capacity will be required.

13.8.2 Plant Tailings Toxicity

CIL plant tailings are treated using the SO_2/O_2 cyanide destruction process, and in the 2015 campaign cyanide levels, less than 1 ppm were consistently achieved.

In 2018, sodium metabisulphite (SMBS) was used as the oxidant rather than liquid SO_2. The CN destruction targets of cyanide levels less than 5 ppm were achieved. The two operational campaigns demonstrated that either liquid SO_2 or SMBS are viable at Phoenix.

13.8.3 Tailings Management Facility Effluent

The cyanide in the CIL tailings is destroyed using the SO_2/O_2 cyanide destruction process. Cyanate ions are produced as a product of the destruction process. The cyanate breaks down, producing ammonia. Ammonia is a regulated discharge parameter that must be kept within provincially regulated limits.

During initial operation in 2015, ammonia concentrations in the TMF exceeded the allowable discharge limits. The sources of ammonia were identified to be the cyanide destruction circuit, and the mine water, which was pumped from underground to the TMF. To comply with discharge limits, Rubicon implemented several mitigation measures; these include eliminating the use of ANFO explosives underground, locally treating mine water with zeolite in the mine, and using two tanks in the mill CIL circuit as reactors to create a temporary ammonia removal system using zeolite to lower ammonia to meet discharge limits. As a result, 91,237 m^3 of TMF effluent was successfully treated and discharged to the environment between September 2015 and November 2015.

In 2016, 66,281 m^3 was discharged from a variety of runoff collection ponds on the property, with the permission of the provincial environmental authority. This allowed for some dewatering to occur, despite ammonia levels in the TMF remaining above approved discharge limits. Sewage sludge was added to the pond to increase bacterial degradation of ammonia, and this increased the rate of destruction. In 2017, ammonia levels fell below discharge limits, and 221,158 m^3 of water was discharged from the TMF, returning the water elevation to levels not seen since mid-2015.

In 2018, 22,012 m^3 was discharged from the TMF. Discharge only occurred for a short time, late in the season due to the upgrades being done to the water treatment plant (a metals treatment component was added in 2017 and commissioned in 2018). As was the case in 2015, the cyanide destruction circuit contributed a significant amount of ammonia to the TMF. Consecutive ammonia samples in effluent surpassed the discharge objectives, though the plant had already been shut down and discharging to the environment had stopped when the last sample results were received.

The existing TMF is adequate for the short term and near future. An Actiflo® system has been installed to ensure compliance with effluent discharge limits, and a metals removal circuit was added in 2017 and commissioned in 2018. The sourcing of an acceptable ammonia treatment system remains a long-term objective. Mitigation measures taken to manage ammonia at the source (such as eliminating ANFO use in the mine) have been effective in lowering ammonia contributions to the TMF. The additional ammonia generated from cyanide destruction during short-term mill operating campaigns can be managed naturally. An ammonia treatment system may be required during continuous mill operation.

13.9 Comment on Mineral Processing

The Mineral Processing QP is of the opinion that the mill was operated in accordance with generally acceptable industry standards. Belt scales were routinely calibrated, while standard sampling and metallurgical account practices were followed; (as examples of acceptable standards). The recovery

estimates are appropriate for the mineralized material extracted from the High-Ti Basalts within the F2 Gold Zone. The accuracy of the future recovery and production estimates for the Phoenix project will depend on the mine's ability to predict the grade of the mineralized material that will be milled.

The QP is not aware of any processing factor or deleterious elements in the mineralization of the current Mineral Resource estimate that could have a significant effect on potential economic extraction of gold at the Phoenix project. The author is aware of an alteration zone in the area, which, if mined in the future would require metallurgical testing to determine potential metallurgical performance and determine process parameters specific to treating this material.

In the event that subsequent mine exploration should discover mineralization outside of the F2 Gold Zone, metallurgical testing will be required to determine the impact on the performance of the process plant. Mitigating actions, such as changes in equipment or procedures, can then be implemented prior to processing.

Future mill operation will benefit from providing basic on-site analytical capability to allow operators to make informed operating decisions in a timely manner.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

In March 2019, TMAC prepared a new Mineral Resource estimate for the Phoenix Gold Project. The 2019 Mineral Resource estimate for the Phoenix Project consists of Measured, Indicated, and Inferred Resources (the 2019 Mineral Resource Estimate). Mr. Tim Maunula, P.Geo., Principal Geologist for TMAC, was the QP responsible for the completion of the 2019 Mineral Resource estimate for the Phoenix Gold Project. The effective date of the 2019 Mineral Resource Estimate is March 18, 2019.

The 2019 Mineral Resource Estimate is based on drill hole data provided by Rubicon from surface and underground diamond drill programs completed between 2008 and 2018. The cut-off date for assay data used in the 2019 Mineral Resource Estimate was December 17, 2018. All data received was in the local grid coordinate system for the Phoenix Gold Project, which is rotated 45 degrees to the east of magnetic north, with grid coordinates 10000E, 50000N corresponding to the location of the Phoenix shaft.

The gold mineralization for the Phoenix Gold Project was modelled in four zones of mineralization, Zone 1 through Zone 4. The stratigraphy, or rock type groups, provided was modelled based on a nearest-neighbour indicator for the three principal rock types: High-Ti Basalt, Ultramafic, and Felsic Intrusive. TMAC updated the interpretation based on the new drilling completed post-2017. Gold grades were estimated by rock type within each zone separately.

The software used for the 2019 Mineral Resource Estimate was Geovia GEMS 6.7.3 Desktop (GEMS). The Mineral Resource Estimates were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). The 2019 Mineral Resource Estimate was reported at a 3.0 g/t Au cut-off grade for Mineral Resources which are amenable to underground extraction.

14.2 Database

The 2019 Mineral Resource Estimate for the Phoenix Gold Project is based on diamond drill hole data consisting of gold assays, geological descriptions, and density measurements. Underground development and bulk sample stope data were also taken into consideration.

Data was provided to TMAC by Rubicon in electronic formats—CSV and DXF files—and imported into GEMS. The database was additionally verified using the validation tool in GEMS to determine errors and overlapping or out-of-sequence intervals. Minor errors were noted, and the database was updated.

The drill hole database received from Rubicon consisted of 1,631 drill holes totalling 551,811 m of core drilling. The database includes all drilling on the Phoenix Gold Project in proximity to the interpreted mineralization zones. The historical data of the McFinley deposit was not included in the database for the 2019 Mineral Resource Estimate. This current Mineral Resource Estimate included an additional 106 drill holes (Table 14-1) that were drilled post-2017. This selection consists of 104,308 gold assays that were used for modelling Mineral Resource estimation.

Table 14-1: Drill Hole Header Table Coding

MA Code	Count	Description
-1	25	Drill holes within mineralization zones but excluded from Mineral Resource
0	187	Drill holes outside mineralization zones
2	1313	Drill holes used in 2018 Golder Mineral Resource
3	106	Additional drill holes used in 2019 TMAC Mineral Resource

As recommended by Golder (2018) in the previous Technical Report, TMAC also excluded 20 surface and five underground drill holes. These drill holes were excluded to minimize potential bias. Table 14-2 lists the drill holes excluded from this current Mineral Resource estimate.

Table 14-2: List of Drill Holes Excluded from the Mineral Resource Estimate

Drill Hole Numbers			
F2-01	F2-09	F2-41	244-09-04
F2-02	F2-10	F2-42	305-05-HQ1
F2-03	F2-11	F2-57	305-18
F2-04	F2-12	F2-60B	305-29
F2-06	F2-13	F2-61B	
F2-07	F2-21	F2-2012-06A-W1	
F2-08	F2-22	244-09-03	

14.2.1 Bulk Density

A total of 5,982 SG measurements were provided from on-site drill core measurements. A full description of the measurement process is provided in Section 11.4.

An additional 50 SG measurements were collected in 2017 to confirm the density values assigned in the block model (Section 11.4). Check-sample results confirmed the mean density values used (Table 14-3). Density values were assigned in all zones in the block model based on the rock type unit.

Table 14-3: Comparison of Bulk Density (t/m³) Values

Rock Type Unit	Assigned Value	As Logged	2017 Check	Based on 2019 Rock Type Update
High-Ti Basalt	2.96	2.96	2.95	2.94
Ultramafic	2.90	2.90–2.92	2.92	2.90
Felsic Intrusive	2.67	2.67	2.62	2.70

14.3 Geological Domaining

The gold mineralization for the Phoenix Gold Project is enveloped within four zones: Zone 1 to Zone 4. Gold mineralization within the deposit is primarily hosted within the High-Ti Basalt units; however, gold mineralization also occurs within Ultramafic units and Felsic Intrusive rocks.

Rubicon identified three main basalt lenses hosting mineralization, including, from west to east: Hangingwall Basalt, West Limb Basalt, and F2 Basalt. These lenses make up most of the mineralization contained within Zone 1 (F2 Basalt) and Zone 2 (Hangingwall and West Limb Basalt). Zone 3 is a small, narrow zone primarily in Felsic Intrusive rocks between the F2 Basalt and West Limb Basalt, Zone 1, and Zone 2, respectively. Zone 4 includes an extension of F2 Basalt to the north of the main mineralized area of Zone 1.

Additionally, a small zone of High-Ti and Ultramafic rocks were modelled slightly outside to the east of Zone 2 (at approximately 4,600 m elevation). This small area of gold mineralization was attributed to Zone 2.

Figure 14-1 presents the four mineralization zones of the Phoenix Gold Project system.

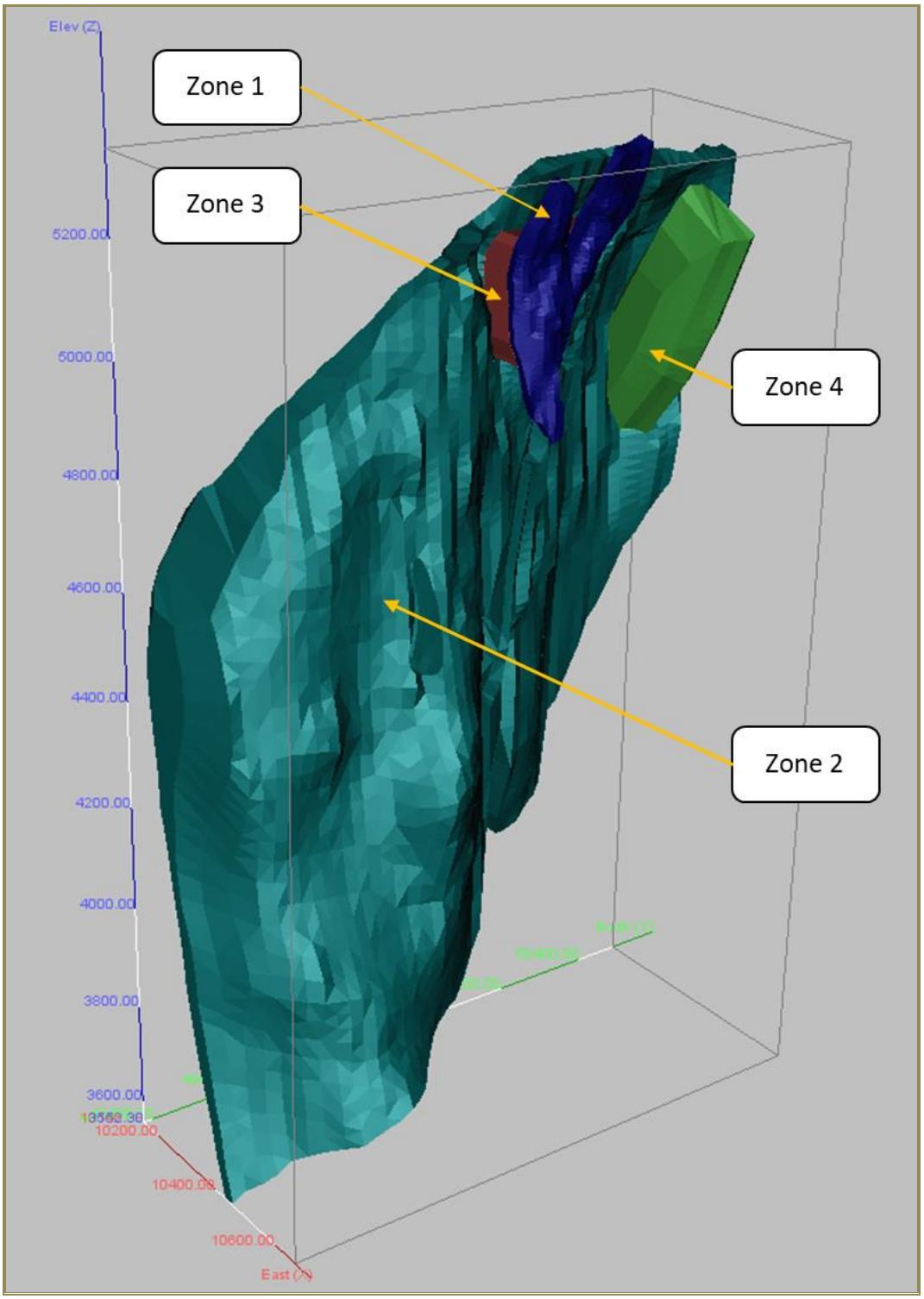


Figure 14-1: Mineralization Zones (Oblique View Facing Northwest)

The rock units comprising these mineralized zones have been subjected to deformation, and as a result, have complex shapes and distributions with variable continuity. Due to contrasts in the physical characteristics (competency contrasts) between rock units, as described previously in Section 7, the soft and locally talcose-altered ultramafic unit is interpreted to have deformed in a ductile manner around more competent basalt and felsic units which are believed to have deformed in a brittle–ductile manner and possibly pulled apart (boudinaged) locally. Due to the complexity of the host rock units and associated mineralization, Golder (2018b) chose to model mineralization domains as broad low-grade envelopes based on the three main lithologies. TMAC reviewed the geological model and updated it based on the new drilling information provided post-2017.

Table 14-4 presents the zone and rock type codes assigned to the block model for the Phoenix Deposit.

Table 14-4: Codes Used for the Phoenix Deposit

Zone Code	Rock Type	Description	ZR Codes
1	7	Ultramafic	107
1	9	High-Ti Basalt	109
1	17	Felsic Intrusive	117
2	7	Ultramafic	207
2	9	High-Ti Basalt	209
2	17	Felsic Intrusive	217
3	7	Ultramafic	307
3	9	High-Ti Basalt	309
3	17	Felsic Intrusive	317
4	7	Ultramafic	407
4	9	High-Ti Basalt	409
4	17	Felsic Intrusive	417

Figure 14-2 illustrates the geology block model for 5,100 m elevation.

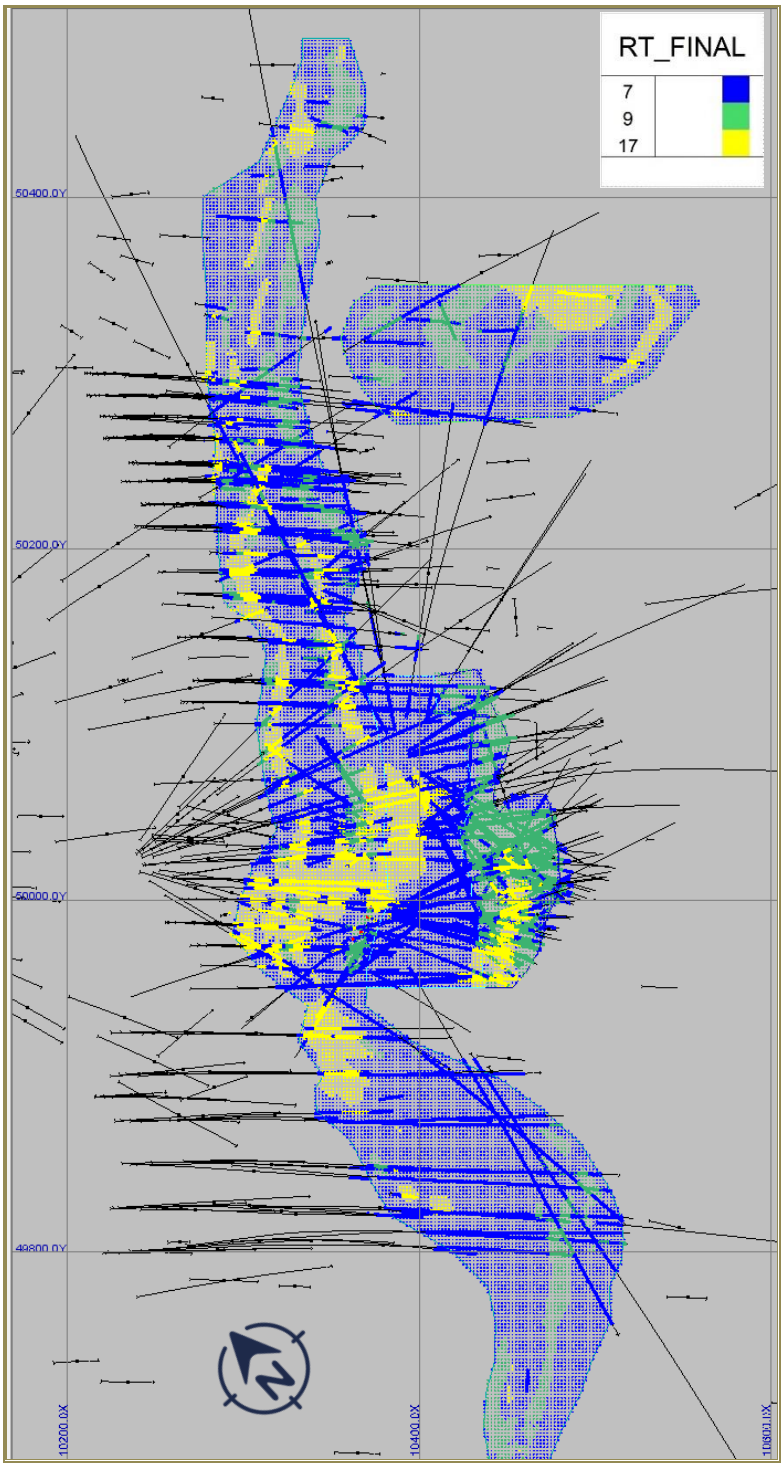


Figure 14-2: Geology Model Plan View (5,100 m Elevation)

14.4 Exploratory Data Analysis

14.4.1 Raw Assays

The Mineral Resource model only includes gold assays from drill holes. Capped and uncapped grades are reported in the exploratory data analysis. Analysis of the gold assay values was conducted on raw drill-hole data selected by mineralization zone and lithology to determine the nature of the gold grade distribution and correlation of grades with individual rock units. Through a combination of descriptive statistics, histograms, probability plots, and box plots, the gold grade values were reviewed to analyze the gold data.

Table 14-5 provides a summary of the descriptive statistics for gold and assay length for the raw sample populations captured from within each mineral zone. Zone 1 and Zone 2 contain most of the assay data. The average assay sample length is slightly less than one metre.

Table 14-5: Descriptive Statistics of Raw Assay Data by Zone

Zone	Unit	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Std. Dev.	CV
1	Au (g/t)	21,407	0.0025	2,305.23	2.41	29.39	12.19
	Length (m)	21,407	0.10	2.80	0.85	0.21	0.25
2	Au (g/t)	74,013	0.0025	3,194.65	1.44	26.01	18.08
	Length (m)	74,013	0.08	6.90	0.90	0.20	0.22
3	Au (g/t)	6,242	0.0025	185.26	0.63	4.21	6.71
	Length (m)	6,242	0.20	2.00	0.94	0.16	0.17
4	Au (g/t)	2,646	0.0025	457.43	0.65	9.47	14.55
	Length (m)	2,646	0.27	1.34	0.92	0.17	0.19

Notes: Std. Dev. = Standard Deviation; CV = Coefficient of Variation

Table 14-6 discriminates by rock type within each zone. The coefficient of variation (CV) is high (>2) and indicates the potential for estimation bias using linear estimation methods. Capping of outliers is one method to manage the CV.

The highest average gold grade is reported for High-Ti Basalt in Zone 1 and Zone 2. However, within Zone 2, the Ultramafic and Felsic Intrusive units have high-grade outliers in excess of 2,000 g/t Au.

Table 14-6: Descriptive Statistics of Raw Assay Data by Rock Type per Zone

Zone	Rock Type	ZR Code	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Std. Dev.	CV
1	7	107	4,338	0.0025	895.54	1.29	15.26	11.86
	9	109	15,175	0.0025	2,305.23	2.95	33.90	11.48
	17	117	1,863	0.0025	111.19	0.63	3.31	5.27
2	7	207	24,762	0.0025	2,581.67	1.27	29.99	23.57
	9	209	21,411	0.0025	2,620.70	2.14	26.69	12.49
	17	217	27,756	0.0025	3,194.65	1.05	21.26	20.29
3	7	307	3,166	0.0025	185.26	0.52	5.32	10.14
	9	309	30	0.0025	5.96	0.59	1.38	2.34
	17	317	3,046	0.0025	86.65	0.73	2.62	3.57
4	7	407	994	0.0025	73.94	0.24	2.43	10.19
	9	409	1,227	0.0025	114.53	0.71	4.13	5.81
	17	417	425	0.0025	457.43	1.44	22.25	15.45

Notes: Std. Dev. = Standard Deviation; CV = Coefficient of Variation (See Table 14-4 for Rock Type Code)

Box plots are shown in Figure 14-3 and Figure 14-4; these illustrate the grade distribution graphically.

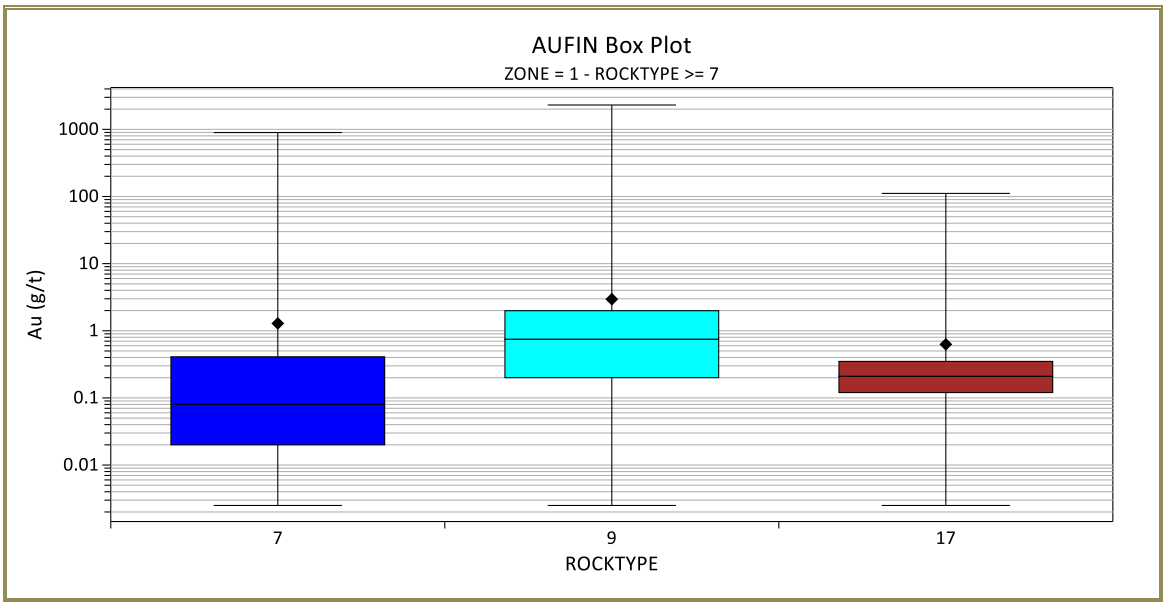


Figure 14-3: Box Plot of Rock Type for Zone 1; Uncapped Gold (g/t Au)

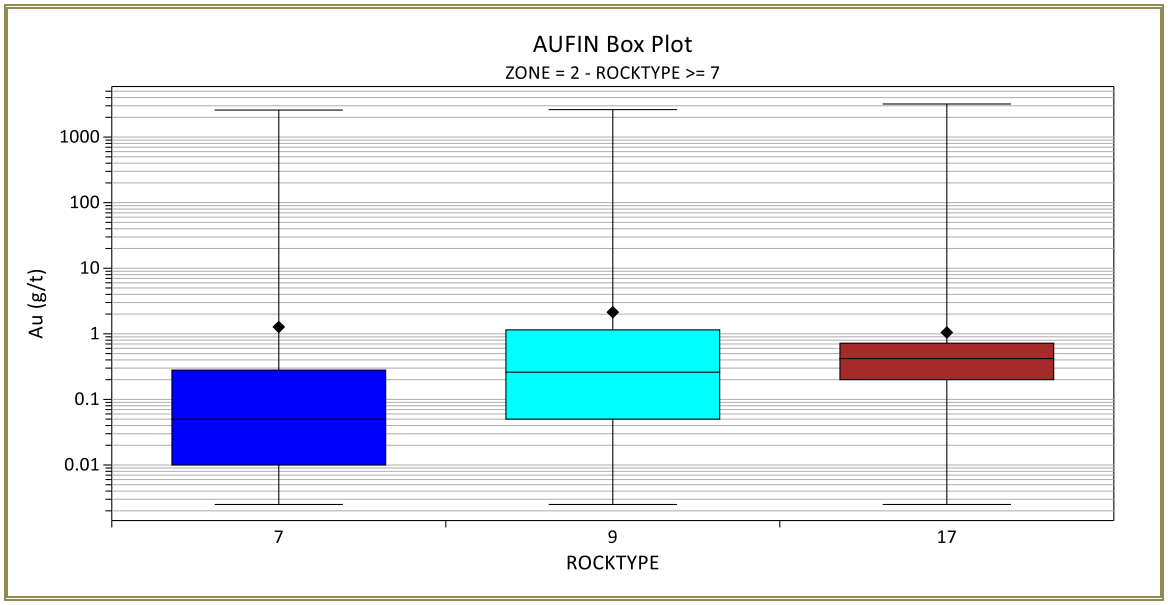


Figure 14-4: Box Plot of Rock Type for Zone 2; Uncapped Gold (g/t Au)

14.4.2 Compositing

For purposes of normalizing the assay data for further analysis, the raw assay values were composited to 1 m and 2 m intervals within the interpreted mineralized zone wireframes. Composite lengths were adjusted to avoid short remnants on the hanging wall or footwall contacts of the mineralized zones by equalizing the composite length within the interval. Unassayed intervals were assigned a grade of 0.0025 g/t Au, which is half the lower detection limit. Composite values were then tagged by zone, rock type, and ZR (combination of zone and rock type) codes. Capping analysis was carried out on the 1 m composites. The 2 m composites were used for the NN interpolation and the 1 m composites for inverse distance and OK.

Table 14-7 shows the descriptive statistics of the uncapped 1 m composite values by mineralized zone. Descriptive statistics for gold composite grades by zone and rock type are shown in Table 14-8.

Figure 14-5 and Figure 14-6 present the box plots of the capped 1 m composite values by mineralized zone and rock type for Zone 1 and Zone 2. In general, the CV value was lower for the uncapped composites. In some cases, the CV increased because of the unassayed intervals being assigned 0.0025 g/t Au.

Table 14-7: Descriptive Statistics of 1 m Composite Values (uncapped) by Zone

Zone	Unit	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Std. Dev.	CV
1	Au (g/t)	20196	0.0025	1,211.40	1.74	15.43	8.88
	Length (m)	20196	1.00	2.00	1.03	0.06	0.06
2	Au (g/t)	112149	0.0025	1,918.36	0.66	11.38	17.13
	Length (m)	112149	1.00	2.00	1.03	0.06	0.06
3	Au (g/t)	24363	0.0025	95.97	0.13	1.26	9.75
	Length (m)	24363	1.00	2.00	1.02	0.04	0.04
4	Au (g/t)	6036	0.0025	195.94	0.21	3.02	14.2
	Length (m)	6036	1.00	1.85	1.02	0.04	0.04

Notes: Std. Dev. = Standard Deviation; CV = Coefficient of Variation

Table 14-8: Descriptive Statistics of 1 m Composite Values (uncapped) by Rock Type per Zone

Zone	Rock Type	ZR Code	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Std. Dev.	CV
1	7	107	5864	0.0025	415.86	0.66	6.52	9.85
	9	109	12503	0.0025	1,211.40	2.42	19.04	7.88
	17	117	1829	0.0025	53.90	0.54	1.99	3.71
2	7	207	64248	0.0025	1,918.36	0.33	11.29	34.69
	9	209	18742	0.0025	1,074.34	1.64	13.19	8.04
	17	217	29159	0.0025	1,526.67	0.78	10.22	13.04
3	7	307	20480	0.0025	95.97	0.06	1.18	20.23
	9	309	28	0.0025	3.99	0.37	0.89	2.42
	17	317	3855	0.0025	57.86	0.50	1.58	3.14
4	7	407	4408	0.0025	68.55	0.05	1.06	21.50
	9	409	1158	0.0025	72.88	0.65	3.11	4.78
	17	417	470	0.0025	195.94	0.67	9.07	13.58

Notes: Std. Dev. = Standard Deviation; CV = Coefficient of Variation

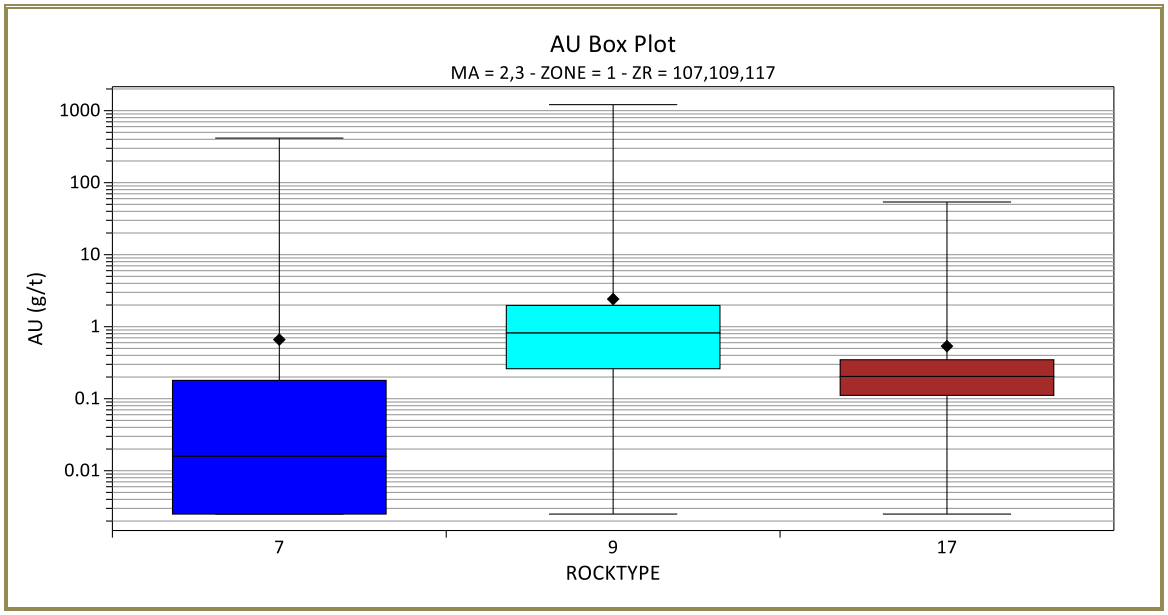


Figure 14-5: Box Plot of 1 m Composite Grades by Rock Type for Zone 1; Uncapped Gold (g/t Au)

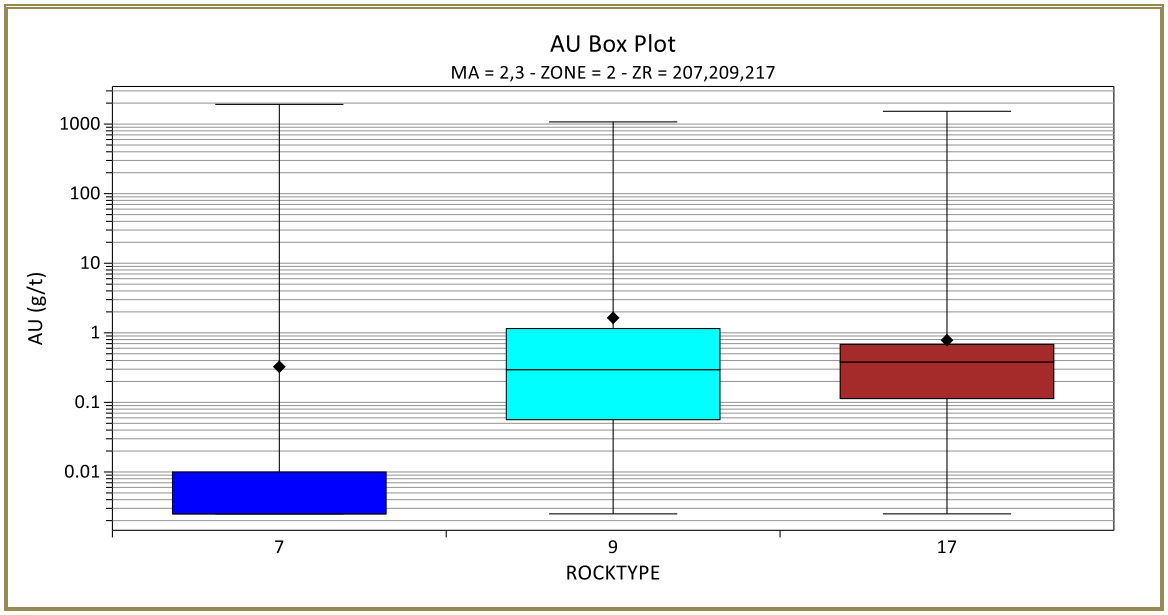


Figure 14-6: Box Plot of 1 m Composite Grades by Rock Type for Zone 2; Uncapped Gold (g/t Au)

14.4.3 Capping Analysis

In mineral deposits having skewed distributions (typically with CV >2), a few high-grade outliers can represent a large portion of the metal content. Often there is little continuity demonstrated by these outliers.

Capping analysis was carried out on 1 m composite values for the three rock types within each zone separately in the form of decile analysis, disintegration analysis, histogram, and log-probability plots.

Disintegration analysis uses a 10% to 15% step function to denote the changes in an ordered dataset and provides a degree of resolution on the plots to see more clearly the value of the population breaks that can be used for capping. It also provides a good look at the continuity of the grade dataset.

Figure 14-7 and Figure 14-8 illustrate the selection of the cap value for High-Ti Basalt in Zone 1 and Zone 2, respectively.

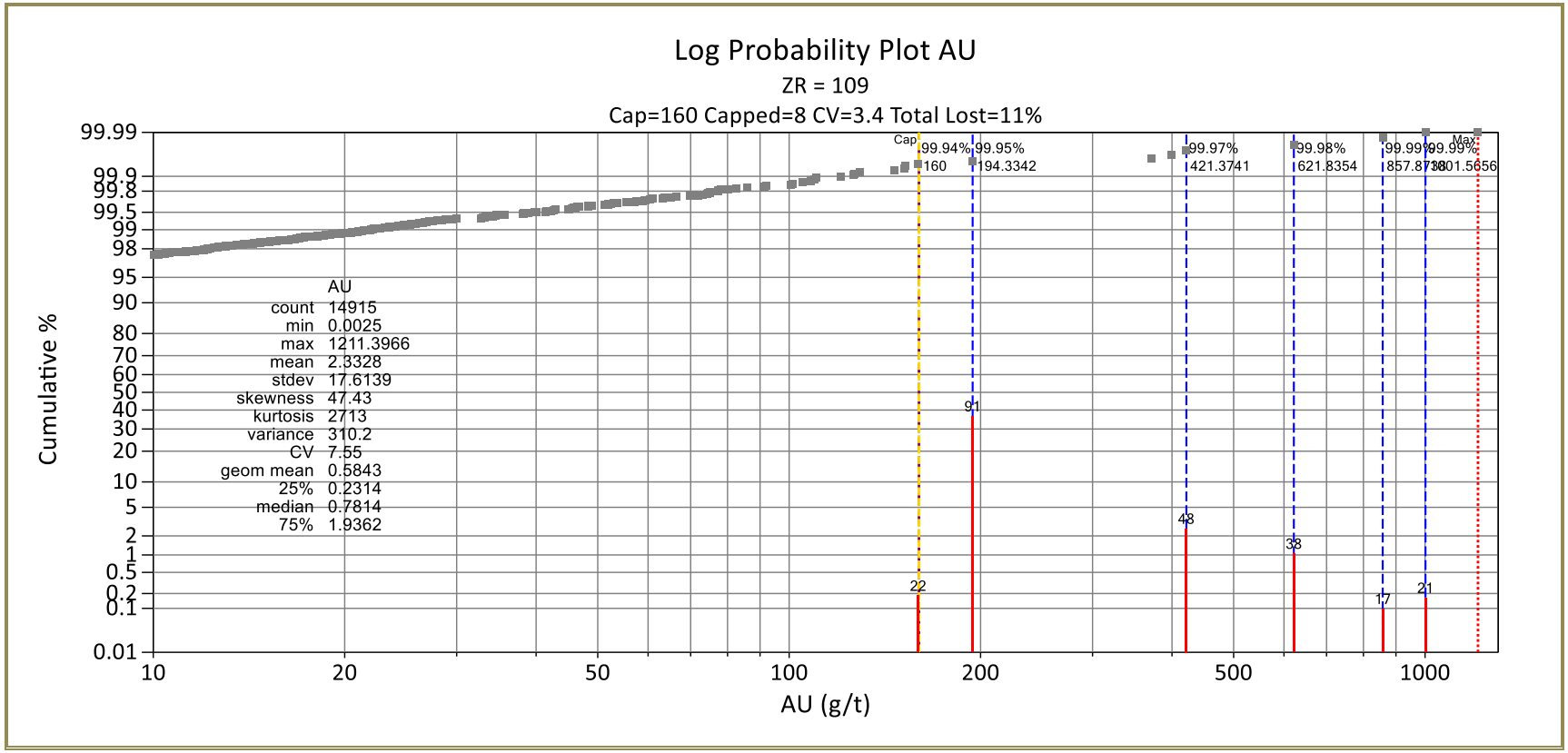


Figure 14-7: Zone 1, High-Ti Basalt Disintegration Analysis

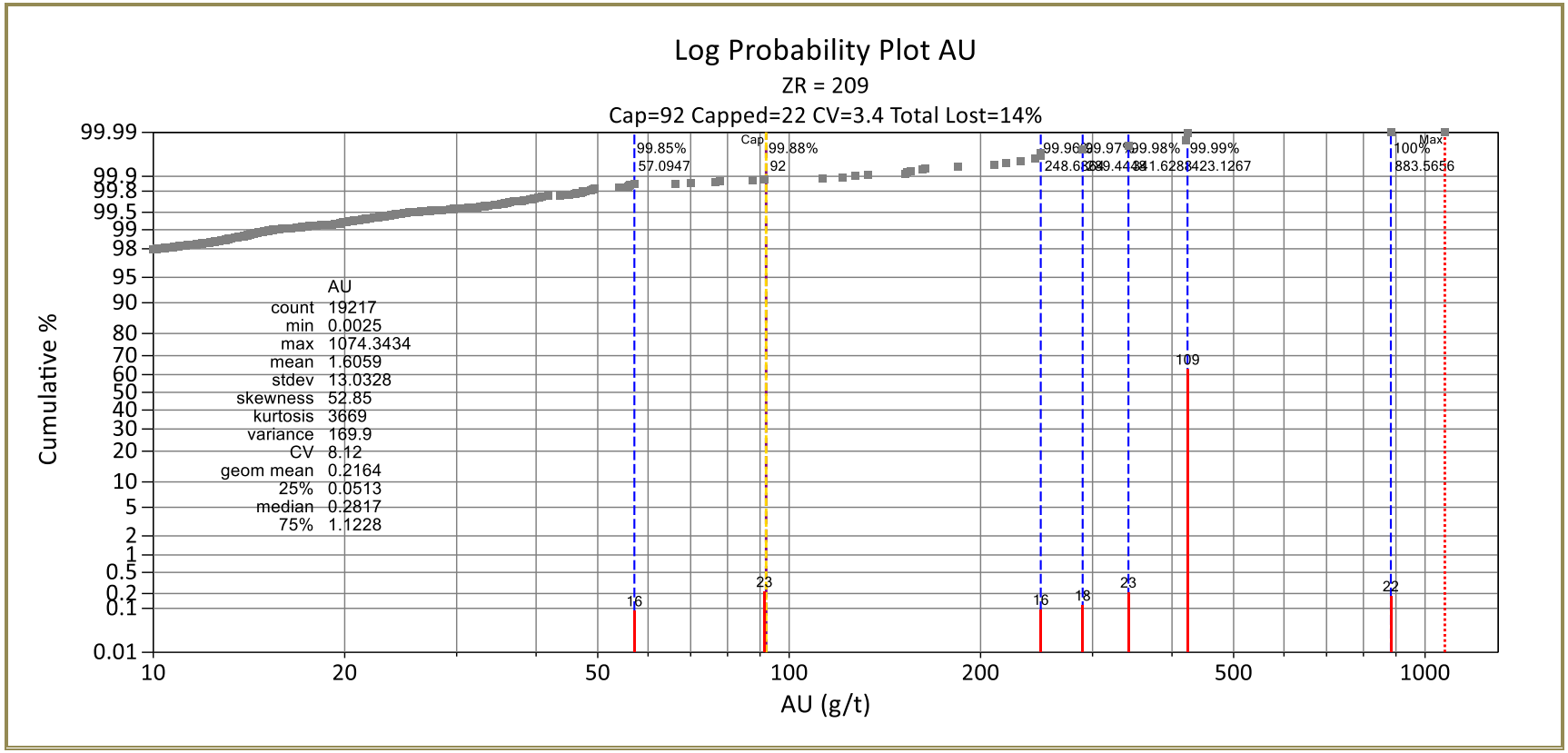


Figure 14-8: Zone 2, High-Ti Basalt Disintegration Analysis

Table 14-9 presents the selected capping levels applied to the 1 m composite values by zone and rock type. Also reported are the number of composites that were capped, and the associated metal loss percentage.

Table 14-9: Capping Levels by Rock Type per Zone

Zone	Rock Type	ZR Code	Capping (g/t Au)	Number Capped	Metal Loss (%)
1	7 (Ultramafic)	107	70	4	12
	9 (Basalt)	109	160	8	11
	17 (Felsic Intrusive)	117	15	5	7
2	7 (Ultramafic)	207	60	35	40
	9 (Basalt)	209	92	22	14
	17 (Felsic Intrusive)	217	80	7	11
3	7 (Ultramafic)	307	24	7	16
	9 (Basalt)	309	4	none	0
	17 (Felsic Intrusive)	317	30	2	3
4	7 (Ultramafic)	407	10	1	27
	9 (Basalt)	409	23	3	12
	17 (Felsic Intrusive)	417	13	1	58

Table 14-10 presents the descriptive statistics for the 1 m capped composite values. The uncapped composite descriptive statistics were reported in Table 14-8 and for uncapped assays in Table 14-6.

Table 14-10: Descriptive Statistics of 1 m Composite Grades (capped) by Rock Type per Zone

Zone	Rock Type	ZR Code	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Std. Dev.	CV
1	7	107	5864	0.0025	70.00	0.59	3.27	5.59
	9	109	12503	0.0025	160.00	2.11	7.21	3.41
	17	117	1829	0.0025	15.00	0.50	1.37	2.75
2	7	207	64248	0.0025	60.00	0.20	1.84	9.41
	9	209	18742	0.0025	92.00	1.41	4.75	3.36
	17	217	29159	0.0025	80.00	0.70	2.43	3.49
3	7	307	20480	0.0025	24.00	0.05	0.67	13.72
	9	309	28	0.0025	3.99	0.37	0.89	2.42
	17	317	3855	0.0025	30.00	0.49	1.16	2.37
4	7	407	4408	0.0025	10.00	0.04	0.28	7.90
	9	409	1158	0.0025	23.00	0.57	1.77	3.11
	17	417	470	0.0025	13.00	0.28	1.03	3.70

Notes: Std. Dev. = Standard Deviation; CV = Coefficient of Variation

Figure 14-9 to Figure 14-12 present the box plots of the 1 m capped composite values for each of the zones. The boxplot splits the data into quartiles to display the grade distribution. Generally, the High-Ti Basalt demonstrates the highest grade. The exception is in Zone 3 where the majority rock type is the Felsic Intrusive.

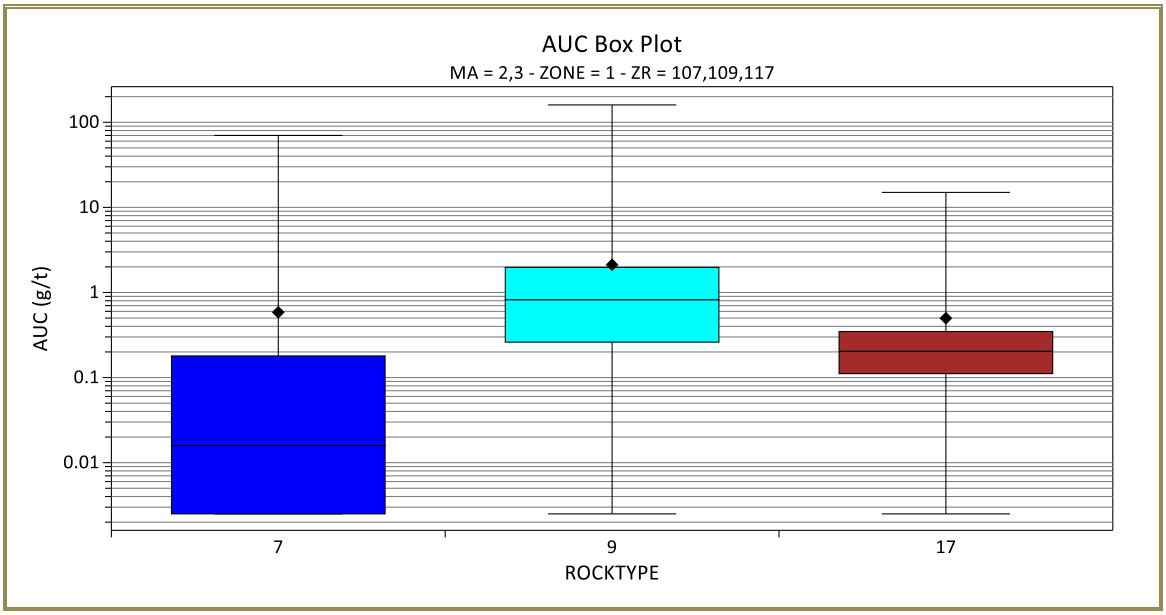


Figure 14-9: Box Plot 1 m Composite Values by Rock Type for Zone 1; Capped Gold (g/t Au)

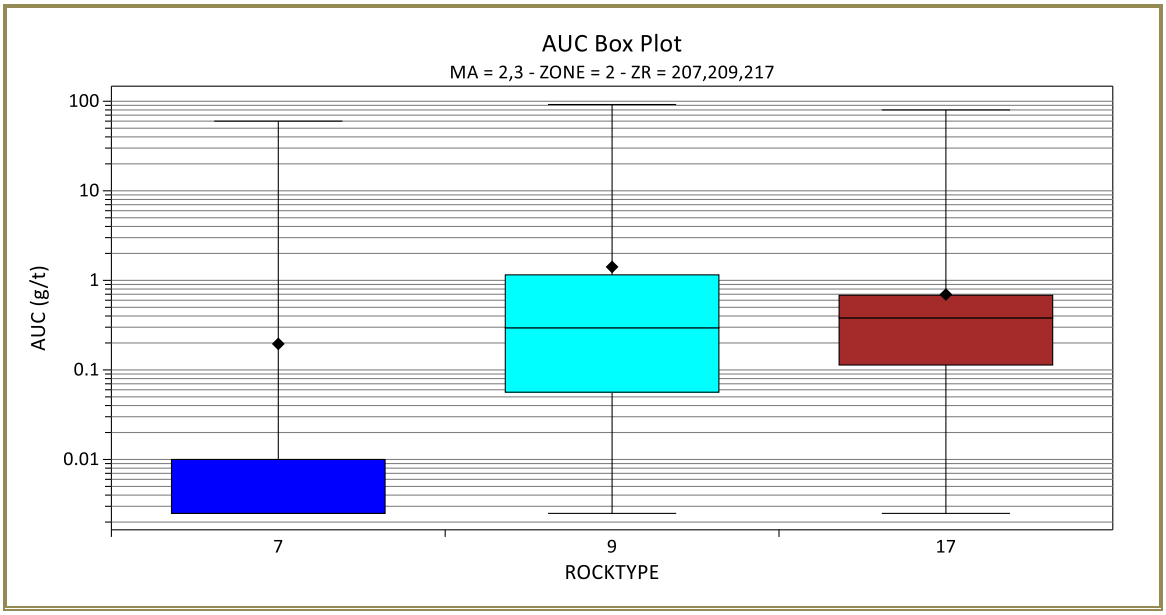


Figure 14-10: Box Plot 1 m Composite Values by Rock Type for Zone 2; Capped Gold (g/t Au)

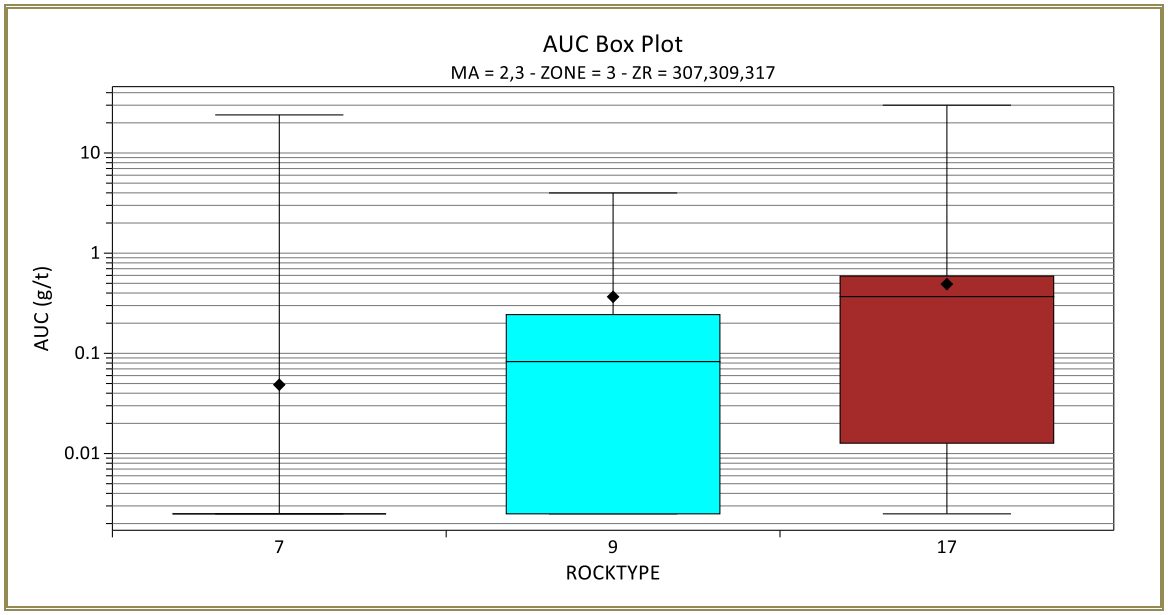


Figure 14-11: Box Plot 1 m Composite Values by Rock Type for Zone 3; Capped Gold (g/t Au)

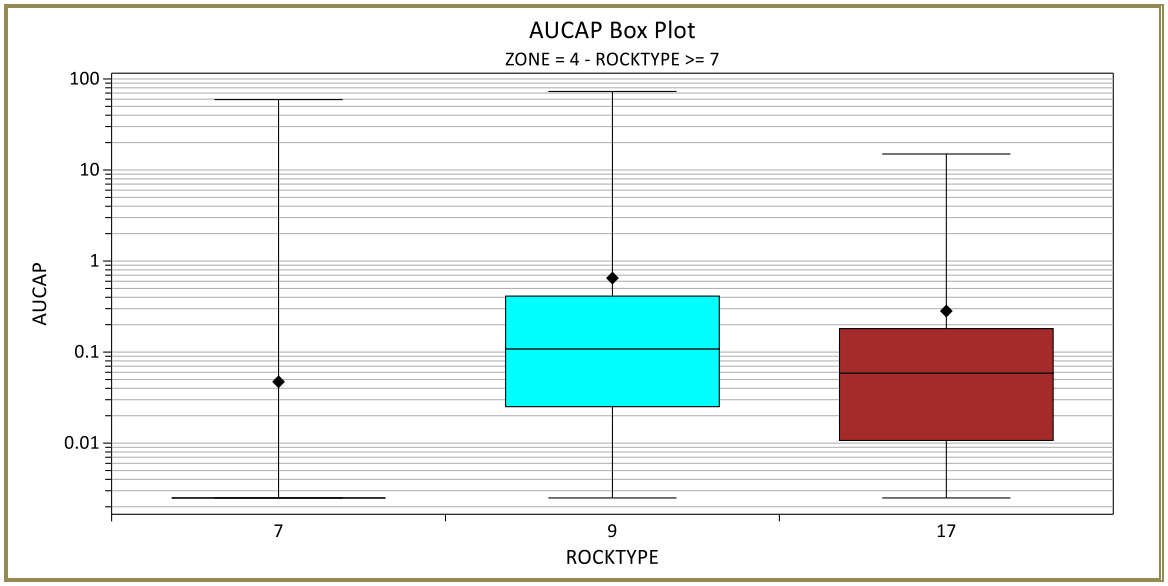


Figure 14-12: Box Plot 1 m Composite Values by Rock Type for Zone 4; Capped Gold (g/t Au)

The CV for the capped 1 m composite samples was compared to uncapped 1 m composite CV values. The CV values for the capped 1 m composite values were found to be within a more appropriate range for spatial analysis and gold grade estimation. Future work should assess the creation of high-grade domains to constrain the extrapolation of the outlier grades.

Table 14-11 presents the comparison of 1 m composite CV values.

Table 14-11: Comparison of 1 m Composite CV Values

Zone	Rock Type	Uncapped Mean 1 m Composite (g/t Au)	CV Uncapped Composite	Capped Mean 1 m Composite (g/t Au)	CV Capped Composite
1	7	0.66	9.85	0.59	5.59
1	9	2.42	7.88	2.11	3.41
1	17	0.54	3.71	0.50	2.75
2	7	0.33	34.69	0.20	9.41
2	9	1.64	8.04	1.41	3.36
2	17	0.78	13.04	0.70	3.49
3	7	0.06	20.23	0.05	13.72
3	9	0.37	2.42	0.37	2.42
3	17	0.50	3.14	0.49	2.37
4	7	0.05	21.50	0.04	7.90
4	9	0.65	4.78	0.57	3.11
4	17	0.6677	13.58	0.28	3.70

14.4.4 Contact Profiles

The contact relationship was analyzed between the three rock types: Ultramafic (7), High-Ti Basalt (9) and Felsic Intrusive (17). The contacts between Zone 1 to Zone 4 were assumed to be hard, as they segregated the interpreted zones of mineralization.

Contact analysis determines the average grade based on the distance between the points. Figure 14-13 to Figure 14-15 illustrate the contact relationship between the three rock types. The High-Ti Basalt is interpreted as a hard contact based on the difference in grade at the contact shown in Figure 14-13 and Figure 14-15. Figure 14-14 confirms the soft contact relationship between Ultramafic and Felsic Intrusive.

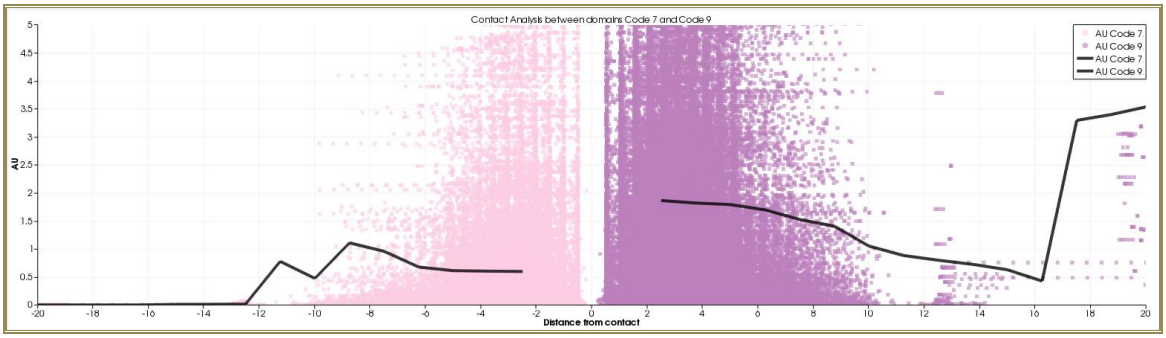


Figure 14-13: Contact Profiles – 7 UM:9 HTB

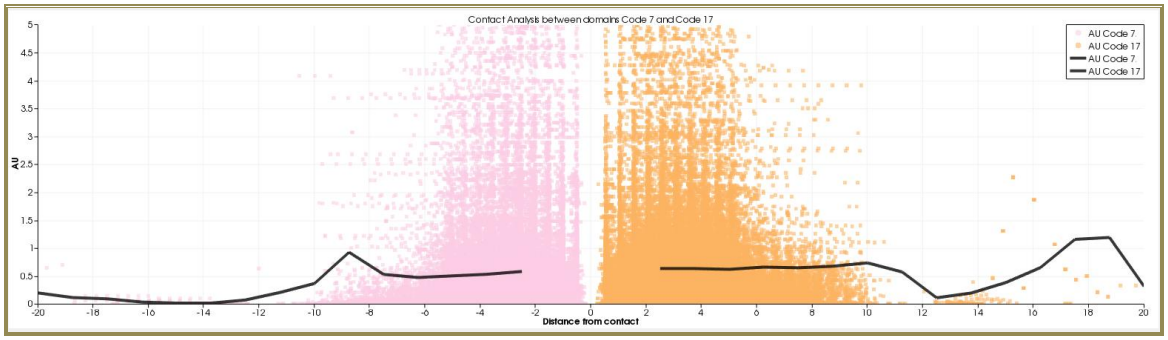


Figure 14-14: Contact Profiles – 7 UM:17 FI

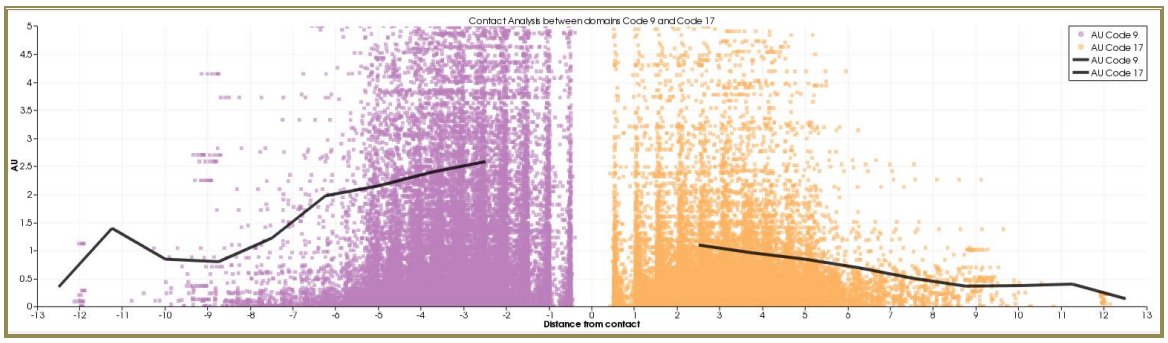


Figure 14-15: Contact Profiles – 9 HTB:17 FI

14.5 Block Model and Mineral Resource Estimation

14.5.1 Block Model

For Mineral Resource estimation, the block model for the Phoenix Gold Project was set up as two block models to cover the upper and lower halves of the deposit. The MA3V2_TOP covers the model elevations between 5,390 m and 4,510 m and MA3V2_BOT covers the model elevations between 4,510 m and 3,530 m. No rotation was applied to the block models. The block matrix was selected in consideration of the geometry of the deposit (narrow zones of mineralization within the specific lithology), drill data density, and selective mining unit (SMU).

Table 14-12 summarizes the block model parameters used in the GEMS project, and Figure 14-16 presents the block models referenced by the interpreted mineralization zones for the Phoenix Gold Project.

Table 14-12: Block Model Parameters for the MA3V2_TOP and MA3V2_BOT Models

	MA3V2_TOP (5390 m – 4510 m)	MA3V2_BOT (4510 m – 3,530 m)
Easting (m)	10,100	10,100
Northing (m)	49,250	49,250
Maximum elevation	5,390	4510
Rotation angle	No rotation°	No rotation°
Block size (X, Y, Z in metres)	2 x 2 x 2	2 x 2 x 2
Number of blocks in the X direction	300	300
Number of blocks in the Y direction	675	675
Number of blocks in the Z direction	440	490

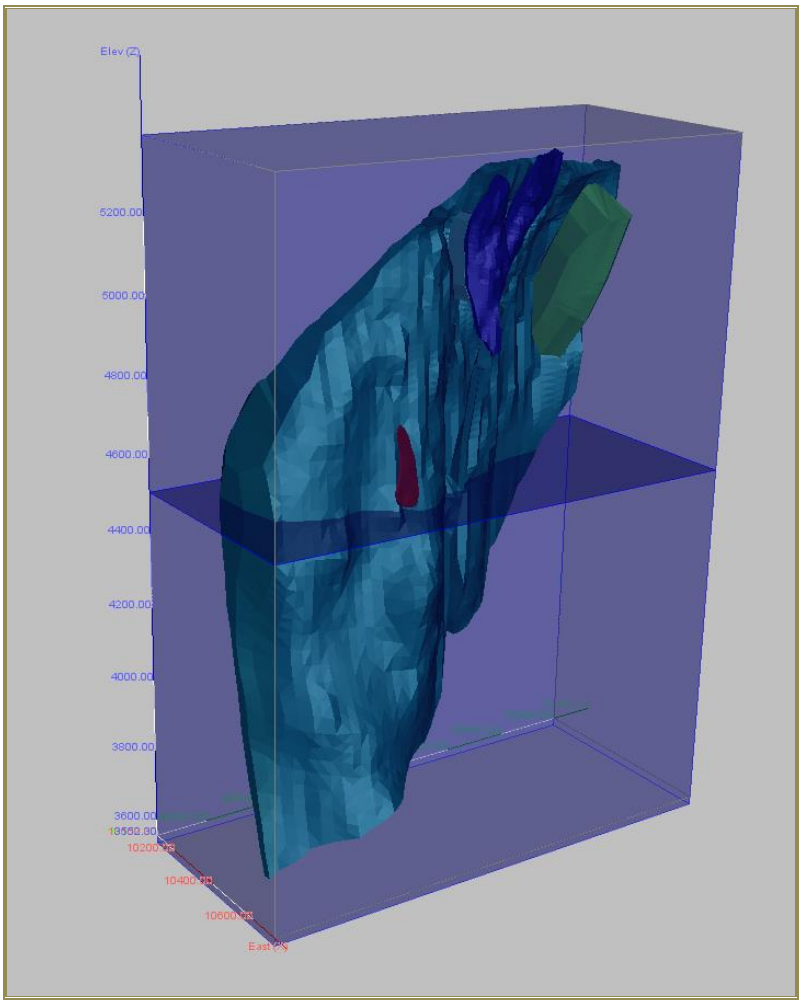


Figure 14-16: MA3V2_TOP and MA3V2_BOT Block Models; Perspective View Looking Northwest

14.5.2 Spatial Analysis

Geostatisticians use a variety of tools to describe the pattern of spatial continuity or strength of the spatial similarity of a variable with separation distance and direction. One of these is the correlogram, which measures the correlation between data values as a function of their separation distance and direction. If we compare samples that are close together, it is common to observe that their values are quite similar and the correlation coefficient for closely spaced samples is near 1.0. As the separation between samples increases, there is likely to be less similarity in the values, and the correlogram tends to decrease toward 0.0. The distance at which the correlogram reaches zero is called the range of correlation, or simply the range. The range of the correlogram corresponds roughly to the more qualitative notion of the range of influence of a sample; it is the distance over which sample values show some persistence or correlation. The shape of the correlogram describes the pattern of spatial continuity. A very rapid decrease near the origin is indicative of short-scale variability. A more gradual decrease moving away from the origin suggests more short-scale continuity. A plot of 1-correlation is made so the result looks like the more familiar variogram plot.

The approach used to develop the variogram models employed Sage2001 software. Directional sample correlograms were calculated along horizontal azimuths of 0, 30, 60, 120, 150, 180, 210, 240, 270, 300, and 330 degrees. For each azimuth, sample correlograms were also calculated at dips of 30 and 60 degrees in addition to horizontally. Lastly, a correlogram was calculated in the vertical direction. Using the 37 sample correlograms, an algorithm determined the best-fit model nugget effect and two-nested structure variance contributions. After fitting the variance parameters, the algorithm then fitted an ellipsoid to the 37 ranges from the directional models for each structure. The anisotropy of the correlation was given by the range along the major, semi-major, and minor axes of the ellipsoids, and the orientations of these axes for each structure. TMAC reviewed the fitted variogram and adjusted to reflect the mineralization.

Table 14-13 and Table 13-4 present the variogram parameters for the block models MA3V2_TOP and MA3V2_BOT. Zone 3 and Zone 4 contained less data than Zone 1 and Zone 2. TMAC reviewed potential variograms and decided to use the Zone 1 variograms.

Table 14-13: Variogram Parameters for Block Model MA3V2_TOP by ZR Code

ZR	Sill = 1.00	Search Anisotropy	Azimuth (°)	Dip (°)	Azimuth (°)	X Range (m)	Y Range (m)	X Range (m)	Variogram Type
107	C_0 = 0.35	ZYZ	83	-10	88				Nugget
	C_1 = 0.55	ZYZ	83	-10	88	3.6	2.2	11.2	Spherical
	C_2 = 0.10	ZYZ	83	-10	88	41.2	55.1	75	Spherical
109	C_0 = 0.50	ZYZ	8	-66	-10				Nugget
	C_1 = 0.40	ZYZ	8	-66	-10	7	10	10	Spherical
	C_2 = 0.10	ZYZ	8	-66	-10	20	40	30	Spherical
117	C_0 = 0.50	ZYZ	5	-60	-10				Nugget
	C_1 = 0.40	ZYZ	5	-60	-10	6	10	5	Spherical
	C_2 = 0.10	ZYZ	5	-60	-10	21	32	12	Spherical
207	C_0 = 0.40	ZYZ	0	-85	-10				Nugget
	C_1 = 0.50	ZYZ	0	-85	-10	10	20	12	Spherical
	C_2 = 0.10	ZYZ	0	-85	-10	25	55	35	Spherical
209	C_0 = 0.50	ZYZ	30	40	-30				Nugget
	C_1 = 0.35	ZYZ	30	40	-30	10	20	12	Spherical
	C_2 = 0.15	ZYZ	30	40	-30	25	50	35	Spherical
217	C_0 = 0.40	ZYZ	30	50	-30				Nugget
	C_1 = 0.50	ZYZ	30	50	-30	10	18	11	Spherical
	C_2 = 0.10	ZYZ	30	50	-30	25	40	30	Spherical

Table 14-14: Variogram Parameters for Block Model MA3V2_BOT by ZR Code

ZR	Sill = 1.00	Search Anisotropy	Azimuth (°)	Dip (°)	Azimuth (°)	X Range (m)	Y Range (m)	X Range (m)	Variogram Type
207	C_0 = 0.40	ZYZ	0	-85	-10				Nugget
	C_1 = 0.50	ZYZ	0	-85	-10	10	20	12	Spherical
	C_2 = 0.10	ZYZ	0	-85	-10	25	55	35	Spherical
209	C_0 = 0.50	ZYZ	30	40	-30				Nugget
	C_1 = 0.35	ZYZ	30	40	-30	10	20	12	Spherical
	C_2 = 0.15	ZYZ	30	40	-30	25	50	35	Spherical
217	C_0 = 0.40	ZYZ	30	50	-30				Nugget
	C_1 = 0.50	ZYZ	30	50	-30	10	18	11	Spherical
	C_2 = 0.10	ZYZ	30	50	-30	25	40	30	Spherical

The rotation convention for the Search Anisotropy is ZYZ (right-hand rule):

- Rotation about Z-axis—Positive rotation X toward Y
- Rotation about Y-axis—Positive rotation Z toward X
- Rotation about new Z-axis—Positive rotation X toward Y.

14.5.3 Grade Interpolation

The Phoenix Gold Project block models were estimated using IDW3 (uncapped-AUCID, capped-AUCID3). OK (capped-AUCOK) and NN (capped-AUCNN) were also run for validation purposes. The block models were estimated in four passes. The NN used the 2 m composites, and IDW3 and OK used 1 m composites for grade interpolation.

Table 14-15 shows the estimation parameters for each pass used to estimate gold grades.

Table 14-15: Summary of Samples Controls for All Zones

Pass	Minimum No. of Samples	Maximum No. of Samples	Maximum No. Samples per Drill Hole	Minimum No. Drill Holes
1	5	11	3	2
2	5	11	3	2
3	4	8	3	2
4	3	8	3	1

14.5.4 Special Models

GEMS uses special models to track interpolation characteristics. These special models were also used to evaluate Mineral Resource classification. TMAC employed the following in their grade interpolation:

- NN Interpolation:
  - DISTNN – Distance to nearest composite
- IDW3 interpolation:
  - AVGDIST – Average distance to composites used for interpolation
  - DIST1 – Distance to nearest composite for Pass 1
  - DIST2 – Distance to nearest composite for Pass 2
  - DIST3 – Distance to nearest composite for Pass
  - DIST4 – Distance to nearest composite for Pass 4
  - NCOMP – Number of composites used for interpolation
  - NDDH – Number of drill holes used for interpolation
  - PASS – Pass number for interpolation
- OK interpolation:
  - OKNCOMP – Number of composites used for interpolation
  - OKNDDH – Number of composites used for interpolation
  - VAR – Kriging variance.

14.5.5 Search Ellipses

Table 14-16 summarizes the search ellipse parameters for the Phoenix Gold Project. These parameters were based on the geological interpretation and variogram analysis. Similar search ellipses were used for IDW3 and OK grade interpolation. The NN estimation used Pass 1 and Pass 4.

Table 14-16: Search Ellipse Dimensions by ZR Code

ZR	Pass	Search Anisotropy	Z (°)	Y (°)	Z (°)	X Range (m)	Y Range (m)	Z Range (m)
107	1	ZYZ	-83	-10	88	20	30	40
	2					40	55	75
	3					60	80	110
	4					100	140	190
109	1	ZYZ	7.6	-66.3	-45	10	20	20
	2					20	40	30
	3					30	60	50
	4					50	100	80
117	1	ZYZ	5	-60	-45	10	20	10
	2					20	35	15
	3					30	50	20
	4					60	90	40
207	1	ZYZ	0	-85	-45	10	30	10
	2					25	55	10
	3					40	80	20
	4					60	140	30
209	1	ZYZ	-30	50	75	10	30	15
	2					25	50	20
	3					40	80	25
	4					60	130	30
217	1	ZYZ	30	40	49	20	10	20
	2					40	20	30
	3					60	25	50
	4					100	30	80

14.5.6 Outlier Controls

Additional outlier controls were applied during the interpolation passes. For Pass 1, the capped outliers were used in the normal grade interpolation. The average of the mean distance of samples used for grade estimation was 15.5 m, and the 99th percentile was 29 m. For Pass 2 through Pass 4, the outliers were excluded and not used for grade interpolation.

14.6 Model Verification and Validation

TMAC distinguishes between verification and validation of the block model:

- Verification is a manual check (i.e., visual inspection) or quasi-manual check (i.e., spreadsheet) of the actual procedure used
- Validation is a test for reasonableness using a parallel procedure, which may be manual or a computer-based procedure (i.e., different interpolation methods).

14.6.1 Visual Verification

The block model was validated by visually inspecting the block model results in section and plan compared with the drill hole composite data. The grades of the blocks by section agreed well with the composite data used in the interpolation.

Figure 14-17 presents a selected 5,100 m elevation plan view that shows the gold grades (AUCID3) in blocks with the gold grades from the 1 m composites. Figure 14-18 illustrates good correlation between block and composite grades on Section 51040N.

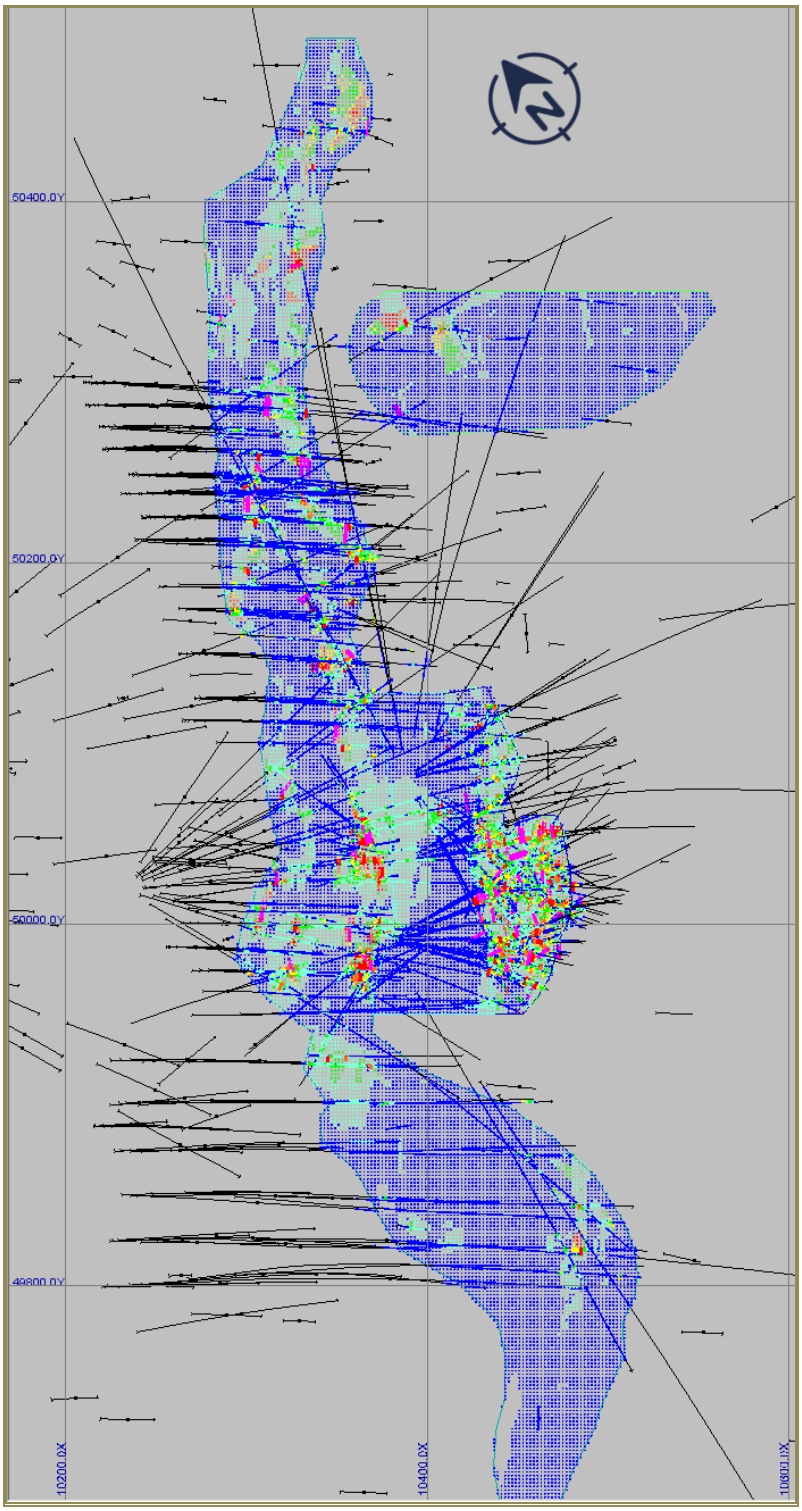


Figure 14-17: Plan View 5,100 m Elevation Comparing Block Grades with 1 m Composites

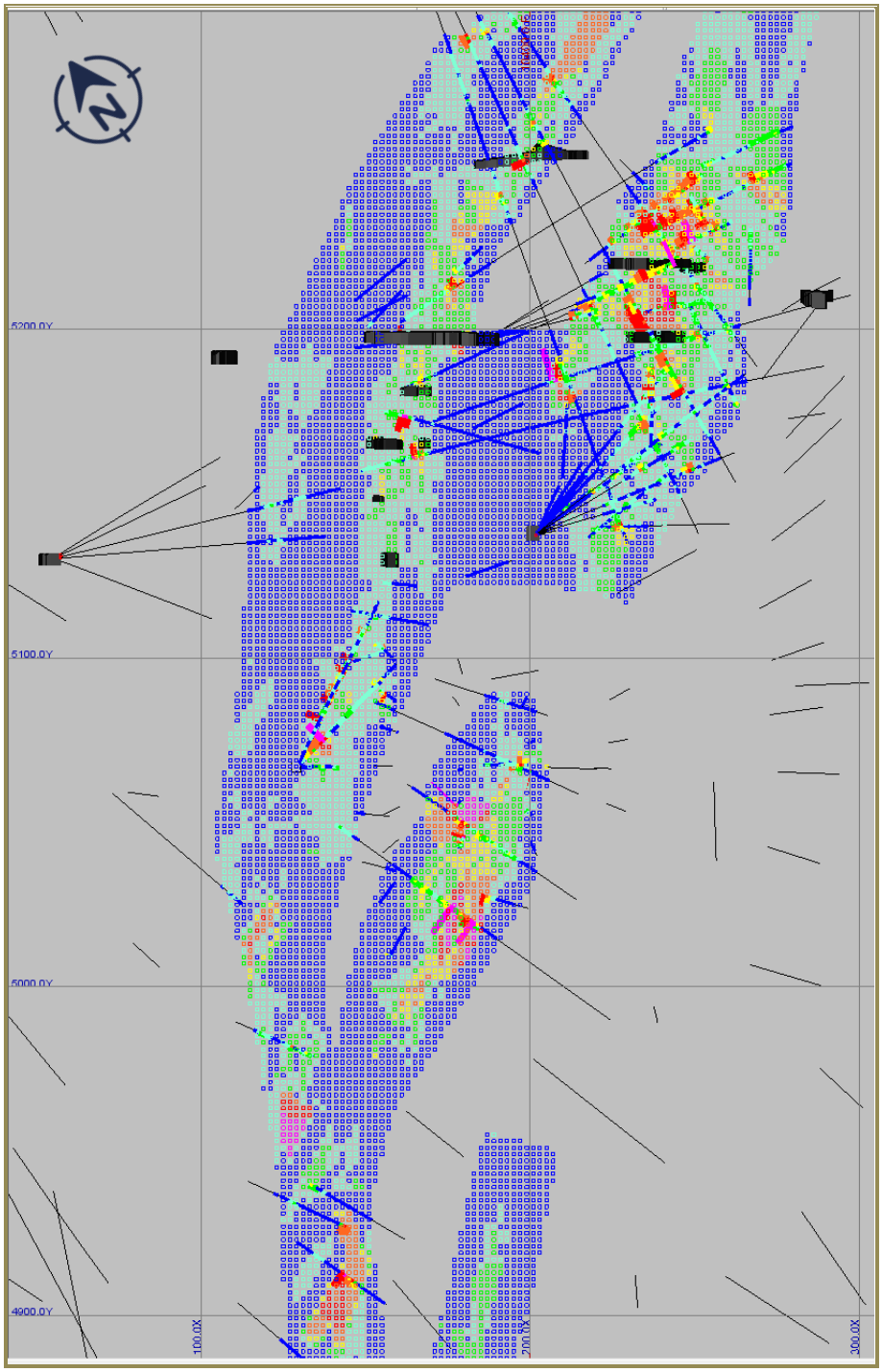


Figure 14-18: Section 50140N Comparing Block Grades with 1 m Composites

14.6.2 Statistical Validation

The block model statistics were reviewed for each rock type in each zone and no bias was found between the different interpolation methods and the 1 m composites.

Table 14-17 presents the average gold grades for Measured–Indicated–Inferred Resource blocks in MA3V2_TOP by rock type. Minor differences are noted between the interpolation methods, but those may reflect data density and statistics generated by block count rather than weighted by tonnes.

Table 14-17: Statistical Comparison by Rock Type of Capped Interpolated Grades for MA3V2_Top

Zone	Rock Type	NN Mean (g/t Au)	IDW3 Mean (g/t Au)	OK Mean (g/t Au)
1	7	0.395	0.411	0.445
	9	1.945	1.966	1.994
	17	0.432	0.438	0.450
2	7	0.261	0.236	0.251
	9	1.174	1.113	1.134
	17	0.606	0.571	0.527
3	7	0.059	0.064	0.073
	9	0.382	0.434	0.367
	17	0.495	0.407	0.443
4	7	0.053	0.065	0.077
	9	0.477	0.475	0.499
	17	0.185	0.187	0.190

14.6.3 Swath Plots

A series of swath plots of gold grades were generated from capped gold grades for the NN, ID^3, and OK interpolation methods. The grades are averaged over 20 m swaths for each of the plots.

Figure 14-19 to Figure 14-21 compared the capped gold grades for NN with the IDW3 models. Figure 14-22 to Figure 14-24 compared the capped gold grades for OK with the IDW3 models. These figures confirm a good correlation between the grade models, independent of interpolation method.

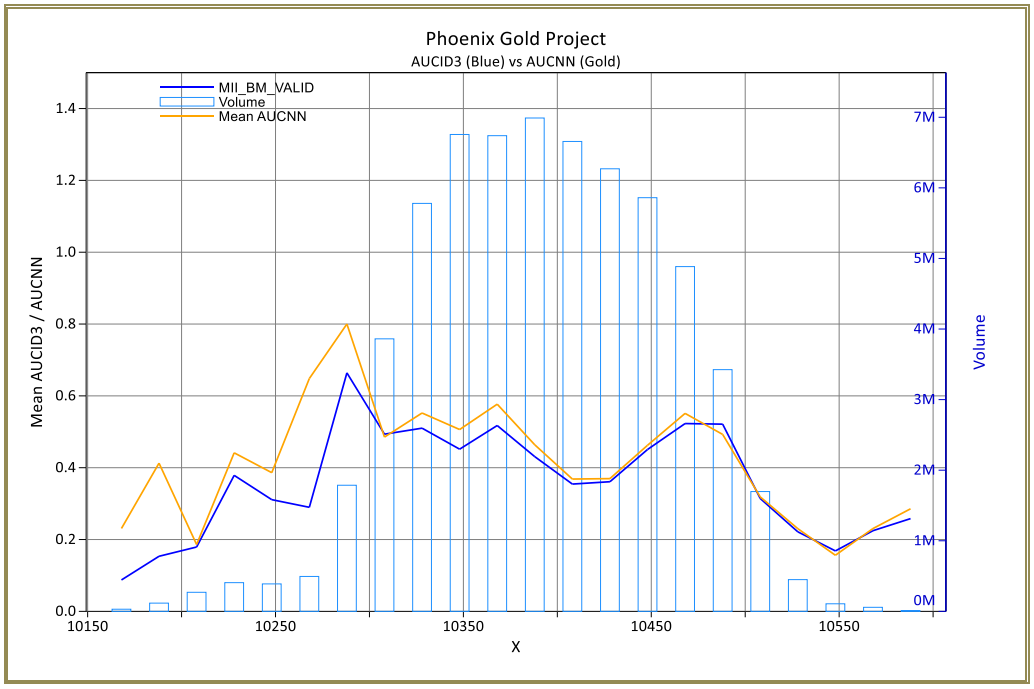


Figure 14-19: Swath Plot by Easting, AUCID3 vs. AUCNN

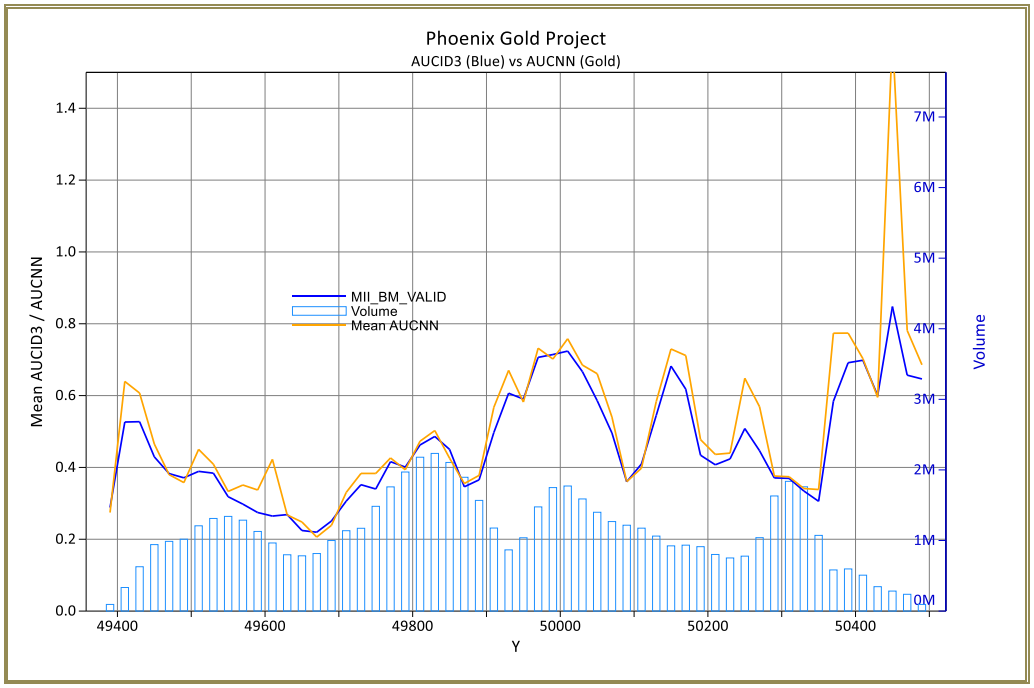


Figure 14-20: Swath Plot by Northing, AUCID3 vs. AUCNN

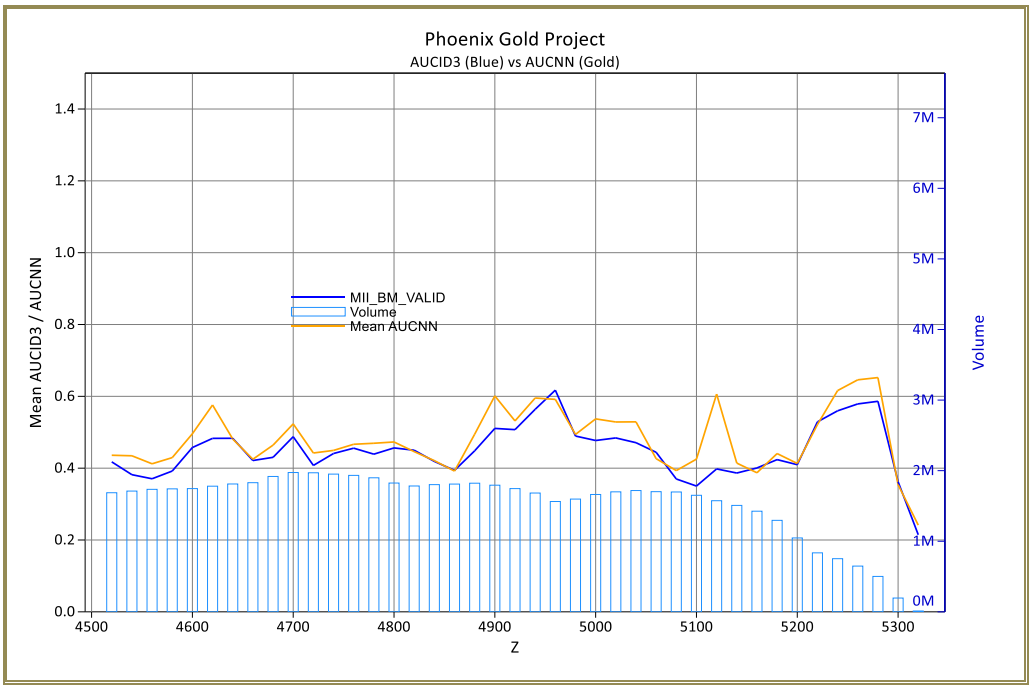


Figure 14-21: Swath Plot by Elevation, AUCID3 vs. AUCNN

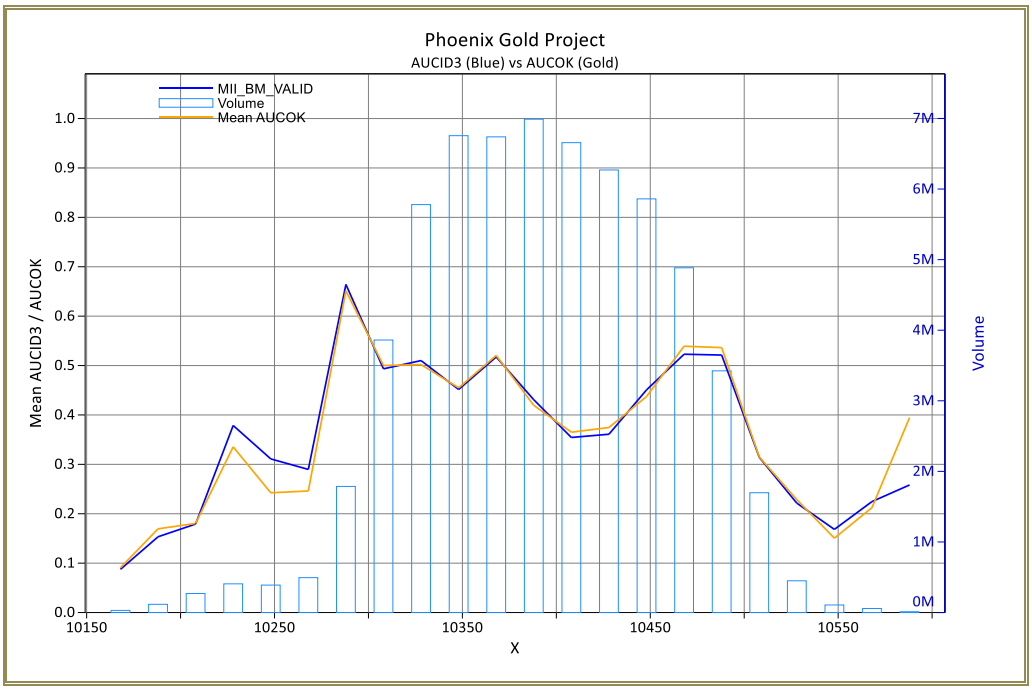


Figure 14-22: Swath Plot by Easting, AUCID3 vs. AUCOK

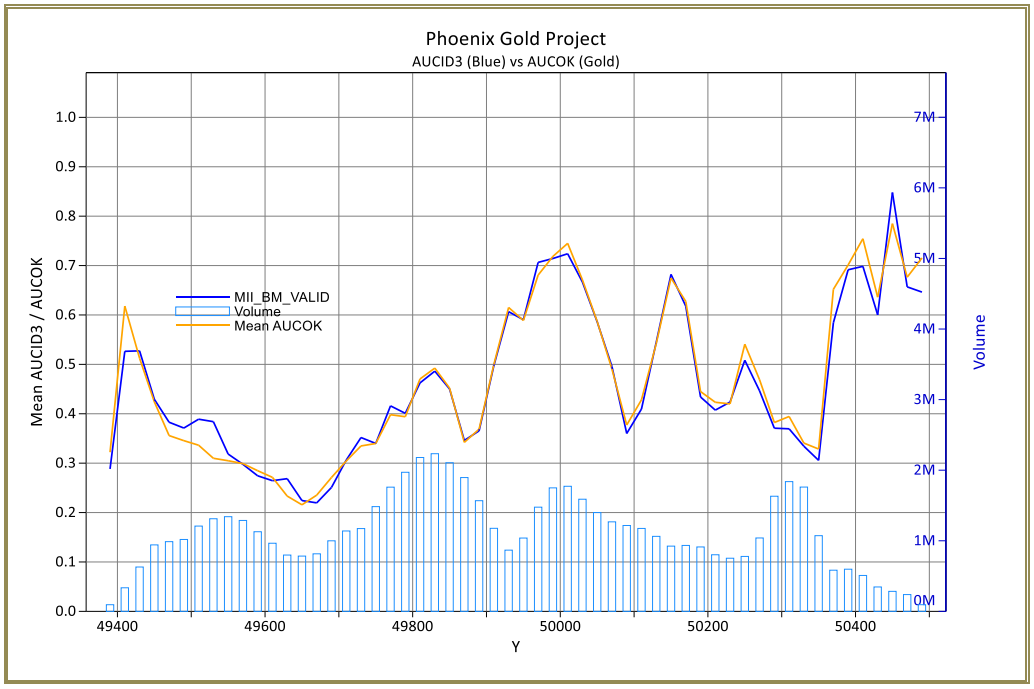


Figure 14-23: Swath Plot by Northing, AUCID3 vs. AUCOK

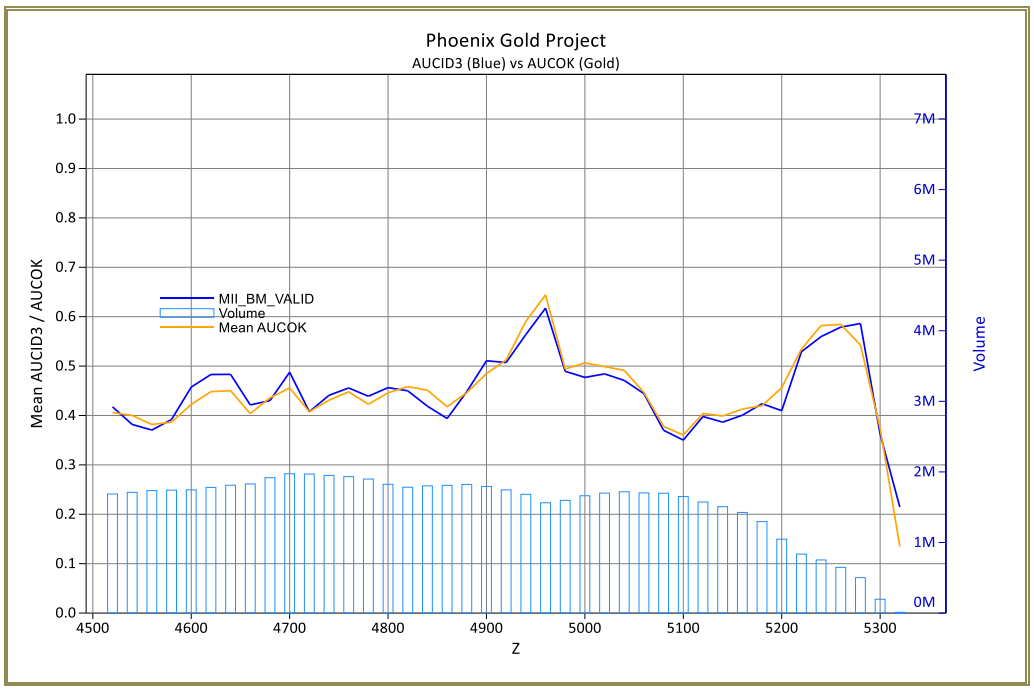


Figure 14-24: Swath Plot by Elevation, AUCID3 vs. AUCOK

14.6.4 Reconciliation of 2018 Test Mining Program

In late 2017, Rubicon commenced mining operations at the Phoenix Gold Project to extract a bulk sample and process it through the existing Phoenix Mill to produce gold-bearing material.

The program consisted of three stopes located in the F2 High-Ti Basalt, one accessed from the 183 m Level and the other two from the 244 m Level. The material was sequentially mined, hoisted to the surface, and stored in separate stockpiles. Each stope was then batch-processed through the mill. The stopes were not commingled with other stope material. There was only one final full mill cleanup at the end of the program.

Golder (2019) compared the tonnes and gold mined, less dilution, to judge the variance against the in-situ Mineral Resource estimate. Table 14-18 compares a calculation of undiluted tonnes derived from process tonnes less measured dilution with the current Mineral Resource model. The tonnes were within 1.5% and the estimated Mineral Resource gold ounces were 12.5% less.

Table 14-18: Calculated Undiluted Bulk Sample vs. Undiluted Resource Model

	Actual Results			Reporting from CMS		
Bulk Sample Stope	**Tonnes**	**Au g/t**	**Au oz**	**Tonnes**	**AUCID3 g/t**	**AUCID3 oz**
244-Z1-015	7,329	3.36	792.3	7,331	2.98	701.7
244-Z1-977	9,985	3.69	1,184.6	9,794	3.48	1,095.4
183-Z1-161 combined	18,420	5.38	3,187.6	19,136	4.42	2,722.0
Total	**35,734**	**4.50**	**5,164.5**	**36,261**	**3.88**	**4,519.2**

14.6.5 Hermitian Correction

The relative degree of smoothing in the block model estimates can be evaluated using the Discrete Gaussian or Hermite polynomial change of support method (Journel & Huijbregts, 1978).

With this method, the distribution of the hypothetical block grades can be directly compared to the estimated (OK or ID) model by using pseudograde/tonnage curves. In general, the estimated model should be slightly higher in tonnage and slightly lower in grade, compared to the Herco distribution at the projected cut-off grade. These differences account for selectivity and other potential ore-handling issues that commonly occur during mining.

The Herco distribution is derived from the de-clustered composite grades that have been adjusted to account for the change in support from smaller drill hole composite samples to the large blocks in the model. The transformation results in a less skewed distribution, but with the same mean as the original declustered samples.

The Measured and Indicated Mineral Resource estimate within the High-Ti Basalt in Zone 1 and Zone 2 are smooth relative to the Herco distribution. As shown in Figure 14-25, the IDW3 model displays less smoothing than the OK model for Zone 1 High-Ti Basalt—the tonnes were underestimated about 3% and the grade −5.3% at the 3.0 g/t Au cut-off grade.

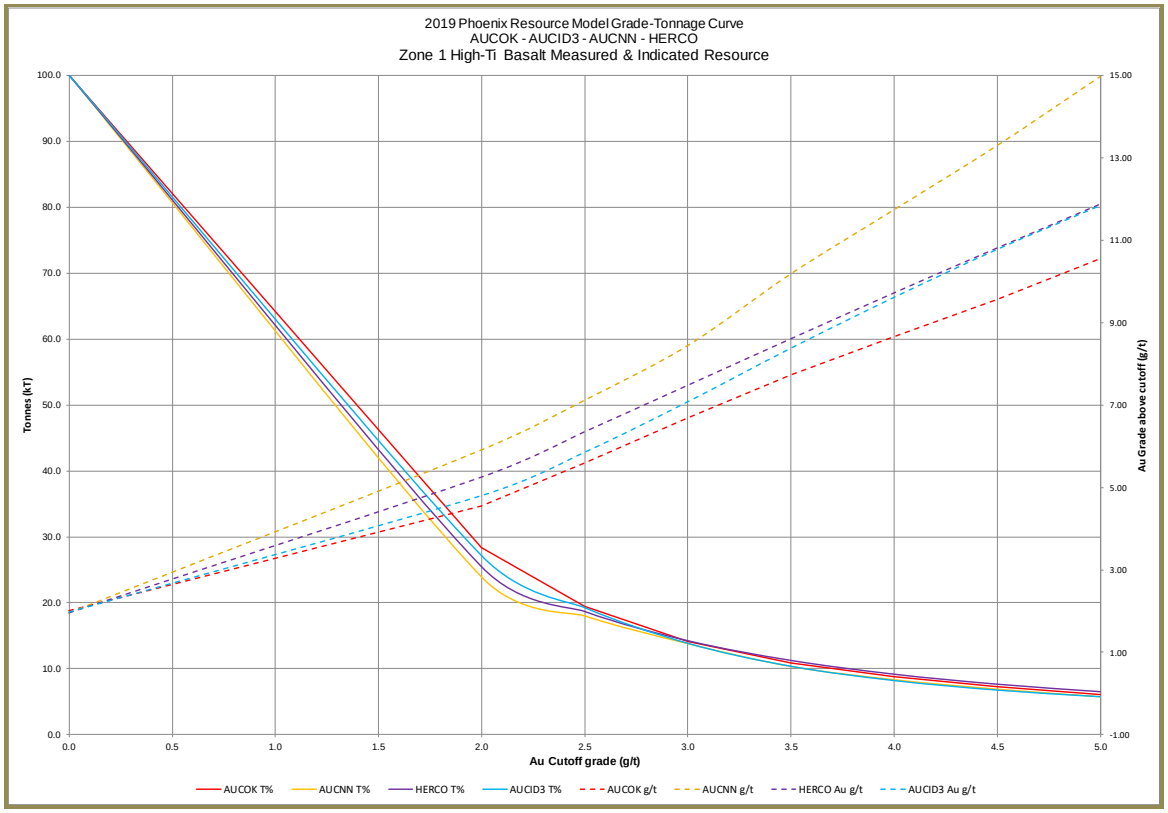


Figure 14-25: Zone 1 High-Ti Basalt Herco Grade-Tonnage Curve

Similarly, in Figure 14-26, the ID model displays less smoothing than the OK model for Zone 1 High-Ti Basalt—the tonnes were underestimated about 1% and the grade -11.7% at the 3.0 g/t Au cut-off grade.

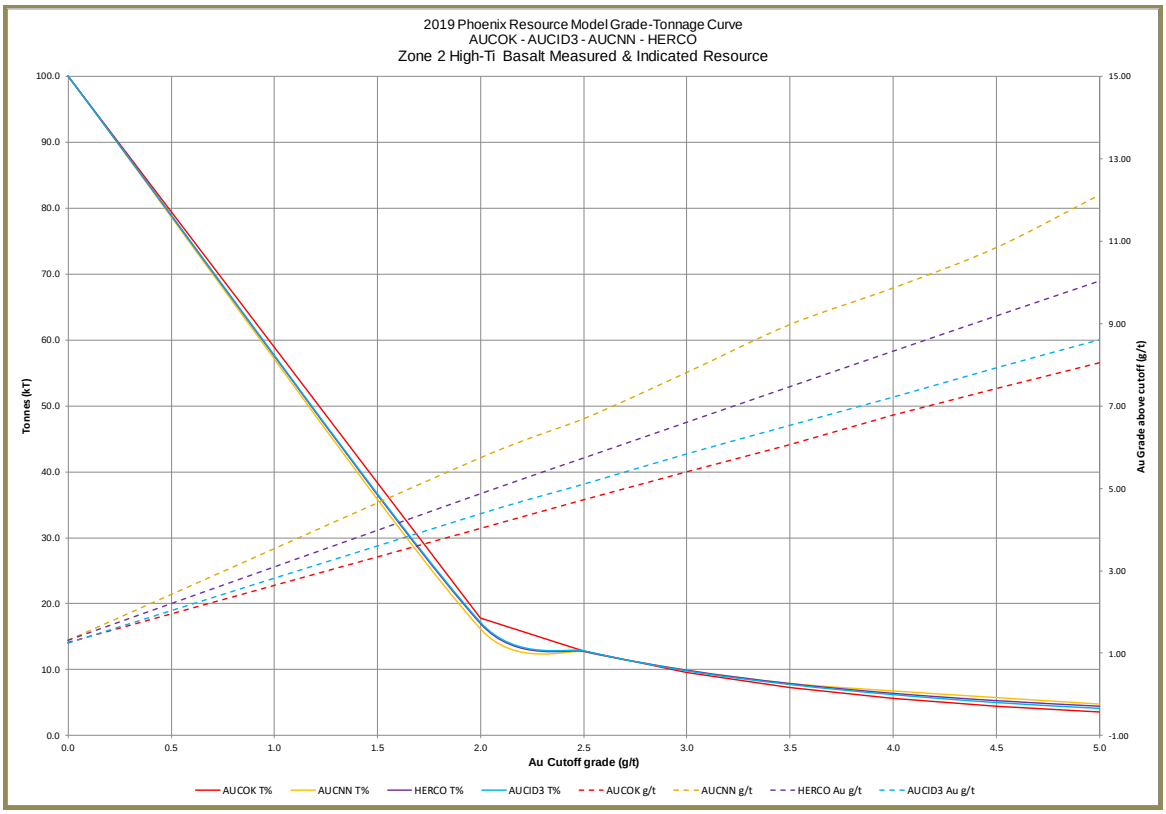


Figure 14-26: Zone 2 High-Ti Basalt Herco Grade-Tonnage Curve

14.7 Mineral Resource Estimates

14.7.1 Mineral Resource Classification

Mineral Resource estimates were classified in accordance with definitions provided by CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). Mineral Resource estimates have an effective date of March 18, 2019.

Mineral Resource estimates were initially assigned based on data density in coordination with mineralization continuity. Mineral Resource classification was refined based on the interpolation statistics collected during interpolation. All Mineral Resource estimates were interpolated using a minimum of two drill holes and a minimum of three composites. The nominal spacing for Measured Mineral Resource estimates, based on distance to nearest composite, was 6 m, with 99% less than 15.7 m. For Indicated Mineral Resource estimates, the nominal spacing was 13 m, and for Inferred Mineral Resource estimates 26 m. Additional statistics are reported in Table 14-19.

Table 14-19: Additional Interpolation Statistics Reported by Resource Class

Resource Class	Avg. Distance to Nearest Composite	99th Percentile Distance to Nearest Composite	Avg. of Mean Distance of Composites Used	99th Percentile Mean Distance of Composites Used	Min. No. of Drill Holes	Avg. No. of Drill Holes Used	Min. No. of Composites Used	Avg. No. of Composites Used
Measured	6	15.7	10	21	2	4	5	11
Indicated	13	32.3	18	34	2	3	3	9
Inferred	26	60.1	33	57	2	3	3	7

Figure 14-27 illustrates the Mineral Resource classification for Zone 1 and Zone 2 in plan view relative to underground development (305 m Level).

Mineral Resource estimates were also constrained by proximity to development. Measured Mineral Resource estimates were constrained between the 122 m Level and 60 m below the 305 m Level.

Figure 14-28 illustrates the location of the Zone 1 Mineral Resource classification relative to underground development. The Indicated Mineral Resource estimate was constrained within 90 m from the shaft bottom at the 732 m Level. The Inferred Mineral Resource estimate was assigned to interpolated blocks down to the 1,525 m Level.

Figure 14-29 illustrates Mineral Resource classification within Zone 2 in a longitudinal view relative to existing underground development.

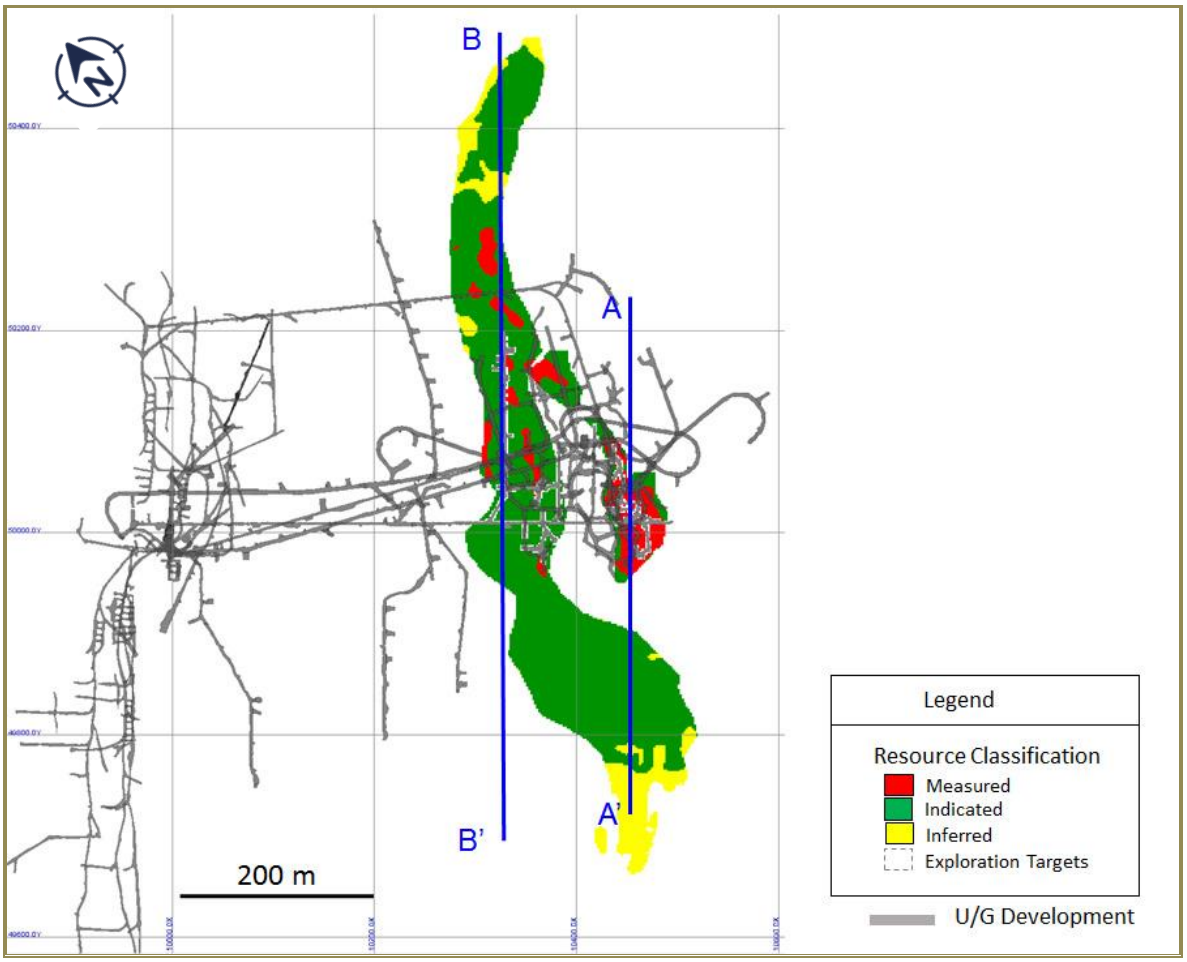


Figure 14-27: 2019 Mineral Resource Classification—Plan View, 305 m Level

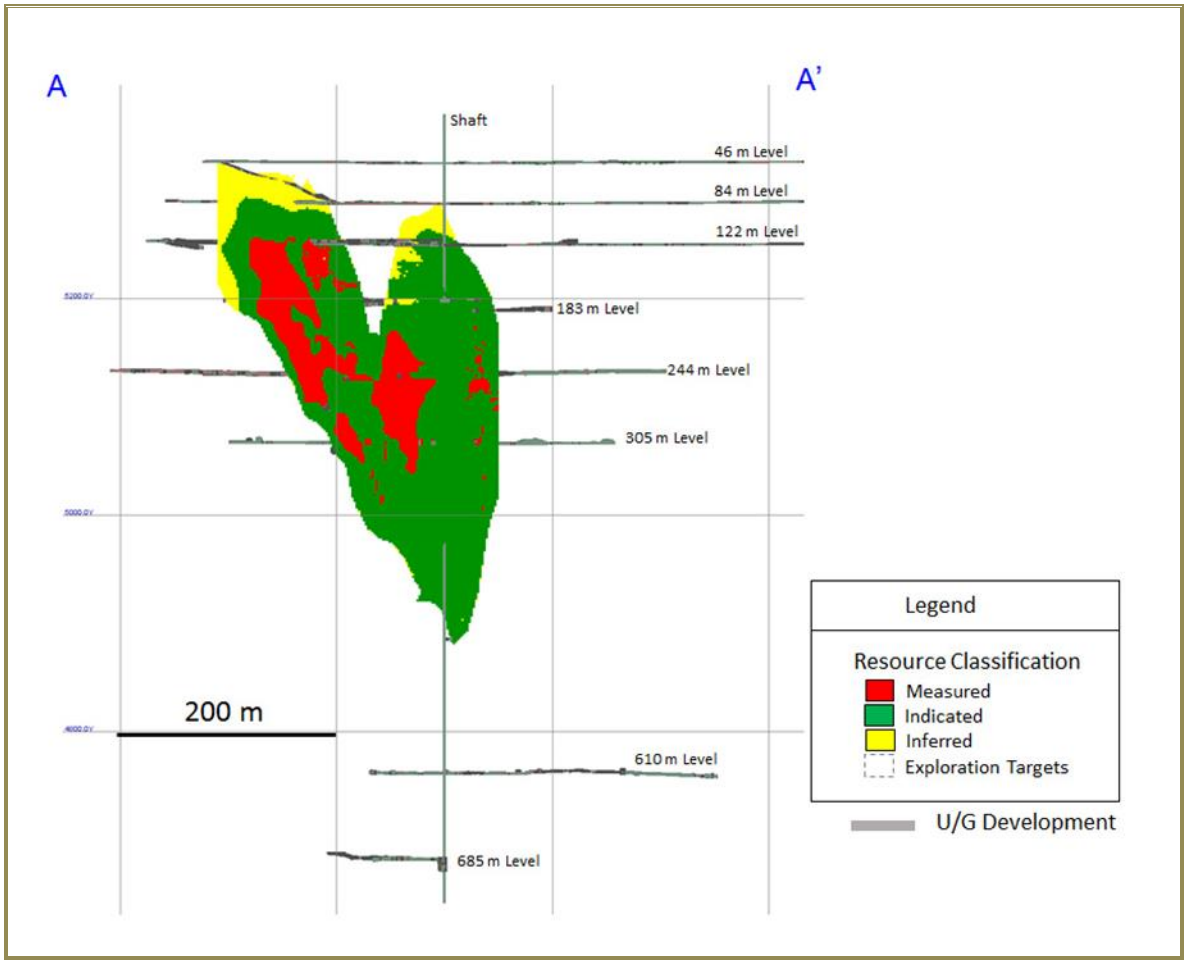


Figure 14-28: 2019 Measured, Indicated, and Inferred Mineral Resource Classification, Zone 1 (Longitudinal View Looking Mine Grid East)

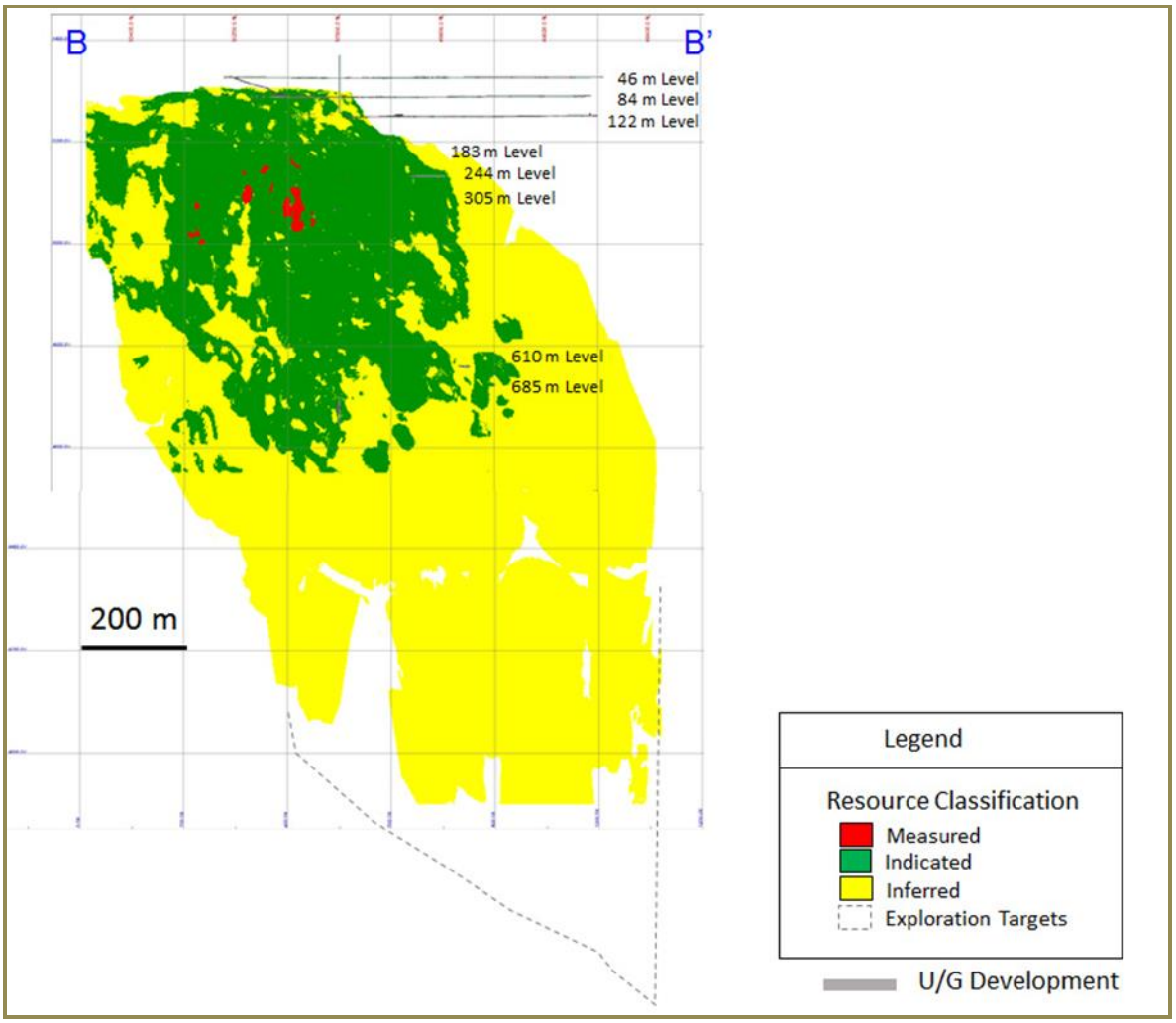


Figure 14-29: 2019 Measured, Indicated, and Inferred Mineral Resource Classification, Zone 2 (Longitudinal View, Looking Mine Grid East)

Table 14-20 reports the 2019 Mineral Resource inventory by underground level for Measured + Indicated and Inferred Mineral Resource.

Table 14-20: Classified 2019 Mineral Resources Reported by Underground Level (Cut-off Grade 3.0 g/t Au)

Metre Level	Measured and Indicated Resource			Inferred Resource		
	Tonnes ('000s)	Grade (g/t Au)	Contained Au (oz '000s)	Tonnes ('000s)	Grade (g/t Au)	Contained Au (oz '000s)
84	8	7.70	2	8	4.90	1
122	51	7.90	13	29	9.70	9
183	170	5.50	30	21	3.86	3
244	248	6.27	50	14	7.14	3
305	284	6.39	58	25	5.68	4
366	396	7.40	94	14	4.88	2
427	429	6.69	92	61	4.81	9
488	347	6.41	71	136	5.38	23
549	274	5.58	49	96	5.56	17
610	167	5.74	31	100	7.21	23
640	208	5.48	37	147	7.22	34
671	7	5.53	1	7	9.20	2
685	95	5.87	18	79	8.23	21
732	99	5.01	16	60	4.93	9
793	124	5.79	23	162	5.88	31
854	21	5.05	3	166	5.37	29
915	-	-	-	322	6.49	67
976	-	-	-	273	8.11	71
1037	-	-	-	177	6.27	36
1098	-	-	-	112	6.59	24
1159	-	-	-	218	6.00	42
1220	-	-	-	120	6.61	26
1281	-	-	-	49	4.51	7
1342	-	-	-	71	8.88	20
1403	-	-	-	53	10.37	18

14.7.2 Cut-Off Grade

The cut-off grade used for the 2019 Mineral Resource estimates is 3.0 g/t Au based on Rubicon's estimated break-even operating expenditure (OPEX) mining cost of C$146.83/t as outlined in Table 14-21. The OPEX cost is based on assumptions of US$1,400/oz Au, a US$ to C$ exchange rate of 0.77, and a gold recovery of 95%. Mineral Resource estimates can be sensitive to the reporting cut-offs used.

Table 14-21: Summary of Assumptions

Item	OPEX (C$/t)
Mining	97.00
Milling	23.79
G&A	6.04
Sustaining CAPEX	20.00
Total	**146.83**

14.7.3 Mineral Resource Statement

The 2019 Mineral Resource Estimate is reported in Table 14-22, as prepared by TMAC for the Phoenix Gold Project (Effective March 18, 2019).

Table 14-22: Phoenix Gold Project 2019 Mineral Resource Estimate Reported at 3.0 g/t Au Cut-off Grade

Resource Category	Quantity (t '000s)	Grade (g/t Au)	Contained Gold Ounces
Measured (M)	442	6.99	99,000
Indicated (I)	2,485	6.13	490,000
M+I	2,927	6.26	589,000
Inferred	2,570	6.53	540,000

Notes: Effective date for this Mineral Resource estimate is March 18, 2019.
Mineral Resource estimate uses a break-even economic cut-off grade of 3.0 g/t Au based on assumptions of a gold price of US$1,400/oz; an exchange rate of US$/C$0.77; mining cash costs of C$97/t; processing costs of C$24/t; G&A of C$6/t; sustaining capital C$20/t; refining, transport and royalty costs of C$57/oz; and average gold recoverability of 95%.
Mineral Resource estimate reported from within envelopes accounting for mineral continuity.
Mineral Resources are not Mineral Reserves and do not demonstrate economic viability.
There is no certainty that all or any part of this Mineral Resource estimate will be converted into Mineral Reserve.
All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.

The 2019 Mineral Resource Estimate is reported at a cut-off grade of 3.0 g/t Au, other cut-off grades are provided to demonstrate tonnage and grade sensitivities.

The 2019 Mineral Resource Estimate excludes mineralization within the crown pillar located between the lake bottom and a depth of 40 m below the lake bottom. In addition, all mineralized development is assigned code 4001 in the block model and removed from the 2019 Mineral Resource estimate. All Mineral Resource blocks outside of Rubicon's claim boundaries are coded as 4002 in the block model and removed from the Mineral Resource estimate.

Mineral Resources are not Mineral Reserves and do not necessarily demonstrate economic viability. There is no certainty that all or any part of this Mineral Resource estimate will be converted into Mineral Reserve.

14.7.4 Gold Grade Sensitivity

Table 14-23 summarizes the sensitivity of the 2019 Mineral Resource Estimate to other potential mining cut-offs. The 2019 Mineral Resource Estimate is reported at a cut-off grade of 3.0 g/t Au, which is highlighted in green in Table 14-23.

Table 14-23: Phoenix Gold Project 2019 Mineral Resource Estimate Sensitivities

Cut-off Grade (g/t Au)	Measured + Indicated Classification			Inferred Classification		
	Quantity (t '000s)	Grade (g/t Au)	Contained Au (oz '000s)	Quantity (t '000s)	Grade (g/t Au)	Contained Au (oz '000s)
2.0	5,237	4.57	770	4,526	4.76	692
2.5	3,861	5.41	672	3,315	5.68	605
*3.0	2,927	6.26	589	2,570	6.53	540
3.5	2,289	7.11	523	2,038	7.39	484
4.0	1,842	7.92	469	1,671	8.19	440
4.5	1,510	8.73	424	1,384	9.01	401
5.0	1,263	9.52	386	1,164	9.82	368

There is no certainty that the Measured and Indicated Mineral Resource estimates will be converted to the Proven and Probable Mineral Reserve categories and there is no certainty that the updated Mineral Resource estimate will be realized. It is reasonably expected that the majority of Inferred Mineral Resource estimate could be upgraded to Indicated Mineral Resources with continued exploration. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. TMAC is unaware of any known environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant factors which could materially impact the 2019 Mineral Resource estimate provided in this Technical Report.

14.8 Comparison to the 2018 Mineral Resource Estimate

The new 2019 Mineral Resource Estimate for the F2 Gold Zone encompasses several changes from the previous 2018 Mineral Resource Estimate (Golder, 2018b). These are:

- An updated interpretation of rock types based on new drilling
- A grooming of isolated blocks in the geological model
- Revised outlier capping and handling during interpolation
- Addition of 106 drill holes completed post-2017
- Addition of a potentially economic area previously modelled outside of Zone 2.

Table 14-24 shows the comparison of the 2018 Mineral Resource Estimate (Golder, 2018b) versus 2019 Mineral Resource Estimate (Effective March 18, 2019). The 2018 Mineral Resource Estimate (Golder, 2018b) is not current and should not be relied on.

Table 14-24: Comparison between 2018 and 2019 Mineral Resource Estimates

Cut-off Grade Classification	Quantity (t '000s)			Grade (g/t Au)			Contained Gold Ounces		
	2019	2018	Change	2019	2018	Change	2019	2018	Change
3.0 g/t Au									
Measured (M)	442	188	135%	6.99	6.8	3%	99,000	41,000	141%
Indicated (I)	2,485	1186	110%	6.13	6.3	-3%	490,000	240,000	104%
Total M+I	**2,927**	**1,374**	**113%**	**6.26**	**6.37**	**-2%**	**589,000**	**281,000**	**110%**
Inferred	2,570	3,884	-34%	6.53	6	9%	540,000	749,000	-28%
3.5 g/t Au									
Measured (M)	335	155	116%	8.18	7.54	8%	88,000	38,000	132%
Indicated (I)	1,954	964	103%	6.92	7.01	-1%	435,000	217,000	100%
Total M+I	**2,289**	**1,119**	**105%**	**7.11**	**7.08**	**0%**	**523,000**	**255,000**	**105%**
Inferred	2,038	3,146	-35%	7.39	6.64	11%	484,000	672,000	-28%
4.0 g/t Au									
Measured (M)	267	129	107%	9.32	8.29	12%	80,000	35,000	129%
Indicated (I)	1,575	779	102%	7.69	7.78	-1%	389,000	195,000	99%
Total M+I	**1,842**	**908**	**103%**	**7.92**	**7.86**	**1%**	**469,000**	**230,000**	**104%**
Inferred	1,671	2,556	-35%	8.19	7.31	12%	440,000	601,000	-27%

Figure 14-30 illustrates the magnitude of the Measured and Indicated (MEA+IND) model changes (2018 depletion and model changes) between the 2018 and 2019 model using a waterfall chart.

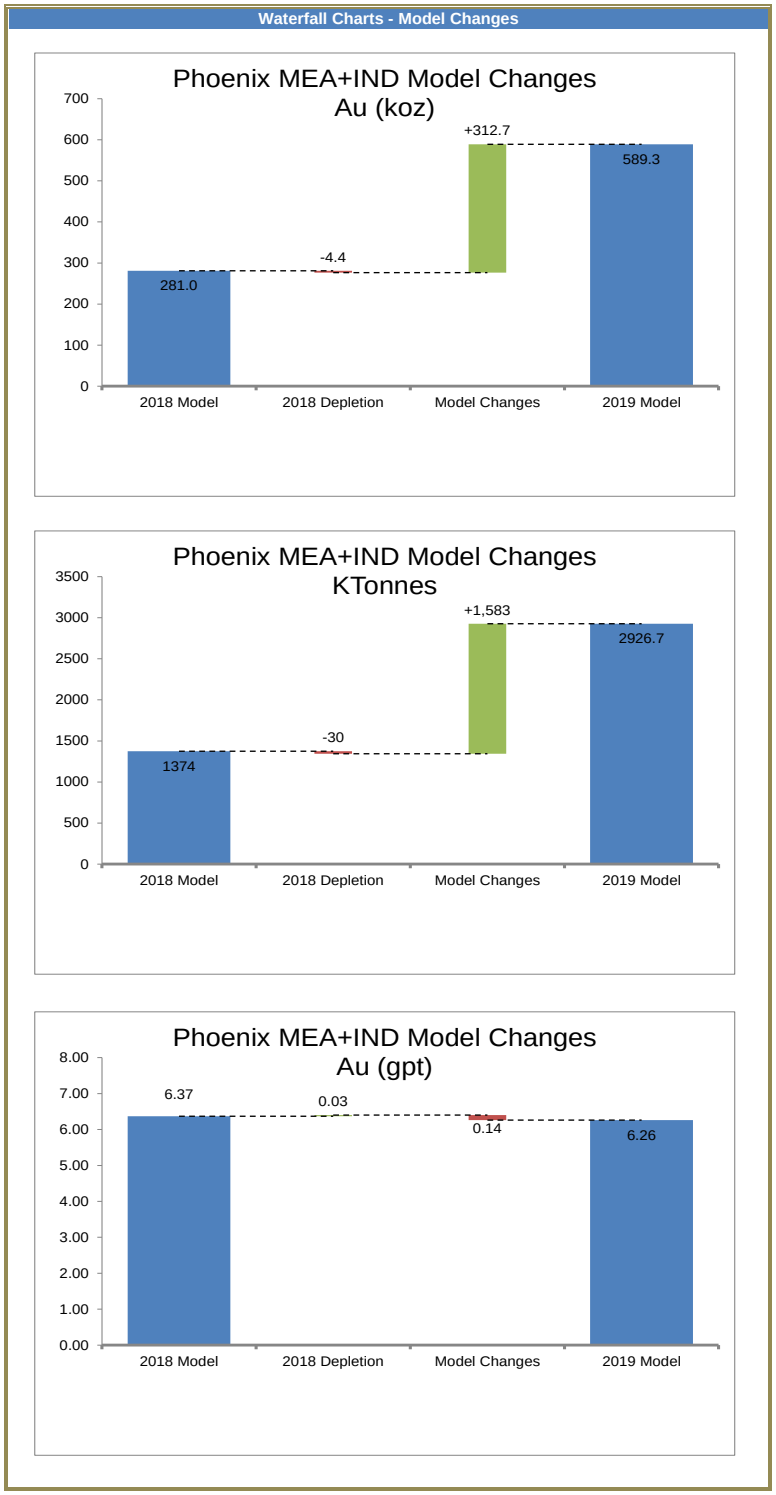


Figure 14-30: Phoenix Model Waterfall Chart

A series of swath plots (Figure 14-31 to Figure 14-33) were generated from capped gold grades to compare the 2019 capped gold grade model (AUCID3) with Golder's 2018 capped gold grade model (AU_IDC). The grades are averaged over 20 m swaths for each of the plots. Locally, differences are noted in the average gold grade but overall the swaths demonstrate a high degree of correlation even though the 2019 Mineral Resource model incorporates additional drill hole data and has modified interpolation parameters and uses different Mineral Resource estimation software than the 2018 Mineral Resource model (Golder, 2018b).

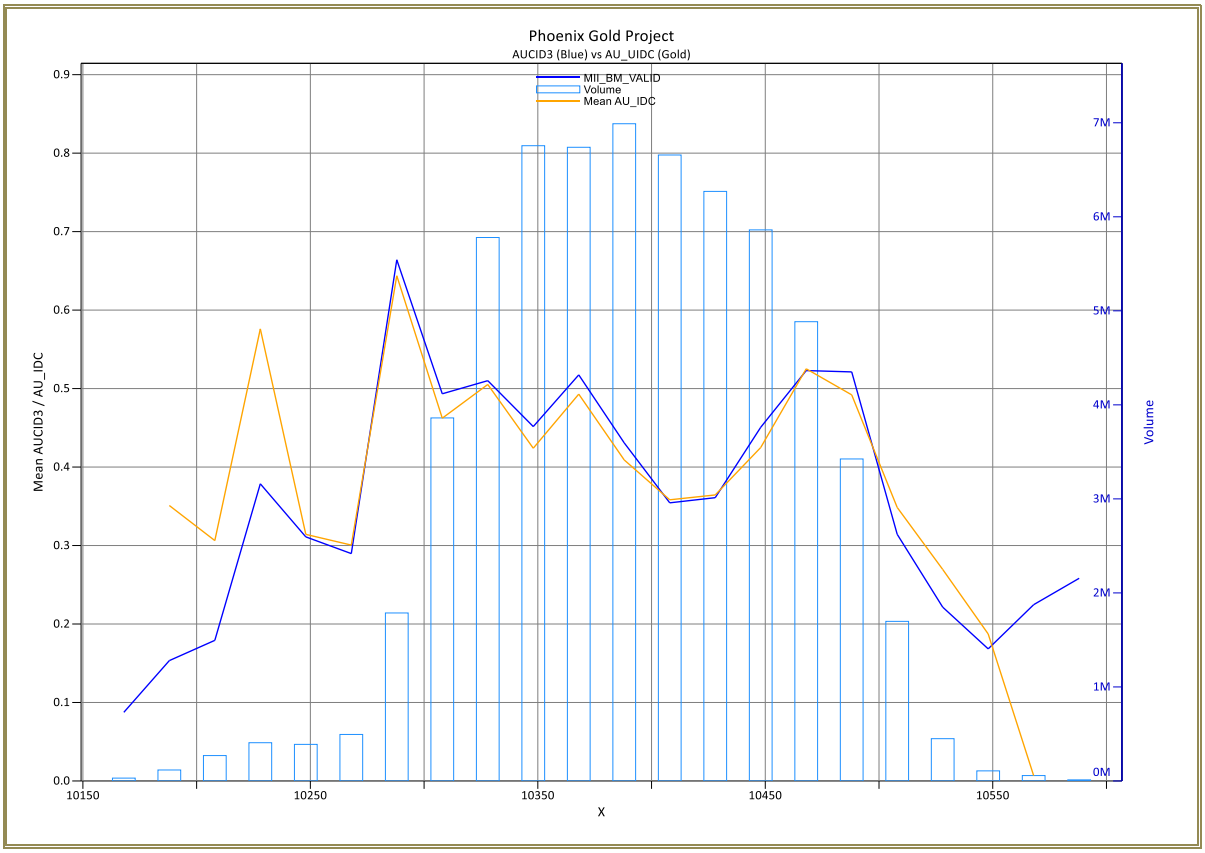


Figure 14-31: Swath Plot by Easting, AUCID3 vs. AU_IDC

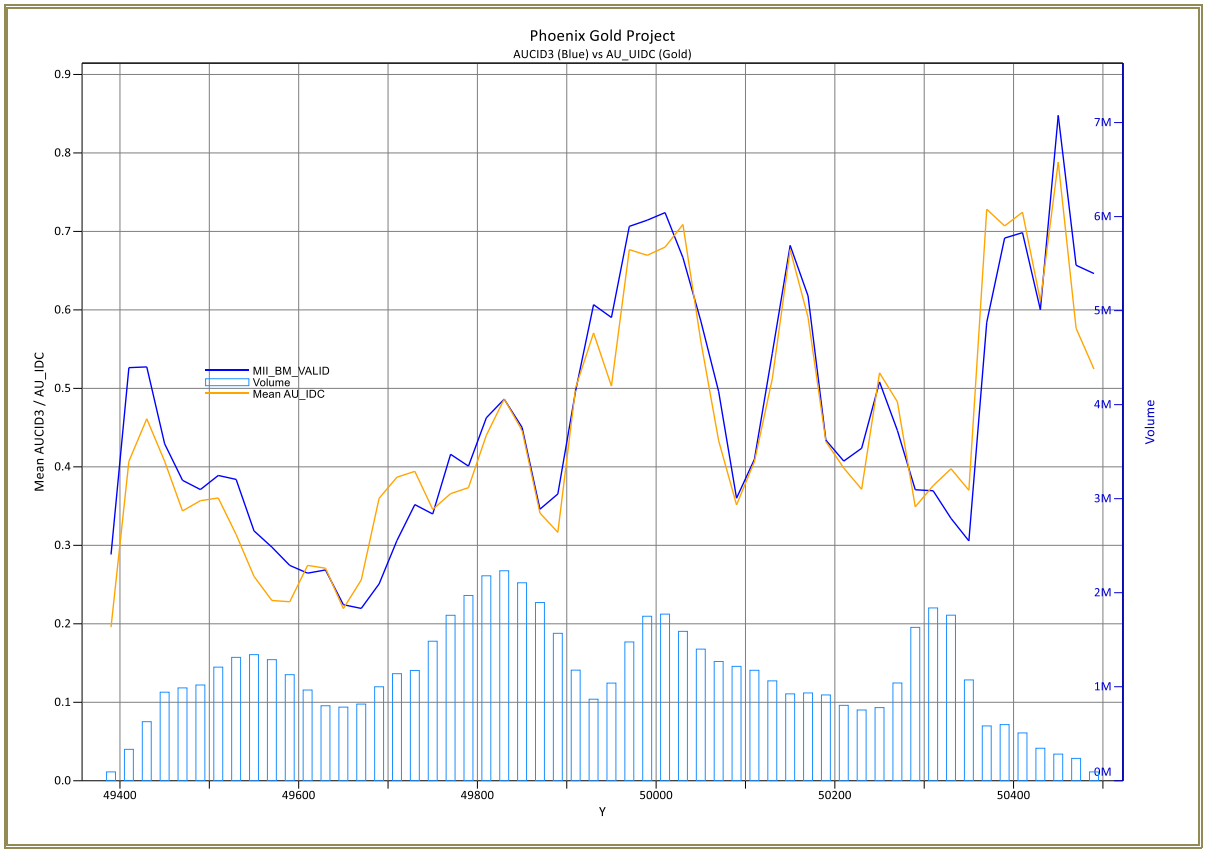


Figure 14-32: Swath Plot by Northing, AUCID3 vs. AU_IDC

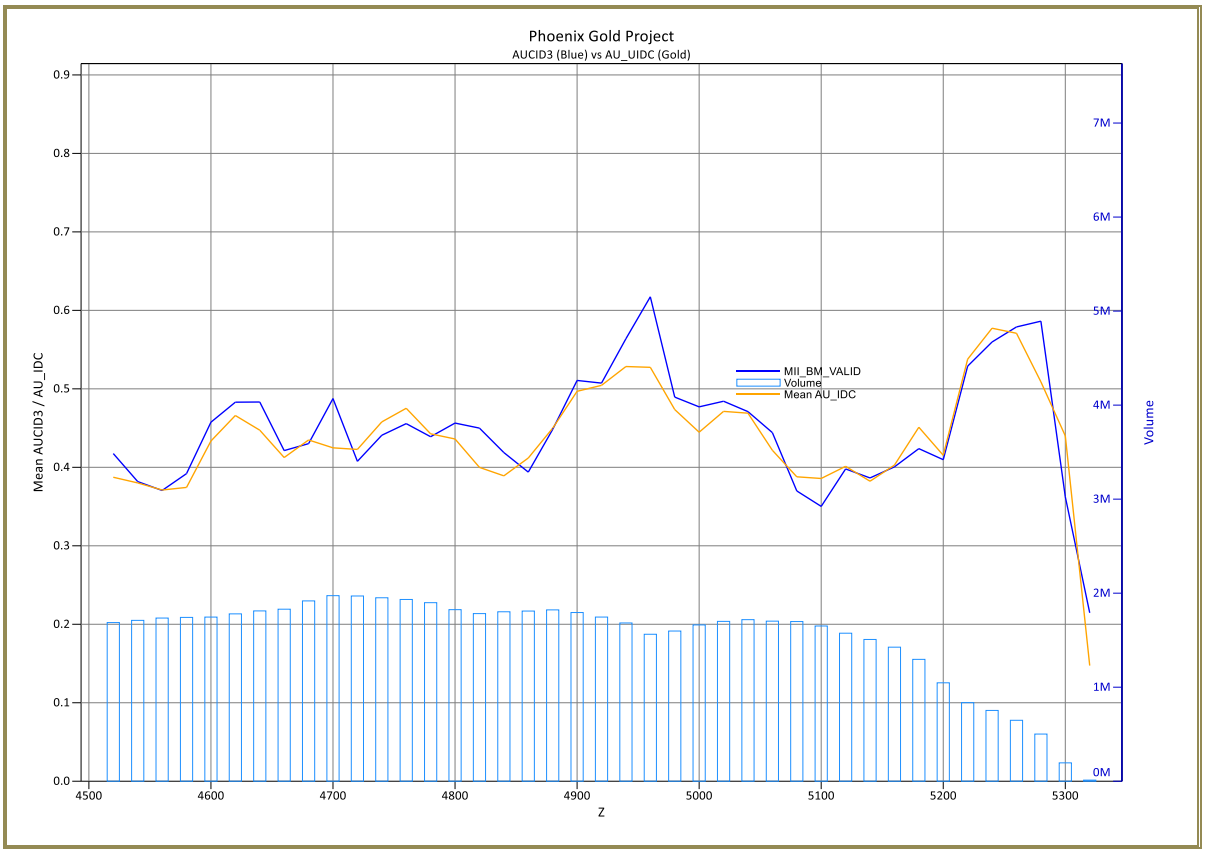


Figure 14-33: Swath Plot by Elevation, AUCID3 vs. AU_IDC

14.9 Comment on 2019 Mineral Resource Estimate

The QP is of the opinion that the Phoenix Gold Project has been interpolated using industry-accepted modelling techniques in GEMS. This included geologic input, appropriate block model cell sizes, grade capping, assay compositing, and reasonable interpolation parameters.

The results have been verified by visual review and statistical comparisons between the estimated block grades and the composites used to interpolate. The IDW3 model (AUCID3) has been selected as the best representation of the grade distribution based on the current geological understanding and gold mineralization. The IDW3 model has been validated with alternate estimation methods: NN and OK. No biases have been identified in the model.

The 2019 Mineral Resource Estimate conforms to the requirements of the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). TMAC is unaware of any known environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant factors which could materially impact the 2019 Mineral Resource estimate provided in this Technical Report. The Mineral Resource estimates are adequate to support mining studies.

Mineral Resources are not Mineral Reserves and do not necessarily demonstrate economic viability. There is no certainty that all or any part of this Mineral Resource estimate will be converted into a Mineral Reserve.

15 MINERAL RESERVE ESTIMATE

The PEA defines no Mineral Reserve estimates applicable to this Technical Report.

The Mineral Reserve definition accepted by CIM is as follows.

"A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined."

16 MINING METHODS

16.1 Introduction

The PEA was commissioned to evaluate the economics of a conservative scenario for extraction of gold mineralization at the Phoenix Gold Project.

Following a site visit, TMAC defined potential mine parameters for the PEA to include:

- Use of existing engineering, infrastructure, facilities, and developments, where it makes sense to do so
- Target throughput above 1,200 t/d but not to exceed 1,850 t/d
- Improve the existing shaft to suit material and personnel transfer
- Consider a surface ramp connection if the shaft is a bottleneck to increased production
- Use conventional diesel equipment
- Include internal ramps to allow inter-level access
- Advance development sufficiently so that mineral production, once commenced, is continuous
- Consider contractor(s) for development, including any raise work, and use company personnel for mine production activities
- Consider only proven C&F, longhole (LH), Uppers, and raise mining methods
- Remove temporary surface buildings and replace with more permanent structures
- Designs and costs to reflect permitting, improvements, and services to satisfy throughput
- Apply first principle engineering and compare productivities and costs to similar gold projects
- Use the 2019 Resource Model (TMAC, 2019) as the basis of a conceptual minable Mineral Resource for stope definition, planning, and scheduling.

Following a review of previous and test stope mining, TMAC engineers and planners applied the following approach to define the conceptual minable portion of the Mineral Resource:

- Upload block model into AMINE, 3D mine design software, and check for conformity when compared to the 2019 Mineral Resource model (TMAC, 2019)
- Perform project-specific hydro-geotechnical and geotechnical literature and computational reviews
- Assign candidate methods to be tested for applicability
- Check methods against geotechnical assumptions for span, strike, and height requirements
- Derive dilution and recovery expectations for each method

- Conduct historical market surveys to define study values for gold price, exchange rates, and mining method costs
- Estimate general and administrative costs (G&A)
- Assess and forecast processing costs and recoveries
- Calculate a general mining cut-off grade
- Approximate level elevation intervals
- Define stopes by mining method and assess a sampling for geotechnical acceptability
- Tabulate tonnes and grades in each stope
- Tabulate dilution grades for each stope by querying stope host grades
- Assign recovery expectations to each stope according to each mining method
- Calculate planned, unplanned, and backfill volumes and dilutions per stope
- Calculate development and/or access costs for each stope
- Run analysis of individual stope mineability based on access, method, tonnes, grade, dilution, and recovery
- Plan and develop takeoffs of each level's access to stoping groups
- Design and quantify takeoffs for ramps, sumps, shops, raises, and other infrastructure to support a mine plan
- Apply a 15% development contingency to ramps and levels to accommodate infrastructure including safety bays, storages, lunchrooms, and so forth
- Create mine plans (schedules) of development, stoping, and backfilling to ramp up to a sustainable mining rate and eventual exhaustion of the conceptual minable Mineral Resource
- Assess environmental and supporting issues including tails impoundment, closure, and permitting costs
- Perform value-added engineering to revisit and revise the model to optimize a conceptual mine plan
- Apply local costs, when known, and budget costs when unknown, for infrastructure, labour, equipment, and power
- Create a schedule of costs and revenues to reflect the PEA.

16.2 Previous Mining

The Phoenix Gold Project property has never been in commercial production to date, though several bulk samples have been taken in the past on both the F2 Gold Zone and the unrelated mineralization that was being assessed at the historical McFinley Zone. Test mining and milling was conducted in 2015 on the F2 Gold Zone at the Phoenix Gold Project. A total of 57,793 tonnes of mineralized

material was processed in 2015, and during the test mining and bulk sample program completed in 2018, another 43,250 tonnes of mineralized material was processed.

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

In 1984, the shaft was reopened as the Phoenix Shaft, and an additional 479 m of lateral development was completed on the 46 m and the 122 m Levels. After a temporary shutdown starting in February 1985, a further 1,151 m of lateral development was completed prior to the decision to take a bulk sample in 1987. The bulk sample program started in July 1988 from prepared stoping areas. Mining exploration activities on the property were terminated in 1989 after test milling of an estimated 2,250 tonnes of material unrelated to the F2 Gold Zone. The level naming convention for the mine was originally measured in feet below the shaft collar. The 400 ft Level was the original bottom Level of the McFinley Zone, and is now referred to by its metric equivalent, the 122 m Level. The Phoenix Gold Project uses the metric system.

Rubicon acquired the property in June 2002 and resumed exploration work. In 2009, the existing shaft was dewatered and reconditioned to support an advanced exploration program. In June 2009, shaft sinking started to deepen the existing shaft to 350 m, and a loading pocket was installed to support development at the 305 m Level, followed by lateral and vertical development on the 244 m and 305 m Levels. This led to two approximately 1,000-tonne bulk samples being excavated on the 305 m Level in 2011, using development methods.

Shaft sinking resumed in July 2012, after upgrading the headframe and hoisting plant. It was slowed significantly due to a zone of squeezing ground encountered during this phase of the shaft sinking through Ultramafic units. Installation of concrete reinforcing rings and other measures were taken to ensure these issues would not cause potential future delays. The shaft was completed to a depth of 730 m in December 2013.

Lateral and vertical development continued from January 2014. In 2015, the Project underwent a period of trial stoping, bulk sampling, and milling. In June 2015, Rubicon announced its first gold pour from the bulk sampling. In November 2015, Rubicon announced it was suspending underground activities at the Project while it enhanced its geological model of the F2 Gold Zone.

In 2017, exploration work recommenced at the site. This consisted of an underground drilling program to provide structural geological information and infill drilling to expand the estimated Mineral Resource. The underground diamond drilling program continued for all of 2018.

Table 16-1 lists total lateral development completed at the Project up to the end of 2018. Total hoisted tonnage in 2015 was 60,077 tonnes, and in 2017 and 2018 was 43,700 tonnes, as accounted for in Table 16-2.

Table 16-1: Underground Lateral Development by Level

Description	Pre-2017 Quantity*	2017-2018	Total by Level
All ramps	1,359	47	1,406
46 m Level	1,742		1,742
84 m Level	1,549		1,549
122 m Level	2,909	13	2,922
183 m Level	1,210	184	1,394
244 m Level	2,022	127	2,149
305 m Level	2,393		2,393
610 m Level	296	240	536
684 m Level	188	8	196
Total (m)	**13,669**	**619**	**14,287**

Note: *To the nearest metre. Diamond Drill and Safety Bays Excluded

Table 16-2: Mineralized Material Hoisted in 2014, 2015, 2017, and 2018

Hoisted (wet tonnes)	2014	2015	2016	2017	2018	Total
Waste	166,383	188,192	-		10,340	364,915
Development Material	503	33,670	-	777	6,043	40,993
Stope Mineralized Material		26,407	-		36,019	62,426
Total	**166,886**	**248,269**	**-**	**777**	**52,402**	**468,334**
Mineralized Material	503	60,077	-	777	42,062	103,419

16.3 Previous Test Mining Description

During the initial test mining completed in 2015, 11 mining blocks were in various stages of development and mining. In November 2015, the decision was made to suspend the underground activities until further evaluation of the deposit was completed. In general, all test stopes were developed for sub-level longhole stope mining. Access to the mining blocks (sub-levels) was gained via an Alimak raise climber (Figure 16-1).

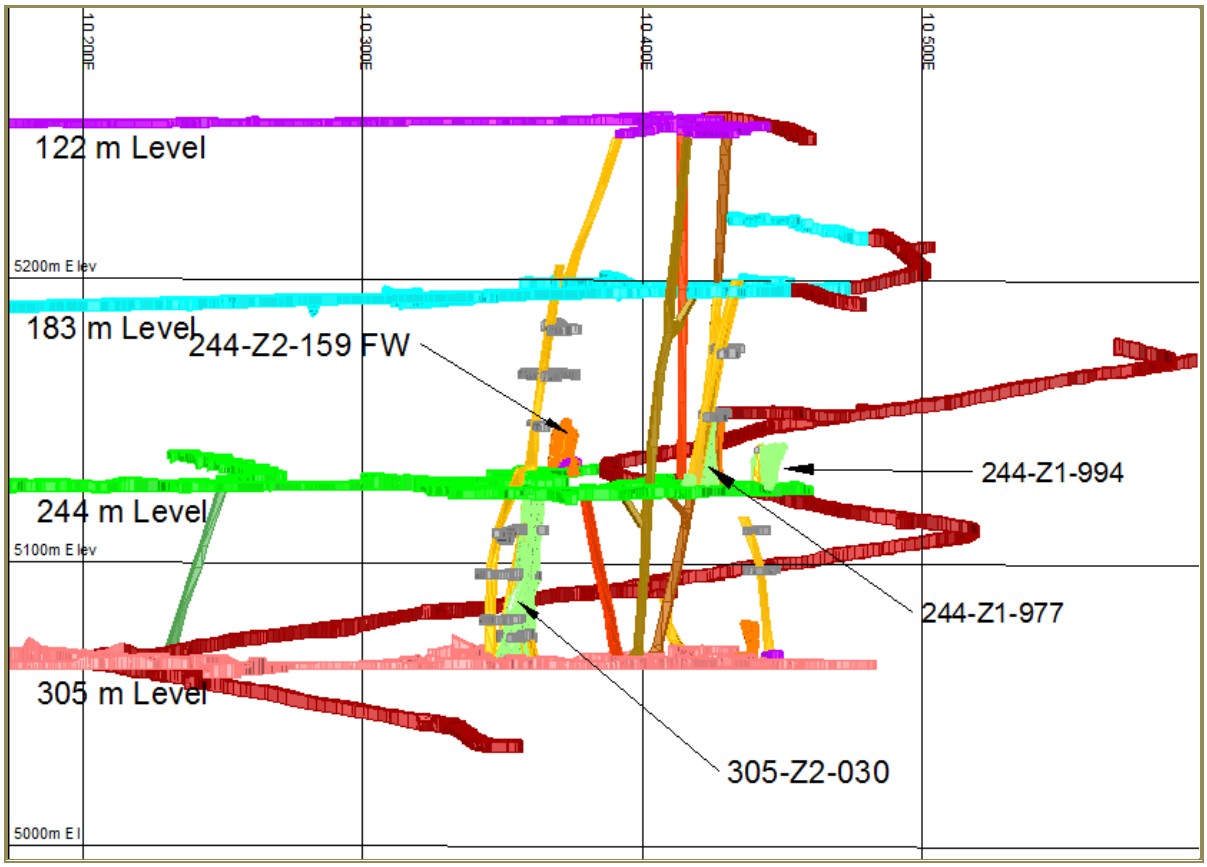


Figure 16-1: Location of Trial Stopes (Looking Grid North)—2015 Test Mining (Rubicon, 2019)

Development of longhole test stopes (e.g., Figure 16-2) followed the general sequence below:

- Delineation of the stope with diamond drilling
- Development of an Alimak raise on the hanging wall contact between the Ultramafic and the High-Ti Basalt from one elevation to the next
- Sub-levels development from the Alimak raise at 15 m intervals, except for the 244 m Level-977 stope, which had a sub-level interval of 20 m; all sub-levels were developed using handheld pneumatic drills and Cavo loaders
- The geology department completed geological mapping and face sampling of the development areas associated with each stope block, and integrated all other relative information to produce a geological shape within the High-Ti Basalt, which then defined the mining block; following a geotechnical evaluation, the engineering department then designed a sequence of extraction that best suited local ground conditions and production efficiencies

- Drilling the mining blocks, from one sub-level to the next, was completed with top-hammer pneumatic longhole drills
- Typically, a slot was opened at one end of the first block and blasted to the mucking horizon where the mined material was removed via a remote-controlled load-haul-dump (LHD) unit; the muck was transported to either the ore pass or direct loaded into ore cars on the 305 m Level
- Following the completion of mining, the excavation was surveyed via a cavity monitoring system to enable comparison of the design shape to the actual excavated opening.

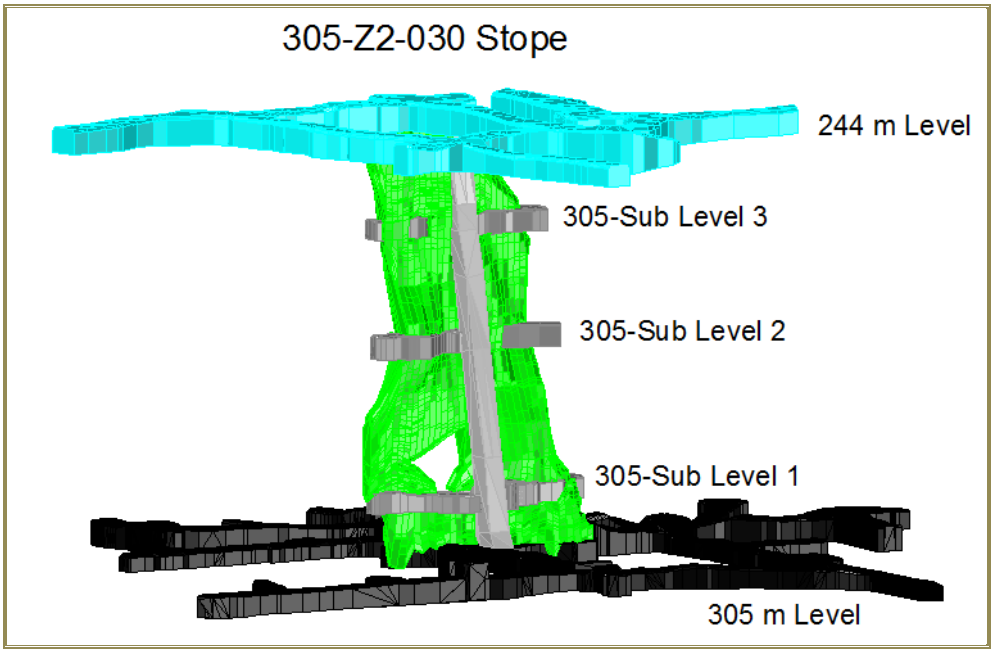


Figure 16-2: 305-Z2-030 Stope (Looking Grid East) Mined in 2015 and Filled in 2018 (Rubicon, 2019)

Mine infrastructure (Figure 16-3) includes muck handling facilities for all levels, a ventilation system, a paste backfill plant, an underground distribution system (partially completed), a mid-shaft loading pocket complete with spill pocket, and a shaft bottom loading pocket. Ramp access has been established between the 305 m and 244 m Levels. Remaining ramp connections from the 244 m Level up to the 122 m Level are within 380 m of completion. A ramp from surface to the 122 m Level has been designed which would be 800 m in length. The cyan outline in Figure 16-3, represents lake bottom and has been included for reference.

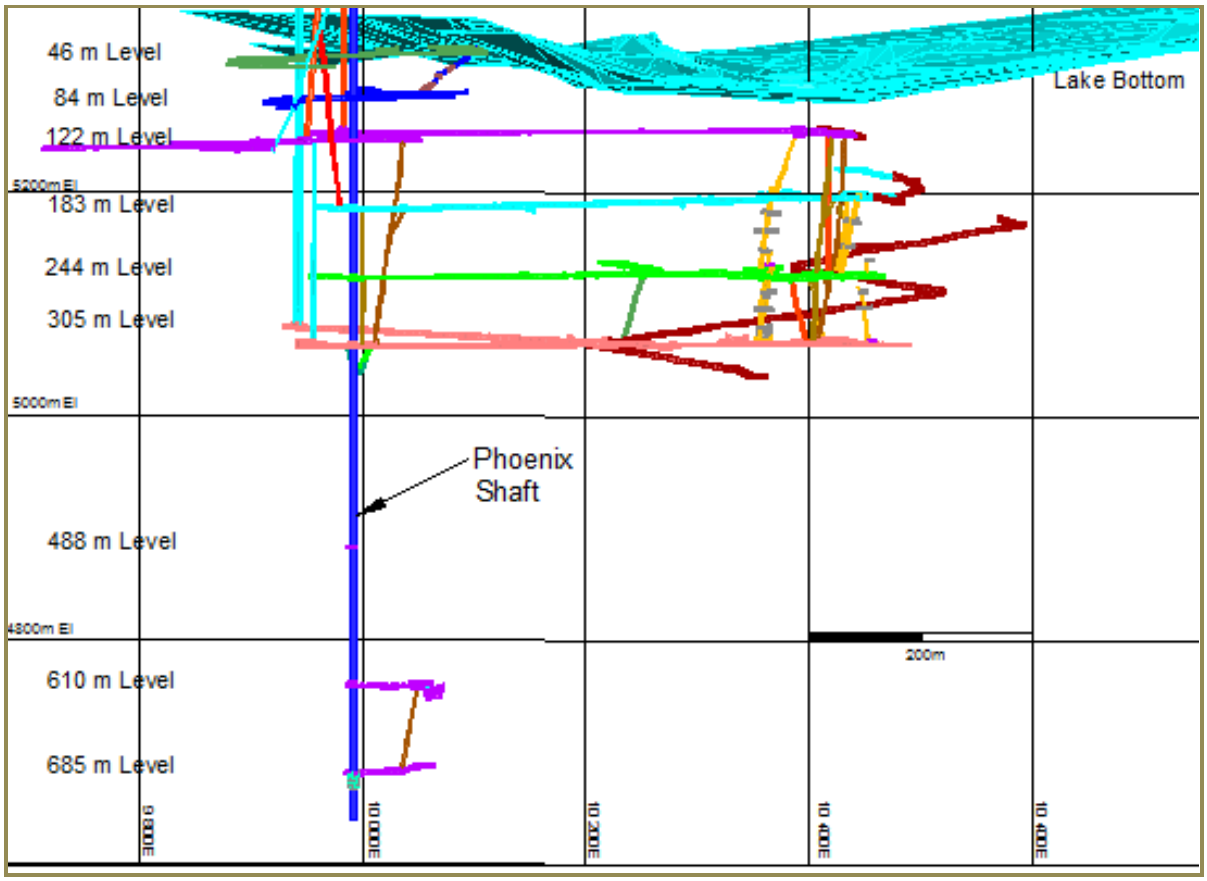


Figure 16-3: Phoenix Gold Project—Underground Workings (Looking Grid North) (Rubicon, 2019)

16.4 Existing Project Infrastructure

This section updates the Project infrastructure at the site. In each section, a brief description of the infrastructure is given with an update near the end of the section.

16.4.1 Surface Infrastructure

The Phoenix Gold Project site is accessed via a dedicated 8-km gravel road from Nungesser Road in the Municipality of Red Lake. The road is nominally 10 m wide within a 50 m right-of-way. Entry into the Project facilities is via a single-entry point onto the property. Access to the property and plant is secured by fencing, gates, and security on 24-hour service. A network of gravel roads on site provides vehicular access to the Project infrastructure.

A significant amount of infrastructure has been constructed. The main surface infrastructure, shown in Figure 16-4 and Figure 16-5, includes:

- Hoist, headframe, and hoist house
- Processing plant
- TMF
- Effluent treatment plant
- Electric power supply and substation
- Propane storage tanks
- Fibre-optic communications cable
- Compressed air supply
- Process and potable water supplies
- Mine ventilation fans and heater house
- Offices and storage buildings
- 200-person camp (currently shuttered).

Hoisting Facility

The Phoenix shaft hoist is a Canadian Ingersoll Rand double-drum hoist, with 4.27 m (14 ft) diameter drums, and two 932 kW (1,250 hp) motors.

The hoist control system, provided by Hepburn Engineering, consists of dual Allen-Bradley programmable logic controllers operating TMEIC fully-regenerative, master/slave IGBT AC drives. The three-compartment shaft was deepened in 2013 to 730 m below surface and includes operational loading pockets at the 337 m Level and 685 m Level. The production conveyances include a skip over double-deck cage combination and second identical skip/cage configuration is operated in balance. Each skip has a capacity of 10 tonnes. Development waste rock hoisted to surface is dumped into a waste bunker beside the headframe. Waste rock is currently stockpiled on site in designated areas. Mined material hoisted to surface can either be dumped into the waste bunker beside the headframe and moved to a designated surface ore storage location or hoisted and dumped in a small coarse-ore bin adjacent to the headframe, from where it can then be conveyed into the mill for processing. The following upgrades or repair work were completed during 2017 and 2018:

- Installed safety railings and screens in hoist room to restrict access to hoist machinery
- Recertified skip and cage conveyances for a five-year duration
- Certified hoist ropes
- Repaired hoist drum, replaced bushing, and re-machined drum brake face
- Completed major hoist servicing – replaced all lubrication and filters

- Installed fire suppression system on hoist lubrication system
- Upgraded hoist control system, including updating of the slack rope system
- Repaired 337 m Level loading pocket and placed back into operation
- Repaired 337 m Level spill pocket and placed back into operation
- Repaired 685 m Level loading pocket and placed back into operation
- Commissioned rock breaker, grizzly, and waste-pass system at 337 m Level loading pocket
- Cleaned shaft timber and all catchment pits in shaft
- Completed enclosure of shaft manway
- Repaired and serviced both ore and waste pass chutes on the 305 m Level.

There are a number of alternatives for access to depths below the current shaft bottom of the 730 m Level. These include a third phase of shaft deepening, sinking of an internal winze closer to the mineralized zone, ramp access, or a new shaft. Economic and logistic viability study of each of these alternatives has not been conducted.

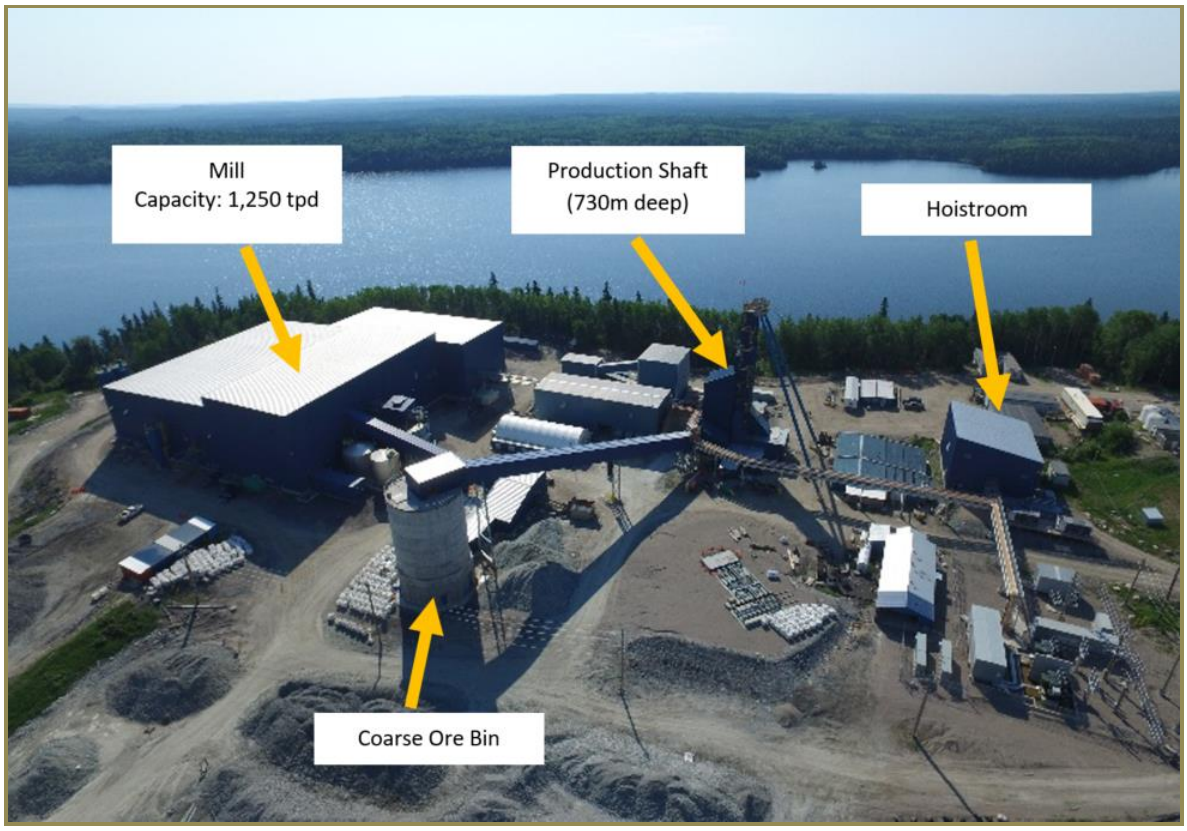


Figure 16-4: Project Site – Looking East (Rubicon, 2019)

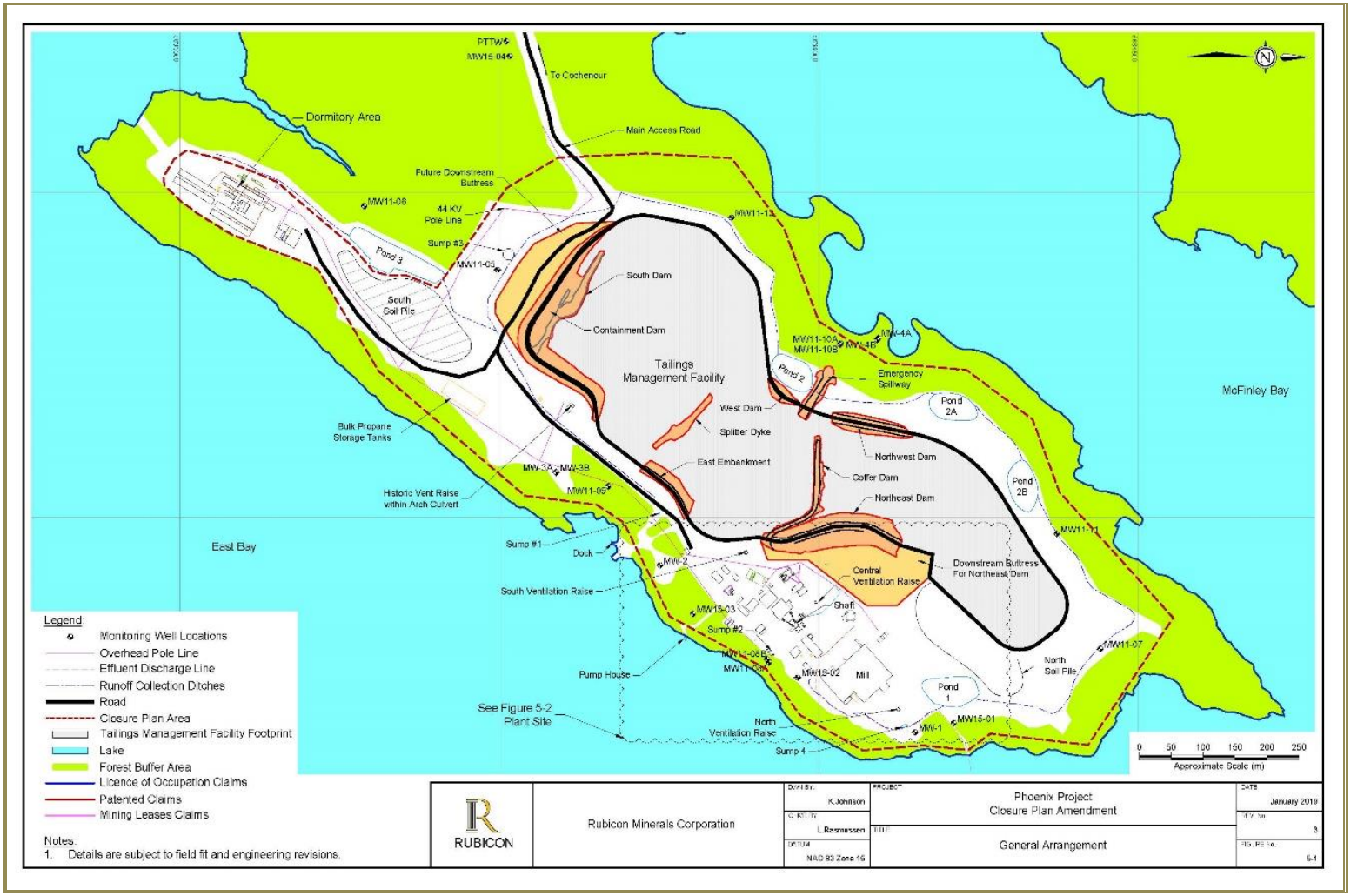


Figure 16-5: Detailed Site Plan of Project Area (Rubicon, 2019)

Processing Plant

The mill is designed for a base processing rate of 1,250 t/d, and can be upgraded incrementally to handle processing rates of 1,800 t/d and 2,500 t/d. Details of the processing facility design and recovery methods are presented in Section 24.8.

The mill houses a paste backfill plant that will produce a cemented paste fill product from the tailings. The paste fill will be pumped underground for placement into voids.

The following upgrades were completed in 2018:

- Install screen guards along SAG Mill feed belt
- Upgrade Lime distribution system
- Update Knelson concentrators
- Modify trash screen system to separate work and plastics from the ore.
- Re-pipe several sections of the mill to provide better control of reagent addition
- Upgrade pre-heat heat exchanger to allow proper flows in elution circuit
- Rebuild service and process water pumps
- Replace SAG mill grates with rubber units
- Replaces gauges and instrumentation throughout mill
- Install Air Dryer and Low-Pressure Compressors to provide air to the various systems throughout the mill.
- Commission mill fire suppression system
- Modify ventilation system in gold refinery.

Tailings Management Facility

The historical TMF consisted of a dam and pond. McFinley constructed the containment pond in 1988 and operated it under a Certificate of Approval. After test milling a bulk sample in 1989, the facility received minimal use. Rubicon upgraded and reactivated the TMF and obtained the necessary government approvals when they took over operations of the site.

The tailings dam will be raised in planned stages periodically over the life of the mine to increase the capacity of the TMF as more tailings are produced. Foundation investigation has been carried out for the current design. For future dam raises, similar foundation investigations will be required to refine the designs. The location of the TMF and related facilities are presented in Figure 16-5.

The TMF design utilizes mine rock that was hoisted to surface for the construction of the TMF dams, buttresses, and other on-site components.

The TMF is designed to withstand a 30-day duration of a 1-in-100-year rain-or-snow event. The mill has a cyanide destruction system that treats tailings slurry prior to discharge to the TMF. Discharge from the TMF is processed by an Actiflo® clarification and metals precipitation system with a capacity of between 780 m^3/d and 3,100 m^3/d. This system is designed to remove total suspended solids and metals from the water prior to discharging it to the environment. Rubicon is permitted to discharge a yearly maximum of 3,100 m^3/d of water to the environment from March 16 to November 30. The metals precipitation component of this system was installed in 2017 and commissioned in 2018.

Power and Communications

Electricity Supply

Rubicon has an agreement with Hydro One to supply power to the site and has been granted an allocation of 5.3 MW of power (this was reaffirmed with Hydro One in late 2018). The on-site electrical supply is from the 44 kV M6 Hydro One line fed from the Red Lake Distribution Station (DS). This feeds the main substation, which contains two 18 MW transformers feeding a common 5 kW bus supplying the site.

The underground electrical distribution system consists of:

- One – 3 conductor 4/0 AWG 5 kV Teck 90 cable installed in the shaft from the surface winch room to the 305 m Level (4160 V)
- One – 3 conductor 350 MCM 5 kV Teck 90 cable installed in the shaft, going from the surface winch room down to the 610 m Level (4160 V)
- One – 3 conductor 500 MCM 1 kV Teck 90 cable from surface to the 122 m Level (600 V).

The underground power distribution system will need to be upgraded once the mine goes into full production. The design necessary for the expansion includes the installation of two 500 MCM 5 kV Tech 90 cables from the surface powerhouse down drill holes to the 122 m Level, continuing down the emergency escape-way to all accessible levels. A disconnect is planned for each of the 122 m Level, 183 m Level, 244 m Level, and 305 m Level. A substation has been installed on the 610 m Level for diamond drilling in the area.

Provision for a future feeder upgrade to 13.8 kV for underground distribution has also been procured with the necessary switchgear and shaft cabling presently being stored on site.

The following upgrades were completed in 2018:

- Commissioning of two diesel generating units (each having 1 MW in capacity) to provide power to various systems on site in case of loss of grid power.

Propane

The main propane tank farm, located at the south end of the property, has a capacity of 226,000 L. This facility is used to provide propane to the Project site for heating ventilation air going underground during the winter and provides heating fuel to all of the buildings used on site. There are also three 6,000-L tanks located at the dormitory (camp), and a 3,000-L tank at the pole barn, but these are not in use at this time.

The following upgrades were completed in 2018:

- Tie in downcast heaters to main propane tank farm.

Natural Gas

Natural gas supply is available in the Red Lake area and could be considered an energy alternative in the future.

Fuel Storage

A 25,000 L above-ground diesel fuel storage tank and dispensing station is currently located beside the electrical warehouse building. The facility has the requisite spill storage capacity and meets other fuel storage requirements of the Technical Standards & Safety Authority (TSSA).

There is a small (4,100 L) gasoline dispensing facility on site, adjacent to the diesel fuel storage tank. The facility has the requisite spill storage capacity and meets other fuel storage requirements of the TSSA.

Communications

Site surface communication is via a VOIP telephone system. The system is connected by a fiber-optic cable installed along the same route as the electrical power supply line. Radios are used for site-wide communications.

Communication systems underground include a leaky feeder system and FEMCO telephones located in shaft stations and refuge stations. The Emergency Control Centre, which is located in the technical services building, is also equipped with a FEMCO phone, as is the security gatehouse.

Fiber-optic cable has been installed throughout the site, including in the shaft. It is in operation with provision for additional expansion for future communications and instrumentation applications on surface and underground.

Compressed Air Supply

The Project currently has two 261 kW (350 hp) air compressors (Sullair TS32-350) rated at 3,186 m^3/h (1,875 cfm) each, and a small back-up compressor unit (Atlas Copco GA160). These units provide adequate volumes for the work presently being completed at site. Additional compressors will need

to be added to the system relative to the tonnages that are planned to be mined in the future. The compressors are housed in a permanent structure, with temperature-controlled louvres to exhaust heat from the building. The compressors operate on a cascading system controlled by local controllers on each unit. The two larger units operate on a continuous basis, and cycle between loaded and unloaded. Status of the underground distribution system is described in Underground Infrastructure, Section 24.3.2.

Process and Potable Water Supply

Lake water is pumped from the adjacent East Bay of Red Lake to feed the process water and underground activities. The authorized pumping rate from the lake through Rubicon's Permit to Take Water 3585-85KGHG, is 695 L/min, with a maximum daily total of 1,000,000 L/d. When the mill is in operation, process water in the mill and water accumulated in the TMF will be recirculated back into the mill process water supply system, thereby minimizing the amount of water pumped from the lake.

The underground dewatering system reports to the TMF and is authorized by Permit to Take Water 3812-9C9KVF for a maximum pumping rate of 2,917 L/min, up to a maximum of 2,100,000 L/d.

Currently, water discharged to the environment from the TMF comes under regulatory control and can be discharged between March to November. It must meet the objectives and limits outlined in Environmental Compliance Approval #1362-AA2HXS.

Potable water for the site is taken from East Bay and conditioned by a Culligan system of nanomembrane modules, UV bacterial disinfection, and chlorine addition prior to use.

Sewage Treatment Facility

The Project's domestic and industrial sewage systems are regulated by Environmental Compliance Approval #1362-AA2HXS.

Currently, all domestic sewage is collected on site at two collection tanks until regular pump-outs are conducted by a third party and the sewage is taken off site to permitted sewage treatment facilities. This process is permitted under Provincial Officer's Order #7655-AMAQDJ.

Mine Ventilation Facilities

Mine ventilation is currently being supplied via a fresh air surface installation on the 122 m Level Fresh Air Raise. The system consists of one 54" diameter (137 cm) 250 hp fan, complete with an associated propane-fired heater and ancillary equipment. This system provides approximately 115,000 cfm to the underground workings and is adequate for the ongoing work program. When the mine is commissioned for full production, it will require up to 370,000 cfm, which will be supplied by two 72" (183 cm) diameter 250 hp fans and their associated heaters and ancillary equipment.

Other Site Buildings

Facilities provided by other buildings near the Phoenix shaft include:

- Muck handling (conveying) and coarse ore storage system for the processing plant
- Mine Dry
- Huddle room for underground crew assignments
- Technical
- Offices for Geology and Engineering
- Core handling facilities and core storage
- Maintenance shop
- Cold storage.

Waste Rock Stockpiles

The waste rock storage area is located on the northwest corner of the peninsula, in a containment area previously referred to as the quarry. All future waste will be placed there for further assessment as potential construction material.

Production Material Stockpiles

Stockpiles for ore, mineralized rock, and waste were re-established for the bulk-sampling program conducted in 2018. All mineralized material processed through the mill was discharged into the TMF. Waste development excavated during the 2018 test mining process was either stockpiled underground, used as stope fill, or hoisted to surface and stored in designated locations. Waste rock was also used as bedding material for the mill during the initial stages of bulk sample processing.

Explosives Magazines

No surface explosive magazines are planned. Upon delivery to site, explosives are moved to authorized magazines underground for storage.

Assay Laboratory

An assay laboratory is located off site in a commercial mall in Balmertown. It has facilities for crushing, pulverizing, fusion, cupellation, acid digestion, and atomic absorption analyses. The two fusion furnaces each have capacity for 42 crucibles, heated to temperatures from 850°C to 1,060°C. The laboratory is capable of processing a maximum of 252 samples every three hours. Currently, the assay lab is closed, and third-party facilities are being utilized for all assay work.

16.4.2 Underground Infrastructure

The underground infrastructure required to support production mining includes material handling facilities, mine dewatering system, paste backfill distribution system, equipment repair bays, ventilation system, supply lines for compressed air and process water, electrical power supply, and miscellaneous facilities.

Material Handling

The material handling system is divided into the upper material handling system from 122 m Level to 305 m Level, the lower material handling system from 366 m Level to 685 m Level, and the material handling below the 610 m Level.

Upper Material Handling System

The upper material handling system consists of a series of connected raises between the 122 m Level and 305 m Level where the ore and waste are then transported by rail. This system allows both ore and waste movement from each level to the mid-shaft loading pocket on the 337 m Level. Construction of ore and waste passes on the 122 m Level is 10% complete. The 183 m Level and 244 m Level's ore and waste passes are operational. Chutes are installed and operational on the ore and waste passes on the 305 m Level. Haulage to the shaft is operational, with two rock breaker/grizzly installations, one for ore and one for waste.

Lower Material Handling System

To date, a 10-tonne loading pocket has been commissioned on the 685 m Level. This system is currently in operation and handling waste material from the 610 m Level and 685 m Level. The future design includes a rockbreaker/grizzly screen combination on 610 m Level, with a chute at the bottom of the waste pass raise on 685 m Level. This chute will transfer waste rock to a conveyor arrangement that will feed the 685 m Level loading pocket.

An ore system is also designed that will accept material from the 610 m Level through a grizzly rockbreaker arrangement down a raise to the 685 m Level, where the sized material goes to a chute on 685 m Level. This chute will transfer the sized mineralized material to a single conveyor arrangement (the same one that moves the waste material) which will feed the 685 m Level loading pocket.

Below 610 m Level Material Handling System

Pending continued exploration, alternatives for accessing the mineralized zone at depths deeper than the 610 m Level will be evaluated.

Mine Dewatering

Main dewatering stations are located at shaft bottom, and the 610 m, 305 m, and 122 m Levels. This system is capable of pumping at a maximum flow rate of approximately 757 L/min (200 US gallons per minute [gpm]) and is adequate for the current Project work (Figure 16-6).

An upgraded system capable of pumping 3,028 L/min (800 US gpm) from the 305 m Level to surface is partially complete. The current Project permit allows dewatering at a rate of 2,917 L/min (771 US gpm) and a maximum of 2.1 million litres per day (ML/d) (0.56 M US gallons per day [gpd]). Further engineering work will be required to finalize the mine dewatering system for production.

Compressed Air Distribution System

The main compressed air line is installed in the shaft and consists of a 150 mm (6") line from surface to the 305 m Level, and a 200 mm (8") line from there to shaft bottom. While adequate for exploration purposes, the system will require additional capacity to accommodate expected production rates. Construction of the compressed air distribution system upgrade is approximately 20% complete and will be finished prior to commissioning of the mine.

Refuge Stations

There are four completed refuge stations located underground, on the 122 m, 183 m, 244 m, and 305 m Levels. Additional refuge stations will be required once mine development progresses. The constructed refuge stations meet Ministry of Labour requirements.

Paste Backfill Distribution System

The paste backfill distribution system is 90% complete for supplying material to workings above the 305 m Level. Piping has been run on all but one level underground and all but one pipe interconnection has been prepared. This work will be completed prior to commissioning of the mine.

The surface plant requires final connections, process control configuration, testing and commissioning before backfill can be consistently delivered underground. The final connections will be completed prior to commissioning of the mine.

Laboratory testing of binder types and mixtures have been completed. Operational testing will be required to achieve optimal binder addition to achieve desired backfill strengths and costs.

Underground Equipment Servicing

There are three service bays areas located on the 183 m, 244 m, and 305 m Levels. Equipment servicing is completed at these locations. A review of the need for an underground repair shop will be completed prior to commissioning of the mine, as it is possible to do major servicing on surface with a ramp completed to surface.

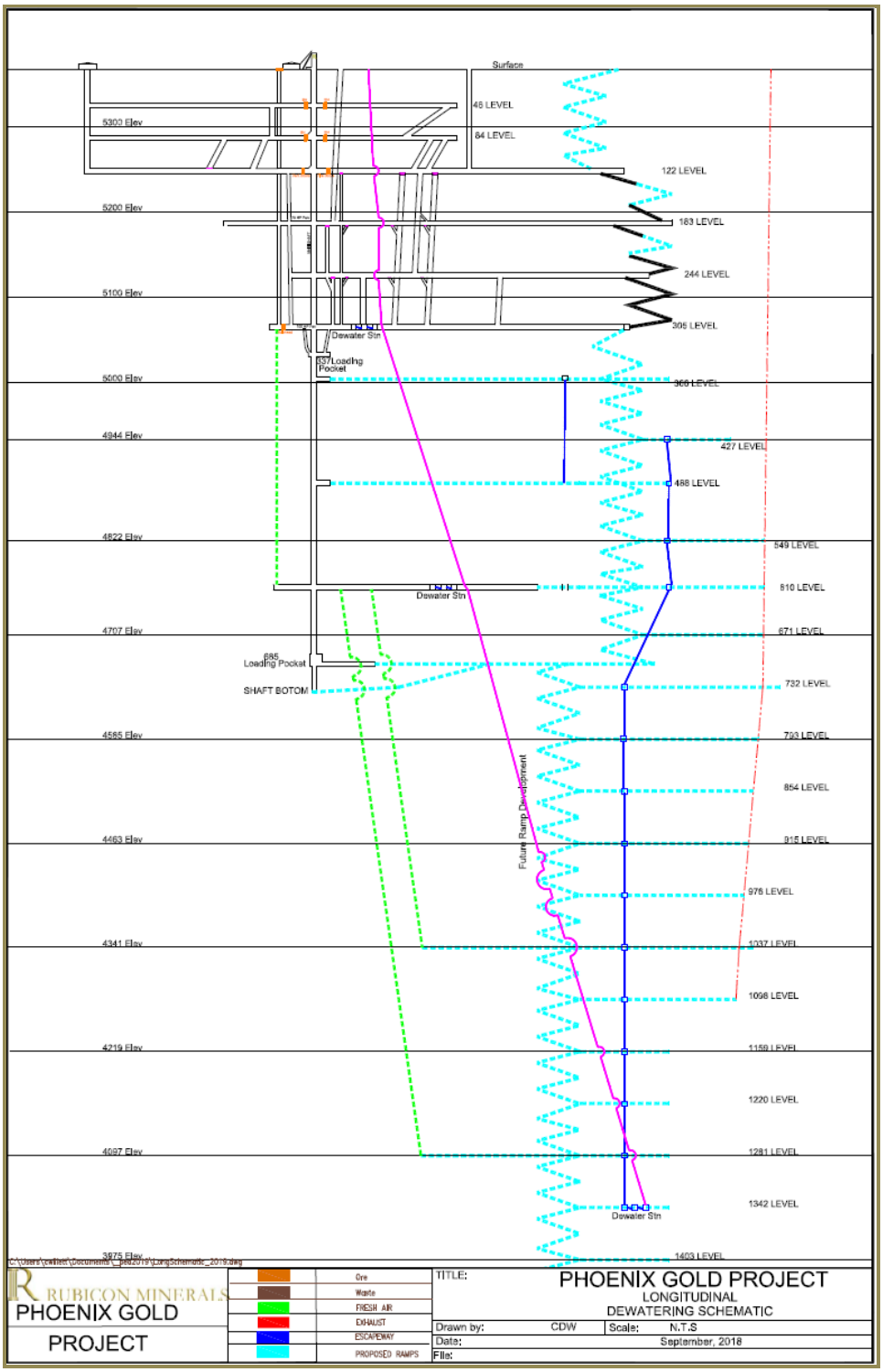


Figure 16-6: Longitudinal Dewatering Schematic

Miscellaneous Facilities

Other underground facilities not covered above include but are not limited to: storage bays for supplies and equipment, electrical substations, diamond drill stations, local electrical panels, charging stations, and toilet facilities conveniently located adjacent to active headings.

16.5 Current Underground Infrastructure

Mine infrastructure currently includes: muck handling facilities on the 305 m and 685 m Levels; a ventilation system capable of supporting 1,250 ore t/d production levels; a paste backfill distribution system extending down to the 244 m Level; a mid-shaft loading pocket below the 305 m Level complete with spill pocket; and a shaft-bottom loading pocket at the 685 m Level. Ramp access is partially completed between existing levels and can be completed following 380 m of further development. Existing infrastructure and project notes include:

- The deposit is located approximately 400 m east of the existing shaft
- All ore and rock not used as backfill will be hoisted to the surface via the existing shaft
  - Shaft accessible levels have been established on 61 m intervals, including 122 m, 183 m, 244 m, 305 m, 610 m, and 685 m Levels
- The 122 m, 244 m, and 305 m Levels are established as track drifts from the shaft station to the F2 Gold Zone area
- Ore passes are in place to service levels above the 305 m Level
- Levels below the 305 m Level will transport ore and rock to the shaft by LHD and/or diesel truck
- Internal ramps connection from the 122 m to the 305 m Level required 380 m to be completed
- Internal ramping from the 305 m Level to shaft bottom needs to be completed
- Internal ramp from shaft bottom to bottom of mine estimated as 1,463 m Level is required to be completed
- Muck handling systems are established on all operating levels except the 122 m Level
- Drainage and staged pumping currently exist on staggered levels
- A dry-fuel line has been installed to the 305 m Level
- The paste backfill system construction is near complete, other than commissioning; following commissioning it will be available to deliver backfill to underground.

Underground infrastructure (Figure 16-7) required to support production mining includes: material handling facilities; a mine dewatering system; a paste backfill distribution system; ventilation system; distribution systems for compressed air and process water; electrical power supply; equipment service bays; and other miscellaneous facilities. Existing and useful lateral development completed prior to 2017/2018 is listed in Table 16-3.

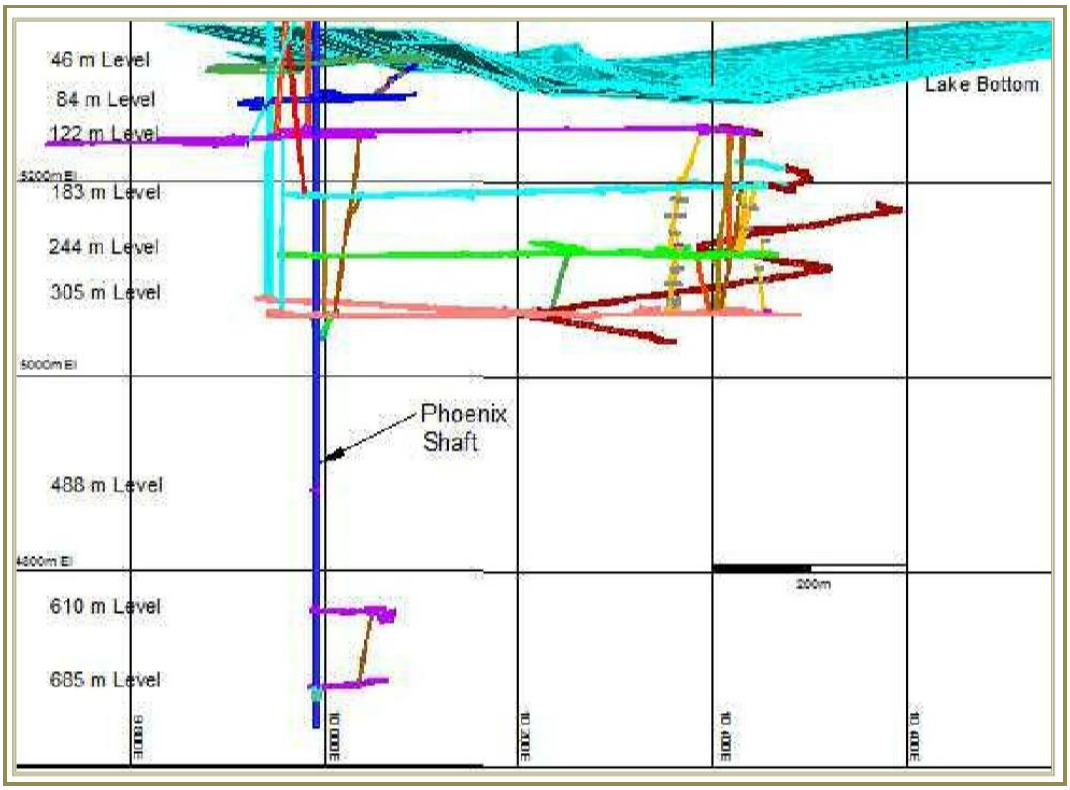


Figure 16-7: Existing Phoenix Underground, Looking North, (Rubicon, 2019)

Table 16-3: Existing Lateral Development

Description	Pre-2017/2018 *Quantity
All Ramps	1,359
46 m Level	1,742
84 m Level	1,549
122 m Level	2,909
183 m Level	1,210
244 m Level	2,022
305 m Level	2,393
610 m Level	296
685 m Level	188
Total (m)	**13,668**

Notes: * To the nearest metre
**Diamond drill and safety bays excluded

Figure 16-7 also shows the existing underground development at the Phoenix Gold Project. This development is currently dry, well ventilated, and in a good state of repair.

16.5.1 Underground Equipment Servicing

There are three service bays areas located on the 183 m, 244 m, and 305 m Levels. Equipment servicing is completed at these locations.

16.5.2 Miscellaneous Facilities

Other underground facilities not covered above include, but are not limited to: storage bays for supplies and equipment; electrical substations; diamond drill stations; local electrical panels; charging stations; and toilet facilities, which are conveniently located adjacent to active headings.

16.6 Planned Development

The Project starts with eight months (approx. 1,760 m) of development prior to commencement of pre-commercial production (Pre-CP). These eight months serve three purposes, including:

- Getting development 12 months ahead of production and creating a definition drilling window
- Completing construction of a surface ramp, 4.5 m wide by 4.5 m high, 800 m long at 15% grade, from just outside the mill down to the 122 m Level, thereby mitigating shaft conveyance time for rock hoisting for the current established levels
- Allowing a fresh air raise, 3 m by 3m, to be constructed from the 610 m Level to the 305 m Level so that mine ramps can be advanced from the 610 m Level up to connect the upper mine and down to meet anticipated production schedules in the lower mine.

Following this initial phase of Pre-CP, development crews will advance a further 5,155 m of ramp (Figure 16-11) and level development below the 305 m Level. Ramp development will connect 122 m, 183 m, 244 m, 305 m, 366 m, 427 m, 488 m, 549 m, 610 m, 671 m Level, 720 m (Shaft Bottom), and 732 m Levels. Vertical raise development, including manways, are scheduled to parallel connections to new levels. A total of 14 levels will be connected by ramp and shaft access.

This 12-month period is part of Pre-CP, which ramps up production from approximately 250 t/d to approximately 900 t/d in the upper mine levels.

During the Commercial Production (CP) period, Month 21 onward, and until the estimated end of mine life, another 19,692 m of lateral development and 2,745 m of vertical development are planned.

Lateral development takeoffs during the Conceptual LOM are summarized in Table 16-4 while vertical developments are listed in Table 16-5.

Table 16-4: Conceptual LOM Lateral Development Schedule (metres)

Surface	Unit	Year->	-2	-1	1	2	3	4	5
	m	400	400						
122	mL	512	512	-	-	-	-	-	-
183	mL	187	187	-	-	-	-	-	-
244	mL	42	42	-	-	-	-	-	-
305	mL	-	-	-	-	-	-	-	-
366	mL	1,350	747	603	-	-	-	-	-
427	mL	2,147	46	1,639	462	-	-	-	-
488	mL	1,447	-	523	924	-	-	-	-
549	mL	2,194	-	570	1,451	173	-	-	-
610	mL	1,852	183	395	756	518	-	-	-
640	mL	-	-	-	-	-	-	-	-
671	mL	1,684	-	374	-	1,310	-	-	-
685	mL	95	-	95	-	-	-	-	-
720	mL	310	-	310	-	-	-	-	-
732	mL	1,026	-	290	100	20	616	-	-
793	mL	1,522	-	-	435	-	1,087	-	-
854	mL	978	-	-	486	-	186	306	-
915	mL	1,702	-	-	209	1,493	-	-	-
976	mL	708	-	-	-	505	203	-	-
1037	mL	1,403	-	-	-	453	950	-	-
1098	mL	1,100	-	-	-	266	833	1	-
1159	mL	1,622	-	-	-	-	448	1,174	-
1220	mL	1,492	-	-	-	-	448	649	395
1281	mL	448	-	-	-	-	252	196	-
1342	mL	1,144	-	-	-	-	-	448	696
1403	mL	928	-	-	-	-	-	289	639
1464	mL	315	-	-	-	-	-	218	97
	m ->	26,608	2,117	4,799	4,823	4,738	5,023	3,281	1,827
	m/d ->	12.0	8.8	13.3	13.4	13.2	14.0	9.1	5.1

Table 16-5: Conceptual LOM Vertical Development Schedule (metres)

Vertical Development	Unit (m)	-2	-1	1	2	3	4	5
Fresh Air Raises								
610 to 305 mL	305	305	-	-	-	-	-	-
610 to 488 mL	122	-	122	-	-	-	-	-
720 to 610 mL	220	-	220	-	-	-	-	-
732 to 720 mL	24	-	-	24	-	-	-	-
793 to 732 mL	122	-	-	122	-	-	-	-
854 to 793 mL	122	-	-	122	-	-	-	-
915 to 854 mL	122	-	-	50	72	-	-	-
976 to 915 mL	122	-	-	-	122	-	-	-
1037 to 976 mL	122	-	-	-	122	-	-	-
1098 to 1037 mL	122	-	-	-	50	72	-	-
1159 to 1098 mL	61	-	-	-	-	61	-	-
1220 to 1159 mL	61	-	-	-	-	61	-	-
1281 to 1220 mL	61	-	-	-	-	-	61	-
1342 to 1281 mL	61	-	-	-	-	-	61	-
1403 to 1342 mL	61	-	-	-	-	-	61	-
1464 to 1403 mL	61	-	-	-	-	-	61	-
Exhaust Raises								
720 to 610 mL	220	-	220	-	-	-	-	-
732 to 720 mL	24	-	-	24	-	-	-	-
793 to 732 mL	122	-	-	122	-	-	-	-
854 to 793 mL	122	-	-	122	-	-	-	-
915 to 854 mL	122	-	-	50	72	-	-	-
976 to 915 mL	122	-	-	-	122	-	-	-
1037 to 976 mL	122	-	-	-	122	-	-	-
1098 to 1037 mL	122	-	-	-	50	72	-	-
Fresh Air Total	1,769	305	342	318	366	194	244	-
Exhaust Total	976	-	220	318	366	72	-	-

Mine development will be conducted by conventional mining equipment according to the size, advance rates, and costs shown in Table 16-6. When possible, development rock will be stored in depleted stopes, otherwise, it will be hoisted to surface and stockpiled, or used to construct tailings dam lifts.

Table 16-6: Waste Development in Conceptual Mine Plan Particulars

Development Type	Length (m)	Single Heading ($/m)	Single Heading (m/d)	Multiple Heading ($/m)	Multiple Heading (m/d)	Waste (t)
Ramp, 4.5 m x 4.5 m, 15%	9,329	6,230	3.1	4,151	4.7	472,281
Level Access & Haulage, 4.5 m x 4.5 m	1,070	6,230	3.1	4,151	4.7	54,169
Level Access & Haulage, 4 m x 4 m	10,401	5,980	3.9	3,984	5.8	526,551
Stope Access, 3 m x 3 m	5,808	5,180	4.2	3,451	6.3	294,030
Raising by Alimak, 3 m dia.	2,745	7,670	2.4			61,763
Total	**29,353**					**1,408,793**
$/m includes indirect cost ~ 40%						

Figure 16-8 shows the distribution of metres for ramp and level development, by year, over the conceptual LOM. The ramp and level mine plan is shown in Figure 16-9, including the proximity of stoping blocks, existing development, pre-commercial production, and sustained development over the LOM.

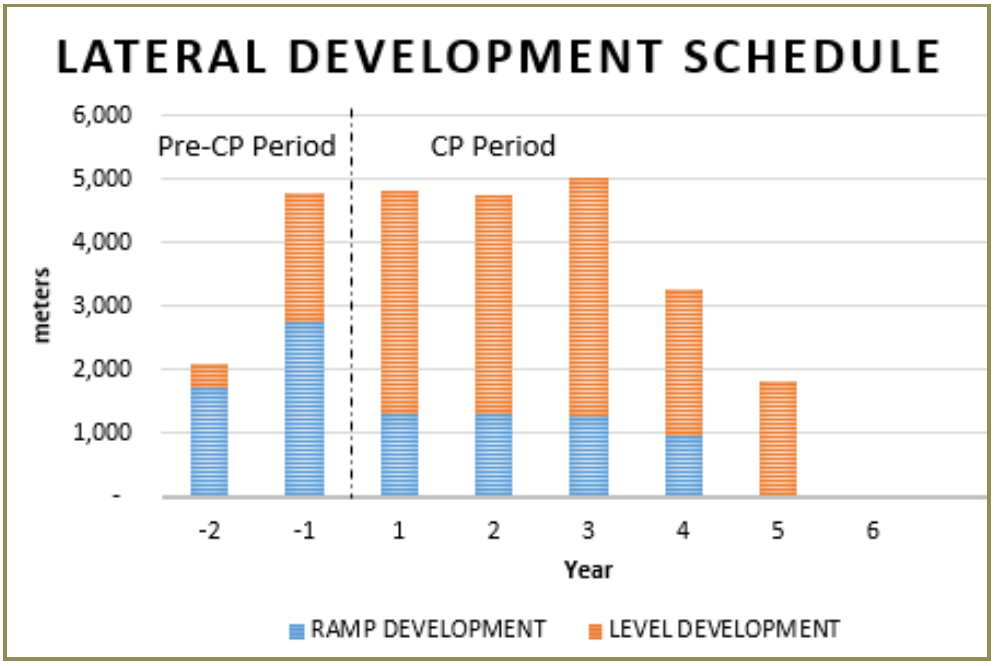


Figure 16-8: Annual Length of Ramp and Level Development over conceptual LOM for Ramps and Levels

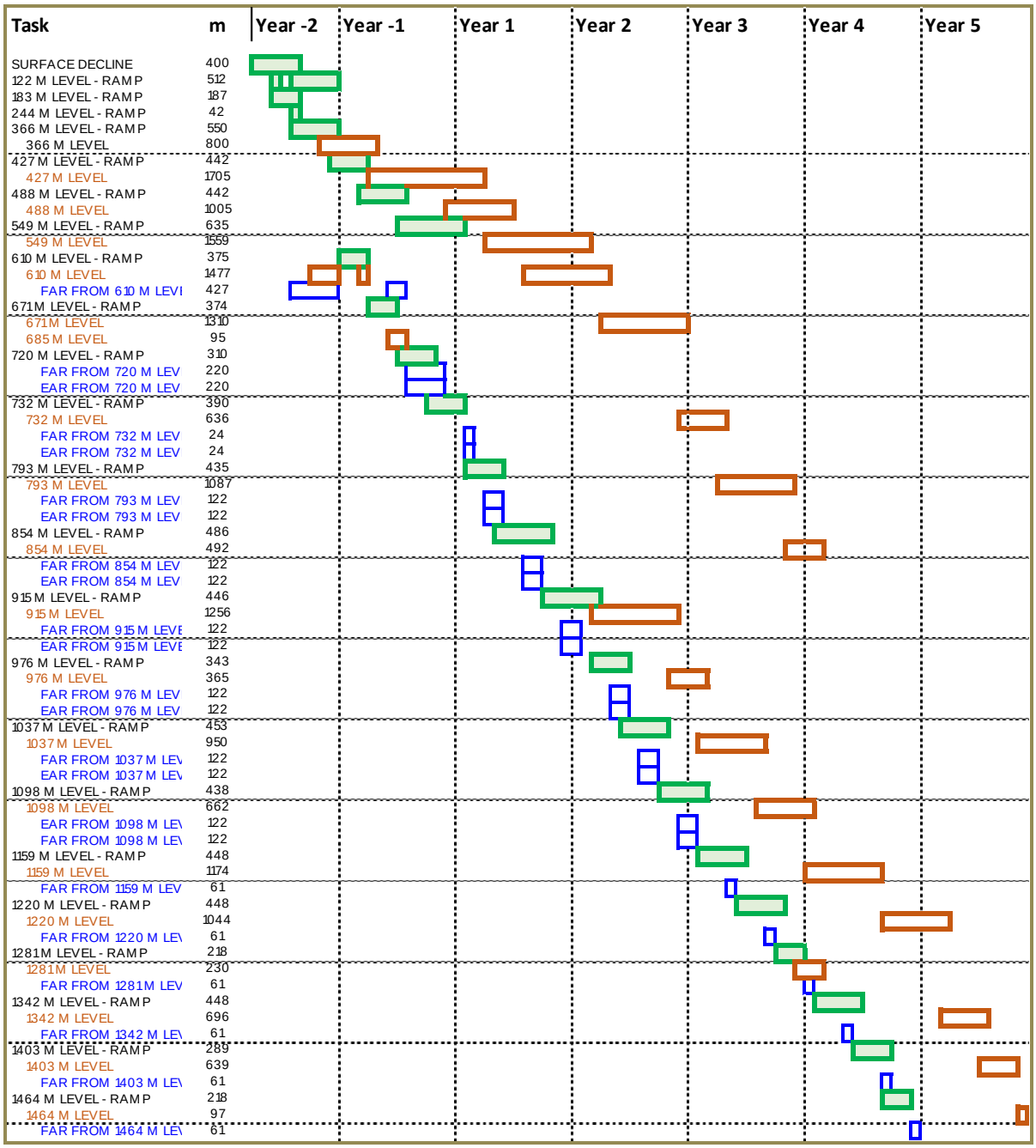


Figure 16-9: Development Schedule Showing Ramp (Green), Levels (Brown) and Air Raises (Blue)

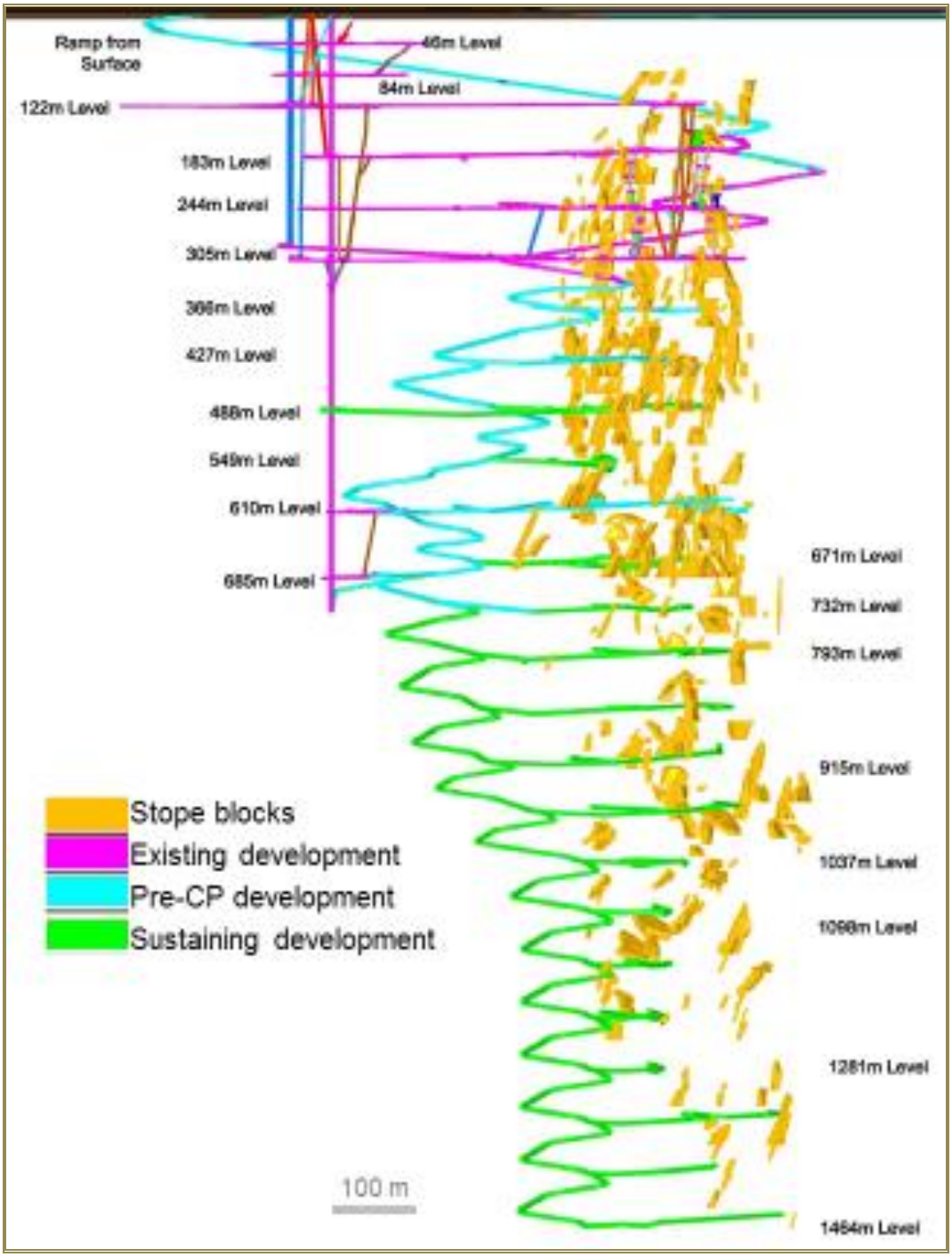


Figure 16-10: Phoenix LOM Development Plan 3D Isometric, Looking North, (Rubicon, 2019)

A 3D isometric view of the Phoenix project is shown in Figure 16-10. Existing development, magenta, is contrasted with Pre-CP development, cyan, proposed to be completed in the first 20-months, setting the project up to keep development 12-months ahead of production over the conceptual LOM. CP development is shown in green and extends down to 1464 m Level.

Examples of three different mining levels are shown in the following three figures. Figure 16-11 shows 366 m Level, access to the ramp and connections to a variety of stoping locations.

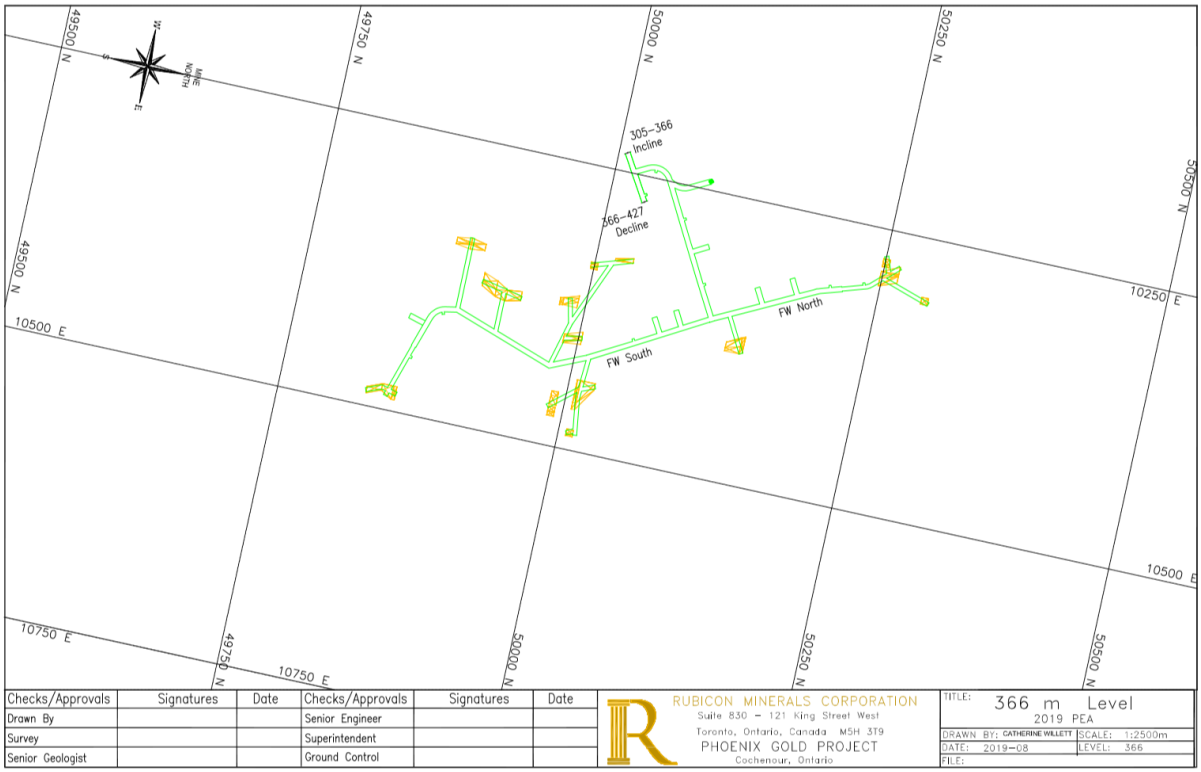


Figure 16-11: Phoenix 366 m Level, Plan View (Rubicon, 2019)

Figure 16-12, 610 m Level, shows shaft and ramp access locations, truck dumps to feed the 685 m Level, loading pocket arrangements, and several mining stopes.

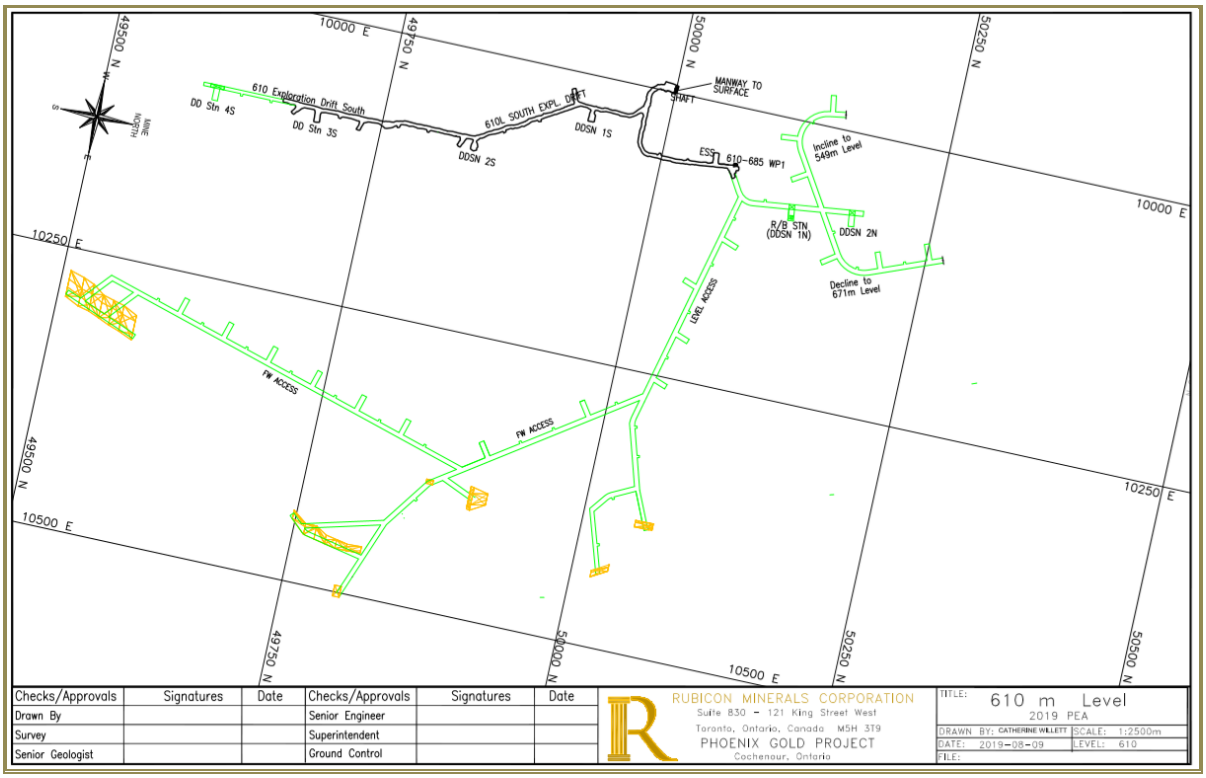


Figure 16-12: Phoenix 610 m Level, Plan View (Rubicon, 2019)

Figure 16-13, shows ramp connection to 732 m Level as well as level haulage and access to a number of potential stoping areas.

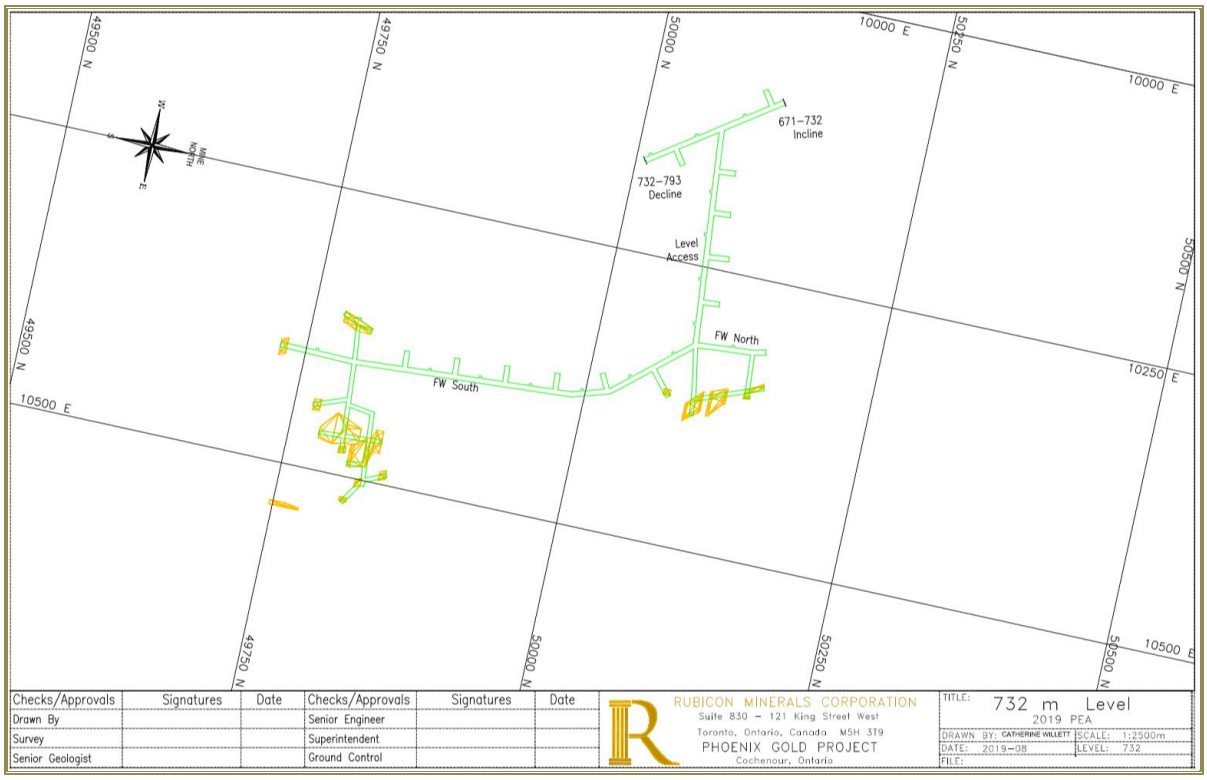


Figure 16-13: Phoenix 732 m Level, Plan View (Rubicon, 2019)

16.7 Geotechnical Evaluation

In general, ground conditions at the Phoenix Gold Project can be considered good, particularly in the F2 High-Ti Basalt Zone. Cavity-monitoring surveys completed in early 2017 on the 305-Z2-030 test stope mined in 2015 confirmed the good ground conditions in this area, with minimal (i.e., less than 10%) unplanned dilution in the stope after two years.

Historical ground stability issues were encountered in the Ultramafic units. These issues are related to geological structures and have been mitigated by applying a planned reduction of development performance and enhancement of ground support, using shotcrete, during transitions by or through these structures. Mine planners have designed all permanent development, including ramps, to minimize interaction with Ultramafic units.

Geotechnical evaluations completed prior to this PEA include a scoping-level evaluation by SRK in July 2013 (SRK, 2013a), a crown pillar assessment by AMC in December 2014 (AMC, 2014), and Ground Support Standards for Rubicon by AMC (2009), which are currently being used at the site.

16.7.1 SRK's Geotechnical Assessment

The geotechnical assessment conducted by SRK is available in the PEA for the F2 Gold Zone, issued August 9, 2013, amended and restated February 28, 2014 (SRK, 2013b). Regular monthly ground support audits of the underground workings are being completed by both external professionals and in-house staff.

16.7.2 AMC's Crown Pillar Assessment

In 2014, Rubicon commissioned AMC to assess the crown pillar, as the gold mineralization extends to the lake bottom. AMC has recommended a conservative minimum crown pillar thickness of 40 m, and certain other risk mitigation options. Special operating procedures are recommended, outlining ground support, backfill, and instrumentation monitoring strategies in the moderate- to high-risk areas. Currently, no mining work is planned for the crown pillar.

16.7.3 TMAC's Geotechnical Assessment

While not a concern at the PEA level of study, the Phoenix conceptual mine plan should be subjected to confirmatory stress modelling as it proceeds through the feasibility level of study. For the PEA, typical stope shapes were modeled using empirical methods, including Mathews modified stability graphs and equivalent-length overbreak slough (ELOS) plots to assess stability and unplanned dilution concerns.

Of note, more than 85% of stope shapes are isolated from each other in either elevation or strike distances. This finding allows planners to minimize the complexity of sequencing primary and secondary stoping. Larger stopes, where primary and secondary extraction is warranted, have been scheduled to reflect a bottom-up chevron mining front.

During planning, over 700 stopes were outlined as potentially minable. Following geotechnical, accessibility, and economic assessment, a total of 341 stopes were defined as viable minable shapes. The average size of these stopes, by mining method, is shown in Table 16-7.

Table 16-7: Average Stope Sizes by Mining Method

Method	# Stopes	Height (m)	Width (m)	Strike (m)	Tonnes/Stope
Uppers	172	16.3	5.8	12.4	4,120
Cut & Fill	60	11.8	6.4	16.2	3,770
Longhole	66	35.7	8.4	25.2	24,170
Mass Blast Raise Method	43	37.3	3.5	22.2	11,950

Assessments for larger shapes were performed to confirm stability. An example stability plot for a 976 North Stope is shown in Figure 16-14 along with a corresponding section, showing the 78° dipping stope (30 m long, 15 m wide, 50 m high), plots with a stable back while the hanging wall plots in the transition zone. TMAC has included mid-span cable bolts, where appropriate, to mitigate stability concerns.

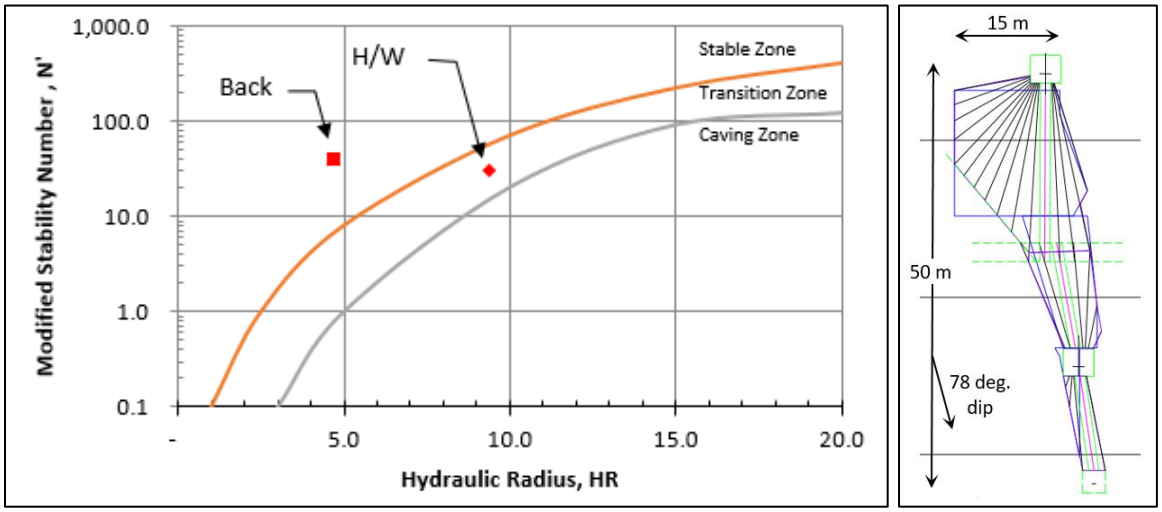


Figure 16-14: Modified Stability Graph For 976 North Stope—Plots Do Not Include Mid-Span Cabling

To check against the empirical observations and CMS finding, <10% dilution in longhole (LH) test stopes, TMAC applied the ELOS method.

The blue-shaded area in Figure 16-15 represents a range of stope sizes from 1.6–8 m in span, from 10–30 m long and 30–60 m high. This ELOS plot parallels the findings in SRK's (2014) report (i.e., <10% dilution anticipated), and importantly corroborates the 2018 CMS evidenced dilution calculations.

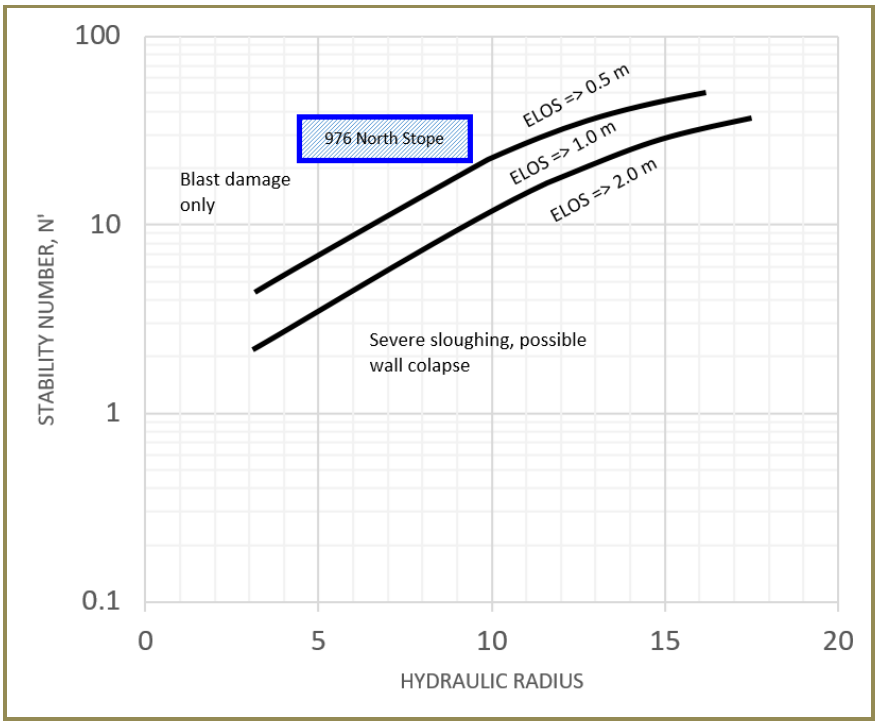


Figure 16-15: ELOS Plot of Rubicon's 976 Stope, Typical Stope, shows >0.5 m Slough Expected

16.7.4 Rubicon's Ground Control Management Plan

Rubicon's ground control management plan incorporates standardized ground support applications for the various ground stability issues that are expected at the Phoenix Gold Project. This is based on the various ground support studies that have been completed at the Project, and also integrates information that has since been acquired from underground development completed, and test stopes developed and mined. Standard ground support methodology used during the last test mining phase in 2018 included the use of resin rebar, split sets, omega bolts, wire mesh panels, and wire mesh straps. Stope stability analysis has been conducted on all stopes to date.

16.8 Mine Methods

Phoenix's mineralization typically comprises underground, tabular, narrow veins, approximately 1.0 to 15 m thick, and dipping from 65° to 90° to the southwest. When selecting narrow mining methods, dilution is always a concern. For that reason, four mining methods were selected: longitudinal longhole (LH), Uppers, Cut-and-Fill (C&F), and mass blasting raise mining (MBRM) stoping. These mining methods were based on favourable geometry, current geotechnical knowledge, and successful test mining. Both the mineralized material and the host rock are sufficiently competent to facilitate the void sizes required for the methods.

The physical characteristics (context) of the gold mineralization relevant to selection of the mining method are:

- The deposit is under a lake; therefore, a crown pillar of 40 m is required
- No extraction of the crown pillar is considered in this Technical Report
- The deposit is located approximately 400 m east of the existing shaft
- The shaft is on the hanging wall side of the deposit
- Zones 1 through 4 are considered in the PEA
- The deposit dips between 75° and 90° and runs from 122 m Level (bottom of crown pillar) to 1586 m Level, a vertical distance of 1,464 m
- In section, the mineralized zone is 150 to 200 m wide, with widths from 1.0 to 15.0 m above cut-off
- The deposit extends to 1,000 m along strike
- Mineralized zones pinch and swell rapidly along strike and down-dip creating discontinuity in that parts can fall below the 3.5 g/t Au cut-off grade
- There are isolated mineralized zones outside the main corridor
- The mineralized zone has contacts that are difficult to identify visually
- Half of the 2019 Mineral Resource Estimate is below the current shaft bottom
- Mineralization is hosted primarily in the High Ti-Basalt, which has demonstrated good to very good geo-mechanical characteristics
- A weak talcose Ultramafic unit, the East Bay Deformation Zone (EBDZ), lies west of the F2 Gold Zone
- The mine is generally dry, with minimal water inflow.

Key mine planning criteria are summarized in Table 16-8.

Table 16-8: Mine Planning Criteria

Parameter	Unit	Value
Operating Days per Year	Days	365
Shifts per Day	Shifts	2
Hours per Shift (Underground)	Hour	12
Work Rotation	Weeks in x Weeks out	3 x 2
Nominal Ore Mining Rate	t/d	1,850
Annual Ore Mining Rate	t/a	675,250
Ore density	t/m^3	2.91
Waste density	t/m^3	2.56
Swell factor		55%

The planning parameters used for each of the methods is listed in Table 16-9.

Table 16-9: Mining Method Planning Parameters

Method	Longhole	Uppers	Cut & Fill	Raise Mining
Level Interval	40 m	15 m	40 m	80 m
Dip	90 to 65°	90 to 65°	65 to 0°	90 to 60°
Stope Strike Length	50 m	50 m	30 m	30 m
Limiting HR (hanging wall)	8	8	2.8	5.6
Minimum Mining Width	1.6 m	1.4 m	2 m	1.2 m
HW & FW Overbreak	Total 0.25 m	Total 0.2 m	Total 0.25 m	Total 0.2 m
Minimum Diluted Width	1.85 m	1.6 m	2.25 m	1.4 m
Maximum Mining Width	15	12	6	13
HW & FW Dilution Gold Grade	0.1 g/t Au	0.1 g/t Au	0.1 g/t Au	0.1 g/t Au
Transverse or Longitudinal	Both	Both	Longitudinal	Both
Direction of Mining	Top Down	Bottom Up	Bottom Up	Bottom Up
Other			Max 8 cuts	No top sill required

TMAC's mine planners used AMINE and SURPAC software to visually define stoping shapes. Datamine's Mineable Shape Optimiser (MSO), operated by Rubicon personnel, was used as a check on planners' efforts.

A discussion of the four methods is discussed below.

16.8.1 Sub-Level LH Retreat

This method is a highly productive bulk mining approach that is usually applied to ore widths of 3 m and greater (Figure 16-16).). It involves development of the ore body at regular vertical intervals (sub-levels), typically every 15 m to 30 m.

Several methods can be employed to develop the sub-levels, from driving raises (as done in 2015), to excavating accesses from main ramps (as done for the test stoping completed in 2018). A blasting slot would be developed at one end of the excavation, and mining of the blocks would retreat along the strike of the stope. Mucking takes place within the undercut of the mining block via remote-controlled LHD equipment. The strike length is dictated by wall stability in the open stope, and is initially determined by empirical design.

This mining method is applicable to thicker areas of the deposit; it was employed on all three of the test stopes in 2018 (183-Z1-161, 244-Z1-015, and 244-Z1-977).

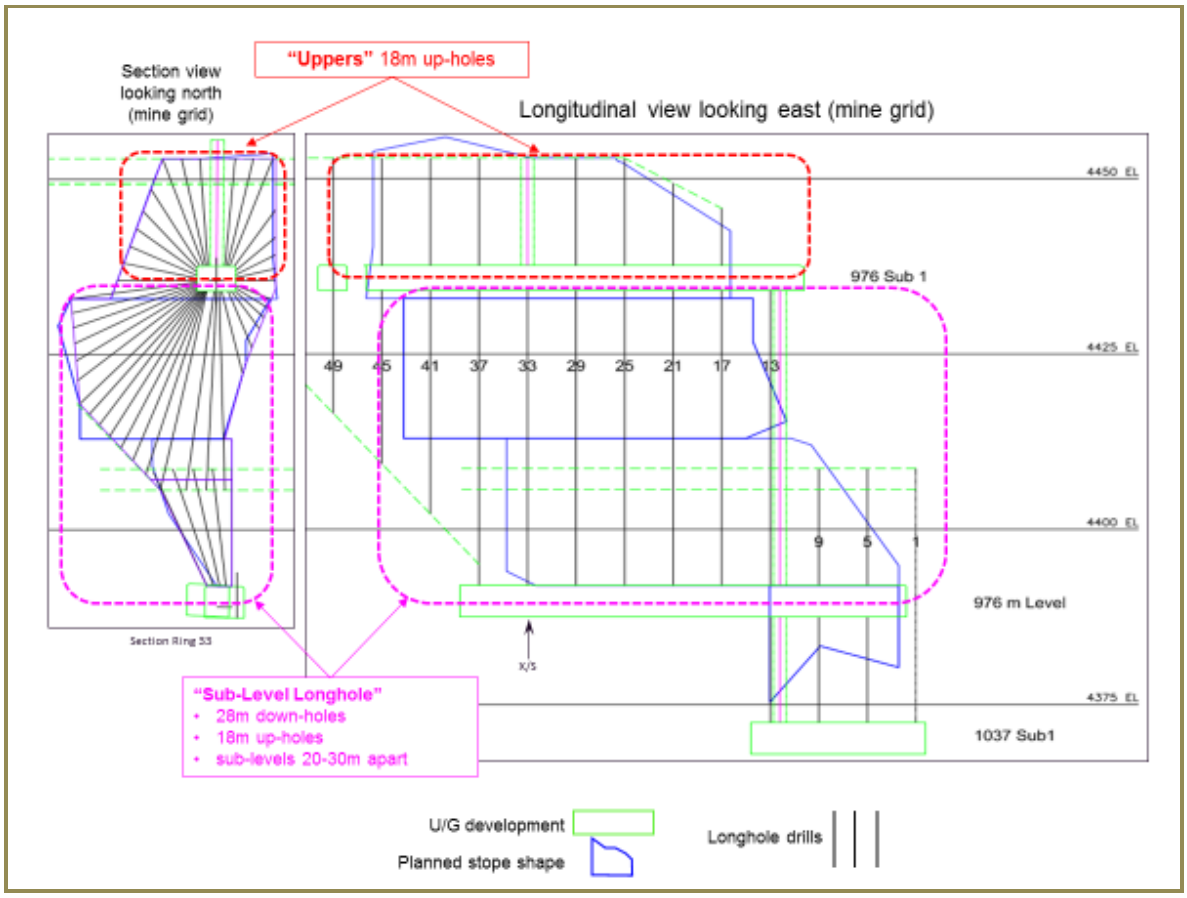


Figure 16-16: Conceptual Diagram of Sub-Level Longhole and Uppers Mining Methods (Diagram Not to Scale)

16.8.2 Uppers LH Method

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

This method is best used where ore continuity is known, and strike length is limited. Often used in conjunction with LH mining, the method can be used separately as part of an overall mining strategy. The Uppers method Figure 16-17 was successfully employed on one of the test stopes in 2018 (183-Z1-161).

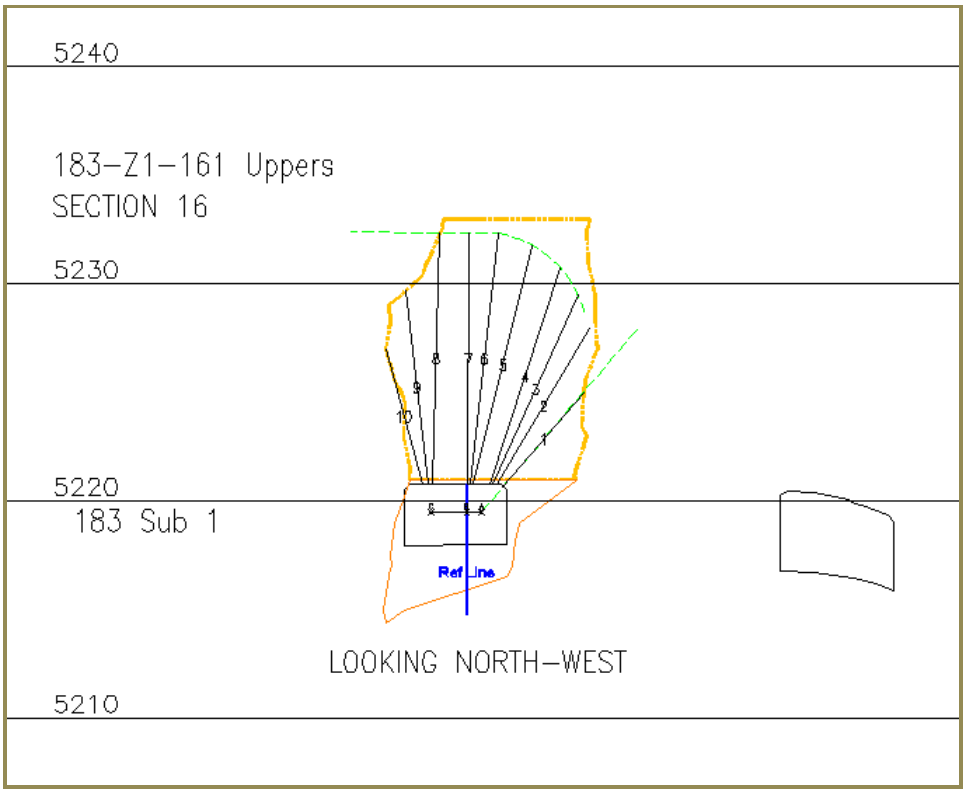


Figure 16-17: Typical Upper Ring Layout 183-Z1-161

16.8.3 Mechanized C&F

C&F (Figure 16-18) is a moderately productive, highly selective mining method for mining widths between 2.4 and 10 m. Muck can be segregated by "resueing" into mineralized material and waste, each to be handled differently based on logistical conditions at the time the material is generated. Muck material can be sent to the surface and processed, sent to the surface and stockpiled, stockpiled underground for future consideration, or left in the stope as backfill. When possible, waste muck is stockpiled underground, or left in another stope as backfill.

The mining sequence begins by driving an attack ramp either from a level or from a nearby ramp. The attack ramp is driven up to a −15% gradient to access the bottom (sill) cut of the mineralized zone near the centre of the stope mass using the same development equipment as used for ramp and level development. The mineralized zone is developed with sill drifts to the extents of the mineralization. A sill mat is installed, if required, prior to backfilling with paste fill, or rock fill if available from the nearby development headings.

After backfilling is complete, a section of the attack ramp is back-slashed and rebolted to gain elevation for access to the next cut. The waste rock broken while doing this will often be left in place or stored nearby to provide roadbed material in the ramp, and rockfill for the next cut. This cycle is

repeated until the designed number of cuts has been mined. Mining continues upward by repeating the process from a new attack ramp to access the mineralized zone at the next higher elevation.

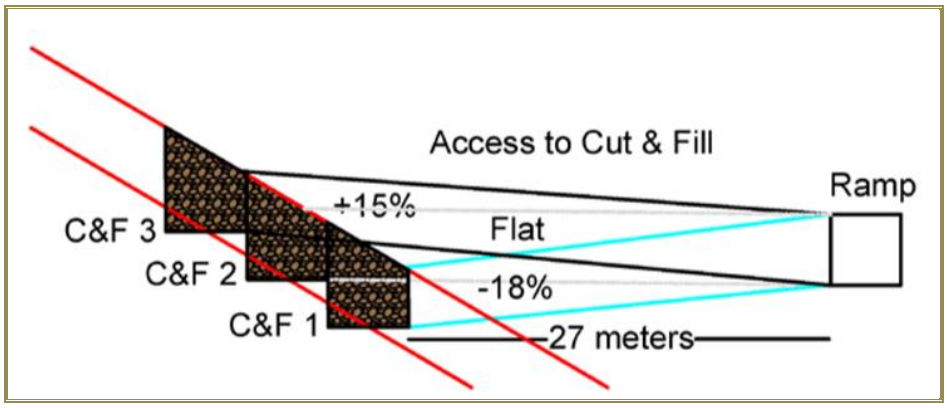


Figure 16-18: Mechanized C&F Method, Long Section

Each of the four methods was applied when deemed to maximize safety, recovery, and economics while minimizing dilution and stoping duration. The relative tonnages allotted to each method are presented in Table 16-10.

Table 16-10: Minable Stope Tonnes by Mining Method

# Stopes	Mining Method	Tonnes*	% of Tonnes
172	Uppers	708,880	23.3
60	C&F	226,420	7.4
66	LH	1,595,921	52.4
43	MBRM	513,974	16.9
341		**3,045,195**	**100**

Note: Tonnes shown are diluted and recovered

16.8.4 Alimak Mass Blasting Raise Mining (MBRM)

A well-suited and proven method for recovery of vein structured orebodies down to 1 m in thickness, with Mass Blasting Raise Mining (MBRM) has strong economic and production considerations for extraction of narrow mineralized seams without requiring access to a top sill. MBRM is used in 43 of the 341 stopes planned in this PEA.

Several Canadian mine contractors are experienced using this method. Variations on the method are employed worldwide, but Barrick Gold's Williams Mine near Marathon, Ontario, appears to have evolved the method, over the last 20 years, to its most advanced and reliable state. TMAC and Rubicon visited Barrick's operation in June 2019 to observe suitability, reliability, and safety of Manroc's operators and engineers. During that visit, Williams's senior production and technical

personnel gave favourable testimonials supporting the method's low cost, narrow excavation applications, and high reliability and productivity. On average, 20% of all mining at the Williams site is performed as MBRM, and on occasion, MBRM has provided more than 50% of the mine's annual gold production.

Michael Willett, P. Eng., Director of Projects for Rubicon, and Andrew MacKenzie, P. Eng., Principal Mine Engineer at TMAC, visited Manroc projects at the Hemlo mining camp, 40 km east of Marathon, Ontario; they concluded that the method is applicable and accessible for consideration at Phoenix.

Manroc reviewed three of Rubicon's larger stope shapes and proposed budget MBRM methodologies, schedules, and prices, which are the foundation of schedules and prices used for this method in this PEA.

The method begins with conventional lateral development of a bottom sill, stope undercut, to allow for LHD mucking, as well as to create void space for production blasts. Typically, undercuts are 4 m wide by 5 m high to provide adequate void for initial blast swell.

Drawpoints and Alimak nests are established at the hanging-wall contact. The use of mass blasting techniques precludes the need to develop top sill access in stopes that are maintained under 80 m in height. A central raise is driven to the upper stope limit, typically 3 by 3 m, so that an LH drill, later used for production drilling, can be table rotated within the raise. After the raise is driven, it is screened and bolted subject to ground conditions and for the safety of production crews. Resin-grouted cable bolts are installed from the raise into the hanging wall. These cables decrease dilution since they reduce the unsupported span by dividing the hanging wall in half. Production blast holes, typically 12 m long and 50 mm in diameter, are then drilled horizontally on either side, from top to bottom of the raise, out to the stope boundary. To prevent drilling into adjacent workings, 2 m pillars are left for stope separation. Cavity Monitoring Systems (CMS) show that a portion of these pillars crush and are later recovered during secondary stoping.

The blast holes are then loaded with explosive, and, using Orica's WebGen 100 wireless initiation system, (through-the-earth initiation system) blasted to deliver the full stopes production to the draw point in a single load/blast effort. Undercuts and raise volumes need to account for the 30% void requirements for mass blasts. If void volume is less then 30%, then the WebGen is initiated in only the lower part of the stope. Mucking of this initial blast provides void space, allowing the already loaded main stope to be blasted later without having to re-enter the stope.

Mucking, remote when required, continues until waste material reports to the drawpoints. The blasted muck provides confinement to wall rock while being drawn down, thereby minimizing dilution from spall. Once mucking is complete, the stope is plugged at the lower drawpoint(s) and paste filled by inclined boreholes or, if accessible, boreholes from higher elevations.

The method is highly productive, in that the full stope volume is blasted and available for mucking without repetitive explosive loading cycles. Importantly, the method is safe, since primary and secondary stope miners are remote from stope voids.

Figure 16-19 shows a longitudinal section of a typical MBRM stope and a cross-section through A – A'.

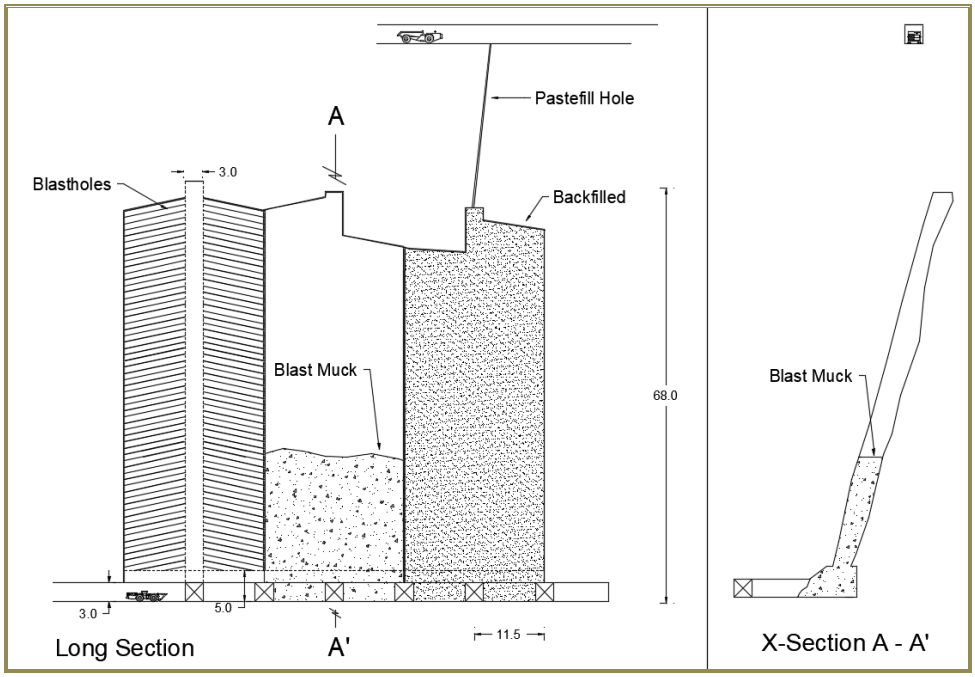


Figure 16-19: MBRM, Stopes, (From Left to Right) Development, Production, Backfilled

For more information regarding Alimak Mining, the reader is referred to Manroc's website at http://www.manroc.com/.

16.9 Initial Cut-Off Grade Calculation

Following general assessment of the underground Mineral Resource Estimate, geological structures, potential mining methods, and limitations, TMAC calculated a cut-off grade to define what areas might become minable. Cut-off grade is the minimum grade required for a mineral or metal to be economically mined (or processed). Material below this grade is considered waste. Table 16-11 shows TMAC's calculation to derive the 3.5 g/t Au cut-off which was applied to the Mineral Resource block model to allow mine planners to identify minable stope shapes.

Table 16-11: Minable Resource Au Cut-Off Calculation (US$:C$ 1.33) (C$:US$ 0.7519)

Description	Value	Value	Net
Spot Price (C$/g)	1,325	56.66	56.66
Payable (%)	99.98	0.01	56.65
Refinery (C$/g)	0.02	0.02	56.63
Royalty (%)	1.50	0.85	55.78
Transport (C$/g)	0.06	0.06	55.72
Mine Recovery (%)	95.0	2.83	52.89
Mill Recovery (%)	95.0	2.69	50.19
Mining (C$/t)			98.00
Mill (C$/t)			25.00
Geology (C$/t)			2.98
Maintenance (C$/t)			3.91
G&A (C$/t)			22.04
Environmental (C$/t)			1.45
Total OPEX (C$/t)			153.37
COG—Mill-Head Grade (g/t)			3.06
*Dilution (%)			10
COG—In Situ (g/t)			3.36
Used (g/t)			3.50

Note: * Dilution value assumed as typical

16.10 Dilution and Recovery

Two types of dilution besides planned dilution were applied to the selected mine method designs:

- Unplanned dilution—in situ host material that falls into the stope from the geometry of the stope shape
- Fill dilution—run-of-mine waste, and/or paste backfill expected to fall into the stope being mined from adjacent stopes and/or inadvertently scraped off the stope floors during mucking.

TMAC corroborated dilution from test mining as well as from linear ELOS assessments (Section 16.7.3). ELOS outputted 0.5 m of potential overbreak in the largest of the planned stopes. This theoretical estimate corresponds very well with the reconciliation from the test stopes, less than 10% slough or unplanned dilution (Table 16-12).

Paste fill will be used to fill all stopes. In consideration of backfill dilution, factors of 5.0% were applied for LH or MBRM methods, 4.0% for C&F, and 0.0% for Uppers methods.

Three host rocks contributed different gold values, through unplanned dilution, and were recognized by mine planners after detailed geological consultations. Those rock types and grades include Felsic Intrusions (1.0 g/t Au), Ultramafic (1.0 g/t Au), Hi-Ti Basalt in the upper mine (3.44 g/t Au), and Hi-Ti Basalt in the lower mine (1.98 g/t Au). When calculating stope dilution, to derive an individual stope dilution value, the variance of host rock percentages was considered for each stope.

Dilution and recoveries by mining methods and stope types are summarized in Table 16-12 with an example calculation in Table 16-13.

Table 16-12: Dilution and Recovery by Mining Type

Au Dilution	Unplanned Dilution Grade by Host Rock Type				Unplanned Dilution (%)	Mining Recovery (%)
	Felsic 1 g/t	Ultramafic 1 g/t	Hi-Ti Upper 3.44 g/t	Hi-Ti Lower 1.984 g/t		
Uppers	19%	37%	39%	5%	15%	94%
C&F	21%	38%	29%	12%	3%	99%
LH	26%	28%	43%	4%	10%	95%
MBRM	12%	55%	31%	2%	10%	95%
Average	22%	35%	39%	4%	-	-

Note: Backfill dilution assumed as zero grade

Table 16-13: Example Dilution and Recovery Calculation

LH Stope 5127 #13, 183 Level in Upper Mine				(g/t)
Dimensions, L x H x W (m)		31 x 28 x 6		
In Situ Mineralization		15,687	tonnes	7.30
Unplanned Dilution	10%	1,569	tonnes	1.30
Comprising		49%	Felsic	1.00
		39%	Ultramafic	1.00
		12%	Hi-Ti (Upper Mine)	3.44
Backfill Dilution	4%	627	tonnes	0.0
Diluted		17,883	tonnes	6.52
Recovered	95%	16,989	tonnes	6.52

Mining recovery, Table 16-12, is a function of mineralized material left behind due to operational constraints typical in the mining process. The longhole mining method is dependent on accuracy of longhole drilling and explosive detonation to properly fracture the ore. Where holes deviate from the ore limits, some material will remain hung up and may never report to the stope floor for recovery. Lesser factors considered to affect recoveries in longhole mining include ragged mucking floors and

limited visibility for remote mucking. Secondary stopes recognize higher recoveries due to improved probability of blasted mineralization making its way to the stope floor for mucking.

Mining recoveries in MBRM and LH are estimated at 95%, while Uppers is 94%. These values are based on industry norms and operational experience for mucking stopes of similar type. Mineral recovery in C&F stopes is estimated at 99% and is typical for overhand, narrow-vein methods using paste as tight fill.

16.11 Conceptual Mineable Mineral Resource Estimate

The conceptually mineable Mineral Resource estimate, diluted and recovered, comprises 3.05 Mt at a gold grade of 5.31 g/t Au and containing 520,000 oz of gold (Table 16-14 and Table 16-15).

Table 16-14: Conceptual Mineable Mineral Resource Estimate (Diluted and Recovered)

Mineral Resource Categories	2019 MRE Using a 3.5 g/t Cut-Off Grade Sensitivity		New PEA Conceptual LOM Plan (@ 3.5 g/t Au Mining Cut-Off Grade)		Net Conversion of Tonnes (%)
	(t)	In Situ Grade (g/t Au)	(t)	Mill-Head Grades (g/t Au)	
Measured & Indicated	2,289,000	7.11	1,561,000	5.23	68.2
Inferred	2,038,000	7.39	1,484,300	5.39	72.8

Note: Numbers may not add due to rounding. Tonnes were rounded and may not add up to LOM tonnes

Table 16-15: Stopes Designed by Method

Stope Quantity	Mining Method	Tonnage	Percentage (%)	Gold (oz)	Gold (g/t)	Average (tonnes/stope)
172	Uppers	708,880	23.3	121,000	5.30	4,120
60	C&F	226,421	7.4	38,000	5.16	3,770
66	LH	1,595,327	52.4	274,000	5.34	24,170
43	RM	513,975	16.9	88,000	5.31	11,950
341		3,044,603		520,000	5.31	8,930

Note: Figures have been rounded and totals may be affected

16.11.1 Basis of the Conceptual Mineable Mineral Resource Estimate

The minimum stope width considered in this study was 1.2 m. Gold is the only recovered payable mineral. The cut-off grade of 3.5 g/t Au was calculated using input parameters such as commodity price, exchange rate, processing, and marketing costs (Table 16-11). The PEA determined a maximum production rate of 1,800 t/d is achievable, and the operating costs used for cut-off grade calculation were based on that mining and processing rate.

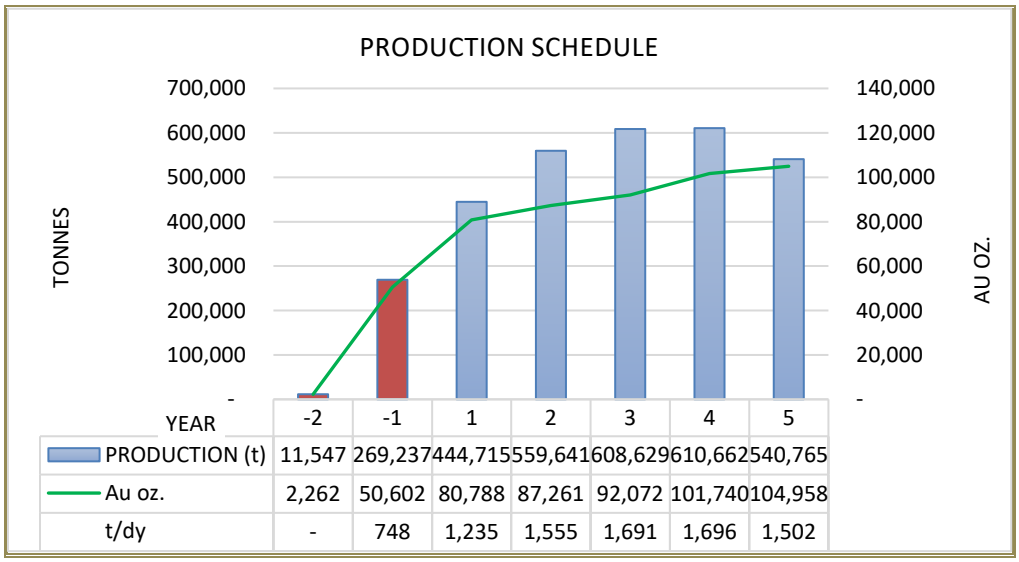
To generate stopes from the 2019 Mineral Resource Estimate, TMAC planners used AMINE and SURPAC to visually identify accessible and economic mineralization based on level intervals of 61 m. During that first pass at stope design, planners identified 484,000 oz Au within the conceptual minable Mineral Resource estimate.

MSO was then used as a check, so planners could perform a value-added review that resulted in an additional 35,000 oz Au identified as conceptually minable, 520,000 oz in total. Keeping in mind the structure of a boudinage style orebody, TMAC recommends a further round of MSO and mine planner stoping assessments for narrower parts of the Mineral Resource estimate.

Parameters used by mine planners are shown in Table 16-9, as well as Section 16.10 Dilution and Recovery.

16.12 Conceptual Production Schedule

Pre-CP is anticipated to begin near the end of Year −2 and to run until the end of Year −1. Production gradually ramps up from 250 t/d to 1,250 t/d which is achieved in Year 1 of the Project; step increased towards 1,800 t/d by years 3 and 4 and; a LOM average of 1,530 t/d being produced, during CP, until the end of Year 5, as outlined in Figure 16-20 and Table 16-16.



YEAR	-2	-1	1	2	3	4	5
PRODUCTION (t)	11,547	269,237	444,715	559,641	608,629	610,662	540,765
Au oz.	2,262	50,602	80,788	87,261	92,072	101,740	104,958
t/dy	-	748	1,235	1,555	1,691	1,696	1,502

Figure 16-20: Phoenix PEA Conceptual Annual Production (Diluted and Recovered)

Table 16-16: PEA Conceptual Annual Production Schedule by Level (diluted and recovered)

Year		-2	-1	1	2	3	4	5	Au oz
Levels	Production (t)								
61	15,850	11,547	4,303						3,530
122	68,441		68,441						13,108
183	140,584		140,584						26,621
244	114,430		55,909	58,521					18,051
305	270,272			270,272					51,833
366	242,331			115,922	126,409				39,558
427	442,055				418,437	23,618			69,530
488	79,984				12,000	67,984			11,807
549	198,471				2,795	195,676			31,485
610	229,638					229,638			33,241
671	188,956					87,843	101,113		28,657
732	59,888					3,870	56,018		7,536
793	99,575						99,575		15,917
854	61,319						61,319		8,253
915	168,635						161,751	6,884	26,464
976	107,639						105,122	2,517	25,944
1037	226,745						25,764	200,981	43,250
1098	58,207							58,207	10,292
1159	155,768							155,768	25,666
1220	36,668							36,668	7,826
1342	54,417							54,417	14,582
1403	23,348							23,348	6,264
1464	1,975							1,975	266
tonnes	3,045,196	11,547	269,237	444,715	559,641	608,629	610,662	540,765	3,045,196
t/d	1,477	-	748	1,235	1,555	1,691	1,696	1,502	
Au oz	519,683	2,262	50,602	80,788	87,261	92,072	101,740	104,958	519,683

Pre-Commercial Production

16.13 Material Handling

The material handling system is divided into two parts. First, the upper material handling system from the 122 m to the 244 m Levels, and second, the lower material handling system from the 305 m Level to the bottom of the mine at the 1,464 m Level.

16.13.1 Upper Material Handling System

The current upper material handling system was built with a series of connected raises, between the 122 m and 244 m Levels, where ore and waste were collected, then transported by rail for hoisting at the mid-shaft loading pocket.

This PEA reflects the addition of a 4.5 m by 4.5 m surface portal/ramp that connects down to the 305 m Level within the Pre-CP phase of this Project. The PEA plan is to have upper mineralized material and waste transported to surface via 40-tonne trucks from levels above the 305 m Level, including the 244 m, 183 m, and 122 m Levels.

16.13.2 Lower Material Handling System

All waste and ore generated from the 305 m Level and down, will be loaded onto 40-tonne trucks and trammed to a planned truck dump on the 610 m Level. The design includes a rock breaker/grizzly screen combination on the 610 m Level with a chute at the bottom of the waste pass on the 685 m Level. This chute will transfer rock to a conveyor arrangement that will feed the 685 m Level loading pocket.

A 10-tonne loading pocket has already been commissioned on the 685 m Level. This system is currently in operation and handling waste material from the 610 m and 685 m Levels.

16.14 Mine Services

16.14.1 Mine Ventilation and Mine Air Heating

Overview

The primary ventilation system is designed as a positive or “push” system delivering 180 m^3/sec (382,000 cfm) to the underground workings. Fresh air is provided through the historical shaft and two parallel internal Alimak raises. Return air is exhaust through the primary ramp, an internal return air raise and the primary return air raises that exhausts to surface.

Fresh air is distributed from the fresh air raises through historical shaft stations, ventilation transfer drifts, and ventilation access drifts. Fresh air will flow across levels, past cross-cuts to the return air raise and ramp. Airflow distribution will be controlled by regulators in the ventilation access drifts.

The primary ventilation system fan requirement is for two parallel 84" half blade fans with a 32" hub and 373 kW (500 hp) motor. At peak ventilation load the primary fans are expected to operate at 90 m^3/sec (191,000 cfm) and 2,750 Pa (11" w.g.) total pressure. Included in the total pressure is an estimated resistance of 500 Pa (2.0" w.g.) for the heater house and surface arrangement.

These fans can be modified with additional blades and larger motors after installation to accommodate mine expansion and increased airflow needs.

Criteria and Assumptions

The following assumptions and design criteria have been used in the design of the ventilation system:

- All historical ramps were driven at 4.0 m x 4.0 m cross-section
- All new ramps are to be developed at 4.5 m x 4.5 m cross-section
- The historical shaft is 3.0 m x 6.0 m and free of major obstructions such as steal sets
- All other raises are driven by Alimak, or Raisebore, with a cross-section of 3.35 m x 3.35 m (11 ft x 11 ft)
- Surface climatic conditions used for fan sizing is assumed to be + 10°C dry bulb temperature with an air density of 1.137 kg/m^3 at surface
- Mine air heaters will be sized for a maximum temperature rise of 40°C.

Ventilation Design Guidelines

Ventilation design guidelines were used for the design of the primary, and auxiliary ventilation systems so that they will meet or exceed federal and provincial requirements, with airflow requirements addressing diesel exhaust emission controls and volume requirements specified by Canmet (Table 16-17).

Air volume requirements have been calculated based on:

- Proposed diesel equipment fleet
- CANMET approved ultra-low diesel engine certification data
- Air density was corrected for a maximum average mining depth of 800 m below collar (auto-compression of 10.0%)
- 10% leakage factor was added
- Applied utilization factors are estimates based on typical equipment availability and operational practices.

Table 16-17: Airflow Requirements

Equipment	Fleet	Canmet Vent Rate (m^3/sec)	Utilization Factor (%)	Utilization Based Ventilation Requirement (m^3/sec)	Utilization Based Ventilation Requirement (cfm)
Elect. Scoop (2 yd)*	2	0	50	-	
Scoop (4 yd)*	1	7.22	100	7.2	15,299
Scoop (2 yd)*	1	3.87	100	3.9	8,201
Scoop (4 yd)*	1	6.07	100	6.1	12,862
Scoop (6 yd)	3	8.07	66	16.0	33,859
Scoop (4 yd)	2	3.35	50	3.4	7,099
Scoop (2 yd)*	1	5.33	100	5.3	11,294
Jumbo (2 Boom)*	1	6.47	0	-	-
Jumbo (2 Boom)	2	2.21	100	4.4	9,366
Jumbo (1 Boom)	2	2.21	100	4.4	9,366
Haul Truck (45 t)	4	14.16	75	42.5	90,015
Scissor-lift*	1	6.47	0	-	-
Scissor-lift	5	3.87	80	15.5	32,802
ANFO Loader	4	3.87	50	7.7	16,401
Mobile Shotcreter	2	3.87	50	3.9	8,201
Boom Truck	2	6.47	50	6.5	13,710
Grader	1	3.87	33	1.3	2,706
Mechanics Vehicle	2	3.87	33	2.6	5,412
Electricians Vehicle	2	3.87	33	2.6	5,412
Supervisor vehicle	4	3.87	50	7.7	16,401
Personnel Carriers	6	3.87	33	7.7	16,237
			Subtotal	**148.5**	**314,643**
Leakage			10	14.8	31,464
Density corrected for 800 m below collar			1.10	16.8	35,498
			Total	**180**	**381,606**

Note: *Equipment already owned by Rubicon

16.14.2 Primary Ventilation and Methodology

Fresh air will be supplied underground through the main downcast raise, which runs from surface to the 305 m Level; (Figure 16-21). The raise will be extended down to the 610 m Level. At the bottom of this raise, a ventilation transfer drift will direct air toward the southeast, southern portion of the lower orebody. Two Alimak raises will be established to direct fresh air to depth. As production in the mine increases below shaft bottom a second parallel Alimak raise will be established to accommodate the increased airflow volume needed and reduce the resistance of the ventilation network. Airflow will enter levels through ventilation access drifts and be controlled through baton board regulators.

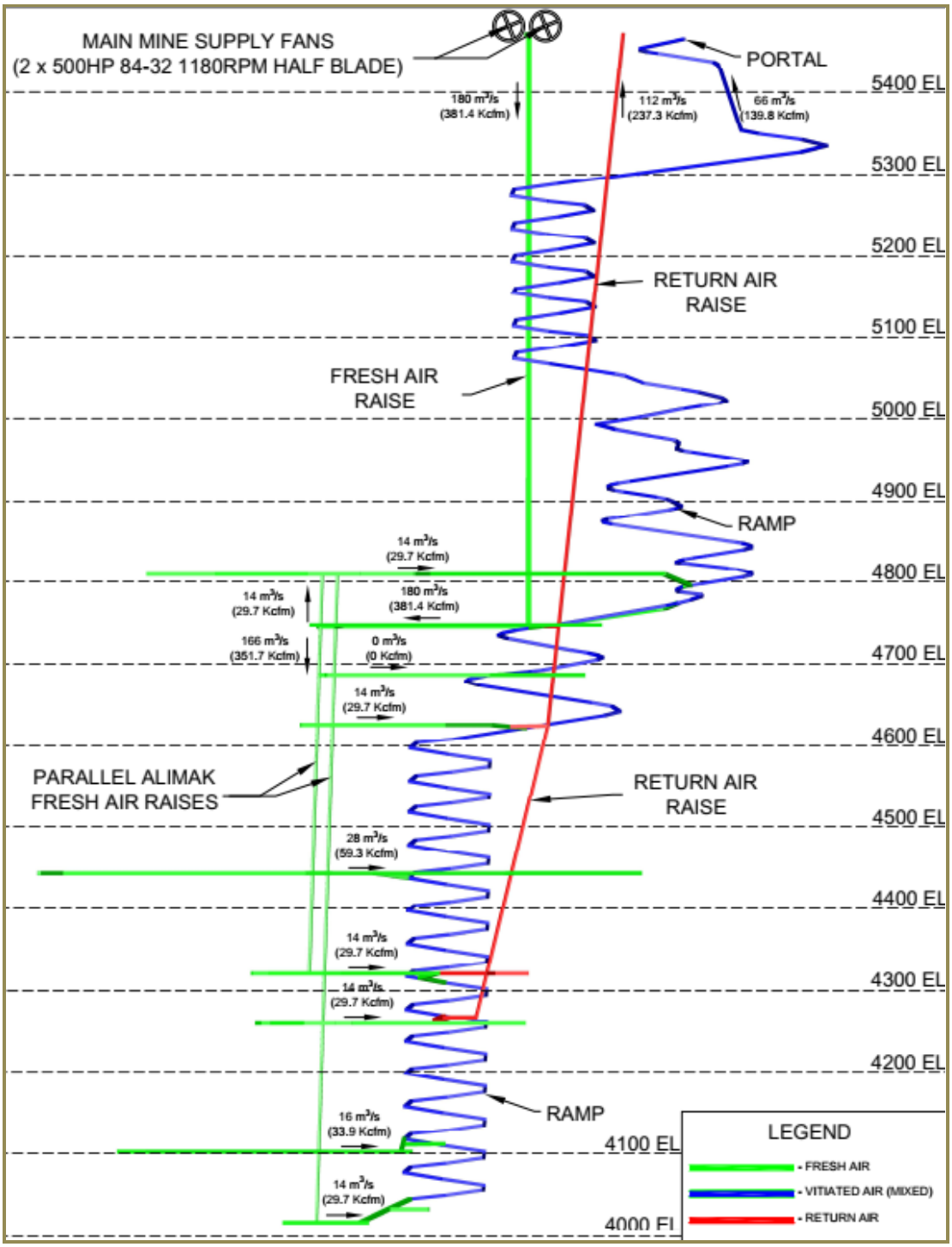


Figure 16-21: Ventilation Schematics – Peak Load

Auxiliary fans and layflat ventilation ducting will direct air from the level access drifts to the drawpoints, overcut drifts, Alimak nests and C&F stopes. See Figure 16-22

Contaminated air will flow north through the level access drift to the primary ramp and return air raise. Return air will be exhausted from the mine through the primary access ramp and return air raise. The return air raise will be driven to a depth of 1,250 m below the collar in multiple stages. The return air raise is required to both reduce the air speed in the upper section of the primary access ramp and total pressure on the primary fans.

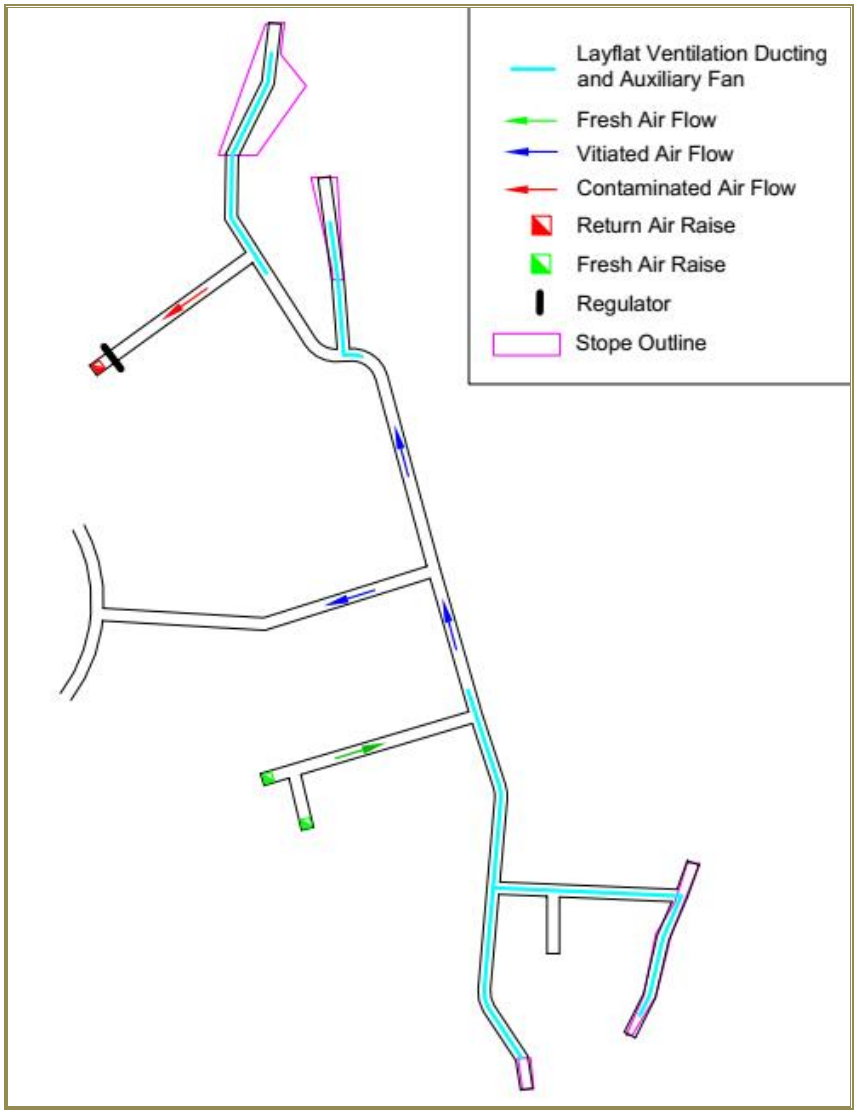


Figure 16-22: Typical Sublevel Ventilation Schematic

16.14.3 Ventilation Modelling

Computer modelling using Ventsim Visual™ was used to create a model of the mine as part of the ventilation planning process. The model was used to determine the pressures required to move sufficient air through the mine workings and meet the airflow requirements described in the Airflow Requirements section.

The model used engineering estimates and standard textbook mine resistance values. The model verifies that the required air distribution can be met and determines the pressures required to move this air.

16.14.4 Mine Air Heating Requirements

Based on average monthly outside temperatures in Red Lake, Ontario, propane consumption and associated costs were estimated. The consumption was based on a mine air intake quantity of 180 m^3/sec (382,000 cfm).

Using a maximum temperature rise 40°C, an estimated 8,583 kWh (29,295,000 BTU/h) total heating system is required on the fresh air fans. January is the coldest month with an average minimum temperature of -26.5°C. An 8,583-kWh heating capacity system would be capable of heating outside air as cold as -37°C up to 3°C. Outside temperatures below -37°C would require additional heating or reduction in airflow to underground.

Propane consumption for the total area is shown in Table 16-18.

Table 16-18: Propane Consumption

Month	Consumption (L)
November	229,257
December	485,011
January	592,947
February	460,277
March	297,174
April	42,053
Total Winter Consumption	**2,106,718**

Note: Temperature Setpoint = 3°C; Airflow = 180 m^3/sec (382,000 cfm).

16.15 Backfill

16.15.1 Rock Fill

Waste rock will be used to fill LH stopes that that do not impose risk on secondary stopes or, to fill the bottom portion of a C&F lift. The waste rock would be produced by development blasting and may

be stored underground where practicable. Otherwise, the waste rock would be trammed directly from development to the areas where the rockfill is needed.

16.15.2 Paste Fill

Paste fill and unconsolidated rock fill is planned for use in all bulk and C&F stopes. The paste-fill distribution system has been essentially completed for supply to workings above the 305 m Level. Compliant piping, Sch 40 and 80, has been installed down to the 244 m Level. Confirmation of routing and calibration of instrumentation needs to be completed as part of system commissioning.

A paste backfill plant has been designed and partially built to meet the needs for backfill and to provide tailings dewatering and distribution for the TMF. A walk-through of the plant, control system, and distribution installations by TMAC supports Nordmin's (2014) report, which concluded that "the paste system will produce around 60 m^3/h and all stopes can be filled using the HA160 configuration at peak pressure." Figure 16-23, extracted from the report, shows that gravity, plus the Putzmeister HA160 pump, generating 4,250 kPa, which substantially exceeds the PEA pressure requirements of 2,000 kPa required to deliver 60 m^3/h of paste fill to 1,400 m depths and/or 1,500 m horizontally.

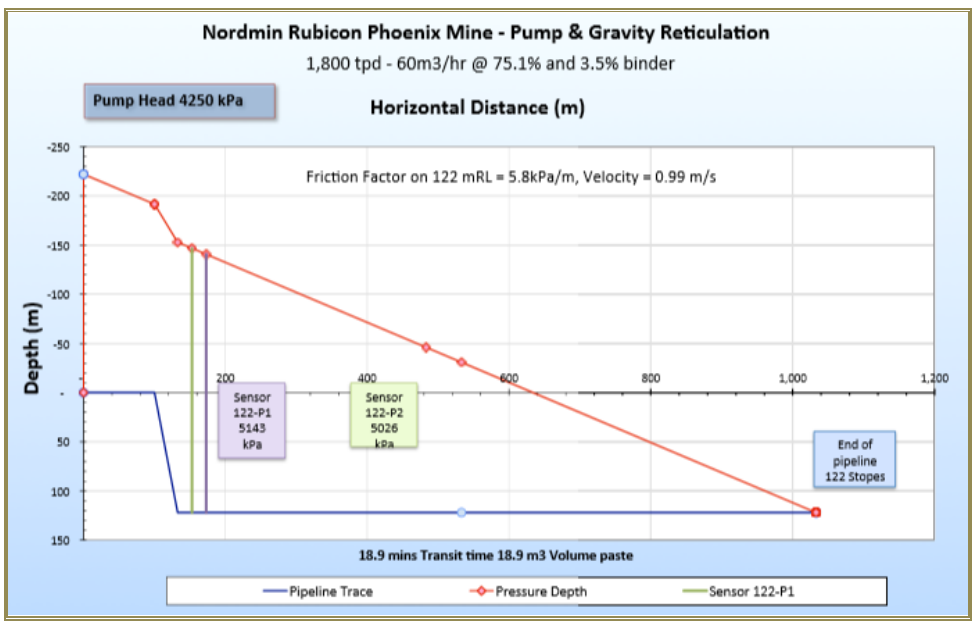


Figure 16-23: 2014 Loop Reticulation Test Results for the Rubicon Phoenix Project

Rubicon's internal laboratory measured unconfined compressive stress (UCS) testing by Lafarge and Paterson & Cooke in 2015 confirmed that Phoenix tails can provide the stope strengths, ranging from 250 to 1 MPa, as required for the four mining methods contemplated in this study (Table 16-19). Operational testing will allow field optimization of binder addition to achieve desired backfill strengths and costs.

Table 16-19: Paterson and Cooke Paste Test Results

Binder Type	Binder (%)	Slump (mm)	Solid Content (%m)	7-day UCS (kPa)	28-day UCS (kPa)	56-day UCS (kPa)
100 % EGU	3	16	76.6	260	251	314
100 % EGU	5	17	76.8	409	494	515
100 % EGU	9	16	76.8	1,263	1,492	1,474
50/50 EGU/Flyash	3	16	76.8	136	230	287
50/50 EGU/Flyash	5	16	76.8	251	446	549
50/50 EGU/Flyash	9	16	77.1	592	1,045	1,516

Note: EGU = Exshaw General Use cement

Field commission was initiated by Rubicon in 2015 but was not completed due to shutdown of operations. A review of the commissioning report shows a short timeline remains to operation.

16.16 Compressed Air Distribution System

The Project currently has two active 261 kW (350 hp) Sullair TS32-350 air compressors rated at 3,186 m^3/h (1,875 ft^3/min) each, and a small Atlas Copco GA160 backup compressor unit. These units provide adequate volumes for the work being completed presently at the site. Two additional compressors will be bought and activated in the system relative to the tonnages planned in the future. The compressors are housed in a permanent structure with temperature-controlled louvers to exhaust heat from the building. The compressors operate on a cascading system controlled by local controllers on each unit. The two units operate on a continuous basis, and cycle between loaded and unloaded.

The main compressed air-line is installed in the shaft and consists of a 150 mm (6") line from surface to the 305 m Level and a 200 mm (8") line from there to shaft bottom. While adequate for exploration purposes, system upgrades have been costed into the PEA to assure capacity to accommodate expected production rates. Construction of the compressed-air distribution system upgrade is approximately 20% complete and will be finished prior to commissioning of the mine.

16.17 Water Supply

Lake water is pumped from the adjacent East Bay of Red Lake to feed the process water and underground activities. Rubicon's permit authorizes pumping from the lake at 695 litres per minute (L/min) with a maximum daily total of 1,000,000 litres per day (L/d).

When the mill is in operation, process water in the mill and water accumulated in the TMF are recirculated back into the mill process water supply system, thereby minimizing the amount of water pumped from the lake. The underground dewatering system reports to the TMF and is authorized by permit to take water for a maximum pumping rate of 2,917 L/min, up to a maximum of 2,100,000 L/d.

Currently, water discharged to the environment from the TMF comes under regulatory control and can be discharged to the environment year-round. It must meet objectives and limits outlined in the Environmental Compliance Approval.

Potable water for the site is taken from East Bay and is conditioned by a Culligan system of nano-membrane modules, UV bacterial disinfection, and chlorine addition prior to use.

Take and discharge water requirements to meet the LOM plan have been costed and scheduled into this PEA.

16.18 Dewatering

Main dewatering stations are located at shaft bottom, the 730 m (shaft bottom), 610 m, 305 m, and 122 m Levels. This current system is capable of pumping at a maximum flow rate of approximately 757 L/min (200 US gpm) and is adequate for the current Project work.

An upgraded system capable of pumping 3,028 L/min (800 US gpm) from the 305 m Level to surface is partially complete. The current Project permit allows dewatering at a rate of 2,917 L/min (771 US gpm) and a maximum of 2.1 million L/d (0.56 M US gpd).

This PEA has included costs and scheduling for the installation of a main sump on the 610 m Level, as well as improvement of the dewatering reticulation system to accommodate 3,000 L/min from the 1,464 m Level, staged, to surface.

16.19 Fuel Storage and Distribution

Underground fuel distribution will be via dry-line delivery to Satstat portable storage units that will be located on the 305 m and 610 m Levels (Figure 16-24).

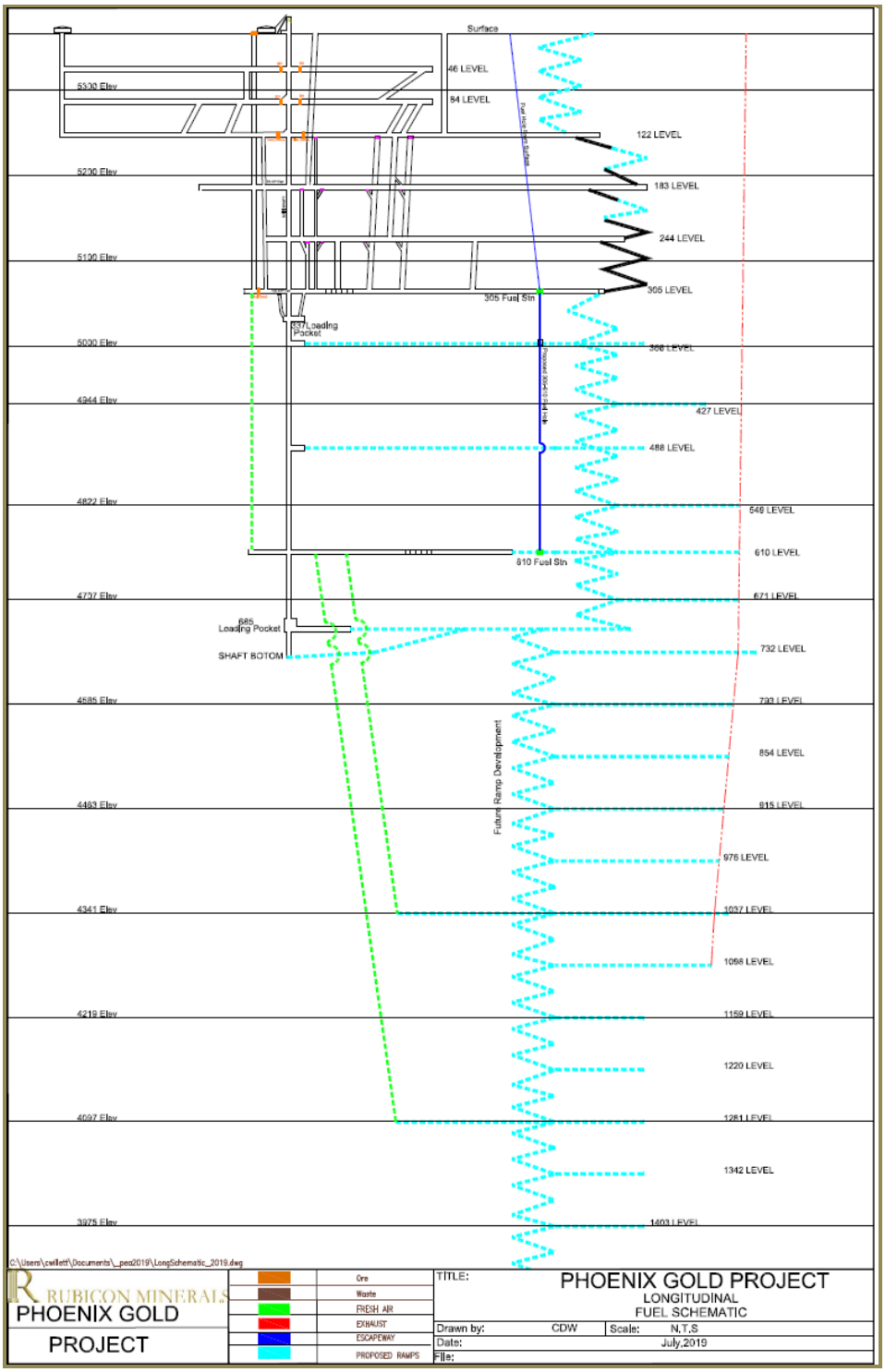


Figure 16-24: Longitudinal Fuel Schematic

16.20 Repair Bays/Shops

Two service bays will be outfitted for use underground. The existing bay on 305 m Level will be upgraded to support minor repairs for mobile equipment. A second is planned on the 610 m Level. Capital has been included in this PEA for the underground service bays as well as the construction of fully equipped repair shops on surface.

16.21 Refuge Stations

There are four completed refuge stations located underground, on the 122 m, 183 m, 244 m, and 305 m Levels. The PEA plan includes portable, self-contained refuge stations that would be dispersed throughout the mine near the main and current work areas. The refuge stations are designed to withstand fire, extreme heat, and prevent gas from entering the unit. They are equipped with potable water, a telephone line, lighting, compressed/fresh air, rations, and clay to seal any cracks in openings to further prevent gases from entering. Using portable units decreases the amount of development required, as they can be put in previously used headings.

16.22 Explosives and Detonator Storage

No surface explosive magazines are planned. Upon delivery to site, explosives are moved to authorized magazines underground for storage.

16.23 Mine Equipment

The current, active equipment fleet, listed in Table 16-20, has been taken into consideration for the overall LOM plan. Some of this equipment will play a role during the development phases of the Project, but the majority is not anticipated to survive to the production phase; thus, a new fleet of equipment has been budgeted for the Project.

The LOM equipment fleet peak occurs in Year 2. Table 16-21 demonstrates the anticipated fleet requirement in that year. All PEA equipment is considered new and will be leased, 10% initial payment plus 3% of new per month over the LOM. Equipment financing considerations in this PEA are treated the same regardless if the new equipment is company- or contractor-owned.

On any transition from contractor to owner operation, Rubicon could acquire the equipment fleet. It is assumed that such equipment would continue to be leased on similar terms so that the unit operating and development costs are not affected. With the LOM being less than six years, it is anticipated that all equipment would survive without having to be replaced.

Table 16-20: Active Underground and Surface Mobile Equipment

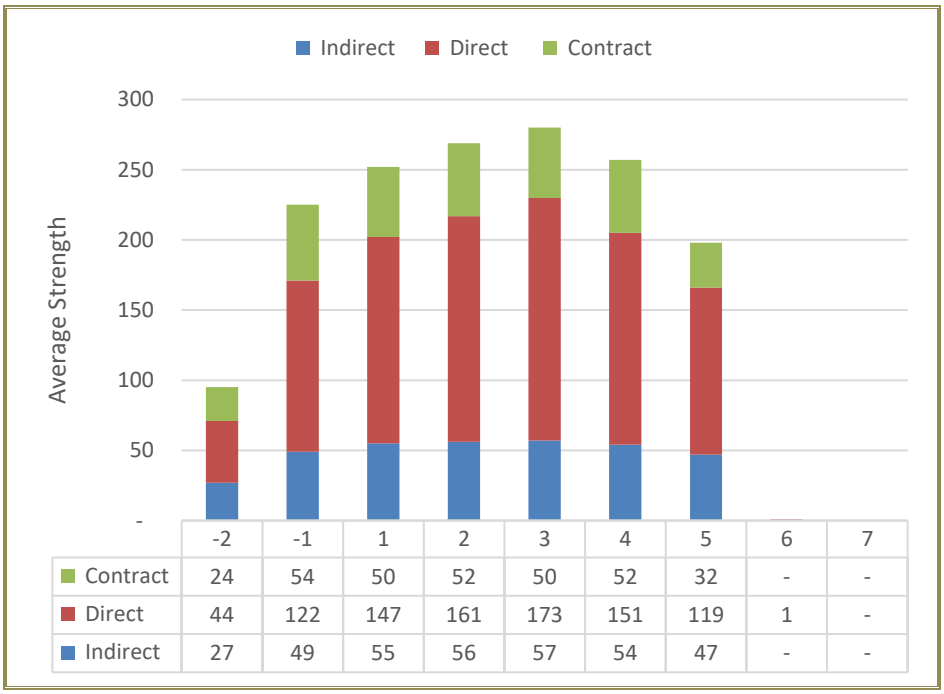
	Year	Make	Model
Active Underground Equipment			
Electric Scoop No. 610	2008	Atlas Copco	Est-2d
Electric Scoop No. 685	2014	Atlas Copco	Est-2d
Scoop No. 244	2015	Atlas Copco	St-7
Scoop No. 610	2012	Atlas Copco	St-2g
Scoop No. 183	2018	Sandvik	Lh307
Jumbo No. 244	2011	Atlas Copco	282
Jumbo No. 183	2014	Atlas Copco	S1d
Haul Truck No. 244	2014	Atlas Copco	Mt2010
Scissor-Lift No. 244	2014	J&S Manufacturing	Slx4100m4d
Diesel Loci (55 Hp) No. 305		Clayton	-
Diesel Loci No.2 44		Clayton	-
Diesel Loci No. 305		Clayton	-
Forklift No. 244		Kubota	420
Excavator No. 244		Bobcat	323
Excavator No. 183		Bobcat	324
Long Tom No. 305			-
Active Surface Equipment Number			
Loader No. Lo001		Caterpillar	924 h
Forklift No. Mfl002		Manitou	Mlt 840-115
Forklift No. Cfl004		Caterpillar	Th255c
Aerial Lift No. Al001		Jlg	800 Aj
Light Plant No. Lp001		Multiquip	Lt12d

Table 16-21: Mining Fleet Peak Requirements

Equipment Type	Peak Units (Year 2)	Unit Cost ($)
2 Boom Jumbo—Ramp and Levels	3	1,021,418
1 Boom Jumbo—Access and C&F	4	617,756
Mechanical Bolter	3	846,296
LHD 6 YD	3	1,014,750
LHD 4 YD	4	712,743
LHD 2.5 YD	4	596,687
40-tonne Truck (Includes a Spare)	6	1,567,621
X-Lift	6	288,659
Anfo Loaders	4	550,903
Mobile Shotcreter	2	498,593
Boom Truck	2	251,250
Grader	1	318,750
Mechanics Vehicle	3	82,000
Electricians Vehicle	3	82,000
Supervisor Vehicle	5	65,000
Personnel Carriers (6-person)	7	95,000
Satstat (SE1004-A 4,550L) Fuel Station	3	450,000
Jacklegs	16	7,898
Stopers	16	7,898
Misc. Drill Steel and Bits	1	250,000
Misc. Tools for Repair bay	1	150,000
Longhole Drill Contractor	0	-
Alimak (by Contractor)	0	-
Alicab (by Contractor)	0	-
Alimack Rail (by Contractor)	0	-
Lamps & Radios	210	1,000

16.24 Mine Personnel

The Project workforce requirements are shown in Figure 16-25 and Table 16-22. Approximately 45% of the workforce will be housed at Rubicon's on-site camp at an anticipated cost per person of $67.50/d. It is anticipated that the remainder of the workforce will be accommodated in the modern communities of Red Lake or Balmertown. Details on wages and burdens are discussed in the operating cost section of this report.



	-2	-1	1	2	3	4	5	6	7
Contract	24	54	50	52	50	52	32	-	-
Direct	44	122	147	161	173	151	119	1	-
Indirect	27	49	55	56	57	54	47	-	-

Figure 16-25: Workforce Loading by Year

Table 16-22: Peak Workforce Breakdown (Year 3)

General	Indirect	Direct	Contract	Total
Mine Management	1	-	-	
Administration	6	-	-	
Safety & Training	1	2	-	
Security	-	4	-	
Yard & Warehouse	3	6	-	
Environmental	2	-	-	
Surface Waste Handling	-	2	-	
Subtotal Surface	13	14	-	27
Mill				
Mill Management	1	-	-	
Mill Supervision	5	-	-	
Plant & Surface Operations	-	28	-	
Assay Laboratory	4	-	-	
Maintenance Mill	-	12	-	
Subtotal Mill	11	40	-	51
Mine				
Mine Supervision	10	-	-	
Maintenance Supervision	5	-	-	
Maintenance	-	24	4	
Mine Technical Services	16	-	-	
Development	-	37	-	
LH Stoping	-	-	25	
C&F Stoping	-	18	-	
Raise Miners	-	-	5	
Trammers	-	14	-	
Shaft & Muck Circuit	-	12	-	
Backfill	-	2	-	
Construction and Utility	-	13	-	
Definition Diamond Drilling	4	-	16	
Subtotal Mine	35	120	50	205
Total Site Personnel	**59**	**174**	**50**	**283**

17 RECOVERY METHODS

Rubicon constructed a processing mill on site, along with ancillary, mine-waste storage, and TMF. Construction of the mill began in 2013 and was essentially completed during 2015. The principal components of the paste backfill system were installed, but piping, electrical, and instrumentation connections remain to be completed and commissioned. The mill was commissioned and operated between May 2015 and November 2015. The average head grade to the mill in 2015 was 3.02 g/t Au. The mill ceased operating on November 21, 2015 and was placed into care and maintenance. A total of 5,610 oz Au were produced. A Technical Report was prepared by SRK (2016) that included results from the 2015 mill operation.

The mill continued to be maintained, and some gold locked in the process equipment was recovered during 2016. In general, during the care and maintenance period, reasonable effort was made to ensure that the integrity of the major equipment was preserved. Prior to start-up of the mill for the bulk sample testing in 2018, all equipment was inspected, and any deficiencies found were corrected. These steps effectively served to recommission the facility.

Rubicon prepared the mill for operation in 2018 to batch process approximately 35,000 tonnes of mineralized material to be mined from three stopes within the F2 Gold Zone. Rubicon successfully batch-processed material mined from three stopes in the F2 Gold Zone along with 7,620 tonnes of low-grade material from various other sources. Concurrently Rubicon identified and mitigated many legacy process issues that could only be identified while in operation. In total, 43,250 tonnes were processed, producing 5,669 oz Au. The 2018 bulk sample processing campaign yielded an improvement in overall gold recovery as compared to the 2015 operation, with the largest gain occurring in gravity gold recovery.

The mill has once again been placed in care and maintenance after the 2018 test was completed. Lessons learned from the previous care and maintenance have been applied to better preserve the integrity of the equipment.

17.1 Process

The process flow diagram for the Phoenix Gold Project is presented in Figure 17-1. The mill was designed with an initial throughput capacity of 2,500 t/d and set up to operate between 1,250 t/d and 1,800 t/d, with provisions in the layout to increase capacity up to 2,500 t/d with additions to the existing equipment.

The mill contains an ore-handling system feeding a two-stage grinding circuit closed by hydrocyclones. Free gold is recovered by gravity concentration in the grinding section, and by cyanide leaching in a CIL circuit. Doré is produced by smelting the gravity circuit concentrate and the electrowinning sludge produced from the CIL circuit.

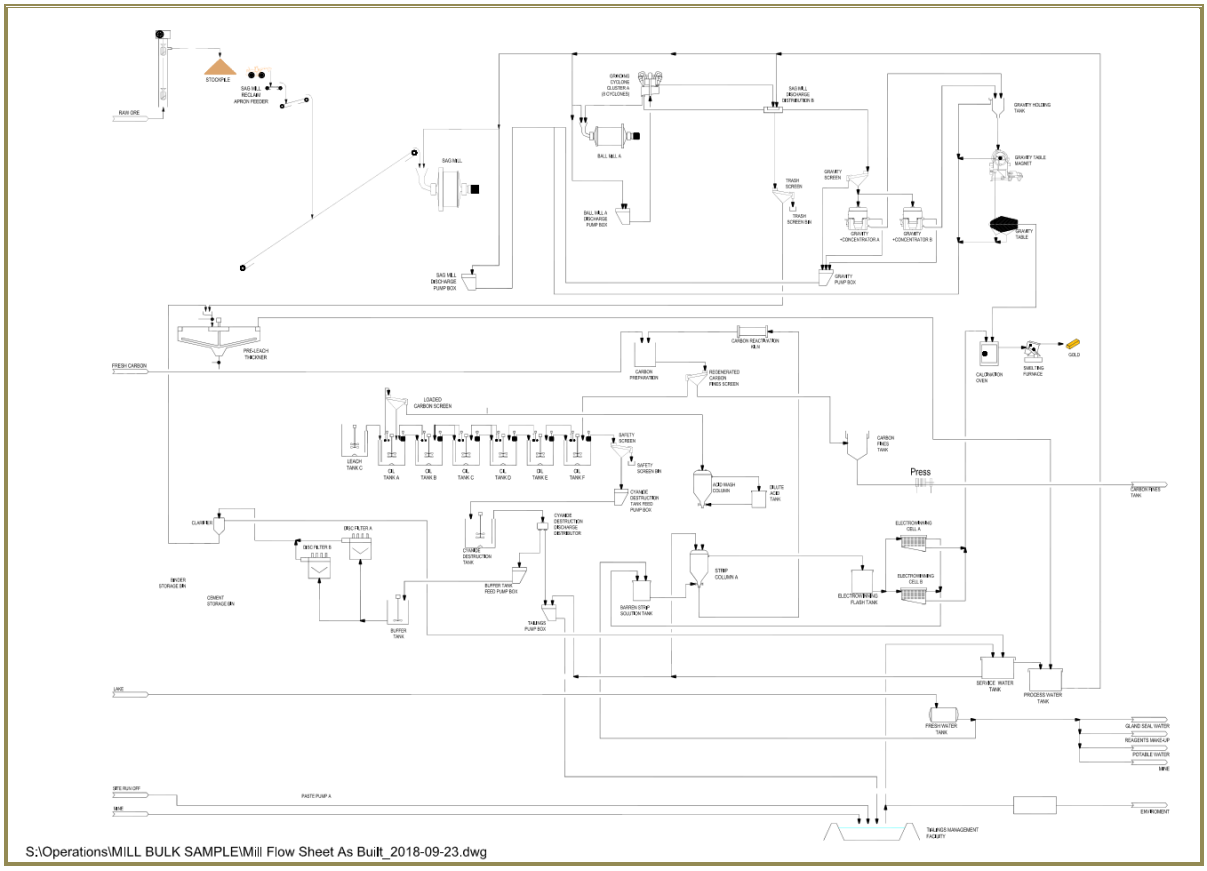


Figure 17-1: 1,250 to 1,800 t/d Simplified Process Flow Diagram Description

For completeness, the 2,500 t/d process flow diagram is presented as Figure 17-2 which shows the outstanding equipment insertions which would be required if it became desirable to transition up to a 2,500 t/d processing facility.

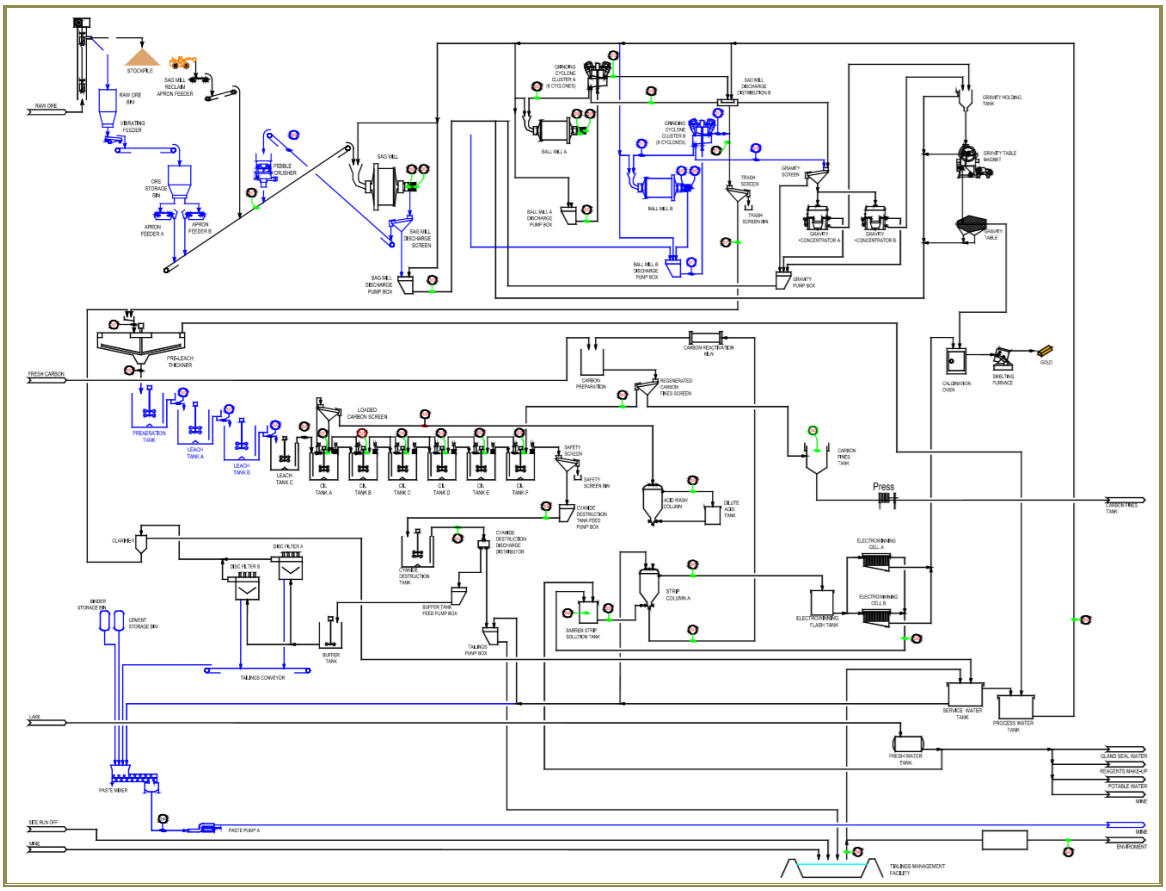


Figure 17-2: Original 2,500 t/d PFD showing Installation Requirements to Transition From 1,800 to 2,500 t/d (in blue)

The process plant construction commenced in 2013 and was essentially complete in the spring of 2015; however, equipment purchased for several unit operations was not installed. The gold recovery plant was commissioned in 2015 and operated intermittently until November 21, 2015, when surface stockpile milling was completed. A paste backfill plant to prepare paste backfill for use underground was also constructed but not commissioned. An Actiflo® metals treatment system was installed in 2017 and commissioned mid-2018 to operate in conjunction with the previous Actiflo® solids removal system and ensure effluent meets environmental discharge objectives.

The existing process consists of a single line, starting with a SAG mill. The discharge from the SAG mill is sent to the ball mill circuit, which uses hydrocyclones in closed circuit for classification. A gravity separation circuit is included to recover and concentrate any gravity-recoverable gold. The remaining gold is extracted in a conventional CIL circuit. The loaded carbon is washed with hydrochloric acid

solution to remove carbonates. Gold is then removed from the loaded carbon by elution (stripping) followed by electrowinning. The electrowinning and the gravity circuit both produce a high-grade gold concentrate that is smelted in an electric induction furnace to produce doré. The stripped carbon is regenerated in a reactivation kiln before being reintroduced to the process. Fine carbon is constantly eliminated (and recovered) from the process to avoid gold loss, with fresh carbon being continuously added to the process.

The cyanide contained in the tailings from the CIL circuit is eliminated in a cyanide destruction tank using the SO_2/O_2 cyanide destruction process. Either liquid SO_2 or SMBS can be used as the SO_2 source. Once the cyanide is destroyed, the tailings are pumped to the TMF for storage.

When paste backfill is required, tailings will be diverted to the paste plant, where they will be filtered to lower the water content. The filter cake will then be mixed with fly ash and cement to produce a paste, which will be pumped to the underground for backfilling. The gold recovery plant, cyanide destruction process, and TMF were commissioned and operational in 2015. The backfill plant has not yet been operated, as the Project has not yet required backfill. Major equipment for the tailings filter plant and the paste plant has been installed. However, some minor piping, electrical, and instrumentation connections remain to be completed before this equipment can be commissioned and operated.

17.1.1 Mineralized Material Storage

An underground grizzly screen on the 305 m Level with standard 23 cm openings (10" x 10") and a rock breaker are used to reduce the mineralized material size prior to hoisting it to the surface. Underground crushing was in the original design and a 48" jaw crusher, associated hoppers, structural steel, and electrical equipment were procured but never installed. The skipped mineralized material is dumped into a 277-tonne capacity raw ore bin adjacent to the headframe, while the waste is dumped into a waste bunker adjacent to the headframe. The mineralized material is discharged from the raw-ore bin via a discharge chute onto a vibratory feeder, which then transfers the mineralized material onto the ore storage bin feed conveyor. A cross-belt magnet is situated above and running perpendicular to the ore storage bin feed conveyor and is used to remove tramp metal from the coarse mineralized material.

The Phoenix mill is proposed to process a minimum of 3 Mt of diluted and recovered Mineral Resource. Annual production forecasts are shown in Table 17-1.

During the 2015 operation, the ore handling system was not commissioned before starting the mill. Mineralized material was fed to the mill through the reclaim feed system, feed rate to the SAG mill could not be controlled evenly due to feeder setup, large material, and tramp steel. In September 2015 the ore handling system was commissioned, however, operating this system remained difficult while feeding uncrushed ore. In 2018 the ore handling system was not recommissioned since ore was stockpiled outside separately by stope.

Table 17-1: Proposed Annual Production – Conceptual LOM

Period	Daily (t)	Annual* (t)	Paste (t)	Tailings (t)
Year -2	0	11,547	4,619	6,928
Year -1	748	269,237	107,695	161,542
Year 1	1,235	444,715	177,886	266,829
Year 2	1,555	559,641	223,856	335,785
Year 3	1,691	608,629	243,452	365,177
Year 4	1,696	610,662	244,265	366,397
Year 5	1,502	540,765	216,306	324,459
Total		**3,045,196**	**1,218,078**	**1,827,118**

Note: *360 days per year

For future operation, the underground crushing concept will be replaced by a modular low-profile semi-mobile surface crusher installed next to the headframe, which will receive mineralized material from the raw-ore bin feed conveyor or direct-dump run-of-mine truck or loader feed to size appropriately for the SAG mill. The mineralized material would then be conveyed to the ore storage bin.

17.1.2 Grinding and Thickening

The raw mineralized material from the ore storage bin is reclaimed by two apron feeders and is discharged onto a first conveyor. The material on the first conveyor is discharged to a second high-angle conveyor equipped with a belt scale, which then transfers the mineralized material to the SAG mill mobile feed chute. Mineralized material reclaimed from stockpiles can be fed through a hopper, feeder belt and transfer conveyor to the SAG feed conveyor, bypassing the ore storage bin.

The grinding circuit is a double-stage circuit consisting of a 6.1 m x 3.35 m SAG mill and a 3.2 m x 4.9 m ball mill. The SAG mill operates in open circuit, while the ball mill is operated in closed circuit, with 381 mm hydrocyclone. Process water is added to the SAG mill feed chute to achieve the correct dilution for grinding. The head grade is measured by pumping all of the SAG mill discharge thru a primary sampler and overflow is combined with ball mill discharge and pumped to the hydrocyclones. The sampler underflow is sized by a sample crusher (ball mill) before it is cut by the secondary sampler. Approximately 75% of the hydrocyclone underflow is directed to the ball mill for regrinding, while the remaining portion goes to the gravity separation circuit. The hydrocyclone overflow pulp flows to the thickening circuit.

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

The pre-leach thickener is fed by the trash screen undersize and the thickening area sump pump. Flocculant is also added to improve the settling rate and lime is added to adjust pH. The thickener overflow feeds by gravity to the process water tank, while the underflow is pumped to the pre-aeration tank in the CIL circuit. During shutdown, the thickener underflow recirculates to the thickener feed box.

In the original start-up of 2015, the mill was difficult to control, had regular unplanned outages, SAG mill grate failure and uncontrolled surges in the trash screen feed which overflowed to the trash bin and leaked to the underflow bypassing screening. The coarse solids accumulated in the pre-aeration tank and the trash contaminated carbon and fouled the carbon handling circuit strip vessel.

Unknown to operations at the time, the process water tank was accumulating fine solids from thickener overflow and reclaim water. The fine slurry was pumping through the process water system, damaging unlined pump casings, control valves, block valves, and piping.

Many issues addressed in the 2018 pre-operation inspection and recommissioning were related to mill setup, unreliable instrumentation, and/or installation. Despite the 2015 problems the Phoenix mill regularly sustained 1,250 to 2,500 t/d rates during the short-day shift windows it operated.

17.1.3 Gravity Separation

The gravity circuit consists of one vibrating screen, two gravity concentrators, a concentrate pumpbox, a transfer pump, a 4-tonne concentrate holding tank, one gravity table, and one gravity-table magnet. The hydrocyclone underflow launder is partitioned to collect underflow from three of the hydrocyclones (optimally two operational and one standby) within the cluster and is sent to the gravity circuit. The underflow from the remaining five hydrocyclones (normally three or four are operational hydrocyclones) is returned to the grinding circuit.

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

The gravity-screen undersize flows to a distribution box, then directed to the gravity concentrator where gravity-recoverable gold is recovered. Dilution water is added by the spray wash directly to the gravity-screen underflow to facilitate the pulp flow into the concentrator and to wash fine solids from the overflow. The gravity concentrate is pumped to the gravity concentrate holding tank, while the gravity concentrator tailings are directed to the gravity tailings pump box, and combined with gravity-screen oversize, then pumped back to the hydrocyclone feed pump box in the grinding circuit.

The gravity concentrate stored in the gravity holding tank is fed to the gravity-table magnet, where magnetic particles are removed. The non-magnetic portion of the stream is sent to the gravity table to produce an upgraded gold concentrate that is then calcined in an oven prior to being smelted into

doré in the on-site refinery. The gravity-table tailings are pumped to the hydrocyclone feed pump box, along with the gravity-screen oversize, and the gravity-concentrator tailings, for reprocessing in the grinding circuit. Magnetic particles from the gravity magnet are captured in pails and stored for future recovery of gold at an outside facility.

The initial mill start-up in 2015 did not go well for gravity recovery of gold. The circuit was started only in July 2015 and operated 42% of the time; average recovery of gold entering the mill was 14%. The primary problems were poor performance of the Knelson concentrators and the difficulty pumping Knelson concentrate to the storage tank above the gravity table in the refinery.

17.1.4 Carbon-in-Leach

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

The six tanks have been sized to provide 36 hours of residence time at the design flow-rate and solids concentration for 1,250 t/d. Each tank is equipped with a single inter-stage screen and a carbon-transfer pump and is agitated to maintain the solids in suspension. Air is injected in the bottom of the pre-aeration tank and in each CIL tank of the other tanks for gold dissolution. Interconnecting tank launders are arranged so that any tank in series can be bypassed without having to shut down the entire CIL circuit.

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

17.1.5 Elution and Carbon Reactivation

Loaded carbon recovered by the loaded carbon disengagement screen gravitates to the loaded carbon tank where it is rinsed to flush fine slurry, which is then pumped to the acid-wash column of the elution circuit. The carbon elution circuit treats a 4-tonne batch in approximately 12 hours. This circuit is designed to process one elution batch per day.

The acid solution is prepared in the dilute acid tank, then pumped through the acid-wash column. Once the acid wash is complete, the spent acid is neutralized with sodium hydroxide. The carbon is transferred from the acid-wash column to the strip column for gold desorption. The solution from

the barren strip solution tank flows through a series of heat-recovery heat exchangers and a heater to reach the correct strip temperature. The solution strips the gold on the carbon, which is then removed through a Johnson screen from the upper-side of the column. This loaded solution then flows to the heat-recovery heat exchanger to preheat strip-feed solution on the way to the electrowinning cells in the refinery for gold recovery.

The stripped carbon is withdrawn from the base of the strip column and is pumped to the carbon reactivation kiln feed hopper. The carbon is dewatered through a drain screen while it is fed by screw conveyor to the reactivation kiln. After reactivation, the carbon is discharged into a quench-tank, and then is pumped over a screen to remove (and recover) fine carbon while reporting to the regenerated carbon storage tank. When required it is pumped from the regenerated carbon storage tank to the final CIL tank. Fresh carbon from the pre-attrition tank is added into the carbon quench tank on a regular basis to replenish the fine carbon that has been removed.

17.1.6 Electrowinning and Refinery

The gold bearing "pregnant solution" from the strip column is pumped through the heat-recovery heat exchanger to the steady head electrowinning flash-tank, then flows by gravity to a pair of parallel electrowinning cells, where the gold is plated onto cathodes. After electrowinning the stripped "barren solution" is recovered and pumped back through the preheat heat exchanger on the way to the barren strip solution tank in the elution circuit.

After a fixed period, the stainless-steel cathodes are cleaned using high-pressure water, and the gold sludge falls to the bottom of the cell. This sludge is then pumped using a diaphragm pump to a filter press to remove excess water. This filtrate is then returned to the barren-solution pump box.

The filtered gold sludge is taken to a calcining oven for drying before suitable fluxes are added (typically borax, soda ash, sodium nitrate, and ultrafine silica sand). This mixture is then placed into the induction furnace crucible and power is turned on. Once the gold has melted, it is sampled as it is poured into doré moulds. After the bars have solidified, they are extracted from the moulds and weighed prior to shipment.

17.1.7 Cyanide Destruction

A safety screen ahead of the tailings pump box is fed from the last CIL tank overflow, to prevent loss of fine broken carbon, which is recovered into an oversized bin.

The screen undersize is pumped to a primary and secondary sampler that discharges into the cyanide destruction tank. Liquid oxygen (produced on site) is added into a dispersion cone at the base tank, and liquid sulphur dioxide (SO_2) is added. Alternatively, a sodium metabisulphite (SMBS) solution can be prepared and used in the reactor. Copper sulphate and lime slurries are then added at the top of the tank.

Once cyanide has been destroyed, the treated tailings are pumped to a distributor. When the paste plant is operating, these tailings are pumped to the buffer tank. However, when the paste plant is not operating, the tailings are directed to the tailings pump box and are pumped to the tailings pond. Service water can be added to the tailings pump box to control pump surging, and to circulate reclaim water to prevent pipelines freezing in winter.

17.1.8 Tailings Filtration (Not Commissioned)

The tailings filtration circuit has not yet been completed or commissioned. This filtration system consists of a pair of disk filters, with two filter feed pumps, two vacuum pumps, two snap-blow receivers, two filtrate tanks, and two filtrate pumps.

When completed, buffered cyanide-free tailings are pumped to the buffer tank, then fed to one of a pair of filtration units, and filtered tailings are discharged onto the tailings conveyor, which feeds the paste mixer. Clean filtrate can then be returned to the service water tank.

17.1.9 Paste Backfill Preparation (Not Commissioned)

The paste backfill plant has also not yet completed or commissioned. The disc-filter tailings cake from the tailings conveyor is mixed with service water in a paste mixer to produce backfill paste. Fly ash and Portland cement can be added to achieve the requirements of underground backfill. These proportions are controlled to achieve the proper concentration in the backfill paste, which is then discharged into the paste-pump feed hopper.

17.1.10 Paste Backfill Distribution (Not Commissioned)

The paste-backfill distribution system is also not completed or commissioned. Once the paste is prepared, one positive-displacement pump will be used to deliver the paste into the underground stopes. Each pump is equipped with a hydraulic unit.

17.1.11 Reagents

Apart from the reagents used in relatively small quantities in the electrowinning and refinery sectors, the reagents discussed below are those used throughout the gold recovery process.

Sodium Cyanide

Sodium cyanide (NaCN) is supplied in 1-tonne bags and is mixed with water in batches on site in a controlled environment, then transferred to the cyanide distribution tank. The sodium cyanide solution is pumped to the CIL circuit and the barren elution solution tank.

Flocculant

Flocculant is used in the pre-leach thickener to improve the solids settling rate. Flocculant is supplied in 23 kg bags. The preparation station consists of a wetting unit, mixing tank, and distribution tank. The flocculant is then pumped into the pre-leach thickener.

Hydrochloric Acid

Hydrochloric acid (HCl) is used for the carbon acid wash. The HCl is supplied in 1 m^3 chemical totes and pumped to the acid storage tank. The acid is pumped to the dilute acid tank in the carbon elution circuit as required. In both 2015 and 2018 acid washing was not used since the mill was started on new carbon in both cases.

Lead Nitrate

Lead nitrate ($PbNO_3$) is sometimes used to improve the gold leaching kinetics in the CIL circuit. A lead nitrate handling and addition system designed for 1 tonne totes has been installed but not used in 2015. Instead, the pumps and piping were re-tasked to provide a high volume of caustic for the temporary ammonia treatment plant installed in CIL. The system was recommissioned for its original purpose and used in processing the 2018 bulk sample.

Sulphur Dioxide

Liquid sulphur dioxide (SO_2) is used as an oxidizing agent in the cyanide destruction process. The SO_2 is delivered by truck and stored in the SO_2 tank. The SO_2 tank is equipped with a pressure system to keep the SO_2 in liquid form and to deliver the SO_2 to the cyanide destruction tank. Liquid SO_2 was the primary oxidizing agent for the cyanide destruction system installed at the Phoenix mill as commissioned and operated in 2015. In July 2015 the piping system failed, and SMBS was used for cyanide destruction for two months while the bulk SO_2 system was repaired. Upon mine and mill shutdown at the end of 2015, this system was decommissioned; a contractor was paid to dispose of the liquid SO_2, flush the system and pressurize it with nitrogen gas at a substantial cost. Liquid SO_2 was not used as the oxidizing agent for the short duration of the 2018 bulk sample processing.

Sodium Metabisulphite (SMBS)

SMBS ($Na_2S_2O_5$) is used as an oxidizing agent in the cyanide destruction process as an alternative to liquid SO_2. The SMBS is delivered by truck in 1 to 1.5-tonne tote bags. The SMBS is mixed with water in a mixing tank that dumps to a stock tank, and the solution is pumped to the cyanide destruction tank. This system was operated during the 2018 bulk test.

Lime

Lime, delivered as quicklime (CaO), is used to control the pH in the grinding, CIL, and cyanide destruction circuits to prevent cyanide gas (HCN) formation. The high-quality lime is delivered in bulk by truck and stored in the lime bin. A screw conveyor fed from the lime-bin flow activator and star feeder transfers the lime to the lime slaker to prepare the milk of lime, which is stored in the lime distribution tank. Distribution pumps deliver the milk of lime to the CIL circuit and cyanide destruction circuits through a closed-loop distribution system. The mechanical seals on the circulation pumps failed in 2015 causing the operators to run the system at maximum continuously to avoid sanding the main line. This problem was solved in 2018 by replacing the lime circulation pumps with peristaltic pumps that greatly reduced lime usage.

Copper Sulphate

Copper sulphate ($CuSO_4$) is used as a catalyst in the cyanide destruction process. Copper sulphate is supplied in 1-tonne tote bags and mixed with water in batches on site in a controlled environment, then transferred to a distribution tank. The copper sulphate solution is pumped to the cyanide destruction tank as required. Copper sulphate is also added dry from a pail to the cyanide destruction tank.

Sodium Hydroxide

Sodium hydroxide (NaOH) is used for carbon stripping and to neutralize the residual acid in the dilute-acid tank and the acid-wash column. The caustic (50% solution) is supplied in 1 m^3 chemical totes and pumped to the caustic storage tank. A distribution pump transfers the caustic to the dilute acid tank and to the barren strip solution tank.

Descalant

A descalant reagent is used to reduce calcium carbonate deposits. The descalant system was not installed as part of the Phoenix Project. A vendor-supplied package for distribution was installed but never used during the 2015 campaign. During the 2015 shutdown the system was removed by the vendor and never replaced. Installing a permanent system (or vendor package) that supplies descalant in totes and pumps to the process water tank and barren strip solution tank is required.

Cement

Cement will be used at the paste plant to enhance the strength of the paste backfill. Cement will be delivered in bulk by truck and will be stored in a bin. A screw conveyor will deliver the cement to the paste mixer. This system has been constructed but has not yet been commissioned or operated.

Fly Ash

Fly ash will be used at the paste plant to enhance the strength of the paste backfill. Fly ash will be delivered in bulk by truck and will be stored in a bin. A screw conveyor will deliver the fly ash to the paste mixer. This system has been constructed but has not been commissioned or operated.

17.1.12 Utilities

Freshwater

A freshwater system is required to store and distribute freshwater to various areas of the mill and Project site. The existing freshwater tank is situated at the highest topographical location, south of the hoist room. The freshwater tank is fed by the redesigned pump system that draws water from East Bay of Red Lake through a basket strainer. Two freshwater pumps (one operational, one standby) distribute freshwater to the processing plant and various other areas at the Project site. Freshwater is used for reagent preparation, cooling, and washbasins. The basket strainer is designed for particles larger than ½ inch; this system requires regular cleaning in the summer when algae grow thick.

Reclaim Water

The water recovered from the tailings pond (reclaim water) is pumped into the service water tank by one of the two reclaim water pumps located in the pond. The remaining reclaim water pump is either used as a spare or as an alternate for feeding the water-treatment plant for the treatment and discharge of surplus water from the TMF to the environment.

Service Water

The service water tank is used to store reclaim water that contains low levels of cyanide. It is fed by reclaim water from the tailings pond, and by freshwater when required. The service water tank overflows into the process water tank and serves as make-up process water. The service water is also pumped and distributed throughout the concentrator for gland water and Knelson concentrator fluidization and flush water.

Process Water

The process water is stored in the process water tank, located on the west side of the pre-leach thickener to allow any overflow from the thickener to gravitate into the process water tank. The process water tank is also fed by the service water tank overflow if additional water is required. Two process water pumps (one operational, one standby) distribute the water to various process areas. Process water is used in the grinding, gravity, and thickening circuits.

Domestic Water for Emergency Showers

Domestic (potable) water feeds the domestic water heaters. Two domestic water pumps (one operational, one standby) distribute domestic water to the emergency showers throughout the concentrator as well as the rest of the Project site.

Air Service

Mine air compressors supply compressed air at 125 pounds per square inch gage (psig) to the process plant as service air, and to an air dryer. The air dryer supplies dry air to a dry-air receiver, which stores and supplies dry air for instrumentation requirements. Two low-pressure air blowers are used for air distribution to pre-aeration, leach, and CIL circuit, with one blower in service and the other on standby.

17.1.13 Operating Criteria

Table 17-2 shows the mill, gravity, and CIL operating criteria developed from the 2018 bulk sample. Criteria are referenced from Rubicon's Bulk sample Processing Report, 28 February 2019.

Table 17-2: Mill, Gravity, and CIL Operating Criteria

Parameter	Value	Unit	Production Target	
Plant				
Feed Characteristics			*1250 // 1800*	
Gold Head Grade (Nominal)	8.10	g/t		
Ore Moisture	3.0	% w/w		
Ore Specific Gravity	2.90			
Operating Schedule				
Scheduled Operating Days	365	d/y		
Operating Hours	24	h/d		
Plant Availability	92	%		
Production Rate				
Plant Feed Rate (Nominal)	1,250	t/d	*52 // 75*	
Plant Feed Rate (Operation)	1,359	t/d	*56.6 // 81.5*	
Gold Recovery	92.5	%	94.1 // 95.6	
Grinding				
SAG Mill				
SAG Mill Fresh Feed	57	t/h	57 // 85	
Conveyor Capacity	167	t/h		
SAG Mill Power (Required)	1,474	kW	*500 // 670*	8.77 // 8.93
SAG Mill Solids Feed Rate	57	t/h	*57 // 85*	
SAG Mill Discharge Grate Opening at Reduced Throughput	12	*mm // in (polymet)*	*12 // 1/2*	

Parameter	Value	Unit	Production Target	
SAG Mill Feed Size (F_{80})	152	*mm // in*	*152 // 6*	
Max Feed Size	178	*mm // in*	*255 // 10*	
Product Size (T_{80})	874	*µm // in*	*874 // 0.0344*	
Operating Speed	73.0	% of critical speed	*70 // 73.6*	
Bearing Pressure		kPa	*15,500 // 15,800*	*@ 26 %v/v*
Mill Charge	26	% v/v		
Ball Charge	12	% v/v	*4.8 // 9.1*	
SAG Mill Ball Size	127	mm //in	*127 // 5*	
SAG Mill Ball Consumption	0.5	kg/t	*0.28*	
SAG Mill Discharge Pump A/B				
SAG Mill Discharge Pump A/B Discharge Solids %	63.0	% w/w		
Ball Mill				
Ball Mill Power (Required)	429	kW		
Ball Mill Feed Solids %	70	% w/w		
Product Size (P_{80})	105	µm	*74 // 0.0029*	
Bond Ball Mill Work Index for HiTi Basalt	11.5	kWh/t		
Ball Mill Ball Size	38	*mm // in*	*51 // 2*	
Ball Mill Ball Consumption	1.2	kg/t	*0.69*	
Cyclone				
Number of Cyclones (in Operation)	4	unit	3 // 4	
Number of Cyclones (for Gravity Circuit)[1]	1	unit	2 // 2	
Operating Pressure	62.0	kPag	62 // 9.4	
Cyclone Overflow				
Overflow D80	100	µm	74 // 0.0029	*D74 and D93*
Gravity				
Feed Characteristics				
Cyclone Underflow to Gravity Circuit[2]	25	% w/w	*25 // 35*	
Gravity Circuit Solids Feed Rate	42.5	t/h	*56.4 // 108*	
Gravity Screen				
Gravity Screen Type	Vibrating			
Gravity Screen Solids Feed Rate Capacity	84.9	t/h	*56.4 // 108*	
Gravity Concentrator				
Gravity Concentrator Type	Centrifugal			
Number of Concentrators (available)	2	unit	*2*	
Number of Concentrators (in operation)	1	unit	*2*	
Percentage of Feed to Gravity Concentrator A	100	% w/w	*50%*	
Percentage of Feed to Gravity Concentrator B	-	% w/w	*50%*	
Gravity Concentrator Solids Feed Rate (Total)	40.3	t/h	*56.4 // 108*	
Gravity Concentrator Feed Solution pH[3]	10.2 - 11.0		*8.5 to 10.5*	
Flush Frequency	60	min	*30*	

Parameter	Value	Unit	Production Target	
Number of Flushes per Hour	1.0	nb/h	*2*	
Time Required per Flush	1.0	min	*1*	
Concentrate Transfer Time from the Gravity Concentrate Pump to the Gravity Holding Tank[4]	2.0	min		
Mass Pull per Flush per Unit	25	kg	*25*	
Gravity Concentrate Mass Pull (Total)	25	kg/h	*100*	
Gravity Holding Tank				
Gravity Holding Tank Residence Time	3	d	*1.5*	
Gravity Holding Tank Capacity (Required)	1.80	t	*2.4*	
Gravity Table Magnet				
Magnetic Reject gold content		g/t	*300*	
Gravity Table				
Gravity Table Operating Time	3.0	h/d	*5*	
Gravity Table Gold Recovery	151	oz t/d	*200*	
Cyclone Overflow Primary Sampler				
Delay Between Cuts	8.5	min	*6*	
Thickener Underflow Solids %[5]	50	% w/w	*44*	
Flocculant Dosage	10	g/t	*30*	
CIL				
CIL Circuit				
Total CIL Circuit Residence Time - Leach and CIL (Installed)	73.5	h	*40*	
Leach Tanks				
Number of Leach Tanks[6]	3	unit	*1*	
Leach pH	10.5		*11*	
Leach Tank C Air Requirement	145	Nm^3/h	*230 - 300*	
Leach Tanks Cyanide Concentration (NaCN) *(LEACH TANK C)*[7]	250	ppm	*0*	
Sodium Cyanide Addition to Leach Tank A	0.25	kg/t	*0*	
Lime Addition to Leach Tank A	0.025	kg/t	*0.25*	
CIL				
CIL Feed Grade (Nominal)	4.1	g/t	*1 // 4*	
Gold Dissolution	85	%	*0*	
Number of CIL Tanks in Series	6	unit	*6*	
Residence Time (Total for All CIL Tanks)	19.5	h	*19.5 // 13.5*	
CIL pH	10.5		*11.5 to 11*	
CIL Cyanide Concentration (NaCN) ***(CIL Tank 1 (A) only)***	250	ppm	*250*	
CIL Oxygen Concentration	8.0	ppm	*8*	
CIL Air Requirement (for Each Tank)	118	Nm^3/h	*230*	
Sodium Cyanide Addition to the CIL Tank A ***TANK***	0.13	kg/t	*0.22*	
Sodium Cyanide Addition to the CIL Tank C Launder	0.13	kg/t	*0*	
Lime Addition to the CIL Tank C Launder	0.025	kg/t	*0.025 // 0.1*	

Parameter	Value	Unit	Production Target	
Carbon				
Gold Loading	1,376	g/t	*1800*	
Loaded Carbon Screen				
Water Flowrate to the Loaded Carbon Screen	9.0	m^3/h	*3*	
Metabisulphate		mg/L	*2.12*	
$CuSO_4$ Dosage	20	g/t (Cu/ore)	*46*	
Lime Dosage	0.8	g/gSO_2		
Lime Addition to the Cyanide Destruction Tank	0.75	kg/t		
O_2 Flow Capacity (Required)	29.3	Nm^3/h	*49*	
Carbon Regeneration and Attrition				
Carbon Attrition				
Carbon Daily Consumption[8]	82.2	kg/d	*209*	
Fine Carbon Recovery	63.8	kg/d	*209*	
Safety Screen Carbon Recovery	15	kg/d	*26*	
Total Carbon Loss (in Tailings and Carbon Fines Loss)	67.2	kg/d	*235*	
Stripping				
Stripping Time	12	h	*17*	
Stripping Temperature[9]	140	°C	*127*	
Stripping Solution Flowrate (Bed Volume)	2.1	BV/h	*1.48*	
Stripping Solution Flowrate	17.0	m^3/h	*14*	
Reagent Preparation and Consumption				
Flocculant (Pre-Leach Thickening)		*g/t*	*28.98*	
Lead Nitrate ($Pb(NO_3)_2$)				
Lead Nitrate Overall Consumption	0.25	kg/t	*0.32*	
Sodium Cyanide (NaCN)				
Sodium Cyanide Overall Consumption	0.50	kg/t	*0.42*	
Sodium Cyanide Proportion to Leach Tank A	49.0	%	*0*	
Sodium Cyanide Proportion to CIL Tank A Launder	25.0	%	*50*	
Sodium Cyanide Proportion to CIL Tank C Launder	25.0	%	*0*	
Quick Lime Overall Consumption	1.15	kg/t	*1.37*	
Caustic Soda Daily Consumption	60	kg/d	*150*	
Copper Sulphate ($CuSO_4$*$5H_2O$) Overall Consumption	0.079	kg/t	*0.46*	
Sodium Metabisulphite Overall Consumption	1.60	kg/t	*2.12*	

Notes: [1]Requires two regardless of feed rate; [2]Screen wash requires optimization; [3]Service water when clean preferred; [4]Pump system required; [5]Densitometer not calibrated; [6]Used as pre-aeration tank; [7]Pre-aeration only; [8]Start-up w/o pre-attrition of fresh carbon; [9]Heating elements, HX scaled analcite.

17.1.14 Observations on 2018 Mill Operation During Bulk Sample Processing

Mineralized Material Storage

The bulk samples from each stope were stockpiled separately, an outside contractor with a mobile crusher was retained to crush ore to nominally minus 6" and remove mine scrap metal with a cross-belt magnet. During the 2018 batch trial feeding the plant required the use of the reclaim feed system. To eliminate problems experienced in 2015 the feeder conveyor controls were overhauled, and the variable-speed control set to operate within a 40–60 t/h range. The mill weightometer was recalibrated against weighed truckloads of mineralized material.

Milling

During the 2018 Bulk Sample, the SAG–ball mill arrangement performed well. Lower-than-designed power consumption, ball charge, grind size, and pump capacity were experienced, implying the initial design was conservative. The correction of the SAG mill discharge sump level and cyclone head pressure reading, and the use of variable-speed control for mill start-up and grind-out eliminated surging on downstream equipment such as the trash screen. The starting SAG mill charge was 4.8% (v/v), 60% less than the design for the planned operating rate, resulting in only half the design estimate for grinding steel consumption. METSO's Poly-Met 12.7 mm (½") grate openings were more than adequate for the range of operating rates.

Ore hardness typically was less than design, with the SAG mill's optimum performance while operating at 72.5% of critical speed, and 15,000 kPa bearing pressure. Maximum throughput peaked at 90–95 t/h before being backed down to the average design rate of 75 t/h (1,800 t/d). No spills were observed downstream at the trash screen and/or gravity screen coarse overflow. Since the mill piping and conveyor were initially sized to accommodate 2,500 t/d, it was not surprising to observe that the mill, operated at 75 t/h (1,800 t/d), required just 75% of the designed ball charge and 46% of the connected mill power. The maximum feed rate was achieved without attempting to increase SAG ball charge as in 2015. Processing of 250 mm (10") run-of-mine mineralized material was possible in the new feed setup, but the mineralized material handling and tramp metal removal was problematic when mineralized material exceeded 200 mm (8"). Plans to install a modular semi-mobile surface crusher prior to the ore storage bin will keep mineralized material sizes below 200 mm and free of scrap metal.

Although the SAG mill has excess capacity at high throughput, the ball mill does not. Based on the performance with ½" Poly-Met grates, operators are even considering reducing SAG mill grate openings to 3/8" to balance mill power distribution between mills.

The trash screen performed well after surges in the cyclone overflow were eliminated. The trash-screen wash sprays could operate better if the low-volume high-pressure water wash sprays were

installed. A temporary trash dewatering screen used for the bulk sample will need a permanent solution, and a similar spray wash is required to deslime trash and keep floors free of spill.

Head sampling by sampling SAG mill discharge proved difficult due to the size reduction step performed by running primary sampler underflow through a ball mill sample crusher prior to secondary sampling. During both 2015 and 2018, using SAG mill discharge sampling for head grade always resulted in under-reporting head grade due to accumulation of gravity-recoverable gold in the ball mill and the time required to grind coarse material. A quick improvement would be to install a larger secondary sampler and bypass the sample crusher. However, to ensure all mill head grade information is accurate a conveyor transfer discharge sampler and sample handling system is recommended.

Gravity Circuit

The designed gravity recovery was 46% of gold fed to the mill (50% of recoverable gold). The sizing of the gravity circuit ranged from one Knelson concentrator operating for 1,250 t/d to two operating for 1,800 to 2,500 t/d at 25 kg mass pull per flush cycle per unit (600 kg/d/unit on 60-minute [min] batch cycles). Initially, the gravity circuit was operated with one cyclone and one concentrator, which frequently overflowed requiring both concentrators to operate with one cyclone underflow producing 1,200 kg of concentrate per day. With persistence and regular cleaning of the concentrators and feed distributor, concentrators could operate with two cyclones feeding two concentrators. In the 2018 "Operating Criteria" Gold was observed to accumulate in cyclone underflow subsequently causing gold lockup in the ball mill. This was remedied by increasing mass pull from each concentrator from 25 kg per hour to 50 by reducing the time between wash cycles from 60 to 30 min. The improvement was successfully trialed, resulting in exceeding design, reaching 49.4% recovery for one cyclone feed and up to 57.3% poured gravity recovery for two cyclones feeding, which corresponded to a 16% decrease in cyanide consumption compared to design. Operating under the two-cyclone condition requires further modification of equipment sizing and screen performance.

Gemini table tailings are pumped back to the mills for further grinding to liberate finer gold particles. Magnetic concentrate, typically 2,800 g/t, was not returned to grinding to reduce metal scrap recirculating in the ball mill circuit.

The gravity-tails pump box was difficult to operate since the coarse solids from scalping screen overflow was not readily mixed with the Knelson tailings slurry, and would stack up, plugging the screen overflow chute or slough, and plugging the pump suction. With increased volume of cyclone underflow running through Knelson concentrators, it is important to mix slurry from Knelson with screen overflow and returning gravity tails to the ball mill feed through the cyclone underflow tub. Malleable-copper blasting wire from the underground operation formed copper flakes during the milling process, which were recovered in the gravity circuit and, due to the gravity table design, captured in the final concentrate. Assays report that the table concentrate has, on occasions,

contained up to 60% copper, which is an opportunity the technical team will refine by adjusting the separation parameters.

During the 2018 test mining, the pair of Knelson concentrators operated on 30-minute cycles (50 kg/h per unit), with optimum results obtained when a single cyclone fed each concentrator. The technical team observed that low-pressure water sprays caused excess dilution of the concentrator feed, so high-pressure low-volume sprays are planned as a remedy.

While not critical, transfer issues were encountered when pumping the Knelson concentrate to the refinery. The design pumps had insufficient head, and the backup slurry pumps from 2015 were unlined and their casings failed. Air diaphragm pumps used for underground dewatering were tried, but were susceptible to grit and changes in air pressure. Mechanical diaphragm pumps or an open impeller sludge pump (Eddy pump) are amongst several alternative transfer systems that are being considered and should be installed and tested before moving to full operation.

Initial tabling of Knelson concentrate resulted in low grades due to high levels of metal waste from blasting wire, large grit and ball chips entering the table concentrate. To improve quality, the table feed was screened at 20 mesh (841 microns [µm]) increasing the table concentrate gold content from 48% to more than 65%, which is the minimum grade that should be fed to the induction furnace.

The 2018 operation of the gravity circuit was successful in reducing in-process gold inventory in the ball mill. It also reduced metallic gold reporting to CIL, which is always a concern. The direct benefit of increased gravity recovery is not overall recovery, but:

- An overall reduction of in-process inventories (such as gold that accumulates in the mill and gold on carbon), which improves cashflow
- Lower CIL head grade which reduces cyanide cost
- Increased accuracy of metallurgical accounting since inventory in gold traps cannot be predicted accurately.

Based on this experience, Rubicon is considering expanding the gravity circuit to increase mass pull to over 3,000 kg/d, developing piping of slurry to mix with gravity-screen overflow, and pumping gravity tails to the cyclone underflow tub before restarting the mill.

Thickening and Pre-aeration

A 14 m high-rate thickener dewaters the cyclone overflow slurry from 35% to 50% solids ahead of cyanidation. Additionally, the pre-aeration tank was operated in 2018 with cyanide only added to the CIL tank A and again between CIL tanks C and D.

Designers were cognizant that much of the water in Northern Canada drains from peatbogs and is acidic. This raw water will react with cyanide to produce hydrocyanic acid which is even more toxic. It also increases the corrosion when left stagnant in equipment/pipes and wear of iron balls that are used for grinding media. Lime addition to the thickener feed box and to the pre-aeration tank neutralizes this acid and maintains the slurry between 11.5 and 12 pH.

The surplus water is recycled to the "process water tank" for reuse in the grinding circuit. The rate of leaching depends on both the cyanide content and dissolved oxygen levels. Abrasion by grinding allows oxidation of the exposed mineral surfaces with a reduction of the dissolved oxygen content of the slurry. The pre-aeration tank is thus crucial in restoring the dissolved oxygen level to 8 milligrams per litre (mg/L) ahead of leaching. The 499 m^3 pre-aeration tank provided four hours of residence time (at 1,250 t/h) ahead of the CIL circuit for the destruction of flocculent to prevent fouling of the carbon. Lead nitrate additions were originally included in the circuit, and improved aeration further.

There is an equilibrium between gold in solution and gold deposited on carbon. A low gold loading ensures maximum removal of gold from solution so that the loss of gold to tailings is minimized, however, it can necessitate moving and regenerating large quantities of carbon; thus, it is necessary to maintain a high gold loading on the carbon to maintain a cost-effective process.

This equilibrium relationship between gold on carbon and gold in solution is demonstrated in Figure 17-3.

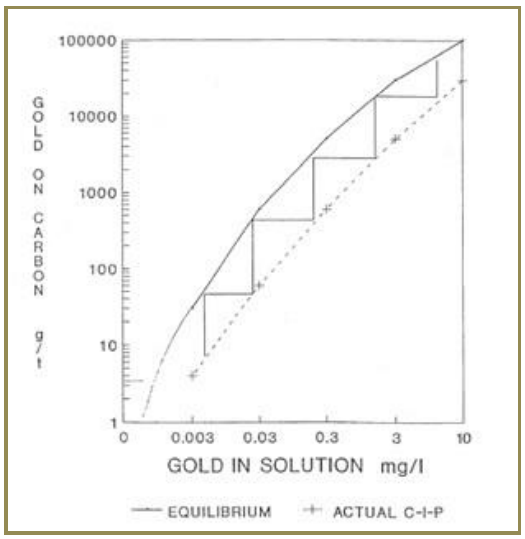


Figure 17-3: Equilibrium Relationship between Gold on Carbon and Gold in Solution (Rogans, 2012)

Another example of the variance in gold and silver extraction between CIP and CIL operations is provided in Figure 17-4 which reflects the results observed at Balabag in 2013 (TVI Pacific Inc., 2013). In that study it was observed that CIL provides a faster time to mineral dissolution over CIP, and subsequently less capital cost for additional tank capacity.

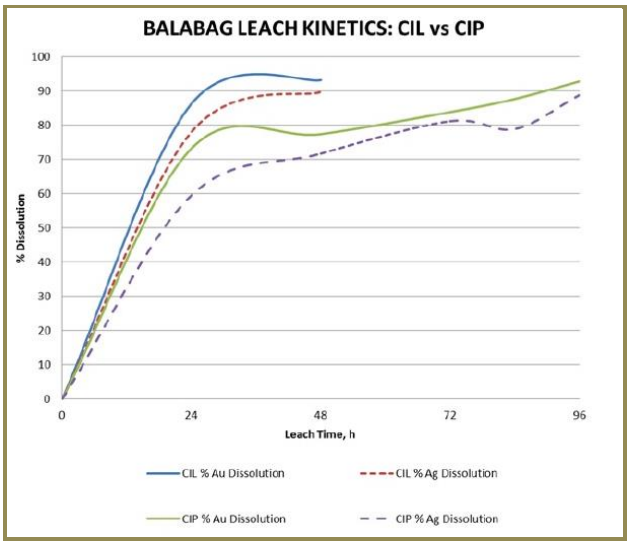


Figure 17-4: Balabag Leach Kinetics: CIL vs. CIP (TVIA Pacific Inc., 2013)

The Balabag example might not be a good comparison for the Canadian situation, due to dissimilar climate. Plants built in, or near, the tropics are typically roofless and unenclosed, and aerated cyanidation processes pose far less risk of noxious gas exposure. By contrast, in Canada, cyanide leaching facilities are located outside the main building, while gold is recovered in CIP tanks within the mill building. It is recognized that CIL has a higher toxic risk, which increases the process operating cost by about C$20,000/a, or C$0.05/t. Since the outside leaching was stricken from the 2015 construction the 2018 Rubicon CIL tests included in-mill, stage-wise additions of cyanide, which resulted in high residual free cyanide concentration in the final CIL tanks (which feed the detoxification process). A capital allowance has been allotted in this PEA to accommodate outside tank erection, tanks already purchased, cyanide monitoring and control, and cyanide destruction.

In general, processors maintain pulp densities close to the density of wet carbon (1.3) to maintain the carbon in suspension and ensure homogeneous mixing. This generally requires a slurry density of around 40%–45% solids leaving the pre-leach thickener. However, in CIL operations aeration bubbles can cause the carbon to float, which impairs mixing. At Rubicon's 2018 tests, CIL slurry densities occasionally exceeded 55%, and reagent floating was observed. Although the pulp density in the thickener underflow and pre-aeration tank at Rubicon were designed for 49% solids, this density is reduced when carbon is periodically removed and washed on the loaded carbon screen. Many of these issues should be resolved with the erection of the dedicated external pre-aeration and leach tanks.

Also, during the 2015 commissioning, rocks over 178 mm (7") and poor control of cyclone pressure caused the trash screen to be overloaded and coarse rock fragments to enter the circuit and become inseparably mixed with the carbon. This issue was resolved after approximately 200 t of gravel were removed from CIL Tank B using a vacuum pumper truck. With the improvements to mill instrumentation and control of SAG mill feed made during the 2018 tests, this gravel issue should not reoccur.

The finer the carbon, the faster the gold loading; however, to ensure separation, the carbon needs to be coarser than the largest particles in the slurry. Kemix-style basket screens (700 µm) permit the slurry to pass to the next CIL tank, but prevent passage of the coarse "6—12" carbon (1.7–3.38 mm). Allowing for agitation and aeration, each tank provides about six hours of leach residence time, but as one tank is often offline due to screen cleaning and repairs, the total leach time can be less than the design. Based on the 2018 results this occurrence will not be an issue as the Kemix stainless steel baskets were replaced with Derrick screen conversion kits and replaceable panels were installed. The modifications to the gravity circuit will maintain the CIL head grade at a minimum, further reducing residence time required for leaching in CIL.

The Rubicon circuit design specified fresh and regenerated carbon to be added to the final CIL tank with 10.2 tonnes of carbon in each CIL tank, at a distribution of 20 g/L.

During the 2018 trial, total gold (solids and solution combined) in the final CIL tails averaged 0.23 g/t, and ranged from 0.06–0.33 g/t. Gold recoveries ranged from 96.7% for stope 161 "Up Holes," to 92.9% for the low-grade high tonnage, 1,800 t/d, test. Gold loading on carbon peaked at 1,218 g/t lower than the 1,600 g/t obtained in 2015. However, Rubicon stripped daily by improving elution time to minimize in-process gold inventory. Regardless, operations expect that, through carbon management, loaded carbon grades of 1,376 to 2,751 g/t in the design criteria for operating rates of 1,250 to 2,500 t/d can be easily achieved.

Elution and Carbon Reactivation

Prior to stripping gold, the loaded carbon is washed to remove residual cyanide and fine slurry, then rewashed in 3–5% hydrochloric acid to remove any carbonate, which will block the pores in the carbon granules. The industry is divided over whether hydrochloric acid should be used before eluting the gold, or nitric acid after elution, when the cyanide content has been reduced and there is a lower risk of creating toxic hydrocyanic acid.

Hydrochloric acid also incurs a risk of chloride cracking of any stainless steel used in the circuit, such as the regeneration kiln liner, and electrowinning anodes, leading to higher-cost materials having to be used, such as operating electrowinning cells at a pH 12.5 instead of 10 to avoid corrosion.

During the "Pressure Zadra" elution stage the loaded gold solution is heated to 140°C at a pressure of 400 kPa. Caustic soda and cyanide are used over a 12-hour period to remove the gold and other metals from the carbon. This "loaded solution," after passing through a heat-recovery heat exchanger, is then pumped to the electrowinning cells located in the refinery. In the 2018 batch trial, the elution circuit was extended up to 36 hours due to undersized barren solution return-pumps feeding the recuperation heat exchanger. The barren solution pumps did not conform to the FLSmidth & Co. (FLSmidth) manual and did not overcome the heat exchanger pressure drop. It required a doubling of the exchange plates to lower the pressure drop to achieve 15 m^3/h flow rate sufficient to

elute four-tonne batches of carbon in 12- to 14-hour cycles. This increased the average strip efficiency from 82.5% to 90%, although further improvement may be possible. In 2018 tests the heat transfer plates were repaired, which had been badly corroded during the 2015–2018 stoppage; in addition, extensive instrumentation damage was repaired.

After the gold has been eluted, the stripped carbon is regenerated in a kiln at 650°C, and fine carbon removed by screening. During the 2018 Batch trial, it was realized that more fine carbon was being created than the fine-carbon storage tank could handle. Careful to avoid any unnecessary losses, all fine carbon was captured, as opposed to 2015 when fine carbon was not captured. It is unknown if these excess fines were created from weak carbon or due to transfer pumping (6 transfers per batch); the latter is the most probable cause. Dealing with the quantity of carbon fines far exceeded the capacity of the installed 4.75 m^3 (1,260 gal) system and the 1.5 h/ week assigned to this task. Further analysis is being undertaken by Rubicon's operators prior to restart of production.

After quenched carbon is returned to the carbon feed tank, any shortfall is made up by adding fresh carbon. Washing fresh carbon to remove fines requires vigorous agitation at 10% to 20% solids for one to two hours, but the Soutex design is only for a 5.7 m^3, tank which is barely adequate for 500 kg lots. Operators should consider a volume increase before stepping the plant up from 1,250 to 1,850 t/d.

Electrowinning and Refinery

Pregnant gold-bearing solution from the elution column is pumped into a pair of electrowinning cells operated in parallel during the elution cycle, and the temperature is maintained above 90°C, as this promotes diffusion of the gold and cyanide ions and improves gold deposition. The pH is kept above 10 to prevent corrosion of the stainless-steel anodes, although a pH above 12.5 must be maintained when chlorides exist in the solution. Increasing caustic levels (1.3%–1.5%) can be beneficial and should never be allowed to fall below 0.75% or rapid corrosion can occur.

The rate of gold deposition depends on the cathode area, the rate at which gold is delivered to the cathode and uniformity required to prevent solution bypassing. These criteria are well addressed in the FLSmidth electrowinning cell design installed in the Phoenix Mill. Gold is therefore plated onto stainless stainless-steel cathodes in two parallel electrowinning cells.

The soft gold deposition enables it to be removed by pressure washing, and the dried cathode sludge is then smelted in the induction furnace.

After the gold and other metals have been extracted the barren electrolyte returns to the elution column. A sampler routinely collects drip samples from the pregnant solution, which indicates how far the elution has progressed. A barren solution sampler also monitors the performance of the electrowinning cell.

Cyanide Destruction

Once the gold solution has been adsorbed onto the carbon the residual slurry exits the final CIL tank through a 500-µm safety screen installed to recover any entrapped carbon particles. Normally, only 5% of the attrited carbon is caught on this screen, and 41% of fine carbon often passes through it (Marsden & House, 2009). Carbon exiting CIL occurs when carbon gets smaller and/or stainless-steel screen openings get larger which occurs during transfer pumping, abrasion or by operator cleanup. In 2015 carbon losses to tails were high, since the safety screen was bypassed and fine carbon from carbon handling was sent to CIL from floor sumps and regenerated carbon return. In 2018 operations commissioned and operated the safety screen which effectively contained all fine carbon sources from outside the CIL process. Included in the PEA capital cost is equipment to alter flow patterns to minimize abrasion damage in tanks while maintaining solid suspension; for improving safety screen performance; and maintaining interstage screen apertures.

The Gekko TAC-1000 detoxification system installed at Rubicon was found incapable of maintaining the required 1 ppm cyanide in tailings when the total inlet cyanide exceeded 150 ppm, although it had been designed to achieve this with an inlet feed of 250 ppm total cyanide. The control formula algorithm underestimated the SMBS and SO_2 addition rates and needs repair or replacement.

With two cyanide addition points in the CIL circuit, the residual cyanide can be expected to be high. While maintaining a high cyanide level throughout the CIL circuit achieves the required recovery, it does incur an additional cost to destroy unused cyanide. During 2018, campaign operations demonstrated that high recovery could be achieved by controlling free cyanide so CIL tank F total cyanide was 150 ppm or less, therefore justifying replacing the Gekko TAC-1000 with a unit that can be used to monitor and control free cyanide as well as controlling the Gekko cyanide destruction arrangement. When the paste plant is not operating, the treated waste slurry flows by gravity to the tailings pump box and is pumped to the tailings management area. In 2018, tailings were spread evenly across the dam face with spigots to push water away from the dam and to reduce turbidity at the reclaim water pump suction intake. This feature should be incorporated into future tailings operations.

Tailings pumping is subject to repeated surging since the installed equipment is designed to handle higher discharge head. Future operations may require adding some flow restriction to smooth pump operation and improve accuracy of tailings flow measurement.

Tailings and Paste Plant

The Phoenix Mill discharge at 80% passing 75 µm at 1,250 t/d is better than the design 80% passing 105 µm; while this may increase gold extraction, it will incur problems for paste production, as less than 15% should be finer than 20 µm. The current design does not include any secondary sizing units after the mill cyclones, and a more thorough size grading analysis should be obtained.

Utilities and Reagents

Fresh water is supplied from a freshwater reservoir tank located south of the hoist room and draws water from Red Lake. Fresh water is used for reagent make-up, cooling, gland water, and wash basins, and can be used to supplement fire water.

Reclaim water is pumped from the tailings dam decant pond for service water, which also includes water from the mill clarifier, or (when cyanide level is low) can be treated for discharge through the water-treatment plant.

Process water is collected from the thickener overflow, and can be supplemented with reclaim and service water. During the 2018 pre-operation and inspection, 20% of the process water tank was filled with compacted silt. The silt was removed, and wells installed to keep solids back from the process water pump suction intakes. Pumping this silt through process water damaged equipment, valves and piping of which operations replaced only control valves and pipes that had failed. Provisions to repair this system and install a method to capture solids outside of the process water tank are included in the PEA.

Average reagent use during the batch trial was above the design targets, although cyanide usage was about 16% below the 0.50 g/t operating criteria estimate.

Lime losses were minimized after the original pumps were replaced by peristaltic pumps and hose delivery lines. This also eliminated choking and indicated a 68% reduction below 2015 cost.

Actual reagent and grinding steel consumption compared to the design criteria are in Reagent and Grinding Steel Consumption Comparison.

SMBS and SO_2 consumption compared to design was excessive in 2015 due to incorrect calibration of the DCS outputs that overdosed cyanide. This was corrected in 2018; however, SO_2 from SMBS was only optimized near the end of operation. Corrections to the Gekko algorithm and replacement of the TAC-1000 will allow operations to meet or exceed the designed consumption.

Cyanide addition progress improved from 2015 to 2018. Further improvements from cyanide monitoring brought cyanide consumption to as low as 0.22 kg/t and averaged 0.42 kg/t for the last three stope runs. Cyanide measurement and control will respond immediately to maintaining minimum free cyanide, and further improve cyanide consumption.

Flocculant consumption was high during actual operations, and due to surges in 2015 and operating on manual control while running low tonnage rates in 2018, caused regular fluctuation in underflow density. During the high tonnage test, the thickener could be operated automatically with no issues, indicating the design addition rate may be possible at higher tonnages.

Table 17-3: Reagent and Grinding Steel Consumption Comparison

Milling Consumables	2018 Actual (kg/t avg.)	2018 Optimized (kg/t)	2015 Actual (kg/t avg.)	Design Criteria
Sodium Metabisulphite	2.76	2.12	3.59	2.32
Sulphur Dioxide	-	-	1.33	0.71
Cyanide	0.57	0.42	1.32	0.50
Magnafloc 351	0.03	0.03	0.04	0.01
Lead Nitrate	0.32	0.32	-	0.25
Caustic Pearl	0.04	0.04	0.04	0.03
Copper Sulphate	0.46	0.46	0.54	0.09
Lime (bags)	0.09	0.09	0.08	0.05
Lime (bulk)	2.08	1.37	4.52	1.18
Caustic (cube 50%	0.24	0.24	0.69	0.06
Grinding Balls (2")	0.69	0.69	2.06	1.20
Grinding Balls (5")	0.28	0.28	2.14	0.5
Granulated Carbon 6x12				
Antiscalant				
Fine Carbon	0.21	0.14	1.90	0.07

Copper sulphate addition far exceeded design and is suspected to be associated with the Gekko system algorithm. No work was done to investigate copper sulphate consumption during past operating campaigns.

Lime consumption improved after installing peristaltic pumps. The optimum consumption remains higher than design, which is suspected to be the result of maintaining pH in CIL tanks during the two weeks the mill was shut down between stope runs, due to moisture in the aeration air.

Excess caustic consumption was due to problems experienced in the elution circuit.

Grinding steel consumption was significantly less than design, and would have been lower if it were not that balls were added to the SAG mill for mine site power-demand verification.

Carbon consumption measured by fine carbon is excellent considering 60 tonnes of fresh carbon was added without pre-attrition which typically removes 3% to 5% by weight. Excluding these fines, and the fact operations was stripping carbon to minimize in-process carbon gold inventory, future carbon consumption close to the design is expected. No descalant has been used in operations. The system installed in 2015 was never commissioned before the mill shut-down, and the equipment was returned to the supplier.

Descalant and 50% caustic soda are received in 1 m³ chemical cubes, whereas lead nitrate, copper sulphate, flocculant, and SMBS are received in tote bags. Problems occur when the reagent-bag cutters permit fragments of the bag to enter the mix tanks which impedes the level control system and chokes pumps. During the 2018 test, screens were added to prevent bag fragments from entering the mix tanks.

17.2 Plant Refurbishing and Operating Costs

Several months of dismantling and repairs have already been completed; however, several issues need to be addressed.

Past problems with oversized mineralized material will be mitigated by installing a modular surface crusher. Additional crushing has been considered and included in this review.

Attention to the SAG Mill feedback control is required. The SAG Mill discharge pumps suffered the most during the 2015 operations and subsequent outage. Pump liner life is 7,000 tonnes between liner replacement and 18,000 tonnes before casings fail. Cast white metal casings and/or liners are required to extend time between rebuilds. Overall, pipes are aligned vertically or horizontally with rubber-lined piping and sweep 90° elbows on all slurry lines. Given that mill piping is from original installation, it has been estimated that 20% to 25% of this should be replaced. This includes acquiring spares for the ceramic elbows that METSO recommends for replacing the current steel cells on the SAG Mill discharge.

Ball chats scats and steel fragments discharged from the SAG mill severely damaged the rubber liners of the ball mill, so removing any ball chips remaining in the SAG mill prior to restarting is recommended. In any case, the mill operations have concluded the ball mill liners removed in 2018 can be reused, and a new set is in inventory, stored in the mill.

As a rule of thumb, to minimize abrasion or bedding while keeping the slurry homogeneous, slurry velocity in a pipe is maintained between 1.2 and 1.8 m/sec (4 ft/sec and 6 ft/sec). At Rubicon, gravity discharge duplicate pipes have been installed so that the smaller pipe takes the flow when operating at 1,250 t/d, and the larger pipe at 2,500 t/d. However, for the proposed 1,250–1,800 t/d operating range, standard 100 mm (4") pipe should be adequate in the majority of locations. In the case of the gravity discharge pumps, there are two pipelines— 150 mm (6") and 75 mm (3"), both of which are non-standard sizes and thus incur higher pipe replacement costs. The discharge was designed to feed the 75 mm line until the flow velocity forces use of the 150 mm line. Operations plan to re-pipe gravity tails to the cyclone underflow tub and eliminate the dual pipe configuration.

There are several discrete occurrences observed where sanding up of pipes can occur: where a primary and backup pump feed a common line; where isolation valves are located too far from the junction; where leakage of the operating pump into the dead leg of another pump occur. Operators are aware of these issues and will mitigate as required.

Piping costs in the SRK (2013b) report were estimated at $4.3 million, equivalent to about $4.8 million in 2019 (with inflation of 2.5% pa). pump and pipe refurbishing will be required prior to start-up and includes changes and extensions to piping in the leaching section

17.2.1 Mill Operations and Mineralized Material Handling

Essential components of the original design for mineralized material handling that were not installed include the dust control system, a levelling gate for the reclaim feeder, and a calibration chain for the weightometer.

Repairs to the ball mill lines and SAG mill grate alterations in the modification criteria include replacement of the unreliable primary sample crusher and secondary sampler with a new unit from Heath & Sherwood. The sampler upgrade eliminates the sample crusher (ball mill) that does have some gold lockup, however, if head samples continue to be unreliable a transfer-point discharge sampler and processing station should be considered.

The trash screen is sized for 2,500 t/d; high levels of coarse grit in the CIL circuit could have occurred only due to failure of the SAG Mill screen under extreme overload conditions, such as frequently plugged cyclone underflow. In the 2018 operation, trash screen overloading was not observed after the cyclone head pressure sensor was relocated.

The SAG mill bolt removal equipment proved ineffective and must be replaced with a larger unit.

17.2.2 Gravity Circuit

There has been a proposal to expand the gravity circuit. While gravity is less hazardous and costly than cyanidation (roughly $2.87/t), enlarging the two existing Knelson units, or adding a third may not increase overall gold recovery; however, increasing gravity recovery directly reduces cyanide consumption in CIL as demonstrated in the 2018 operation. The above modifications and replacement of the undersized and warped Gemini table in the refinery should be done prior to the next mill start up. The cost savings from increasing mass pull must be adjusted to include adding at least one part-time senior operator in the refinery, of which operations has addressed in operating costs.

An InLine Leach Reactor (ILR) sized for the high-mass pull tested in 2018 should be acquired prior to start up. Provisions are included in layout of the grinding and electrowinning circuits to install process equipment; however, costs for electrowinning, installation, piping, electrical, and controls must be added. Although intense agitated leaching gold from gravity concentrate increases cyanide consumption and costs, the other benefits of gravity recovery remain (i.e., reduced gold lockup in grinding, and CIL carbon), and the risk of gold theft is reduced.

Thickening and Pre-aeration

Installation of the pre-aeration tank (not constructed) will be required. This addition includes foundations, containment, equipment, and enclosures equivalent to original construction. The new

tank should also include the CANAMIX agitator upgrade that operates at a 30% speed reduction to reduce wear and power cost.

17.2.3 Leaching

The three original leach tanks need to be installed. Two of these would be new construction, and the third, previously used as a temporary pre-aeration tank, can be refurbished. These tanks should include the CANAMIX agitators and the use of the existing aerators and agitator gearboxes purchased under the original construction (that were not installed).

Modifications need to be done to leach tanks A through F and include increased ventilation, installing the CANAMIX agitation system, modifying Kemix screen wet ends with the Derrick interstage screen conversion, and replacing the loaded carbon screen with a Derrick screen to improve cleaning efficiency.

17.2.4 Elution and Carbon Regeneration

Operators stated that the service water freshwater source was limited due to the existing basket strainers. This may account for the severe scaling in the heat exchanger. Water quality in elution and for carbon wash screens is critical. Scaling problems in the heating system make it difficult to achieve required strip temperatures. A higher pump pressure provided some relief, but a more thorough analysis is needed. Descalant was not used, but must be installed for future operations. Also recommended are installing fittings and a small pumping system so heat exchanges and solution heaters can be acid washed without dismantling the equipment. Failure to thoroughly strip carbon has led to installing temporary piping to return high-gold-content carbon to the strip column, avoiding returning to the CIL circuit and the risk of potential gold losses.

The elution process requires several cycles (10 to 12-bed volumes) with barren solution returned to the elution column over a 12 h period. During the bulk sample trial, the strip cycle took 36 h instead of the designated 12 h because the initial maximum barren solution return was only 5.5 m^3/h instead of the designated 17 m^3/h. Poor carbon stripping and slow elution also cause the gold inventory on site to be increased.

CIL operations have a reputation for generating a high volume of carbon fines. The carbon handling circuit installed at Rubicon is an FLSmidth P-5006 package unit; however, most of the tanks appear undersized, making it necessary to install additional tank volume for fine carbon handling. A vertical settling tank sized to recover fine carbon from all flows is recommended.

Given that four years have passed since the plant was installed, some suppliers may be reluctant to undertake repairs under warranty. The future operation-needs estimate is based on zero manufacturer warranty and includes funding for carbon regeneration for acid wash, and for elution flows and heat transfer. Note: carbon handling installed is sized for 2,500 t/d and more. The carbon is regenerated in 4-tonne batches after having been screened and fines extracted, then bagged and shipped off site for gold recovery.

17.2.5 Gold Recovery

A main requirement in the elution and electrowinning circuit is to operate at the design parameters, to elute 4 tonnes of carbon in a 12 h period, and to recover this in an electrowinning cell. There is anecdotal evidence of equipment being mislaid and until this is resolved the refurbishing cost is variable. The Gemini cell is undersized and warped. However, if the Knelson concentrate were to be dissolved in a Gekko-type intense leach tank it would provide increased gold security and could eliminate one operator; however, it does increase consumable costs for cyanide.

Ventilation is a problem, as is limited communication. The pregnant solution sampler should be relocated outside the refinery to provide the elution operator with better control of that operation.

The electrowinning system is designed to operate in parallel with each 125 ft^3 electrowinning cell designed to recover up to 250,000 gold equivalent ounces per year.

17.2.6 Cyanide Destruction

During CIL operations the cyanide usually needs to remain active until exiting the last tank, which requires a high free-cyanide content. The cyanide destruction system requires the capability to destroy total cyanide (free plus weak-acid dissociable [WAD]) up to 250 ppm, with an expected level of 234 ppm total cyanide (150 ppm WAD and 84 ppm free cyanide). Actual operation demonstrated compliance performance (1 ppm total cyanide) could be maintained at 150 ppm total cyanide, which appeared to be the upper limit for the TAC–1000. In addition, control of free cyanide at CIL tank F discharge lowered overall cyanide consumption, showing cyanide monitoring and control—as opposed to cyanide measurement—will significantly lower operating cost. Modifications for cyanide destruction includes improvements to safety screen operation, installing the CANAMIX upgrade, and installing cyanide measurement and control.

17.2.7 Reagents

In 2018, significant work was completed to operate the reagent area; however, some work remains for permanent ½" screens on mixing hoppers, pump strainers, descalant, fresh water control, and floor sump piping.

17.2.8 Services

Services needs are significant considering that there is no support equipment for mill maintenance, mobile equipment, or assay and metallurgical lab facilities. In addition, equipment does not exist for operators to generate quick turnaround of solution gold assays or to mix reagents on site. All these items will be required prior to start up.

18 PROJECT INFRASTRUCTURE

18.1 Site Electrical

The Phoenix site currently has a fully operational electrical power distribution system capable of distributing enough power to both the mine and mill complex. Plans are in place to expand the system to support a long-term production rate of 1,800 t/d.

To meet long-term production requirements a formal agreement with Hydro One for the supply of approximately 11.5 MW of electrical power must be negotiated. Discussions are currently underway.

18.2 Power Supply

Mine and mill electrical power is supplied by Hydro One from its North of Dryden system as shown in Figure 18-1. The Red Lake subsystem, which is part of the North of Dryden system, currently has a load meeting capability (LMC) of 61 MW, of which the Phoenix site has been allocated 5.3 MW until July 2021 ((Hydro One letter of notification, November 7, 2018)).

A 44 kV, 9 km long overhead power line extends from the junction of Nungesser Road, along the mine access road to the site's main substation. The line has a carrying capacity of 455 amperes (A) (40°C, with sun, and wind at 0.61 m/sec [2 ft/sec]) that corresponds to a full load-rating of approximately 33.9 MW, which is significantly higher than long-term requirements.

Hydro One has acknowledged in their North of Dryden Integrated Regional Resource Plan, January 27, 2015 (Hydro One, 2015) which covers the area illustrated in Figure 18-1 that "the electricity transmission system serving the area is at capacity and is unable to accommodate demand growth."

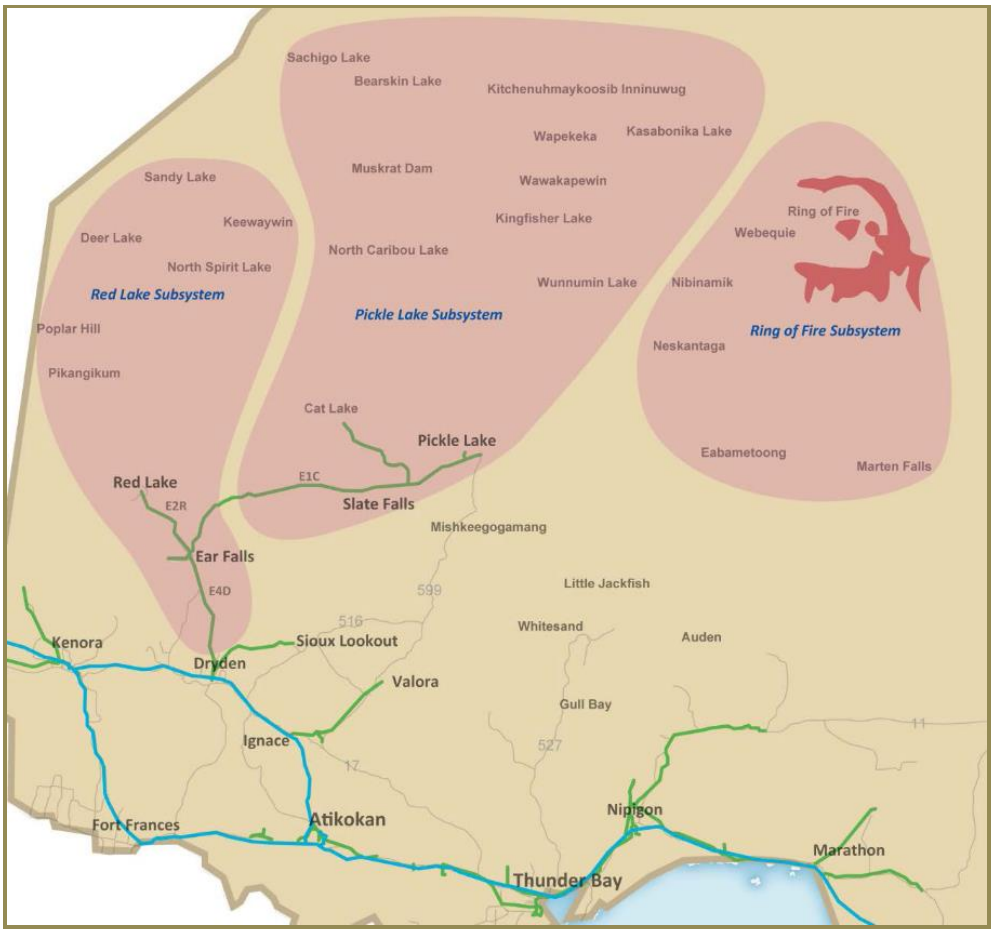


Figure 18-1: North of Dryden Planning Zones (Hydro One, 2015)

Since 2015, Wataynikaneyap Power LP (WPLP) has undertaken plans to construct a 230-kV transmission line from the Dryden area to Pickle Lake. This tie will off-load Pickle Lake, resulting in increased capacity of the Red Lake Subsystem without the need to upgrade existing transmission lines. The planned in-service date is expected to be Q4 2020.

Without a definitive contract in place Hydro One's policy is to assign capacity based on a customer's highest rolling three-month average peak electrical demand over a period of the last three years. If no request for capacity increase is submitted and approved by Hydro One, Rubicon's allocated capacity will be significantly reduced.

As a result, Rubicon has taken proactive steps to mitigate this risk. Discussions have been initiated with Hydro One to secure an additional 5.7–6.7 MW of demand, well in advance of production ramp up. A formal request for a capacity increase is in the process of being prepared and all indications are, that additional capacity will be made available to the Phoenix site when required.

18.2.1 Expected Demand

Mine and mill electrical demand has been estimated to peak at 9.8 MW when operating at a production rate of 1,200 t/d (Bestech, 2014) (Figure 18-2).

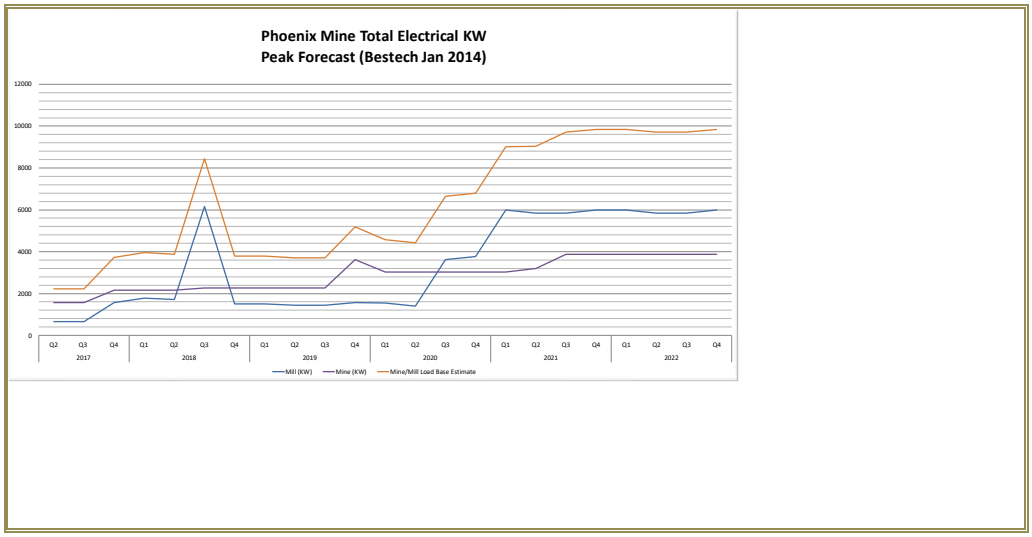


Figure 18-2: Peak Electrical Load Forecast

However, this demand does not include additional loads identified in this PEA. These loads are listed and estimated in Table 18-1.

Table 18-1: Additional Load Demand

Load Description	Estimated Demand (MW)
Dewatering System Upgrades	0.1
Paste Fill System	0.6
Surface MMD Sizer	0.4
Misc. Shops, Office, Dry, Warehouse, Camp	0.6
Total Additional Demand	**1.7**

Inclusion of these loads given in Table 18-1, raises the expected peak power demand of the site to 11.5 MW. This is more than double the amount currently allocated to the project by Hydro One.

18.3 Main Substation

The incoming power line, which connects to Hydro One's M6 feeder at the Red Lake Transformer Station terminates within the Phoenix Project's main substation on a dead-end structure containing the necessary isolating devices (Figure 18-3).

Figure 18-3: Main Substation Incoming Power Line

The substation contains two transformers rated 46 kV- 4.16 kV, 18/24 MVA in a double-ended configuration, allowing for ease of maintenance while also providing full redundancy (Figure 18-4).

Figure 18-4: Main Substation Transformers

The main transformers feed power to a 5-kV switchgear lineup located inside a modular prefabricated building. This lineup is also connected to 10 reactive megavolt amperes (MVAr) of compensation to provide power factor correction.

Power to the mill, underground and auxiliary areas is effectively distributed via a network of cables and overhead power lines.

18.4 Site Power Distribution

The underground electrical distribution system consists of:

- Single, 3-conductor 4/0 American wire gauge (AWG), 5 kV Teck 90 cable installed in the shaft from the surface winch room to the 305 m Level (4160 V)
- Single, 3-conductor, 350 thousand circular mils (MCM), 5 kV Teck 90 cable installed in the shaft, which goes from the surface winch room down to the 610 m Level (4160 V)
- Single, 3-conductor, 4/0 AWG 1 kV, Teck 90 cable from the surface to the 122 m Level (600 V).

The underground power distribution system is planned to be upgraded prior to the mine going into full production. Two options exist and are covered in this report, in Sections 18.4.1 and 18.4.2, respectively.

The mill electrical feed consists of a combination of four 3-conductor, 350 MCM, 5 kV Teck 90 cables feeding a dual 477 MCM aluminium-conductor steel-reinforced cable (ACSR) overhead power line capable of providing all necessary mill power. Distribution within the mill is accomplished through a series of 5 kV and 600 V switchgear substation, and motor control centres.

18.4.1 5 kV Distribution Expansion

The current approach for the expansion includes installing two 500 MCM, 5kV Teck 90 cables from the surface powerhouse down bore-holes to the 122 m Level, continuing down the emergency escape-way to all accessible levels. A disconnect is planned for each of the 122 m, 183 m, 244 m, and 305 m Levels. Costs associated with this option have been included in pre-production capital requirements.

18.4.2 13.8 kV Distribution Expansion

Installation of a 13.8 kV system for the underground distribution has been considered. A portion of the required equipment (switches, cable, and connectors) has been procured and is presently stored on site. Items missing include primary 44 kV/13.8 kV transformers, switchgear, and mine portable substations. Costs associated with this 13.8 kV option are unnecessary and are not included in this report. The planned 5 kV expansion remains the most cost-effective approach.

18.4.3 Back-up Power

Two, Atlas Copco 1 MW containerized generator sets are installed on site to supply power during major power outages. These units are tied into the main 5 kV switchgear line-up that allows power to be distributed selectively to any area of the site (Figure 18-5).

Figure 18-5: Dual Backup Generators

18.5 Tailings Management Facility

The design basis and operating criteria for the project were developed with information provided by and input from Rubicon and further developed by our team. Table 18-2 presents the anticipated tailings tonnage on an annualized basis.

Table 18-2: Forecasted Annual Tailings

	Daily (t)	Annual (t)	Paste (t)	Tailings (t)
Year -2	0	11,547	4,619	6,928
Year -1	748	269,237	107,695	161,542
Year 1	1,235	444,715	177,886	266,829
Year 2	1,555	559,641	223,856	335,785
Year 3	1,691	608,629	243,452	365,177
Year 4	1,696	610,662	244,265	366,397
Year 5	1,502	540,765	216,306	324,459
Total		**3,045,196**	**1,218,078**	**1,827,118**

The main capacity criteria for the project are as follows:

- Total tonnage of ore to be milled is 3.045 Mt
- Tailings mass reporting to the TMF is 1.827 Mt (60% of the total ore tonnage)
- Tailings solids content for Slurry Tailings is 37% (possibly up to 43%) w/w

- Specific gravity of the tailings' solids and underground mine waste rock ranges from 2.89 to 3.03 t/m^3
- Initial density of the slurry tailings in the TMF is estimated to be 1.4 t/m^3
- Final average settled dry density of the slurry tailings is estimated to be 1.6 t/m^3
- TMF capacity is estimated at 1.14 Mm^3 (1.83 Mt) tailings to design Elevation 373 m (Stage 3).

The tailings will be pumped to the TSF in a slurry form thickened to up to 43% w/w. Previous mill testing resulted in a slurry density of about 37% w/w. Although a thicker density is preferred the TSF can handle 37% w/w. Water will be returned to the mill from the tailings pond for which a causeway and pump house have already been built and used.

The design by Knight Piésold dated 5 March 2015 is used as the baseline for the calculations herein. The various subsequent Wood (AMEC) tailings dams designs focused on finalizing Stage 1 and the spillway only and did not significantly vary from the Knight Piésold design. The ongoing volume considerations are presented in the Knight Piésold work and were reviewed by the author. Wood, however, states that by incorporating a spillway at early stages they can capture more volume in the facility at that stage. Based on the required total volume it is evident that the facility need only be built up to Stage 3 (elevation 373 m) of the original Knight Piésold design. Therefore, the current tailings facility will store the current required volume if built up to Stage 3. The efficiencies established by Wood should apply to this situation if the stated density is not achieved in the short timeframe being considered. Therefore, the PEA considered raising the dams in 2 stages to elevation 373 m on the existing footprint.

The capital cost for Stage 1 has already been incurred and is not included here. The capital costs for two additional raises (Stages 2 and 3) have been included as has the construction of a final spillway for Stage 3. A spillway for Stage 2 was not included, as the need can be circumvented by proceeding to Stage 3 immediately or other allowances be made. Allowance has been made in sustaining capital for an additional investigation required for the final design of Stages 2 and 3. During the feasibility study it would be prudent to consider TSF Stages 2 and 3 as one stage.

The TMF will utilize waste mine rock either hoisted or trucked to surface. Considering the reduced quantities, it is not expected to be efficient to hydro-cyclone tailings to obtain dam construction material and has therefore not been considered. Similarly, it may not be advantageous to use the paste facility to deposit paste during the final years of the TSF life. These options could, however, be evaluated in a subsequent detailed design phase.

18.6 Electrical Conclusions and Recommendations

The Phoenix Project's current electrical power infrastructure is modern, automated, and robust. It is well suited to eventually support a mine and mill complex operating consistently at 1,800 t/d. The current configuration can be readily expanded into new areas as mine development moves forward and can easily support potential increased production rates in excess of 1,800 t/d.

Rubicon has engaged in a process that will ensure the Phoenix site has available, the necessary electrical power required from Hydro One for project ramp up.

Additionally, the Project could benefit from the implementation of the 5-kV distribution expansion over that of the 13.8 kV approach since it is not evident that the added cost required to construct and integrate a new 44-13.8 kV substation on surface would provide a significant benefit. The available 13.8 kV rated equipment however, can be used in a 5-kV deployment providing some cost savings.

19 MARKET STUDIES AND CONTRACTS

Gold will be the only concentrate produced at the Phoenix Gold Project. While a specific marketing study was not undertaken, gold is freely traded, at prices that are publicly known and the prospect for the sale of any gold production is virtually guaranteed. The price of gold is usually quoted in U.S. dollars per troy ounce. Figure 19-1 represents the gold price and the American dollar to the Canadian dollar exchange rate over the past five years. For the basis of the PEA, a gold price of US$1,325/oz and an exchange rate of US$1 = CAD$1.33 was used.

Figure 19-1: Gold Prices and U.S. Dollar Correlation – 5 Years

Doré bars would be produced as part of the existing milling process at the Project. The doré would be transported via a contracted security company from the mill in Red Lake to a contracted refinery. An agreement would be required with a refinery to sell doré and produce gold bullion.

The services of outside consultants knowledgeable in the areas of marketing, finance, and law, as well as logistics and contract management, may be employed to guide the company in the process of selling the gold produced at the Project.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Project is located on McFinley Peninsula in the East Bay area of Red Lake, a valued recreational lake. The surrounding lands and water are used for wilderness and recreation, as well as Mineral Resource development and forestry. The Project itself is a brownfield site, developed in the 1980s and acquired by Rubicon Minerals in 2002.

The Project commenced with advanced exploration in 2009, development and production from 2011 to 2015, then was placed in temporary suspension at the end of 2015. The Project returned to development and production status in July 2018, to process a 40,000-tonne bulk sample and continue mine development activities. The Project is currently undergoing a PEA with the goal of becoming a producing gold mine.

Rubicon holds a certified Closure Plan for the Project, filed with ENDM, which addresses and provides financial assurance for all known environmental liabilities. The Project also holds all but one permit, all licences, and approvals in support of Project operation. In Q2 2019, Rubicon submitted a Notice of Material Change and an amendment to the filed certified Closure Plan that when filed will support the ongoing operation of a 1,250 t/d mine.

20.1 Environmental Regulatory Setting

The Project is subject to Canadian, Ontario, and municipal permitting related to metal mining. Both the federal and provincial governments have well-established formal environmental assessment processes to review major projects that fall within their respective legislative requirements. Environmental assessments are designed to identify potential adverse environmental effects and then to minimize or mitigate these potential environmental effects. They also incorporate environmental factors into decision making. Following environmental assessment, a project requires permits, licences, or approvals for specific environmental discharges to air, water, and soil and to address other legislative requirements. A project is then regulated through all life-cycle phases (construction, operation, closure, and post-closure), by both federal and provincial agencies.

A formal requirement of both environmental assessment and permitting is to consult with Indigenous and local communities on their environmental, social, and cultural concerns to protect the surrounding environment from physical, chemical, biological, and social effects of potential off-site discharges to air, water, and soil.

20.1.1 Current Regulatory Status

Rubicon currently holds a certified Closure Plan filed with the MENDM, which addresses all known environmental liabilities and provides $7.7 million of financial assurance for meeting its obligations

under the Ontario Mine Rehabilitation Code. The Project also holds several other environmental and land use permits, licences, and approvals required to permit the activities described in the Closure Plan. These regulatory documents are subject to required amendments prior to resuming ongoing mine production (Table 20-1).

On September 8, 2015, Rubicon received a Director's Order from the Ministry of the Environment and Climate Change (last amended on January 25, 2016). The immediate requirements of this Order were completed within the specified timeframes. However, if the Project proceeds to mine production status under the Ontario *Mining Act,* the requirement to install and commission an ammonia treatment plant remains outstanding.

In May 2017, Rubicon was issued a Provincial Officer's Order (No. 7655-AMAQDJ) regarding the non-performance of the domestic sewage treatment at the site. The Ecoflo® treatment system was inoperable during the winter months due to low flows into the system (the Project was in temporary suspension), and the other approved treatment systems were not installed due to the suspension of activities in 2015. The Order permits Rubicon to use existing septic tanks as temporary holding tanks. A licenced sewage hauler has been contracted to pump out these tanks on a regular basis. This system of managing domestic sewage is approved up until Rubicon resumes mine production. Prior to production resuming, Rubicon must apply for and receive approval to install and operate appropriate sewage treatment systems for the on-site dormitory (camp facilities) and operational buildings.

In July 2018, Rubicon submitted a Notice of Project Status to the ENDM to end the Project's temporary suspension and resume limited production status. The ENDM accepted this application, allowing Rubicon to undertake the described underground development and process a 40,000-tonne bulk sample through the mill. Rubicon also obtained relief from the Ontario Ministry of the Environment, Conservation and Parks (MECP) on the Director's Order and Provincial Officer's Order for the duration of the underground development and bulk sample described in the Notice of Project Status and related documents. The ammonia treatment plant and the site sewage works are required to be operational prior to the Project resuming regular, ongoing production.

Rubicon completed the bulk sample processing in the second half of 2018, in compliance with all regulatory requirements. It continues to do underground exploration.

In Q2 2019, Rubicon submitted a Notice of Material Change and an amendment to the development and production Closure Plan, to operate a 1,250 t/d (annual average) gold mine, in accordance with Section 141 of Ontario's *Mining Act*.

Rubicon is currently in material compliance and has fulfilled the monitoring and reporting obligations of the permits, licences, and approvals listed in Table 20-1. Rubicon is also in material compliance with obligations under federal legislation including the *Canadian Environmental Assessment Act* and Metal Mining and Effluent Regulations under the *Fisheries Act*.

Table 20-1: Current Environmental Permits and Approvals

Permit	Regulatory Agency	Relevant Legislation	Date of Issuance	Rationale
Phoenix Gold Project (Production) Closure Plan	Ministry of Northern Development and Mines	*Mining Act*	Dec. 2, 2011 (amended Jun. 16, 2016; amendment submitted Q2 2019)	Legislated requirement for advanced exploration, construction, operation, and closure of a mine in Ontario
Permit to Take Water 3812-9C9KVF	Ministry of the Environment	Ontario *Water Resources Act*	Dec. 11, 2008 (Renewed Nov. 20, 2013)	Permit withdrawal of water from mine shaft
Permit to Take Water 3585-85KGHG	Ministry of the Environment	Ontario *Water Resources Act*	November 19, 2008 (last amendment May 2010)	Permit withdrawal of water from East Bay of Red Lake
ECA 1362-AA2HXS	Ministry of the Environment and Climate Change	*Environmental Protection Act*	August 5, 2016	Approve Industrial and domestic sewage works
ECA 6656-8RVMES	Ministry of the Environment	*Environmental Protection Act*	January 27, 2009 (last amendment February 28, 2012)	Approve air emissions, noise, and odour
ECA 0244-8YWLBB	Ministry of the Environment	*Environmental Protection Act*	December 21, 2012 (ownership transferred to Rubicon Minerals effective October 16, 2014)	Approve air emissions, noise, and odour from off-site laboratory
Easement over Crown Land	Ministry of Natural Resources	*Public Lands Act*	September 2, 2011	Approve easement over Crown-owned surface rights for access corridor
LRIA Approval No. RL-2014-01, RL-2014-01C	Ministry of Natural Resources and Forestry	*Lakes and Rivers Improvement Act*	January 23, 2009 (last amended November 6, 2015)	Approve Stage 1 construction of the tailings management facility dams and emergency spillway
Amendment to the Zoning By-Law 1277-10	Municipality of Red Lake	Municipal By-Law 1277-10	Process completed in February 2011	Change the zoning of the Project Site to mineral mining (MM). MM zoning permits mine development, operation, and closure.
Land Use Permit 1204-1010939	Ministry of Natural Resources and Forestry	*Public Lands Act*	October 1, 2015 (Expires September 30, 2020)	Approve use of Crown land for groundwater monitoring wells
Land Use Permit 1204-101951	Ministry of Natural Resources and Forestry	*Public Lands Act*	February 1, 2016 (Expires January 31, 2021)	Approve use of crown land for effluent discharge pipeline

20.1.2 Federal Environmental Assessment Process

In 2011, the Canadian Environmental Assessment Agency (CEAA) confirmed that the Phoenix Project, as described in the Project Description sent to CEAA, would not trigger an environmental assessment pursuant to the Canadian *Environmental Assessment Act* (1992). The mine subsequently became an operating mine subject to other federal legislation including the *Environmental Protection Act* and the Metal Mining Regulations under the *Fisheries Act*.

In 2012, the Canadian *Environmental Assessment Act* was amended to change how projects were selected for review (CEAA, 2012). However, this change in the *Act* did not retroactively affect the decision given to the Project, in 2011.

20.1.3 Provincial Environmental Assessment Process

The Ontario *Environmental Assessment Act* is administered by the Ministry of the Environment, Conservation and Parks (MECP) and the Ministry of Natural Resources and Forestry (MNRF). The *Act* promotes responsible environmental decision making and ensures that interested parties have an opportunity to comment on projects that may affect them. Interested parties may make a designation request to the MECP to have a project referred to an individual environmental assessment. The MECP assesses the merits of the request and may make a recommendation to the Minister, as outlined on the MECP website under the tab titled Environmental Assessments under Designating Regulations and Voluntary Agreements.

There were no designation requests received by the Ministries for an individual environmental assessment of the Project prior to or following the resumption of production in 2018 to process the bulk sample.

Rubicon did undertake Class Environmental Assessments for Resource Stewardship and Facility Development Projects for a portion of the corridor to connect the Project site to Nungesser Road in 2011. It also conducted an environmental assessment related to the shoreline land tenure and the relocation of the effluent discharge line in East Bay resulting in Land Use Permit 1204-1010951. A Class Environmental Assessment was completed in 2011, pursuant to Ontario Regulation 116/01 for the use of <5 MW of diesel generation at the Project site. These generators are now permitted under ECA 6656-8RVMES.

20.1.4 EA Requirements for the Project

There are no outstanding requirements for either federal or provincial environmental assessments for the currently permitted Project. However, should the Project wish to increase production above 1,250 t/d they would be required to submit a Notice of Material Change to the MENDM, under Section 144 (2) of the Ontario *Mining Act* and notify MECP and other agencies regarding potential amendments to permits. Even though the IAA came into force on 28 August 2019 and the 2011

decision by the CEAA to exempt Phoenix from a Federal EA would appear to be grandfathered under the IAA, there remains a possibility that Phoenix may be required to submit a Project Description to the CEAA, under section 8(1) of the Canadian *Environmental Assessment Act*, for review and subsequent decision on whether this change requires an environmental assessment, or not.

20.2 Environmental Approvals Process

This section describes the federal and provincial approvals processes required before resuming operation of a 1,250 t/d mine and for a potential production rate increase to 1,800 t/d in the future.

20.2.1 Federal Approvals Process

At the time of the site visit, (April 2019) CEAA, 2012 and its associated regulations were in effect. The 2011 decision that no federal environmental assessment is required for the current Project also remained in effect. Since April, Bill C-69 passed into law (with significant amendments in the Senate) and the IAA came into force on 28 August 2019. The IAA now supersedes the CEAA. At the time of writing, the 2011 decision by the CEAA to exempt Phoenix from a Federal EA would appear to be grand-fathered under the IAA. This exemption will require confirmation during future studies.

Based on the significant geographical limitations imposed by the Project being on McFinley Peninsula it is unlikely that any expansion of the Phoenix Project would "result in an increase in the area of mine operations of 50% or more."

Based on current regulatory approvals and a non-confrontational relationship with Indigenous peoples and the general public, the potential for "adverse environmental effects or public concerns" are likewise unlikely.

20.2.2 Provincial Approvals Process

Rubicon has completed several Class Environmental Assessments, filed a certified Closure Plan, and holds many permits, licences, and approvals to permit the operation of a 1,250 t/d production mine.

Rubicon is reviewing all of its other permits, licences, and approvals to determine what, if any, amendments are required to permit the activities described in the Closure Plan submitted in Q2 2019. Table 20-2 lists each of these approvals and potential amendments that would be required before the 1,250 t/d mine production may begin. Based on the proposed Project development timeline, applying for, and obtaining the necessary amendments is achievable.

Table 20-2: Anticipated Amendments to Approvals for a 1,250 t/d Mine

Permit	Regulatory Agency	Regulated Activities	Rationale for Amendment
Closure Plan (production)	Ministry of Energy Northern Development and Mines	Construction development, operation, and closure of the project	Submitted amendment in Q2 2019 to consolidate all current information into a single document and to provide detailed cost estimates for the financial assurance
Permit to Take Water 3812-9C9KVF	Ministry of the Environment, Conservation and Parks	Withdrawal of water from Mine shaft; Expires Nov. 20, 2023	No amendment anticipated for 1,200 t/d; renewal required prior to expiry date
Permit to Take Water 3585-85KGHG	Ministry of the Environment, Conservation and Parks	Withdrawal of water from East Bay of Red Lake; expires May 2020	No amendment anticipated for 1,200 t/d; renewal required before expiry date
ECA 1362-AA2HXS	Ministry of the Environment, Conservation and Parks	Industrial and domestic sewage works; last amendment August 5, 2016	Requires amendment for domestic sewage treatment (camp and mill); potable water at camp; adding ammonia treatment and other upgrades to mill effluent treatment system; upgrades to surface runoff management system; additional groundwater wells; fuel storage containment
ECA 6656-8RVMES	Ministry of the Environment, Conservation and Parks	Air, noise, & odour for Project site; last amendment February 28, 2012	Current approval is for 1,200 t/d. Increasing production to 1,250 t/d may require an update of ESDM (Emission Summary and Dispersion Model) or an Administrative Amendment
ECA 0244-8YWLBB	Ministry of the Environment, Conservation and Parks	Air emissions, noise, and odour from off-site laboratory; last amendment December 21, 2012	No amendment anticipated for 1,250 t/d
LRIA Approval No. RL-2014-01, RL-2014-01C	Ministry of Natural Resources and Forestry	Tailings management facility dams and emergency spillway; last amendment November 6, 2015	Any planned modification or changes to the existing tailings management facility or storage method may require an amendment
Land Use Permit 1204-1010939	Ministry of Natural Resources and Forestry	Use of Crown land for monitoring water wells at Project Site; expires September 30, 2020	No amendment anticipated for 1,250 t/d renewal required before expiry date
Land Use Permit 1204-101951	Ministry of Natural Resources and Forestry	Use of Crown land for effluent discharge pipeline; expires January 31, 2021	No amendment anticipated for 1,250 t/d renewal required before expiry date

An increase in production rate to 1,800 t/d in the future would require additional amendments to the Closure Plan and other permits, approvals, and licences. The key steps in the approvals processes prior to increasing production to 1,800 t/d include completing any additional required environmental assessments under the Ontario *Environmental Assessment Act*, submitting a Notice of Material Change, and filing an amendment to the certified Closure Plan under the Ontario *Mining Act*. Subsequently, other permits, approvals, and licences may be obtained under the Ontario *Water Resources Act* and *Environmental Protection Act*, and others.

Table 20-3: Anticipated Amendments to Approvals for an 1,800 t/d Mine

Permit	Regulatory Agency	Regulated Activities	Rationale for Amendment
Closure Plan (production)	Ministry of Energy Northern Development and Mines	Legislative requirement for development, operation, and closure of the project	Increased production rate and modified dimensions of the tailings management facility upon closure, modified financial assurance. The spatial extent of the Project footprint will not be materially affected by the increased production rate
Permit to Take Water 3812-9C9KVF	Ministry of the Environment, Conservation and Parks	Withdrawal of water from mine shaft	An expansion of mine workings may result in increased volume of water to be pumped from the mine
Permit to Take Water 3585-85KkGHG	Ministry of the Environment, Conservation and Parks	Withdrawal of water from East Bay of Red Lake	Increased production rate may require additional withdrawal of water from East Bay
ECA 1362-AA2HXS	Ministry of the Environment, Conservation and Parks	Industrial and domestic sewage works	Increased production rate (administrative amendment), potential changes to water balance; approved engineering design for tailings management facility modifications during late stages of mine life
ECA 6656-8RVMES	Ministry of the Environment, Conservation and Parks	Air emissions, noise, and odour from the site	Modifications to mine ventilation and increased return air volume; additional potential sources of fugitive dust and gaseous emissions
ECA 0244-8YWLBB	Ministry of the Environment, Conservation and Parks	Air emissions, noise, and odour from off-site laboratory	No amendment anticipated for 1,800 t/d
LRIA Approval No. RL-2014-01, RL-2014-01C	Ministry of Natural Resources and Forestry	Construction of the TMF dams and Emergency Spillway	Ongoing tailings management facility construction, and any planned modification or changes to the existing tailings management facility or storage method

20.3 Environmental Studies and Management

20.3.1 Environmental Studies

Environmental studies, baseline monitoring, and compliance monitoring activities are largely mandated by environmental permits, approvals, licences, and legislation (e.g., *Fisheries Act* and its Metal and Diamond Mining Effluent Regulations [MDMER]).

Monthly and annual environmental performance reports generated from the studies and monitoring are submitted to the appropriate regulatory agencies, and are made available for Indigenous and public review and comment. Examples of these submissions include:

- Monthly surface water monitoring since 2007, on or in, the vicinity of the Project
- Semi-annual sampling of groundwater monitoring wells, since 2009
- Archaeological assessment by Ross Associates, 2010
- Annual species-at-risk assessment by Northern Bioscience
- Background conditions studies by BZ Environmental
- Phase 1 MDMER biological assessment by EAG Inc. (EAG)
- Effluent mixing and plume delineation studies by EAG and Story Environmental
- Assessment of risks from the Project to the downstream environment by Novatox
- Hydrogeological characterization by AMEC Earth and Environmental
- Phase 1 and Phase 2 environmental site assessments by True Grit Consulting
- Risk assessment of the groundwater and soils at the Project site by Novatox
- Geochemical characterization of development rock by AMEC Earth and Environmental
- Geotechnical assessments of underground workings by AMEC Earth and Environmental and AMC Mining Consultants
- Project reviews by WESA Consultants and ArrowBlade Consulting Services.

No biological values that would preclude the redevelopment of the Project site have been identified to date, including species at risk, ecologically significant features, regionally significant wetlands, significant wildlife habitat, and environmentally sensitive areas. Ongoing field studies have been conducted with input from the Ministry of Natural Resources and Forestry to ensure adherence to the provincial *Endangered Species Act*, *Public Lands Act*, *Crown Forest Sustainability Act*, and the Provincial Policy Statement that has been issued pursuant to Section 3 of the *Planning Act*.

Consultation to date with Indigenous communities has not identified the presence of cultural heritage values in the vicinity of the Project site. In addition, the desktop and fieldwork by Ross Archaeological Research Associates did not identify any areas with a high potential to host cultural heritage values on McFinley Peninsula (Ross Associates, 2010). As the Project involves the redevelopment of the

existing footprint with only moderate expansion, the potential for impacts to cultural heritage values as a result of the redevelopment of the brownfield Project site is considered to be negligible.

20.3.2 Environmental Management

Rubicon developed an environmental management system (EMS) for the Project (Rubicon, 2007). This document was developed and is used to identify and manage environmental compliance obligations on the Project site, McFinley Peninsula, and East Bay of Red Lake. The key elements of the EMS are:

- Lists of relevant legislation, approvals, agreements, and other documents that contain environmental obligations
- Division of the property into discrete environmental management areas, including environmental obligations and inspection frequency
- Inspection protocols with responsible individuals
- Procedures for addressing non-compliance issues and conditions
- Required documentation, including internal reporting on performance and auditing.

The EMS lists the Project's legal compliance obligations, audit and reporting protocols, community engagement/consultation obligations, commitments registry from Indigenous agreements, and more. The EMS is a living document that is expected to be used as a tool to manage sustainability and social responsibility commitments and obligations.

20.4 Social Setting

Rubicon has been actively involved in Indigenous and public consultation and outreach, since 2008.

20.4.1 Indigenous Consultation

Prior to the approval and development of a mining project, the provincial Crown is responsible for identifying and consulting with Indigenous communities whose Aboriginal and Treaty rights may be impacted. The Crown reserves the right to delegate the procedural aspects of any development to the proponent. The process of identifying and delegating the consultation obligation is legislated primarily by Part VII of the Ontario *Mining Act* and Ontario Regulation 240/00.

Rubicon, under direction from the Crown, and resulting from an independent traditional land- use study (Forbes, 2011), has been consulting with the Lac Seul First Nation and Wabauskang First Nation. In January 2010, Rubicon became the first public company in the mining district of the Municipality of Red Lake to sign an Exploration Accommodation Agreement with the Lac Seul First Nation. In 2014, Rubicon signed an Exploration Accommodation Agreement with Wabauskang First Nation and settled the judicial review of the Closure Plan that was launched in 2012.

In Q2 2019, Rubicon submitted a Notice of Material Change and an amendment to its Closure Plan to the MENDM. The ENDM Director of Mine Rehabilitation will subsequently provide written direction to Rubicon identifying the Aboriginal communities that are to be notified and/or consulted prior to the MENDM's acceptance of the amended Closure Plan.

Over the past several years Rubicon has established and maintained a successful history of consultation with the local Indigenous communities and is committed to continuing consultation over the life of the Project.

20.4.2 Public Consultation

Public information sessions have been held annually in the Red Lake community, since 2008. There are no unresolved negative comments from these public information sessions. Rubicon maintains an open-door policy to proactively identify and address stakeholder concerns regarding the Project. Formal public consultation since 2015 is summarized in Table 20-4.

Table 20-4: Summary of Public Consultation Since 2015

Date	Summary of Public Consultation	Summary of Information Provided	Summary of Received Comments, if any
December 2015	Voluntary Annual Public Information Session. Notice was in general accordance with Sec. 8 of Ontario Regulation 240/00	Local community outreach prior to the information session. Overview PowerPoint presentation to provide an infrastructure update, suspension of mining activities, initiation of Phoenix Gold Project Implementation Plan.	Comments were received regarding employment and business concerns if the Project does not re-start.
January 2017	Voluntary Annual Public Information Session. Notice was in general accordance with Sec. 8 of Ontario Regulation 240/00	Overview PowerPoint presentation of the Project, discussed health, safety and environment, CEO vision and strategy, exploration path forward and proposed development and trial mining	
February 2018	Voluntary Annual Public Information Session. Notice was in general accordance with Sec. 8 of Ontario Regulation 240/00	Overview PowerPoint presentation of the Project, discussed test mining and bulk sample plans for 2018, including move to Mine Development and Production required by the MENDM	
February 2019	Voluntary Annual Public Information Session. Notice was in general accordance with Sec. 8 of Ontario Regulation 240/00	The CEO gave an overview of the exploration drilling and bulk sample results. The Project plans to continue exploration in 2019	Questions were asked about the geology of the surrounding area and the results from the bulk sample. There were no negative comments or concerns.

No Public complaints have been received since the last public meeting in February 2019.

20.5 Tailings Disposal

A TMF consistent with contemporary regulatory requirements received a Certificate of Approval and was constructed at the Project site by McFinley Mines Ltd. in 1988. The site chosen was a large topographic depression lying immediately west of the Project shaft. A retaining dam was constructed to impound tailings and effluent prior to ultimate drainage south into the waters of East Bay. The test milling of an estimated 2,500-tonne bulk sample terminated in 1989, resulting in minimal use of the tailings management facility.

The TMF and other sewage works were subsequently designed for expansion in 2010, and a detailed design for this expansion was completed in 2012. Rubicon received ECAs under the *Environmental Protection Act*. The TMF has been constructed to Stage 1 design elevation in accordance with an approval issued pursuant to the *Ontario Lakes and Rivers Improvement Act*. Approximately 57,000 tonnes of tailings were deposited in the TMF in 2015 before the mine was placed into Temporary Suspension under the Ontario *Mining Act*.

In 2018, approval was received to conduct a 40,000-tonne bulk sample program. The tailings from this bulk sample was also placed in the TMF.

20.6 Environmental Sensitivities

The Project is located on a peninsula in a valued recreational lake. To protect the lake, and seasonal residents using the lake, the Project has included design measures to minimize or eliminate the impact of off-site discharges of water, fugitive dust, and noise.

20.6.1 Water Discharge

The responsible management of water discharges into the natural environment is a priority for Rubicon and a condition of their environmental approvals. Project design features that have been or will be implemented to manage and control water discharges include:

- Engineered runoff collection system around the perimeter of the Project site that directs water to the TMF
- TMF is designed to contain and manage a robust environmental design flood
- Cyanide destruction process (SO_2/O_2 system) in the tailings slurry prior to discharge to the TMF
- Permitted effluent treatment system that treats surplus TMF water to regulated numerical and biological limits prior to discharge to the natural environment
- Mine water pumped from the underground workings and water reclaimed from the TMF will be recycled for use in the mill to minimize withdrawal from the East Bay of Red Lake.

20.6.2 Fugitive Dust

Air emission sources that have the potential to create fugitive dust include diesel-fired equipment; propane– and natural gas–fired heaters; return air from underground operations; vehicle operation on roads; the TMF; and crushing and material handling typically associated with surface and underground mining and milling activities. Rubicon has implemented a best practices management plan for the control of fugitive dust.

Practices to minimize fugitive dust are listed in Ministry of the Environment and Climate Change (2009) and Environment Canada (2009) and are, in part, listed below:

- Minimize vehicle speed, use dust suppressants, minimize fines from material handling areas
- Minimize stockpile size and use wind breaks
- Maintain a wetted tailings surface or use tackifier or binder to prevent wind entrainment
- Enclose material transfer points and use water sprays.

20.6.3 Noise

There are seasonal residential interests on the East Bay of Red Lake with potential for exposure to noise from the Project. Rubicon has designed infrastructure for the Project to reduce and control noise emissions and minimize any disturbance of the seasonal residential interests.

20.7 Closure Plan

Rubicon submitted a Certified Closure Plan that the ENDM accepted for filing on June 16, 2016. This Closure Plan provides a project description and the activities that Rubicon will undertake to ensure chemical and physical stability of the site, revegetation and environmental monitoring during closure and post-closure, according to the Ontario *Mining Act* and Ontario Regulation 240/00, including compliance with the Mine Rehabilitation Code of Ontario.

Rubicon submitted a Notice of Material Change and an amended Closure Plan to the ENDM in Q2 2019 to update the document with the current plans for a 1,250 t/d Au mine. It is expected that this amended Closure Plan will be accepted for filing in due course.

20.7.1 Closure Cost Estimate

The Closure Plan provides for approximately C$7.7 million of financial assurance that was provided to the ENDM in June 2016. The value of financial assurance was recalculated and reconfirmed for the amended Closure Plan submitted in Q2 2019.

21 CAPITAL AND OPERATING COSTS

Conceptual operating costs were derived from 2018 test-mining costs at Rubicon's Phoenix Gold Project, budget estimates from Manroc Developments Inc. and first principles cost calculations by TMAC's technical team. Costs have been compared to similar operations and found to be conservative in nature and quantity. Operating cost estimates (Table 21-1) have been assigned an accuracy level of ± 30% for all items.

Table 21-1: Conceptual Project Operating Costs

	Costs		
	($/t processed)	(C$/oz)	(US$/oz)
Mining costs	88.14	543.79	408.86
Milling costs	32.70	201.76	151.70
G&A	7.82	48.27	36.29
Mining tax	-	-	-
Total	**128.67**	**793.81**	**596.85**

Operating costs have been based on several different sources, including Rubicon's recent test mining and bulk sample processing program completed in 2018, Manroc Developments Inc., OEM suppliers, and other comparable Canadian projects, for labour, consumables, and parts. The cost of power was scrutinized by TMAC for historical costs and forward-looking analysis as provided by both Rubicon and Hydro One Networks Inc. Critical operating cost components are given in Table 21-2.

Table 21-2: Operating Cost Components

Cost Data	Unit	Rate
US Gold Price	$/oz	1,325
Gold Recovery	%	95
Exchange Rate	C$/US$	0.7519
Capital Contingency	%	15
Diesel	$/L	1.20
Power	$/kW/h	0.08
Propane	$/L	0.65
Binder (cement/flyash)	$/t	159.80
Shotcrete	$/m^3	197.60

All waste development is based on costing to allow a mining contractor to complete the work. The development would be completed using: one- or two-boom electric-hydraulic drill jumbos; 2.7 to 6 m^3 bucket LHDs; various-sized haulage trucks up to 40 t capacity; mechanical and scissor lift/bolters; as well as other rubber-tired diesel-powered support equipment.

TMAC detailed development costs using first principle methods and were able to favorably compare the development rates and unit prices to actual experience in Rubicon's 2018 test-mining work. LOM waste development used in the PEA is summarized in Table 21-3.

Table 21-3: Conceptual LOM Waste Development Costs and Performance

Type of Development	Length (m)	Single Heading ($/m)	Single Heading (m/d)	Multiple Heading ($/m)	Multiple Heading (m/d)	Waste (t)
Ramp, 4.5 m x 4.5 m, 15%	9,329	6,230	3.1	4,151	4.7	472,281
Level Access & Haulage, 4.5 m x 4.5 m	1,070	6,230	3.1	4,151	4.7	54,169
Level Access & Haulage, 4 m x 4 m	10,303	5,980	3.9	3,984	5.8	521,589
Stope Access, 3 m x 3 m	5,808	5,180	4.2	3,451	6.3	294,030
Raising by Alimak, 3 m dia.	2,745	7,670	2.4			61,763
Total	**29,353**					**1,408,793**

Note: No contingency included in costs

An example of a summary of task costs to achieve a single heading, 4.5 m x 4.5 m development round is shown in Table 21-4. First principle calculations, vetted by Rubicon operating personnel, confirm an approximately 17.9 h cycle. The principle task distributions for that round are shown in Figure 21-1.

Table 21-4: Detailed Cost Breakdown for Ramp Development – Single Heading Development Round Details (4.5 m x 4.5 m x 3.51 m advance)

Development Tasks	Labour ($/m)	Labour (%)	Labour ($)
Drill	853	58%	495
Blast	663	62%	413
Muck	308	73%	224
Bolt	1,578	54%	853
Truck from Stockpile	503	61%	310
Shotcrete	104	25%	26
Services	440	28%	121
Direct Cost	4,449	55%	2,441
Indirect Costs @ 40%	1,780		
Total	**6,229**	**/metre**	
	87.35	**/waste tonne**	
	21,863.79	**/3.51 m round**	

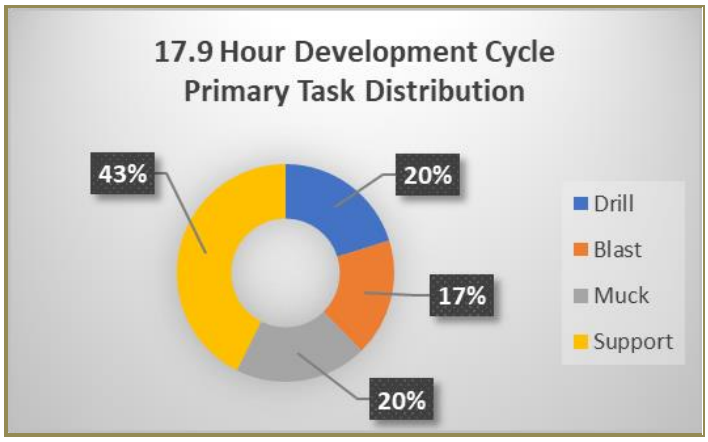


Figure 21-1: Typical 4.5 m x 4.5 m Cycle Task Distribution

The PEA conceptual mine plan identifies four mining methods as best suited to extract the mineralized material—Longhole, Uppers, MBRM, and C&F. Three of the four methods have been used historically with success at the Phoenix Gold Project and in the Red Lake camp in general. The fourth method, MBRM, has been vetted by TMAC and Rubicon following a visit to Manroc Development Inc.'s operations in the Hemlo Camp just east of Marathon, Ontario. Further, following a review of several prospective stoping zones at the Project, Manroc has confidence the method can be applied and has proposed budget pricing and productivity schedules used in this study (Table 21-5).

Table 21-5: Conceptual Mining Method Parameters and Costs

Metrics	Sub-Level Longhole	Uppers	MBRM	C&F
Conceptual LOM tonnes	1,595,921	708,880	513,974	226,420
Conceptual LOM tonnes (%)	52.40%	23.30%	16.90%	7.40%
Total Stopes	66	172	43	60
Average Stope Size (tonnes)	24,181	4,121	11,953	3,774
Average Height x Width x Strike	36 m x 8 m x 25 m	16 m x 6 m x 12 m	37 m x 3 m x 22 m	12 m x 6 m x 16 m
External Dilution*	10%	15%	10%	3%
Average Diluted and Recovered Grade	5.34 g/t Au	5.30 g/t Au	5.31 g/t Au	5.16 g/t Au
Cost per tonne**	C$82.88	C$86.88	C$92.15	C$120.58
Typical Productivity Rates (t/d)	400	300	600	130

Notes: *Waste material from external dilution not at zero grade
**Operating costs include indirect costs but do not have a contingency applied

Throughputs at the mill are expected to vary over the LOM. Unit rates are based on actual costs experienced during the 2018 testing period. Actual costs have been factored using the 6/10^{ths} rule to estimate processing costs. A variety of tonnage scenarios and associated costs are summarized in Table 21-6. The PEA has an average conceptual LOM processing cost of $32.70/t.

Table 21-6: Conceptual Mill Operating Costs at Various Throughputs

	Daily Milling Throughput ->			
	550 t/d ($/t)	**1,050 t/d ($/t)**	**1,550 t/d ($/t)**	**1,850 t/d ($/t)**
Direct Personnel	9.19	8.25	7.06	6.57
Crushing	2.37	2.13	1.82	1.70
Grinding	2.35	2.11	1.80	1.68
Flotation	10.17	9.12	7.81	7.27
Thickening and Drying	0.51	0.46	0.39	0.37
Con/Bullion/Refinery	0.03	0.02	0.02	0.02
Tailings	0.54	0.48	0.41	0.39
Assay Lab	0.64	0.58	0.49	0.46
Site Services (Power)	7.55	6.77	5.79	5.40
Environment	0.62	0.55	0.47	0.44
Warehouse	2.18	1.95	1.67	1.56
Maintenance	0.03	0.02	0.02	0.02
Total	**36.18**	**32.45**	**27.77**	**25.87**

G&A operating costs include costs for maintaining the property in good standing, land taxes, and Mineral Resource usage fees (e.g., water). The G&A operating costs encompass all operating costs associated with operating the site offices and providing materials and supplies for staff.

The average conceptual LOM G&A cost is estimated to C$7.82/t. As an example, typical G&A costs for Year 3 average of 1,691 t/d are summarized in Table 21-7.

The PEA estimates conservative and realistic costs for staff and operating personnel. Conceptual annual cost estimates, including burdens, are shown in Table 21-8 and Table 21-9.

Table 21-7: Example of Year 3 Conceptual G&A Costs

Item	Conceptual Annual Cost ($)
Corporate Costs*	580,000
G&A Equipment Operating Costs	360,677
Environmental Consultation and Permits	385,000
Insurance	205,000
Information Technology	275,000
Accounting	160,000
Human Resources (including safety, security)	25,000
Purchasing and Warehouse	35,000
Site Road Snow Removal	120,000
Ventilation and Heating	1,597,290
Power (other than mill)	217,728
	3,960,662
Tonnes Processed	608,731
Cost per Tonne in Year 3	6.51

Note: *Corporate costs include property insurance and taxes

Table 21-8: Staff Annual Compensation Estimates

Position	Strength at Peak	Annual Salary ($)	Including Burdens ($)
General Manager	1	250,000	400,000
Site Controller	1	160,000	248,000
Administrative Coordinator	1	80,000	104,000
Accountant	1	100,000	130,000
Payroll	1	70,000	91,000
HR Coordinator	1	100,000	130,000
Purchaser	3	75,000	97,500
IT Support	1	75,000	97,500
Mine Superintendent	1	175,000	280,000
Mine General Foreman	1	145,000	217,500
Underground Supervisor	8	122,000	158,600
Mill Superintendent	1	160,000	256,000
Mill General Foreman	1	121,565	158,035
Assay Technician	4	70,000	91,000
Maintenance General Foreman	1	140,000	210,000
Maintenance Supervisor	2	122,000	158,600
Electrical Supervisor	2	122,000	158,600
Manager, Technical Services	1	180,000	279,000
Senior Mine Geologist	1	140,000	182,000
Geologist (PGO)	2	104,000	135,200

Position	Strength at Peak	Annual Salary ($)	Including Burdens ($)
Regional Geologist	1	104,000	135,200
Geologist	4	90,000	117,000
Core Technician Lead	2	53,040	68,952
Core Technician	2	52,000	67,600
Senior Engineer	1	140,000	182,000
Mill Metallurgist	1	140,000	182,000
Planning Engineer	2	104,000	135,200
Mine Technician	2	92,000	119,600
Surveyor	2	75,000	97,500
Health, Safety and Security Coordinator	1	90,000	135,000
Safety/Training Technician	2	75,000	97,500
Environmental Coordinator	1	90,000	135,000
Environmental Technician	1	60,000	78,000
	57		

Table 21-9: Hourly Workers Compensation Estimates

Position	Hourly Rate ($)	Loaded Rate ($)	Position	Hourly Rate ($)	Loaded Rate ($)
Maintenance			***Mill***		
Tradesman Lead Hand	55.00	69.85	Lead Hand Mill	45.00	57.15
Electrician Certified	52.40	66.55	Mill Operator 1	42.00	53.34
Electrician Apprentice 4th Year	47.16	59.89	Mill Operator 2	37.00	46.99
Electrician Apprentice 2nd Year	41.92	53.24	Mill Operator 3	34.00	43.18
Mechanic Certified	52.40	66.55	Mill Operator 4	32.00	40.64
Mechanic Apprentice 4th Year	47.16	59.89	Mill Loader Operator	30.00	38.10
Mechanic Apprentice 2nd Year	41.92	53.24	Labourer	28.00	35.56
Trades Apprentice 1st Year	39.30	49.91	Site Services	34.00	43.18
Millwright Certified	52.40	66.55			
Welders Certified	52.40	66.55	Hoistman	38.00	53.26
Drill Doctor	52.40	66.55	Shaft Miner	36.50	56.36
Underground			Deckman	30.00	48.10
Lead Hand (Spare Supervisor)	40.00	50.80	***Warehouse***		
Miner 1	38.00	83.26	Material Controller 1	28.00	35.56
Miner 2	36.50	74.36	Material Controller 2	24.00	30.48
Miner 3	34.00	63.18	Material Expediter	28.00	35.56
Miner 4	31.00	54.37	Security		
Miner 5	28.00	43.06	Security Officers	25.00	31.75
Construction Miner 1	34.00	58.18			
Construction Miner 2	31.00	54.37			

22 ECONOMIC ANALYSIS

An economic model was developed to estimate the Phoenix Project conceptual LOM plan, comprising mining the Measured, Indicated, and Inferred Resource estimates. The 6.8-year conceptual LOM plan comprises a 20-month Pre-CP period consisting of construction, development, commissioning, and ramp-up to the CP period, which encompasses a 5.1-year production phase. After-tax estimates of Project values were developed to approximate investment value.

The conceptual LOM plan and economic analysis assumes the necessary approvals to exceed currently permitted production of 1,250 t/d, are obtained by the second year of commercial production to enable 1,800 t/d throughput.

Pre-CP work consists of developing a surface ramp to the 122 m Level, improvements in the shaft to achieve nameplate hoist speeds and fresh air raise development from the 610 m Level up to the 305 m Level. Pre-CP work comprises advancement of interlevel ramps to provide a minimum of 12 months lead over production requirements, improvements to surface ventilation fans and internal raises, ramp connection to shaft bottom, advancement of the ramp below the 732 m Level and a gradual production ramp up to 900 t/d, which is 75% of 1,250 t/d permitted throughput.

All costs are in 2019 Canadian dollar nominal terms and inflation has not been considered in the cash-flow analysis. Neither costs nor revenue has been escalated with any Consumer Price Index (CPI) or other base commodities inflation. A C$/US$0.7519 exchange rate has been used in calculating gold revenues in the financial model.

22.1 Economic Parameter Estimates

Economic outcomes of the Project are founded on conceptual LOM model parameters including:

- The Project, on average, is estimated to process 0.55 Mt/a from Year 1 to the end of conceptual LOM
- Average diluted grade processed (both mined and stockpile) is estimated at 5.31 g/t Au over conceptual LOM
- Average process recoveries are estimated to be 95%.

Average gold production of 80,000 oz/a is estimated over the conceptual operating period with peak annual production estimated at 102,000 oz/a in Year 5:

- The PEA conceptual LOM gold price is US$1,325/oz and the Canadian exchange rate is $0.7519
- The Project will produce doré at site
- A 2.0% NSR is payable to Franco-Nevada Corporation and a 1.0% NSR payable to Royal Gold Inc.

The parameters used in the economic analysis have been summarized in Table 22-1.

Table 22-1: Parameters for Economic Evaluation

		Unit	Operating Cost Input Estimates		Unit
Pre-CP, Construction, Development, Commissioning, and Ramp-Up	20	months	Mining Costs	88.14	C$/tonne
Commercial Production Period (CP)	5.1	years	Processing Costs	32.70	C$/tonne
			Site G&A	7.82	C$/tonne
Operating Period (LOM Year)	**6.2**	**years**		**128.67**	
			Capital Cost Inputs		
Assumptions			Development Costs	112.2	C$ million
Gold Price	1,325	US$/oz	Equipment	69.9	C$ million
Exchange Rate	0.7519	C$:US$	Infrastructure Capex	17.7	C$ million
Production Inputs			Surface & Mill Infrastructure	29.3	C$ million
LOM Tonnes Milled (LOM tonne)	3,045,196	tonnes	Closure Costs	7.7	C$ million
CP Tonnes Milled (CP tonne)	2,799,863	tonnes	Contingency (15%)	18.4	C$ million
Diluted Head Grade	5.31	g/t		255.2	C$ million
Gold Recovery	95.0	%	**Cost Summary**		
Total Ounces Recovered	**493,583**	**oz**	**LOM Average Cost**	**624**	**US$/oz**
Average LOM Production	79,610	oz	ASIC	882	US$/oz
Peak Annual Production	101,833	oz	AIC	1,031	US$/oz
Royalty Percentage (LOM)	3.00	%			

Note: Numbers may not add due to rounding

22.2 Capital Costs

Most of the conceptual capital cost estimates (initial and sustaining capital, see Table 22-2) comprise expected underground development (lateral, ramp, and vertical) below the 305 m Level, and connecting the ramp from the 244 m Level to the surface. Including indirect costs, the PEA estimates that the average cost per underground development metre ranges between C$5,500 and C$6,500. This rate is based on actual development costs incurred by the Company during its 2018 test trial mining program and is at the higher end of the range of peer estimates. The equipment fleet assumes equipment lease financing. Other components considered in the conceptual capital cost estimates include:

- Pre-CP capital of C$101.2 million, Table 22-3, is the estimated CAPEX required up to conceptual CP; this includes a 15% contingency
- Capitalized Pre-CP operating cost of C$45.7 million is required to achieve commercial production. During the Pre-CP period, proceeds from sale of Pre-CP gold ounces generated C$77.5 million. After covering the cost of capitalized Pre-CP operating cost, and Royalties, the amount of net Pre-CP operating cash flow is C$28.8 million

- Net Pre-CP capital is C$72.4 (C$101.2 million – C$28.8 million) is required for construction of the Project when considering net Pre-CP operating cash flow of C$28.8 million having been generated from Pre-CP gold production
- Underground development costs include C$32.6 million initial capital and C$79.6 million in sustaining costs
- Equipment costs of C$69.9 million over conceptual LOM reflect equipment lease arrangements based on, over the LOM, 10% down payment plus monthly lease fees, 3% of new cost per month
- Infrastructure costs of C$17.7 million incurred over conceptual LOM include purchases of: ventilation fans; underground electrical work; shaft improvements to return hoist speed to nominal; underground sumps and pumping; mechanical bays; and a truck dump on the 610 m Level
- Mill infrastructure over the conceptual LOM of C$29.3 million includes tailings improvements, two lifts, and improvements in the processing facility to achieve 1,800 t/d
- Pre-production indirect costs of C$1.5 million include diesel costs, generators, trailers, and other miscellaneous items for completion of construction.

Table 22-2: Capital and Operating Cost Estimates - Conceptual LOM

	Total (C$ millions)	Per Unit (C$)	Per Ounce (US$)
Capital Cost Estimates			
(A) Initial Capital (including 15% contingency) (Pre-CP)	101.2	-	-
(B) Capitalized Pre-CP Operating Cost	45.7	186.2/pre-CP tonne	780/pre-CP oz
(C) Royalties (3%) and Other Production Taxes[3] (Pre-CP)	3.0	12.2/pre-CP tonne	51/pre-CP oz
(D) Proceeds from Sale of Pre-CP oz	77.5	-	-
(E) Net Pre-CP Operating Cash Flow: (D) − (B) − (C)	28.8	-	-
Net Pre-CP Capital: (A) − (E)	72.4	-	-
Projected Funding Requirement[4]	80.9	-	-
(F) Sustaining Capital (CP)	154.0	30 million/CP year	258/CP oz
(G) Total LOM Capital Expenditures: (A) + (F)	**255.2**	**41 million/LOM year**	**389/LOM oz**

Notes: [3]Includes community payments.
[4]Based on cumulative net cash flow analysis, factoring in the timing of capital, operating costs, and proceeds from gold sales. C$80.9 million represents the largest net cash flow deficit during the conceptual LOM.

Table 22-3: Pre-CP and Sustaining Capital Requirements - Conceptual LOM

Capital Item	Pre-CP (C$ millions)	Sustaining (C$ millions)
Underground Development and Infrastructure	43.2	86.7
Equipment	16.9	53.1
Surface and Mill (Including TMF, Water Treatment, Crushers, Camp Upgrades, Buildings, etc.)	22.8	6.5
Closure Costs	-	7.7
Contingency (15%)	18.4	-
Total Initial Capital	**101.2**	**-**
Total Sustaining Capital	**-**	**154.0**
Total Conceptual LOM Capital	**255.2**	
Capitalized Pre-CP Operating Costs	45.7	-
Proceeds from Sale of Pre-CP oz	74.5	-
Net Pre-CP Operating Cash Flow	28.8	-
Net Pre-CP Capital	72.4	-
Projected Funding Requirement[8]	80.9	-

Note: Numbers may not add due to rounding

22.3 Operating Costs

The conceptual operating cost estimates in Table 22-4 were derived from the Company's 2018 test trial mining and bulk sample processing program, allowing the Company to collect actual operating cost information. The largest component of operating cost is labour (50%–60%). The majority of the operating costs are anticipated to be in C$. Conceptual mining, milling, and G&A costs of C$128.67/t were estimated based on historical 2018 costs experience during test mining, contractor, and OEM budget quotes, and first principle estimates.

Table 22-4: Conceptual Operating Cost Estimates

	Total (C$ million)	Per Unit (C$)	Per Ounce (US$)
Mining	268.4	88.1/LOM tonne	409/LOM oz
Processing	99.6	32.7/LOM tonne	152/LOM oz
Site G&A	23.8	7.8/LOM tonne	36/LOM oz
(H) Total conceptual LOM operating costs	391.8	128.7/LOM tonne	597/LOM oz
(I) Total commercial operating costs (CP): (H) – (B)	346.1	123.6/CP tonne	579/CP oz
(J) Royalties (3%) and other production taxes[1] (CP)	26.9	9.6/CP tonne	45/CP oz
(K) Total Cash Costs (CP): (I) + (J)	**373.0**	**133.2/CP tonne**	**624/CP oz**

Note: Lettering continued from Table 22-2

22.4 Taxes

The Company currently has approximately C$690 million, Table 22-5, of tax-deductible pools, tax losses, and tax credits (tax loss pools) available for deduction at the potential commencement of CP. Application of these tax loss pools is estimated to result in the payment of no income taxes against the conceptual LOM net income from the Project. In the conceptual LOM plan, Rubicon will utilize approximately C$169 million of federal tax loss pools. The New PEA estimates that the application of tax loss pools improves conceptual LOM free cash flow by approximately C$95.5 million. It is estimated that Rubicon would have C$521 million of unused federal tax loss pools at the end of the conceptual LOM.

Table 22-5: Tax Loss Pools

Existing Tax Loss Pools	C$690
Estimated Remaining Tax Loss Pools after Conceptual LOM	C$521

22.5 Net Present Value

The 40.2% after-tax IRR estimated for the Project is a product of, among other things, significant sunk capital, application of tax loss pools, new and fully operational infrastructure and equipment in place, and a shortened timeline to cash flow from operations.

The after-tax NPV at a 5% discount rate from Pre-CP through to completion of conceptual LOM is estimated at C$135 million and the IRR is estimated at 40.2%, with a payback of 3.9 years from start of production (as summarized in Table 22-6).

Table 22-6: PEA Economic Results

After-Tax Internal Rate of Return (IRR) (%)	40.2%
After-Tax Net Present Value[1] (NPV5%)	C$135.2 million
After-Tax free cash flow potential	C$191.5 million

Note: [1]Calculated at a 5% discount rate.

The Project's financial model is summarized in Table 22-7.

Table 22-7: PEA Conceptual LOM Cash Flow Model

Conceptual LOM Summary	Unit	Summary of Averages	Year								
			-2	-1	1	2	3	4	5	6	7
Macro Assumptions											
Gold Price	US$/oz		1,325	1,325	1,325	1,325	1,325	1,325	1,325	1,325	-
C$/US$		0.7519	0.7519	0.7519	0.7519	0.7519	0.7519	0.7519	0.7519	0.7519	-
Mill Feed											
Throughput	t/d	1,370	133	670	1,225	1,516	1,691	1,699	1,577	823	-
Ore Tonnage	tonnes	3,045,196	4,000	241,333	440,942	545,907	608,731	611,763	567,823	24,697	-
Diluted Grade	g/t	5.31	4.06	5.91	5.67	4.91	4.71	5.17	5.87	7.38	-
Mill Recovery	%	95.0%	95%	95%	95%	95%	95%	95%	95%	95%	0%
Payable Gold											
Gravity	oz	223,411	224	19,713	34,568	37,025	39,587	43,683	46,093	2,518	-
CIL	oz	270,172	271	23,839	41,803	44,775	47,873	52,826	55,740	3,045	-
Total Payable Gold	**oz**	**493,583**	**496**	**43,551**	**76,371**	**81,800**	**87,460**	**96,510**	**101,833**	**5,563**	**-**
Post Tax Cash Flow											
Net Revenue	**C$ '000**	**868,442**	**872**	**76,627**	**134,371**	**143,923**	**153,882**	**169,806**	**179,172**	**9,788**	**-**
Royalties and other production taxes	C$ '000	(29,907)	(78)	(2,924)	(4,656)	(4,943)	(5,241)	(5,719)	(6,000)	(346)	-
Net Revenue Less Royalty	**C$ '000**	**838,534**	**794**	**73,703**	**129,715**	**138,981**	**148,641**	**164,087**	**173,171**	**9,442**	**-**
Mining costs	C$ '000	(268,404)	(4,075)	(25,388)	(41,326)	(45,975)	(49,313)	(49,903)	(49,773)	(2,651)	-
Milling costs	C$ '000	(99,584)	(411)	(11,264)	(15,390)	(17,479)	(18,505)	(18,161)	(17,398)	(976)	-
Site G&A	C$ '000	(23,824)	(1,642)	(2,898)	(3,476)	(3,779)	(3,961)	(3,970)	(3,843)	(255)	-
Mining Tax	C$ '000	-	-	-	-	-	-	-	-	-	-
Total Operating Cost		**(391,812)**	**(6,128)**	**(39,550)**	**(60,193)**	**(67,233)**	**(71,778)**	**(72,033)**	**(71,014)**	**(3,882)**	**-**
Operating Cash Flow	**C$ '000**	**446,723**	**(5,334)**	**34,153**	**69,523**	**71,747**	**76,863**	**92,053**	**102,158**	**5,561**	**-**
Mine Development & U/G Infrastructure	C$ '000	(129,956)	(14,227)	(28,993)	(21,590)	(22,177)	(21,776)	(15,533)	(5,660)	-	-
Equipment	C$ '000	(69,917)	(6,710)	(10,140)	(11,241)	(10,227)	(11,569)	(10,118)	(8,930)	(982)	-
Surface & Mill Infrastructure	C$ '000	(29,309)	(20,639)	(2,145)	(2,525)	(1,500)	(2,500)	-	-	-	-
Income tax	C$ '000	-	-	-	-	-	-	-	-	-	-
Closure Costs	C$ '000	(7,700)	-	-	-	-	-	-	-	(7,250)	(450)

Conceptual LOM Summary	Unit	Summary of Averages	Year								
			-2	-1	1	2	3	4	5	6	7
Contingency (15%)	C$ '000	(18,351)	(7,036)	(11,315)	-	-	-	-	-	-	-
Total Capital Costs		**(255,233)**	**(48,611)**	**(52,594)**	**(35,355)**	**(33,904)**	**(35,845)**	**(25,651)**	**(14,590)**	**(8,232)**	**(450)**
Cash Flow Before Financing Activities	**C$ '000**	**191,489**	**(53,946)**	**(18,441)**	**34,167**	**37,843**	**41,017**	**66,402**	**87,567**	**(2,671)**	**(450)**
Change in Working Capital	C$ '000	(0)	(693)	2,135	2,232	(84)	262	1,295	908	(5,724)	(330)
Post Tax Cash Flow	**C$ '000**	**191,489**	**(54,639)**	**(16,306)**	**36,399**	**37,759**	**41,279**	**67,697**	**88,475**	**(8,395)**	**(780)**
Post Tax NPV Unlevered Cash Flows	5%	**$135,233**									
Post Tax IRR		**40.2%**									
Unit Cost Metrics - 2020 Preproduction EXCLUDED from calculation											
Operating Cash Cost	C$/t	$128.67	-	-	137	123	118	118	125	157	-
Operating Cash Cost (C1)	US$/oz	$623.93	-	-	638	663	662	606	569	571	-
AISC*	US$/oz	$881.55	-	-	987	975	970	806	676	1,684	-
AIC*	US$/oz	$1,031.21	-	-	1,183	1,158	1,141	961	823	4,375	-

Note: *Exempt of Corporate Costs
#Year -2 (stub year) includes only 8 months of construction. Year 6 (stub year) includes only 1 month of operation.

22.6 Sensitivity Analysis

Key economic risks for the Project were examined by running sensitivity analyses. IRR and NPV comparative results are shown in Table 22-8 through Table 22-13.

The PEA base case values are: NPV ($135 million), IRR (40.2%), gold price (US$1,325/oz), foreign exchange (FX) (0.7519), throughput (1,370 t/d), LOM grade (5.31 g/t), OPEX ($128.67/t), and PEA Capex ($255 million) are highlighted by a box outline in the following tables.

Table 22-8: IRR Sensitivity to Gold Price and Exchange Rate

IRR		Gold Price				
		$1,100	$1,200	$1,325	$1,400	$1,500
US$/C$ FX	**0.83**	(4.4%)	9.3%	24.4%	32.8%	43.5%
	0.81	(0.4%)	13.1%	28.2%	36.6%	47.4%
	0.79	4.6%	18.0%	33.1%	41.6%	52.6%
	0.77	8.2%	21.5%	36.6%	45.3%	56.4%
	0.75	11.7%	25.0%	40.2%	48.9%	60.2%
	0.73	16.3%	29.5%	44.8%	53.7%	65.2%
	0.71	20.6%	33.9%	49.4%	58.4%	70.1%

Table 22-9: NPV Sensitivity to Gold Price and Exchange Rate

NPV ($ millions)		Gold Price				
		$1,100	$1,200	$1,325	$1,400	$1,500
US$/C$ FX	0.83	(30)	15	71	105	150
	0.81	(18)	28	86	121	167
	0.79	(1)	46	106	142	189
	0.77	11	60	121	157	206
	0.75	24	73	135	173	223
	0.73	40	91	155	194	245
	0.71	57	109	175	215	268

Table 22-10: IRR Sensitivity to Grade (g/t) and Throughput (t/d)

IRR		Grade (g/t)						
		4.50	4.75	5.00	5.31	5.50	5.75	6.00
(t/d)	1,200	(5.9%)	2.8%	10.9%	20.0%	25.3%	32.0%	38.6%
	1,300	6.9%	15.2%	23.0%	32.3%	37.6%	44.4%	51.2%
	1,370	14.9%	23.0%	30.9%	40.2%	45.7%	53.0%	59.9%
	1,500	28.7%	37.0%	45.1%	54.6%	60.5%	68.1%	75.5%
	1,600	38.5%	47.1%	55.3%	65.6%	71.5%	79.4%	87.2%
	1,700	48.2%	57.0%	65.6%	75.8%	82.3%	90.5%	98.7%
	1,800	57.5%	66.4%	75.6%	86.5%	93.0%	101.4%	109.8%

Table 22-11: NPV Sensitivity to Grade (g/t) and Throughput (t/d)

NPV ($ millions)		Grade (g/t)						
		4.50	4.75	5.00	5.31	5.50	5.75	6.00
(t/d)	1,200	(35)	(7)	20	54	75	101	129
	1,300	7	36	66	102	125	153	183
	1,370	35	65	97	135	159	191	221
	1,500	88	122	156	198	224	258	292
	1,600	128	165	201	247	274	310	346
	1,700	170	208	247	293	324	362	398
	1,800	210	251	293	343	373	408	439

Table 22-12: IRR Sensitivity to OPEX and CAPEX

IRR		OPEX per Tonne		
		$110.00	$128.67	$150.00
CAPEX	-25%	80.9%	65.6%	48.7%
	-15%	67.6%	54.0%	38.8%
	$255	51.6%	40.2%	27.0%
	15%	39.6%	29.2%	17.1%
	25%	32.8%	23.0%	11.6%

Table 22-13: NPV Sensitivity to OPEX and CAPEX

NPV (millions)		OPEX per Tonne		
		$110.00	**$128.67**	**$150.00**
CAPEX	**-25%**	235	189	137
	-15%	213	168	116
	$255	180	135	84
	15%	148	103	51
	25%	126	81	29

23 ADJACENT PROPERTIES

There are no adjacent properties relevant to this Technical Report.

24 OTHER RELEVANT DATA AND INFORMATION

The following risks and opportunities were identified.

24.1 Risks

The PEA does demonstrate that the Project has the potential to be technically and economically viable. Some risks have been identified within the recommendations.

The mining of the Phoenix deposit is technically simple because of the vertical nature of the deposit and shaft plus single ramp direct access to the underground mine. The processing plant uses well proven technologies to achieve gold recoveries. Infrastructure requirements are low risk as the mine is in an area of other economic activity with many regional services.

The main risks to project success are summarized below:

- Until the pastefill system is commissioned, there is some uncertainty with regards to the mechanical and electrical components
- There is a risk that the tailings will require classification prior to effective use as paste underground. This will be resolved in the next phase of study
- Adequate electricity supply from Hydro One is a concern in Northwestern Ontario and while capacity consultations with Hydro One suggest adequate power will be available for this PEA scenario it remains an area of uncertainty for the Project
- While not perceived as a threat, permits for operations and environmental work to achieve 1,800 t/d are not confirmed
- Should mining of new satellite deposits be contemplated in the future to increase throughput, these will require additional permitting
- Availability of miners at compensation levels contemplated within this PEA is subject to change in market demands.

24.1.1 Mineral Resource Estimate

The following risks may affect the Mineral Resource Estimate:

- Revision of geological interpretation impacting rock type and volume
- Possible variation in continuity and grade of gold mineralization, especially regarding outlier grades
- Assumptions for criteria used to determine reasonable prospect for eventual economic extraction
- Poor ground control may reduce recovery within Ultramafic unit.

24.1.2 Mining

Risks that may affect mining are as follows:

- Ramp development is critical to accessing high-grade material and new stoping areas well in advance of production demands; while ramp development has been scheduled at a realistic rate sustainable over the long term (i.e., 3.1 m/d per crew), any significant delays or improvements during initial development are likely to have a direct impact on gold production.
- While stope dimensions, dilution, and level intervals are conservative, based on stope stability and ELOS assessments, it has been assumed that ground conditions will be similar throughout most of the mine; additional geotechnical investigation is recommended in a feasibility level of study.
- Although installed and partially commissioned, the paste fill fabrication and distribution system need to be fully commissioned; difficulties meeting backfilling schedules could impede the production anticipated in the mine plan as well as potentially impact the need for an additional lift at the TMF.
- Labour accounts for 50% to 60% of the total mining costs; therefore, the mining costs are sensitive to changes in the labour market.

24.1.3 Processing

Risks that may affect process are as follows:

- The plant was commissioned during the 2018 bulk mining work, and risks were mitigated effectively at that time. However, variability of the deposit implies some level of process risk for every project.
- The selected gravity and leaching processes are robust, and little variation is expected in the overall recovery; however, the recovery split between the two process steps could vary
- The proposed gravity concentrate leaching process is robust, but work to date on regrinding, concentrate filtration, and cyanide destruction may evolve; future work may impact the estimated capital and operating costs of these sections.

24.2 Opportunities

The following sections highlight the main opportunities to increase Project economics and reduce identified risks. Each section, while as comprehensive as is possible at this time, is not considered exhaustive as more opportunities can arise when considered from alternative perspectives.

24.2.1 Mineral Resource Estimate

In TMAC's opinion, Rubicon can potentially improve or increase the 2019 Mineral Resource estimate with the following recommendations:

- Target infill and step-out drilling in areas containing Inferred Mineral Resources (about 40 m centres drill spacing) to upgrade Mineral Resource classification and Exploration Targets (>80 m centres) to convert to Mineral Resource estimates.
- Drilling is proposed from the 244 m Level, 610 m Level, and 685 m Level; the Exploration Targets could potentially contain[3] between 0.9 to 1.2 Mt, with potential grades between 5.0 and 7.0 g/t Au.
- Evaluate McFinley Deposit and close proximity targets (specifically PEN Zone—within 500 m of current workings) which could potentially be included in a future Mineral Resource estimate.

24.2.2 Mining

- Higher development advance rates. The conceptual mine plan is based on historical, long term, economic, and sustainable development productivities, typically 3 to 4 m/d in single-heading waste development; there are many instances of higher advance rates being achieved and sustained, and any such improvements will provide flexibility in scheduling options.
- Cut-off grade optimization. The conceptual mine plan currently assumes a break-even cut-off grade based on mining, milling, and G&A costs that are all subject to improvements with technological advancements.
- Metal prices and exchange rates. Cut-off grades are heavily skewed by foreign exchange rates and precious metal prices, both of which currently reflect historical strength in the North American economy; sensitivity to a change in either gold price or exchange rate could translate to some fraction of hundreds of TMAC-defined marginal stopes being upgraded to potentially mineable Mineral Resources.
- The increasing availability and implementation of battery-powered equipment provides opportunity to reduce primary ventilation, which is a major consumer of power in the mine; the mine plan is based on leased diesel equipment, especially haul trucks, that could be quickly transitioned to battery-powered equipment.

[3] According to NI 43-101 Section (2)(a), the potential quantity and grade of the Exploration Target is conceptual in nature and there has been insufficient exploration to define a Mineral Resource estimate. It is uncertain if further exploration will result in the Exploration Target material being delineated as a Mineral Resource. The Exploration Target has been defined based on blocks estimated using >80 m drill centres, lower confidence based on decreased data density, and increased cut-off grade with depth.

- MBRM techniques form about 17% of the PEA's production schedule. MBRM does not require top sill or intermediary sill development; if proven appropriate, this method could replace some formerly classified LH stopes, which would increase the rate at which stopes could be put into production.
- The raise method is limited, in this PEA, to stope widths of 1.2 m, but with experience, could evolve to allow narrower stopes, thereby transitioning some fraction of more the more-than-300 existing marginal stopes TMAC has already identified into the minable stope category.
- Optimizing the mining sequence has the potential to reduce the amount of remote mucking, thereby providing productivity and cost benefits; the mine plan incorporates both top-down and bottom-up mining sequences depending on the mining method and nearby proximity to other stopes.
- Opportunities exist to aggregate smaller stopes into larger units that span multiple levels, and according to local geotechnical limits, to improve productivity.

24.2.3 Processing

- The mill is currently designed, constructed, and commissioned to process up to 1,250 t/d. However, the current permit limits capacity to 1200 t/d. The PEA includes changes to the retention cells so that 1,800 t/d can be produced (the mill layout allows for the addition of a second ball mill, a second hydrocyclone cluster, a pre-crushing unit, and a second stripping column if required in the future to achieve 2,500 t/d).
- The mill would benefit from having dedicated assay services available at its metallurgical lab in Balmertown, or on site, to provide timely feedback to the operators; the nominal 24-hour turnaround time for off-site assays is acceptable for the short term, but does not allow for efficient troubleshooting.
- Evaluate ore-sorting technology to potentially increase mill head grade and reduce tonnes to be milled.

25 INTERPRETATION AND CONCLUSIONS

25.1 Interpretations

Rubicon plans to release an updated Mineral Resource estimate in Q4/2019 incorporating drilling results from 2019 with the goal of further increasing the Measured and Indicated Mineral Resource estimate above the current estimate of 589,000 contained ounces.

TMAC concludes that Rubicon should, following completion of the new updated Mineral Resource estimate, initiate further feasibility work, with the intention of progressing the Project to a production decision. The Project's conceptual LOM plan demonstrates robust economics, with an after-tax NPV of C$191.5 million, an after-tax internal rate of return (IRR) of 40.2, and a 3.9-year payback period using a gold price of US$1,325/oz.

TMAC interprets the mineralization at Phoenix to be potentially open to underground extraction using LH, Uppers and Raise Mining, and C&F mining methods. At a 3.5 g/t Au mining cut-off, the conceptually minable Mineral Resource is estimated to contain estimated Measured and Indicated Mineral Resources of 2,289,000 tonnes at a grade of 7.11 g/t Au plus estimated Inferred Mineral Resources estimated at 2,038,000 tonnes at a grade of 7.39 g/t Au.

Following mining accessibility investigations and dilution and recovery calculations, the conceptual LOM demonstrates the potential to recover 68.2% of the estimated Measured and Indicated Mineral Resource (1,561,000 tonnes, at a grade of 5.23 g/t), plus 72.8% of the estimated Inferred Mineral Resource (1,484,000 tonnes at 5.39 g/t).

The Phoenix Gold Project has economic potential as an underground mining operation, utilizing an existing and permitted processing plant to produce gold doré. The PEA identifies potential mill feed estimated to be 3.045 Mt, inclusive of mining dilution and loss factors, averaging an estimated 5.31 g/t Au and estimates total conceptual gold production of 493,583 oz, of which 44,047 oz of conceptual production is anticipated during the Pre-CP period.

The Project has an estimated Pre-CP capital cost during the first 20 months, of CAN$101.2 million. During that same 20-month period, potentially C$77.5 million in proceeds from the sale of pre-CP gold ounces results in net Pre-CP capital requirements of C$72.4 million.

The results of the PEA demonstrate that the Phoenix Gold Project has the potential to be technically and economically viable.

26 RECOMMENDATIONS

26.1 Mineral Resources

TMAC recommends the following:

- Target infill and step-out drilling in areas containing Inferred Mineral Resources (about 40 m centres drill spacing) to upgrade Mineral Resource classification and convert Exploration Targets (>80 m centres) to Mineral Resources.
- Drilling is proposed from the 244 m Level, 610 m Level, and 685 m Level; the Exploration Targets could potentially contain[4] between 0.9 to 1.2 Mt, with potential grades between 5.0 and 7.0 g/t Au.
- Evaluate McFinley Deposit and close proximity targets (specifically PEN Zone—within 500 m of current workings) which could potentially be included in a future Mineral Resource estimate.
- Update the Mineral Resource estimate to incorporate drilling completed up to third quarter 2019; this would also include a re-evaluation of the geological model and confirmation of structural model (Table 26-1).

Table 26-2 summarizes the recommended exploration budget of $2,460,136 for the F2 Gold Zone opportunity to expand the 2019 Mineral Resource Estimate. The combined McFinley Deposit and PEN Zone exploration budget is included in Table 26-3.

Table 26-1: Recommended Additional Resource Update Costs (C$)

Re-assessment of Zone 2 Geological Model	$25,000
Incorporating New Drilling and Update Mineral Resource Grade Model and Classification	$75,000
McFinley Zone—exploration program and Mineral Resource estimate	$1,100,000
Close Proximity Target—Pen Zone exploration program and Mineral Resource estimate	$500,000

[4] According to NI 43-101 Section (2)(a), the potential quantity and grade of the Exploration Target is conceptual in nature and there has been insufficient exploration to define a Mineral Resource estimate. It is uncertain if further exploration will result in the Exploration Target material being delineated as a Mineral Resource. The Exploration Target has been defined based on blocks estimated using >80 m drill centres, lower confidence based on decreased data density, and increased cut-off grade with depth.

Table 26-2: 2019 Exploration Budget—F2 Gold Zone

Items	Levels	Metres	Units	Cost/Unit (C$)	Total Cost (C$)	Grand Total (C$)
Drilling	244 m Level	825	-	90	74,250	
	610 m Level	12,475	-	90	1,122,750	
	685 m Level	7,830	-	90	704,700	1,901,700
Assaying	244 m Level	-	413	40	16,500	
	610 m Level	-	6,238	40	249,500	
	685 m Level	-	3,915	40	156,600	422,600
Consumables	244 m Level	-	-	-	5,304	
	610 m Level	-	-	-	80,196	
	685 m Level	-	-	-	50,336	135,836
						2,460,136

Table 26-3: 2019 Exploration Budget—McFinley Deposit and PEN Zone

Items	Levels	Metres	Units	Cost/Unit (C$)	Total Cost (C$)	Grand Total (C$)
Drilling	685 m Level (McFinley)	1,170	-	90	105,300	-
	244 m Level (PEN Zone)	2,700	-	90	243,000	348,300
Assaying	685 m Level (McFinley)	-	585	40	23,400	-
	685 m Level					
	244 m Level (PEN Zone)	-	1,350	40	54,000	77,400
Consumables	685 m Level (McFinley)	-	-	-	7,521	
	244 m Level (PEN Zone)	-	-	-	17,357	24,879
					-	**450,579**

26.2 Mine and Mill

Given the positive results of the PEA and the potential for enhancement, TMAC recommends the following for the mine and mill:

- Completing a new NI 43-101 Mineral Resource model to further update the existing Mineral Resource estimate
- Confirming/updating structural geology model by incorporating 2018–2019 drilling
- Initiate feasibility level studies including further engineering and design work
- Perform tailings design, includes comparison of phase 2 and 3 dams, and costing

- Hydrological study
- Geotechnical modelling
- Testing the MBRM method at Phoenix
- Remain in compliance with environmental reporting, monitoring, and auditing during future exploration and development
- Continue to advance permitting amendments including Mine Closure Plan and ECA for production, ramp, and development advancements.

Estimated costs for -non-exploration recommendations are summarized in Table 26-4.

Table 26-4: Recommended Non-exploration Work Costs (C$)

Engineering studies and design work, including feasibility studies	$1,250,000
Confirm/update pastefill strength tests	$50,000
Confirm/update structural geology model	$50,000
Tailings design, includes comparison of phase 2 and 3 dams, and costing	$300,000
Geotechnical modeling	$250,000
Procurement work to improve accuracy of CAPEX and OPEX	$200,000
Test the MBRM method at Phoenix	$2,500,000
Advance permitting amendments towards 1,800 t/d	$300,000
	$4,950,000

27 REFERENCES

AMC Mining Consultants (Canada), 2009. Phoenix Gold Project – Provisional Ground Support Standards; Internal Report for Rubicon Minerals Corporation, 13 pages.

AMC Mining Consultants (Canada), 2011. F2 gold deposit - Phoenix Gold Project. Bateman Township. Red Lake, Canada. Technical report prepared for Rubicon Minerals Corporation. August 8, 2011. Available at www.sedar.com.

AMC Mining Consultants (Canada), 2013. 712022 Rubicon Phoenix PEA Optimization: Preliminary Stope Stability Assessment. Internal report prepared for Rubicon Minerals Corporation.

AMC Mining Consultants (Canada), 2014. Rubicon Crown Pillar Assessment, 10 December 2014. Internal report prepared for Rubicon Minerals Corporation.

AMEC Foster Wheeler Environment and Infrastructure, 2016, January, Tailings Management Facility North-East Stage 1 Dam and Emergency Spillway Design Report, Final, Prepared for Rubicon Minerals Corporation.

AMEC Foster Wheeler Environment and Infrastructure, 2016, March, Tailings Management Facility Stage 1 As-Built Report, prepared for Rubicon Minerals Corporation

Andrews, A.J., Hugo, H., Durocher, M., Corfu, F., and Lavigne, M., 1986. The anatomy of a gold-bearing greenstone belt: Red Lake, North-Western Ontario; Proceedings of GOLD '86, An International Symposium on the Geology of Gold, (ed.) A.J. Macdonald, Consult International Inc., Toronto, Ontario, p. 3-22.

Card, K.D. and Poulsen, K.H., 1988. Geology and mineral deposits of the Superior Province of the Canadian Shield. In: Geology of the Precambrian Superior and Grenville Provinces and Precambrian Fossils in North America. Geological Survey of Canada; Geology of Canada, Number 7, pages 15-232.

Dubé, B., Balmer, W., Sanborn-Barrie, M., Skulski, T., and Parker, J., 2000. A preliminary report on amphibolite-facies, disseminated-replacement style preliminary report on amphibolite-facies mineralization at the Madsen gold mine, Red Lake, Ontario. Geological Survey of Canada, Current Research 2000-C17, 12p.

Dubé, B., Williamson, K., and Malo, M., 2001. Preliminary report on the geology and controlling parameters of the Goldcorp Inc. High Grade Zone, Red Lake mine, Ontario. Geological Survey of Canada, Current Research 2001-C18, 13p.

Dubé, B., Williamson, K., and Malo, M., 2003. Gold mineralization within the Red Lake mine trend: example from the Cochenour-Willans mine area, Red Lake, Ontario, with new key

information from the Red Lake mine and potential analogy with the Timmins camp; Geological Survey of Canada, Current Research 2003-C21, 15p.

Dubé, B., Williamson, K., McNichol, V., Malo, M., Skulski, T., Twomey, T., and Sanborn-Barrie, M., 2004. Timing of Gold Mineralization at Red Lake, Northwestern Ontario, Canada: New Constraints from UPb Geochronology at the Goldcorp High-Grade Zone, Red Lake Mine, and the Madsen Mine. Economic Geology. 99. 1611-1641. 10.2113/99.8.1611.

Forbes and Associates, 2011. A Desktop Review of Traditional Land Use (TLU) For the Red Lake Area with Specific Interest to the Phoenix Gold Project Location. Internal report prepared for Rubicon Minerals Corporation.

G&T Metallurgical Services, 2010. Preliminary Evaluation of the Phoenix Project, Report KM2550, July 2010.

G&T Metallurgical Services, 2011. Bulk Sample Grade Assessment – Phoenix Gold Project, Report KM2814, October 2011.

G&T Metallurgical Services, 2012. Analysis of Gold from Bulk Samples – Phoenix Gold Project, Report KM3264, March 2012.

GeoEx Limited, 2011a. Mineral Resource and Geological Potential Estimates, F2 gold system – Phoenix Gold Project. Technical Report prepared for Rubicon by George, P.T., dated January 11, 2011. Available at www.sedar.com.

GeoEx Limited, 2011b. Mineral Resource and Geological Potential Estimates, F2 gold system – Phoenix Gold Project. Technical Report prepared for Rubicon by George, P.T., dated 11 April 2011. Available at www.sedar.com.

Golder Associates Ltd. (DeWolfe, J., 2017). Phoenix Gold Project – Supplemental Core Logging Protocol; Report Number: 1671445-008-R [In-house draft report, March 2017.]

Golder Associates Ltd., 2017. Phoenix Gold Project: 2017 Updated Structural Geology Report. Report No. 1671445-008-R-Rev0, dated February 9, 2018. 44 p.

Golder Associates Ltd., 2018a. Phoenix Gold Project: 2017 Updated Structural Geology Report, Report Number: 1671445-008-R-Rev0, dated February 8, 2018, 387 p.

Golder Associates Ltd., 2018b. National Instrument 43-101 Technical Report for the Rubicon Phoenix Gold Project. Golder Project No. 1671445-Rev B. June 13, 2018. 206 pages.

Golder Associates Ltd., 2019. Phoenix Gold Project Bulk Sample Gold Reconciliation. Golder Project No. 18105476, dated: January 7, 2019.

Groves, D. I., Goldfarb, R. J., Gebre-Mariam, M., Hagemann, S. G., Robert, F., 1998. Orogenic gold deposits: A proposed classification in the context of their crustal distribution and relationship to other gold deposit types; Ore Geology Reviews, V. 13, p. 7-27.

Harris, J.R., Sanborn-Barrie, M., Panagapko, D.A., Skulski, T., and Parker, J.R., 2006. Gold prospectivity maps of the Red Lake greenstone belt: application of GIS technology. Canadian Journal of Earth Sciences, Volume 43, pages 865-893.

Hogg, G. M., 2002b. Technical Report on the McFinley Mine Property of Rubicon Minerals Corporation, Red Lake, Ontario. 15 October 2002. Internal report prepared for Rubicon Minerals Corporation.

Hogg, G.M. & Associates, 2003: Exploration Activities of Rubicon Minerals Corporation on the McFinley Property, Red Lake, Ontario by D.M. Rigg and G.M. Hogg, dated May 12, 2003; 71 p.

Hogg, G.M., & Associates Ltd., 1983. A report on the McFinley Red Lake Gold Property of Sabina Industries Ltd. and McFinley Mines Ltd., Bateman TWP., Ontario.

Hogg, G.M., 2002a. Technical Report on the McFinley Gold Project of Rubicon Minerals Corporation, Red Lake, Ontario. 17 May 2002. Internal report prepared for Rubicon Minerals Corporation.

JKTech Pty, 2011. JKDW Test Report on Two Samples from Rubicon Phoenix Project, JKTech Job No. 11017/P4, March 2011.

Journel, A.G. and Huijbregts, C. J, 1978. Mining Geostatistics, Academic Press.

Knight Piésold Ltd, 2015, March 5, Tailings and Water Management Design Summary, Rev 1, Prepared for Rubicon Minerals

Marsden J.O., & House, D. I., 2009. The Chemistry of Gold Extraction 2nd Ed. Littleton, Colorado: The Society of Mining, Metallurgy, and Exploration, Inc.); p 333.

McNutt, A., 2019. Mill Preparation, Commissioning and Startup for Phoenix Project Process Facility Bulk Sample Processing. Prepared for Rubicon Minerals Corp by Adrian C. McNutt Consulting Metallurgical Engineer, February 28, 2019.

Nordmin, 2014. Phoenix Red Lake Paste System Design Assistance, Reticulation Design review. Prepared for Rubicon Minerals Corporation by AMC.

Novatox, 2011A. Risk Assessment for the Phoenix Advanced Exploration Project. Prepared for Rubicon Minerals Corporation. Novatox Project 10846.

Novatox, 2011B. Assessment of Risk to Humans and the Environment from Phoenix Project Effluent. Internal report prepared for Rubicon Minerals Corporation.

Ontario Geological Survey 2011. 1:250 000 scale bedrock geology of Ontario; Ontario Geological Survey, Miscellaneous Release-Data 126-Revision 1.

Hydro One Networks Inc., 2015. Customer Service Contract & Capital Cost Recovery Agreement between Rubicon Minerals Corporation and Hydro One Networks Inc. for Phoenix Mine Expansion,. March 6, 2015, 16 p.

Parrott, D.F., 1995. The Gold Mines of Red Lake, Ontario, Canada. Dirksen Printers, Steinbach, Manitoba, 256 p.

Pirie, J., 1981. Regional geological setting of gold deposits in the Red Lake area, northwestern Ontario. Miscellaneous Paper. Ontario Ministry of Northern Development and Mines, Ontario Geological Survey.

Ross Archaeological Research Associates (Ross Associates), 2010. Stage 1 Archaeological Assessment and Property Inspection, Rubicon Minerals Phoenix Mine Property. Internal report prepared for Rubicon Minerals Corporation.

Rogans, J., 2012. Activated Carbon in Gold Recovery, Kemit (Pty) Ltd., June 22, 2012, 34 p.

Sanborn-Barrie, M., Skulski, T., and Parker, J., 2004. Geology, Red Lake Greenstone Belt, western Superior Province, Ontario. Geological Survey of Canada, Open File 4594.

Smee and Associates Consulting Ltd., 2009a. Results of a Quality Control Data Review, Phoenix Project, Red Lake, Ontario. Internal report prepared for Rubicon Minerals Corporation.

Smee and Associates Consulting Ltd., 2009b. Results of an Audit of the SGS Laboratory, Red Lake, Ontario. Internal report prepared for Rubicon Minerals Corporation.

Smee and Associates Consulting Ltd., 2011: Results of an Audit of the SGS Laboratory, Red Lake, Ontario. Internal report prepared for Rubicon Minerals Corporation in October 2011.

Soutex Inc., 2013. Phoenix Gold Scoping Study, Bulk Sample Processing. Phoenix Gold Scoping Study (draft version), Project 2220; internal report prepared for Rubicon Minerals Corporation, July 11, 2013. 181 p.

SRK, 2012. Interim Report on 3D Structural, Lithological and Alteration Modelling of the F2 Core Zone, Red Lake, Ontario, prepared by SRK Consulting (Canada) Inc. (SRK) in December 2012

SRK, 2013a. Preliminary Geotechnical Assessment of the Phoenix Gold Project. Report prepared for Rubicon Minerals Corporation, July 2013.

SRK, 2013b. Preliminary Economic Assessment for the F2 gold deposit, Phoenix Gold Project, Red Lake, Ontario; Report (amended and restated) Prepared for Rubicon Minerals Corporation, Effective date: June 25, 2013, Issue date: August 9, 2013. Amended and Restated: February 28, 2014. 263 p. Available at www.sedar.com.

SRK, 2016. Technical Report for the Phoenix Gold Project, Red Lake, Ontario; Report for Rubicon Minerals Corporation, Effective date: January 11, 2016, Issue date: February 25, 2016. 221 p. Available at www.sedar.com.

TVI Pacific Inc. 2013, TVI Pacific Reports Pilot Plant Metallurgical Test Work Results for its Balabag Gold-Silver Project, Philippines, Press Release, Available at www.tvipacific.com.

Vancouver Petrographics Ltd., 2008. Petrographic Report on Samples: CRL Series: 16511, 16516, 14735, 20340, 15422, 9324 (3 sections), 19137, 19156. 19145 (2 sections), 19331 (3 sections); Internal report prepared for Rubicon Minerals Corporation. 48 p.

Internet References

Data products, 2016 Census. https://www12.statcan.gc.ca/census-recensement/2016/dp-pd/index-eng.cfm

Ontario Ministry of Northern Development and Mines (OMNDM), 2013. MDI File MDI52N04SE00005 report. http://www.geologyontario.mndm.gov.on.ca/dtSearch/dtisapi6.dll?cmd=getdoc&DocId=497596&Index=%2a%7baa6fcec0bc49cb0cca8bfcb488289a8a%7d%20MDI&HitCount=3&hits=5+17+c13+&SearchForm=%2findex%2ehtml.

28 CERTIFICATE OF AUTHORS

28.1 Tim Maunula, P.Geo.

I, Tim Maunula, P.Geo., of Chatham, Ontario, a Qualified Person (QP) of this Technical Report titled "National Instrument 43-101 Technical Report for the Phoenix Gold Project Preliminary Economic Assessment, Cochenour, Ontario," dated September 21, 2019, do hereby certify the following statements:

I am Principal Geologist of T. Maunula & Associates Consulting Inc., 15 Valencia Drive, Chatham, Ontario, N7L 0A9, Canada.

I graduated with a H.B.Sc. degree in Geology from Lakehead University in 1979. In addition, I have obtained a Citation in Geostatistics from the University of Alberta in 2004.

I am a member of the Association of Professional Geoscientists of Ontario (Registration Number 1115).

I have worked as a Geologist for a total of 40 years since my graduation from university.

I have read the definition of QP set out in NI 43-101 and certify that by reason of education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43-101.

I am responsible for Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 14 and portions of 25 and 26 of this Technical Report.

I have visited the Property between 2017 and 2019 with the most recent on February 18, 2019.

I have prior involvement as the author of portions of the 2018 and 2019 National Instrument 43-101 Technical Report for the Rubicon Phoenix Gold Project.

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

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

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

I have read National Instrument 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with that instrument and form.

Dated this 23rd day of September 2019 in Chatham, Ontario.

"Original Signed and Sealed"

Tim Maunula, P.Geo.

28.2 Andrew MacKenzie, P.Eng.

I, Andrew MacKenzie, P.Eng., of Keswick, Ontario, a Qualified Person (QP) of this Technical Report titled "National Instrument 43-101 Technical Report for the Phoenix Gold Project Preliminary Economic Assessment, Cochenour, Ontario," dated September 21, 2019, do hereby certify the following statements:

I am an Independent Professional Mining Engineer with an address at 10 Joliette Place, Keswick, Ontario, Canada, L4P 3Y9.

I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument"). My qualifications are as follows:

- Mining engineer graduate from Queen's University in Kingston, Ontario in 1994.
- I have met the Professional Engineers of Ontario licensure requirements (License No. 90470477) and, have been practicing professional engineering since 1996 as follows:
  - Mine Planner – INCO Ltd. (1994–1999)
  - Paste Consultant - Paste Systems Inc. (1999–2004)
  - Senior Estimator/ Manager of Technical Services, Dynatec Mine Contractors (2004-2009)
  - Principal Mine Engineer, Mine Designer and Reserve Estimator, SRK (2009-2010)
  - Divisional Manager of Mining, Tetra Tech Engineering (2010–2014)
  - Manager of Mining – WorlyParsons (2014–2015)
  - Self-Employed Mining Consultant/Mine Costs – Design and Reserve Estimator, 2015–2019.

I have visited the Phoenix Project site in the spring of 2018 and most recently on March 20 to 22, 2019.

I am the author and responsible for Sections 1, 2, 3, 15, 16, 19, 21, 22, 23, 24 and the co-author of Sections 18, 25, and 26.

I am independent of the issuer as described in Section 1.5 of the instrument.

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

I have read the instrument. The parts of the Technical Report for which I am responsible have been prepared in compliance with this Instrument.

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

Dated this 23rd day of September 2019 at Keswick, Ontario.

"Original Signed and Sealed"

Andrew MacKenzie, P.Eng.

28.3 Karlis Jansons, P.Eng.

I, Karlis Jansons, P.Eng., of Toronto, Ontario, a Qualified Person (QP) of this Technical Report titled "National Instrument 43-101 Technical Report for the Phoenix Gold Project Preliminary Economic Assessment, Cochenour, Ontario," dated September 21, 2019, do hereby certify the following statements:

I am an Independent Geotechnical (Tailings) Engineer with an address 2100 Bloor Street West, Suite 6223, Toronto, Canada, M6S 5A5.

I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument"). My qualifications as a qualified person are as follows:

I graduated from the University of Toronto in 1981 in Geological Engineering and Applied Earth Sciences. I have over 38 years of experience in engineering and am registered as a Professional Engineer with Professional Engineers Ontario, registration number 21839501.

I have been to the Phoenix project site and visited on March 20 and 22nd, 2019.

I am responsible for the tailings portion of Section 18.5 Tailings Management Facility

I am independent of the issuer as described in Section 1.5 of the instrument.

I have no prior involvement with the property or any reports associated with it.

I have read the instrument. The parts of the Technical Report for which I am responsible have been prepared in compliance with this Instrument.

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

Dated this 23rd day of September 2019 at Toronto, Ontario.

"Original Signed and Sealed"

Karlis Jansons, P.Eng.

28.4 Peter Broad, P.Eng., FEC., CEng, MIMMM

I, Peter Broad, P.Eng., of London, Ontario, a Qualified Person (QP) of this Technical Report titled "National Instrument 43-101 Technical Report for the Phoenix Gold Project Preliminary Economic Assessment, Cochenour, Ontario," dated September 21, 2019, do hereby certify the following statements:

I am an Independent Mining Engineer with an address at 730 Wonderland Road North (apt505) London, Ontario N6H 4Y9.

I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument"). My qualifications as a qualified person are as follows:

I have been to the Phoenix project site and visited on March 20-22nd 2019

I am responsible for Sections 13 & 17.

I am independent of the issuer as described in Section 1.5 of the instrument.

I have no prior involvement with the property that is the subject of this Technical Report

I have read the instrument. The parts of the Technical Report for which I am responsible have been prepared in compliance with this Instrument.

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

Dated this 23rd day of September 2019 at London, Ontario.

"Original Signed and Sealed"

Peter Broad, P.Eng.

28.5 Charles Tkaczuk, P.Eng.

I, Charles Tkaczuk, P.Eng., of Harriston, Ontario, a Qualified Person (QP) of this Technical Report titled "National Instrument 43-101 Technical Report for the Phoenix Gold Project Preliminary Economic Assessment, Cochenour, Ontario," dated September 21, 2019, do hereby certify the following statements:

I am an Independent Electrical Engineer with an address 9492 Wellington Rd 6, Harriston, Ontario, N0G 1Z0.

I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument"). My qualifications as a qualified person are as follows:

I am a graduate of University of Waterloo, (BSc., 1982).

My relevant experience is a combined 40 years working directly in the mining industry of which 32 years have been as a Professional engineer, supporting the design, construction, maintenance, and operation of mining (open pit and underground) and mineral processing plants.

I have been to the Phoenix project site and visited on March 20-22nd, 2019.

I am responsible for Section 18.1 through 18.4 and 18.6.

I am independent of the issuer as described in Section 1.5 of the instrument.

I have no prior involvement with the Property that is the subject of the Technical Report.

I have read the instrument. The parts of the Technical Report for which I am responsible have been prepared in compliance with this Instrument.

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

Dated this 23rd day of September 2019 at Harriston, Ontario.

"Original Signed and Sealed"

Charles Tkaczuk, P.Eng.

28.6 Ian Horne, B.Sc. Biology

I, Ian Horne, B.Sc., of Oakville, Ontario, a Qualified Person (QP) of this Technical Report titled "National Instrument 43-101 Technical Report for the Phoenix Gold Project Preliminary Economic Assessment, Cochenour, Ontario," dated September 21, 2019, do hereby certify the following statements:

I am an Independent Biologist with an address, 2255 Dunedin Rd., Oakville ON, L6J 5V4

I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument"). My qualifications as a qualified person are as follows:

I have been to the Phoenix project site and visited on March 20 to 22, 2019.

I am responsible for Section 20.

I am independent of the issuer as described in Section 1.5 of the instrument.

My prior involvement with the property that is the subject of the Technical Report is as follows: I have had no prior involvement with the property.

I have read the instrument. The parts of the Technical Report for which I am responsible have been prepared in compliance with this Instrument.

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

Dated this 23rd day of September 2019 at Oakville, Ontario.

"Original Signed and Sealed"

Ian Horne, B.Sc.