Report to:

ATAC Resources Ltd.

Technical Report and Preliminary Economic Assessment for the Tiger Deposit, Rackla Gold Project, Yukon, Canada

Document No. 219242-01-RPT-001

TETRA TECH

Report to:

ATAC RESOURCES LTD.

TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT FOR THE TIGER DEPOSIT, RACKLA GOLD PROJECT, YUKON, CANADA

EFFECTIVE DATE: FEBRUARY 27, 2020

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REVISION HISTORY

REV. NO	ISSUE DATE	PREPARED BY	REVIEWED BY	APPROVED BY	DESCRIPTION OF REVISION
00	2020/04/08	All QPs	All QPs	Steve Ip	Draft report to client
01	2020/04/09	All QPs	All QPs	Steve Ip	Final Report

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GLOSSARY

Units of Measure

annum (year)	a
billion	B
billion tonnes	Bt
billion years ago	Ga
centimetre	cm
cubic centimetre	cm^3
cubic metre	m^3
day	d
days per week	d/wk
days per year (annum)	d/a
degree	°
degrees Celsius	°C
dollar (American)	USD
dollar (Canadian)	CAD
foot	ft
gram	g
grams per litre	g/L
grams per tonne	g/t
greater than	>
hectare (10,000 m^2)	ha
hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a
inch	in
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilovolts	kV

kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	kWh/t
kilowatt hours per year	kWh/a
less than	<
litre	L
litres per minute	L/m
megawatt	MW
metre	m
metres above sea level	masl
metres Baltic sea level	mbsl
metres per minute	m/min
metres per second	m/s
microns	µm
milligram	mg
milligrams per litre	mg/L
millilitre	mL
millimetre	mm
million	M
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
ounce	oz
parts per million	ppm
parts per billion	ppb
percent	%
pound(s)	lb
second (plane angle)	"
second (time)	s
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
three-dimensional	3D
tonne (1,000 kg) (metric ton)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
volt	V
week	wk
weight/weight	w/w

ABBREVIATIONS AND ACRONYMS

acid-based accounting	ABA
acid potential	AP
acid rock drainage	ARD
Acme Analytical Laboratories Ltd.	Acme
All-In Sustaining Cost	AISC
ammonium nitrate/fuel oil	ANFO
Archer, Cathro & Associates (1981) Ltd.	Archer Cathro
Archer, Cathro & Associates (1981) Ltd.	AC
ATAC Resources Ltd.	ATAC
all-terrain vehicles	ATV
atomic emission spectroscopy	AES
azimuth pointing system	APS
bacterial oxidation	BOX
Blue Coast Research	BCR
Bond Ball Mill Work Index	BWi
Bureau Veritas Minerals Laboratories	BVM
Canadian Council of Ministers of the Environment	CCME
closed-circuit television	CCTV
Canadian Development Expense	CDE
CDN Resource Laboratories Ltd.	CDN
community and economic development allowance	CEDEA
Canadian Exploration Expense	CEE
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
cumulative net cash flow	CNCF
Canadian Pension Plan	CPP
carbon-in-leach	CIL
carbon-in-pulp	CIP
CDN Resource Laboratories Ltd.	CDN Resource
certified reference material	CRM
Cominco Limited	Cominco
comma separated values	.csv
continuous vat leaching	CVL
cumulative net cash flow	CNCF
cyanide leachable gold	AuCN
depth/area/capacity	DAC
distributed control system	DCS
Dynamic Secondary Ionization Mass Spectrometry	D-SIMS
emission spectroscopy	ES
Employment Insurance	EI
engineering, procurement and construction management	EPCM
Exploration Cooperation Agreement	ECA
fire assay gold	AuFA

First Nation of Nacho Nyak Dun	NNDFN
G&T Metallurgical Services	G&T
general and administrative	G&A
global positioning system	GPS
Golder Associates Inc.	Golder
Heritage Resource Impact Assessment	HRIA
Heritage Resource Overview Assessment	HROA
Hesca Resources Corporation Ltd.	Hesca
induced polarization	IP
inductively coupled plasma	ICP
International Organization for Standardization	ISO
interim freshwater sediment quality guidelines	ISQG
internal rate of return	IRR
Kappes Cassidy and Associates	KCA
Knight Piésold Consulting	Knight Piésold
life-of-mine	LOM
mass spectroscopy	MS
metal leaching	ML
Microsoft-SQL Server® database	the Database
motor control centre	MCC
Mine Development Associates	MDA
North American Datum	NAD
National Instrument 43-101	NI 43-101
National Topographic System	NTS
NDU Resources Ltd.	NDU
net cash flow	NCF
First Nation of Nacho Nyak Dun	NNDFN
net present value	NPV
neutralization potential	NP
North American Datum	NAD
National Topographic System	NTS
operator interface station	OIS
Ordinary Kriging	OK
preliminary economic assessment	PEA
pressure oxidation	POX
PricewaterhouseCoopers	PwC
Probable Effects Levels	PEL
Qualified Person	QP
quality assurance	QA
quality control	QC
Quantitative Evaluation of Minerals by Scanning	QEMSCAN®
real time kinematic	RTK
reduced intrusion related gold deposit	RIRGD
Registered Retirement Savings Plan	RRSP
rock quality designations	RGD

run-of-mine	ROM
semi-autogenous	SAG
SGS Mineral Services Vancouver, British Columbia	SGS Vancouver
SGS Minerals Lakefield, Ontario	SGS Lakefield
Surface Science Western	SSW
Sodium metabisulfite	SMBS
Tetra Tech Inc.	Tetra Tech
tailings management facility	TMF
the Rackla Gold Project	the RGP
the Rau Property	the Property
the Tiger Deposit	the Project
Universal Transverse Mercator	UTM
variable time-domain electromagnetic	VTEM
vibrating wire piezometers	VWP
waste rock management facility	WRFM
weak acid dissociable	WAD
work breakdown structure	WBS
Workers' Compensation Board	WCB
Yukon Environmental and Socio-economic Assessment Act	YESAA
Yukon Environmental and Socio-economic Assessment Board	YESAB
z-axis tipper electromagnetic	ZTEM

1.0 SUMMARY

1.1 INTRODUCTION

ATAC Resources Ltd. (ATAC) retained Tetra Tech to prepare a National Instrument 43-101 (NI 43-101) preliminary economic assessment (PEA) for the Tiger Deposit (the Project) of the Rackla Gold Project (the RGP), located in Central Yukon, Canada.

The effective date of this report is February 27, 2020 and the effective date of the Mineral Resource estimate is January 3, 2020.

1.2 PROPERTY DESCRIPTION

The Project is one of many mineralized showings within the Rau Property (the Property) and is the primary focus of this report. It is located in east-central Yukon, northeast of Mayo, and forms the western half of the RGP. The Property comprises 3,315 contiguous quartz mineral claims that are 100% owned by ATAC, totaling 665.9 km^2 (66,590 ha). There are no underlying royalties on the Property.

Access to the Property is currently via a 900-m airstrip, located 7.5 km southeast of the Project. The Wind River Trail, a winter road, starts at McQuesten Lake, near Keno City and crosses the central portion of the Property, 13 km west-northwest of the Project.

1.3 HISTORY

The earliest reported exploration in the area occurred in 1922, following the discovery of silver mineralization at Keno Hill. Prospectors first identified mineralization at Carpenter Ridge, located in the far northwest corner of the Property.

A modest amount of work was completed on the Property between 1922 and 2006. This work mostly focused on silver and Mississippi-valley-type lead-zinc mineralization. ATAC initially staked 64 claims in 2006 to cover a drainage where an isolated high gold value was reported by a regional-scale stream sediment geochemical survey.

1.4 GEOLOGICAL SETTING AND MINERALIZATION

The Property lies within a band of regional-scale thrust and high-angle reverse faults that imbricate rocks of Selwyn Basin and Mackenzie Platform. The Tombstone, Dawson, and Robert Service thrust faults affect stratigraphy along the trend of the Property. All of the thrust faults verge north-easterly.

The Project is hosted within Ordovician to Silurian carbonates belonging to the Bouvette Formation, which is part of a thrust package bound to the south by the Dawson Thrust and to the north by the Kathleen Lakes Fault. Stratigraphic units within this package form open folds that are aligned parallel to the thrusts and plunge gently to the southeast.

The Rackla Pluton, a McQuesten Suite intrusion, is located 3 km southeast of the Project. This intrusion post-dates regional thrusting events and is thought to be the source of mineralization.

Carbonate replacement gold-mineralization at the Tiger Deposit is thought to be a distal variety of the reduced intrusion related gold deposit model. The Tiger Deposit is 900 m long, 40 to 150 m wide, and up to 150 m thick. Mineralization is developed within and adjacent to a regionally extensive corridor of highly-strained carbonate rocks.

Gold occurs in both sulphide and oxide facies mineralization. Sulphide mineralization is accompanied by, and developed within, limestone that is replaced by ferruginous dolomite and iron carbonate minerals. Sulphide species consist of disseminated to banded pyrite, with subordinate arsenopyrite and pyrrhotite, and minor bismuthinite and sphalerite.

Oxide mineralization is completely devoid of sulphide minerals and ranges from very competent, weakly porous limonitic mud to rubbly porous limonitic grit. Complete oxidation extends up to 150 m from surface. The highest-grade and deepest oxidation occurs where northerly trending extensional faults intersect the northwest trending regional shear structure.

1.5 EXPLORATION AND DRILLING

In 2006, ATAC staked 64 claims to cover the Rackla Pluton, and an anomalous drainage. Work in 2007 comprised geological mapping, prospecting, grid soil sampling, and airborne geophysical surveys.

A gold anomaly identified near the edge of the 2007 soil sample grid lead to the discovery of the Project and further expansion of the Property. Between 2008 and 2019, exploration on the Property primarily focused on the northwest trending package of favourable stratigraphy that hosts the Project.

Exploration to date includes the collection of 28,163 soil samples, 2,344 rock samples, and completion of airborne geophysical surveys and diamond drilling. Since 2008, this work has discovered sixteen new showings within the Property and identified many new geochemical anomalies that require additional follow-up work. All of the exploration programs on the Property were managed by Archer, Cathro & Associates (1981) Limited (Archer Cathro) on behalf of ATAC.

The Project has been the primary focus of exploration conducted on the Property to date. A total of 28,610 m of drilling in 166 holes has been completed here and forms the basis for the current Mineral Resource estimate.

1.6 MINERAL RESOURCE ESTIMATE

The Mineral Resource estimate was completed using 6,861 assays taken from 166 diamond drillholes, totalling 28,610 m. The modeling and estimation of the Mineral Resources were completed under the supervision of Mr. Steven J. Ristorcelli, a qualified person with respect to Mineral Resource estimations under NI 43-101. The effective date of this Mineral Resource estimate is January 3, 2020.

Rock units and gold and tungsten domains were explicitly modelled on cross-sections and long-sections. This produced pseudo-solids which were used to code the model. Gold and tungsten domains were defined based on population breaks on cumulative probability plots and grade changes in the drill-hole samples. Capping of gold and tungsten was conducted within each domain separately based on a review of detailed statistics. Samples were composited to 3m intervals down-hole honoring domain boundaries.

Inverse distance estimation with a power of three was used to estimate grade. The search parameters were, in part, based on variography.

The Mineral Resources were classified into potential open pit and potential underground material based on technical and economic factors chosen to meet the "*reasonable prospects for eventual economic extraction*" test.

Mineral Resources are reported at a 0.75 g Au/t cut-off for open pit material and a 1.5 g Au/t cut-off in underground material, as shown in Table 1.1. These cut-off grades were selected based on a review of economic parameters for similar deposits.

Table 1.1 Combined Oxide and Sulphide Resources by Mining Method

Type	Classification	Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
			Open Pit				
Oxide	Indicated	0.75	1,980,000	3.74	238,000	282	559
Sulfide	Measured	0.75	799,000	2.92	75,000	171	137
Sulfide	Indicated	0.75	847,000	2.68	73,000	164	139
Ox + S	M+I	0.75	3,626,000	3.31	386,000	230	835
			Underground				
Oxide	Indicated	1.50	165,000	3.09	16,000	253	42
Sulfide	Measured	1.50	29,000	2.06	2,000	188	5
Sulfide	Indicated	1.50	706,000	2.64	60,000	167	118
Ox + S	M+I	1.50	900,000	2.70	78,000	183	165
			Open Pit + Underground				
Ox + S	**M+I**	**Variable**	**4,526,000**	**3.19**	**464,000**	**221**	**1,000**
			Open Pit				
Oxide	Inferred	0.75	20,000	1.54	1,000	139	3
Sulfide	Inferred	0.75	7,000	2.41	500	123	1
Ox + S	Inferred	0.75	27,000	1.73	1,500	*135	4
			Underground				
Oxide	Inferred	1.50	41,000	2.62	3,000	112	5
Sulfide	Inferred	1.50	97,000	2.26	7,000	94	9
Ox + S	Inferred	1.50	138,000	*2.37	10,000	101	14
			Open Pit + Underground				
Ox + S	**Inferred**	**Variable**	**165,000**	**2.17**	**11,500**	**109**	**18**

* These two grades are calculated slightly differently than all other grades in the resource tables. They are calculated based on full-precision tonnages, whereas all other grades are based on rounded tonnages. This was done to remove what looked like an inconsistency resulting from rounding small tonnages.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.7 MINERAL PROCESSING AND METALLURGICAL TESTING

Mineral processing, metallurgical, and mineralogical programs have been ongoing on the Project since 2010. The discussion in this report focuses on work conducted subsequent to 2016, and metallurgical projections used for this study. For details additional to those described in this section, on testwork conducted prior to 2016, the reader is referred to the 2016 Technical Report.

1.7.1 Oxides

Mineralogical work showed that the oxide samples were dominated by quartz and dolomite, with lesser goethite, talc, calcite, mica, and hematite. Gravity testwork recovered up to 18% of the gold to concentrate, suggesting there is a small amount of coarse gold.

In 2016, Blue Coast Research studied a composite and a suite of variability samples from 8 different drill holes. These composites varied in head grade from 1.4 g/t to 13 g/t. A master composite was created from all samples. This was leached at 172 and 65 microns, with the leach recovery being 90% and 92%, respectively, indicating a small benefit can be gained from finer grinding. The arithmetic average leach extraction from the eight variability composites was 88%.

Tungsten recovery was tested on an oxide sample using a Knelson concentrator and tabling. While only preliminary work was conducted, 37% of tungsten present in the sample was recovered, suggesting that at least some of the tungsten is amenable to gravity recovery.

1.7.2 Sulphides

The sulphide mineralogy is comprised mostly of pyrite and arsenopyrite. Non-sulphide mineralization is dominated by carbonates, comprising mostly dolomite and ankerite. Gold occurs both as solid-solution in sulphides and as discrete mineralization.

In 2018, eight samples were submitted to Surface Science Western in London, Ontario to establish the gold content contained in solid solution in the key sulphides. On average across the eight samples, 16% of the gold was contained in solid solution in the sulphides, though this varied widely, from 3% to 33%. Pyrite was not extensively studied but based on available data the gold content is probably close to 10%, ranging from as little as 5%, up to 36%. The grades of gold in the sulphides were low, averaging 9 g/t in the arsenopyrite and 0.9 g/t in pyrite.

The individual samples were combined to create four composites. Gravity recovery, flotation and leach testwork was conducted on various composites and sub-samples. Gravity testwork recovered between 10% and 42% of the contained gold, suggesting the sulphide mineralized materials may contain slightly coarser discrete gold than the oxides. Flotation testwork yielded high gold recoveries, between 95-97%, but with mass pulls around 40%, suggesting flotation does not appear to be a useful process for this mineralized material.

Bottle roll leach testwork yielded recoveries ranging from 18% to 91%, generally correlating with mineralogical and AuCN assay data. This work suggests that sulphide recovery through cyanide leaching will be highly variable, depending on the relative distribution of gold between refractory and free-milling constituents.

Tungsten recovery was tested on a sulphide sample, but the presence of dense sulphides interfered with the test. Sulphides would need to be floated ahead of gravity, which is not likely to be practical given the cost of such a circuit and the tungsten content of the mineralized material.

1.7.3 METALLURGICAL FORECAST

Gold leach recovery from oxide samples has been regressed against a host of candidate geochemical parameters, without exposing any useful algorithms for predicting gold extraction. Significantly, evidence from this data of a head grade effect on recovery is weak so a fixed gold recovery of 90.5% (including solution and carbon losses) has been used for the 2020 PEA.

For sulphide material, a relationship between AuCN assay data and bottle roll leach recovery has been established. Accounting for losses to solution and carbon, recovery of gold from sulphide material is projected to be 7.4% higher than the AuCN/AuFA assay ratio, which has been estimated as a parameter of the block model.

1.8 MINING METHODS

The mining study is based on a nominal process capacity of approximately 1,500 t/d.

The open pit mine will utilize a conventional truck-and-excavator fleet. Based on the geotechnical recommendations provided by Golder (2016), blasting will be performed on non-oxide rock only, while oxide material will be excavated directly by a hydraulic excavator.

Pit optimization and production scheduling were completed using the Measured, Indicated and Inferred Oxide and Sulphide Mineral Resources. The Project's LOM is approximately seven years, including one year of pre-stripping followed by six years of mill production. Over the seven-year LOM, the pit will produce 2.7 Mt of mineralized material and 14.4 Mt of waste rock. The overall LOM average gold grade of oxide and sulphide material is 3.82 g/t. The LOM stripping ratio (defined as waste material mined divided by mineralized material mined) is 5.3.

1.9 RECOVERY METHODS

The proposed 1,500 t/d processing plant will utilize conventional crushing, grinding, cyanidation by carbon-in-pulp (CIP), and gold recovery from loaded carbon to produce gold doré from the Tiger mineralization. A conventional cyanidation circuit consisting of grinding and agitated cyanide leaching for both the sulphide and oxide resources is proposed for this study.

The processing plant will co-process two types of mineralization: oxide mineralization and sulphide mineralization. The overall design philosophy was to select proven equipment, with a simple and single line flowsheet that can be operated and maintained effectively in a cold environment.

A mineral sizer in the crushing circuit will reduce the run-of-mine (ROM) material to a particle size of approximately P_{80} 120 mm.

The crushed material will be transported by a conveyor to a 1,500-t surge bin and then reclaimed and fed to a primary grinding circuit consisting of a semi-autogenous (SAG) mill and a ball mill in closed circuit with hydrocyclones. The grinding circuit will further reduce the crushed mill feed to a particle size of P_{80} 75 µm.

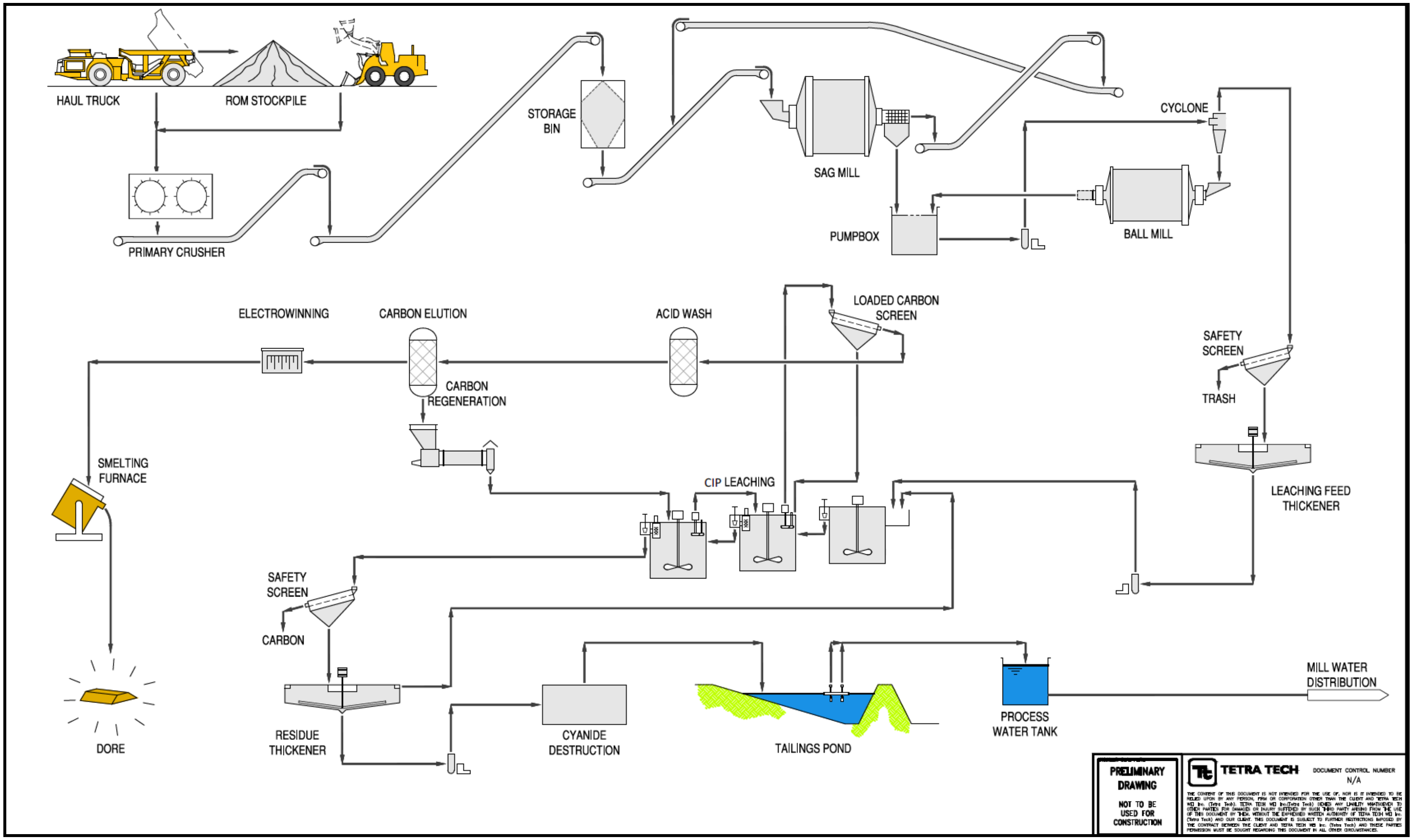
The hydrocyclone overflow from the primary grinding circuit will be thickened, and the underflow of the thickener will be diluted with process water to the optimum solid density, and then cyanide leached in a CIP circuit to recover the gold from the mineralization.

The loaded carbon from the CIP circuit will be washed with a diluted acid solution and eluted by a modified Zadra pressure stripping process. The gold in the pregnant solution will be recovered by electrowinning. The barren solution from the elution circuit will be circulated back to the elution/leach circuit. The gold sludge produced from the electrowinning circuit will be smelted to produce gold doré bullion that will be shipped off-site for refining.

The residue from the leach circuit will be thickened to recover the leach solution for reuse as process water in the cyanidation circuit. The thickener underflow will be sent to a cyanide destruction circuit employing a sulphur dioxide/air process to destroy the residual weak acid dissociable (WAD) cyanide. The treated residue slurry will be pumped to the lined tailings management facility (TMF) for storage.

The simplified flowsheet is shown in Figure 1.1. The average gold production to doré is estimated to be approximately 44,500 oz/a.

Figure 1.1 Simplified Process Flowsheet



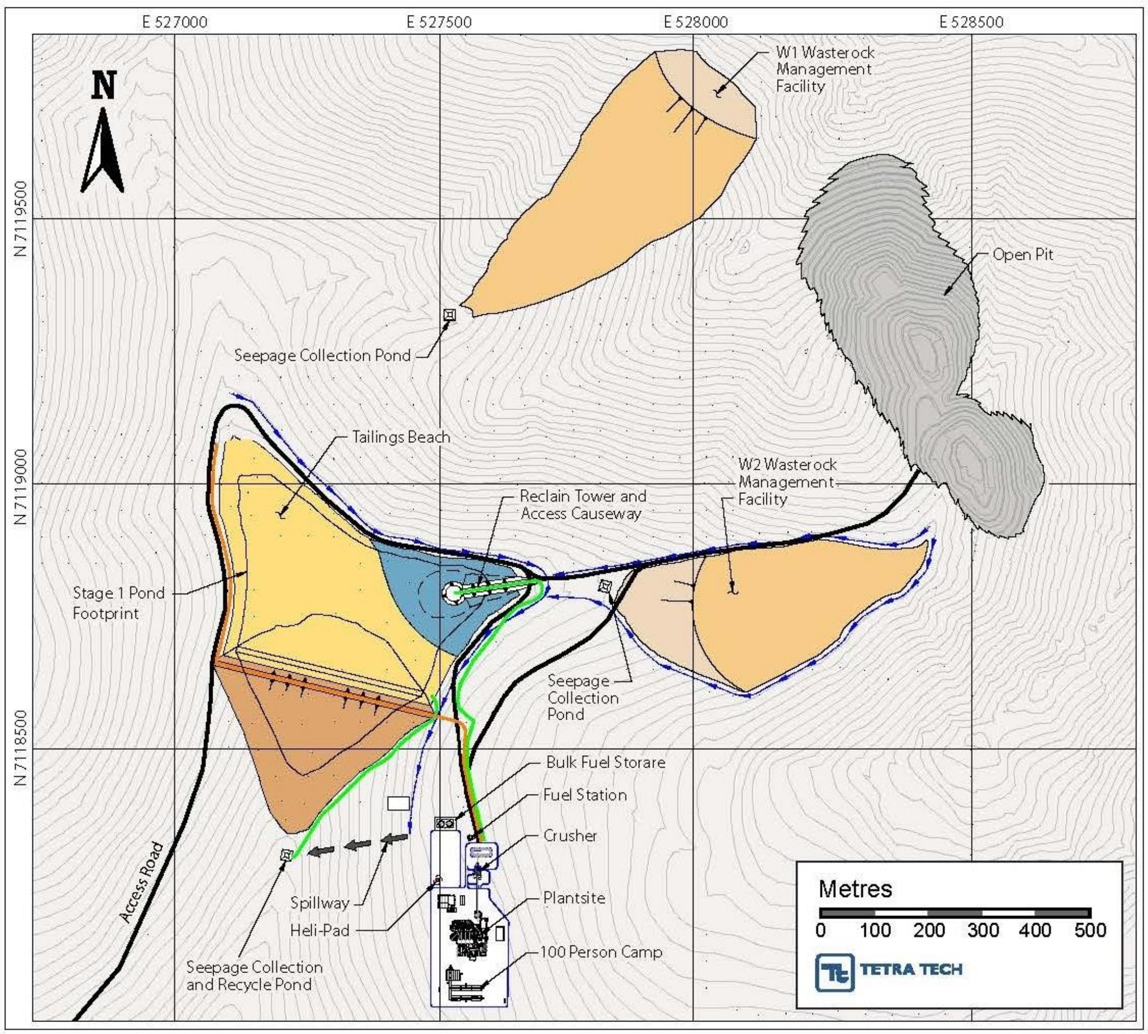
1.10 PROJECT INFRASTRUCTURE

The proposed on-site infrastructure for the Project will include:

- a process plant
- a permanent camp
- an emergency vehicle building with vehicle maintenance shop and warehouse
- administration offices
- power generation units
- main electrical substation and power distribution system
- potable and fire water storage and distribution system
- plant and camp sewage treatment facilities
- a laydown and container storage yard
- fuel storage and fueling station
- a TMF
- two waste rock management facilities (WRMFs)
- access and site roads.

The general site layout of the Project is provided in Figure 1.2.

Figure 1.2 General Site Layout



1.11 Environmental Studies, Permitting, and Social or Community Impact

The Project is located within the Yukon Territory, approximately 143 km northeast of Stewart Crossing, 98 km northeast of the community of Mayo, and 55 km northeast of Keno City. The Project will be subject to an environmental and socio-economic assessment at the Executive Committee level under the *Yukon Environmental and Socio-economic Assessment Act* (YESAA), administered by the Yukon Environmental and Socio-economic Assessment Board (YESAB).

A year-round, all-weather access road will provide site access, and is being assessed/permitted separately to support advanced exploration throughout the district.

Baseline environmental studies were initiated in 2007 and are deemed adequate for this stage of project development. Environmental and baseline study gaps and areas requiring further development for future project development, assessment, and permitting stages have been identified.

A significant potential risk associated with the Project relates to permitting and environmental compliance; these potential risks are common with most mining projects and are mitigated to the greatest extent possible through technical studies, engineering, planning, and with proactive management.

Prior to August 2019, ATAC was conducting mineral exploration and associated activities at the Rau Property, including the Tiger Deposit, under the terms and conditions of Class 3 Exploration Permit LQ00260C (Mining Land Use Regulations). That permit was renewed on August 14, 2020 under authorization LQ00531 and is valid until August 13, 2024.

The Project is located within the Traditional Territory of the First Nation of Nacho Nyak Dun (NNDFN). ATAC has developed a good working relationship with NNDFN and in January 2014 the parties renewed an Exploration Cooperation Agreement (ECA) which was first signed in 2010.

1.12 Capital and Operating Cost Estimates

1.12.1 Capital Cost Estimate

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $110.1 million. A summary breakdown of the initial capital cost is provided in Table 1.2. This total includes all direct costs, indirect costs, Owner’s costs, and contingency. All costs are shown in Canadian Dollars unless otherwise specified.

Table 1.2 Capital Cost Summary

Description	Initial Capital Cost ($ Million)
Overall Site	3.2
Open Pit Mining	10.4
Materials Crushing and Handling	2.0
Process	30.4
TMF	8.0
On-Site Infrastructure	5.2
External Access Roads	11.6
Project Direct Costs - Subtotal	**70.9**
Project Indirect Costs	20.8
Owner's Costs	1.3
Contingencies	17.2
Total Initial Capital Cost	**110.1**

On average, the LOM on-site operating costs for the Project were estimated to be $78.94/t of material processed. The operating costs are defined as the direct operating costs including mining, processing, surface services, general and administrative (G&A), and freight costs (Table 1.3).

Table 1.3 LOM Average Operating Cost Summary

Area	LOM Average Operating Cost ($/t milled)
Mining	23.18
Process	29.88
TMF	0.64
G&A	15.33
Site Service	4.68
Camp and Genset Leasing Cost	1.68
Equipment Leasing Cost	3.55
Total Operating Cost	**78.94**

1.13 ECONOMIC ANALYSIS

A PEA should not be considered to be a prefeasibility or feasibility study, as the economics and technical viability of the Project have not been demonstrated at this time. A PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results as reported in the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

An economic evaluation was prepared for the Project based on a pre-tax financial model. The following pre-tax financial parameters were calculated using the base case gold price of USD1,400/oz and an exchange rate of USD0.77:CAD1.00 (all currency units are Canadian dollars unless otherwise specified):

- 54.5% IRR
- 1.24-year payback on $110.1 million initial capital
- $118.2 million NPV at a 5% discount rate.

ATAC commissioned PwC in Vancouver, British Columbia to prepare a tax model for the post-tax economic evaluation of the Project with the inclusion of applicable income and mining taxes (see Section 22.4 for further details).

The following post-tax financial results were calculated:

- 42.6% IRR
- 1.40 year payback on $110.1 million initial capital
- $85.4 million NPV at a 5% discount rate.

Sensitivity analyses were conducted to analyze the sensitivity of the Project merit measures (NPV, IRR, and payback periods) to changes in gold price, exchange rate, operating costs, and capital costs. The Project's pre-tax NPV, calculated at a 5% discount rate, is most sensitive to exchange rate and gold price followed by on-site operating costs and capital costs. The Project's pre-tax IRR is most sensitive to exchange rate and gold price followed by capital costs and on-site operating costs. The payback period is most sensitive to gold price followed by exchange rate, capital costs and on-site operating costs.

1.14 RECOMMENDATIONS

It is recommended that the Project proceed to the feasibility level of study. The total cost for future recommended work is $3.6 million; Table 1.4 shows the cost breakdown by discipline.

Table 1.4 Recommended Costs for Future Work

Area	Budget Amount ($)
Geology and Mineral Resources	1,000,000
Geotechnical and Hydrogeological	1,100,000
Mineral Processing and Metallurgical Testing	200,000
Mining	500,000
Process & Infrastructure	200,000
Environmental	600,000
Total	**3,600,000**

2.0 INTRODUCTION

ATAC commissioned a team of engineering consultants to complete this PEA update, in accordance with NI 43-101 Standards of Disclosure for Mineral Projects.

Components of this PEA were completed by the following consultants:

- Tetra Tech Inc. (Tetra Tech): overall project management, mining methods, mineral processing and recovery methods, project infrastructure, environmental studies, permitting, and social or community impact, capital and operating cost estimates, and economic analysis.
- Mine Development Associates: (MDA): sample preparation, data verification, and Mineral Resource estimate.
- Archer, Cathro & Associates (1981) Limited (Archer Cathro): project description and location, accessibility, history, geological setting, deposit types, exploration, drilling, and adjacent properties.
- Blue Coast Metallurgy Ltd. (Blue Coast): mineral processing and metallurgical testing.
- Knight Piésold Consulting (Knight Piésold): tailings and waste rock management (including capital costs).

The effective date of this study is February 27, 2020 and the effective date of the Mineral Resource estimate is January 3, 2020.

2.1 QUALIFIED PERSONS

A summary of the Qualified Persons (QPs) responsible for this report is provided in Table 2.1. The following QPs conducted site visits of the Property:

- Raj Priyadarshi, P.Eng. completed a site visit on October 1, 2019.
- Matthew Dumala, P.Eng., completed a site visit from July to August 2011.

Table 2.1 Summary of QPs

Report Section			Company	QP
1.0	Summary		All	Sign-off by Section
2.0	Introduction		Tetra Tech	Hassan Ghaffari, P.Eng
3.0	Reliance on Other Experts		Tetra Tech	Hassan Ghaffari, P.Eng
4.0	Property Description and Location		Archer Cathro	Matthew Dumala, P.Eng.
5.0	Accessibility, Climate, Local Resources, Infrastructure and Physiography		Archer Cathro	Matthew Dumala, P.Eng.
6.0	History		Archer Cathro	Matthew Dumala, P.Eng.
7.0	Geological Setting and Mineralization		Archer Cathro	Matthew Dumala, P.Eng.
8.0	Deposit Types		Archer Cathro	Matthew Dumala, P.Eng.
9.0	Exploration		Archer Cathro	Matthew Dumala, P.Eng.
10.0	Drilling		Archer Cathro	Matthew Dumala, P.Eng.
11.0	Sample Preparation, Analyses and Security		MDA	Steven Ristorcelli, CPG, Peter Ronning, P.Eng.
12.0	Data Verification		MDA	Peter Ronning, P.Eng.
13.0	Mineral Processing and Metallurgical Testing		Blue Coast	Christopher Martin, C.Eng, MIMMM
14.0	Mineral Resource Estimates		MDA	Steven Ristorcelli, CPG
15.0	Mineral Reserve Estimates		Tetra Tech	Suraj (Raj) Priyadarshi, P.Eng.
16.0	Mining Methods		Tetra Tech	Suraj (Raj) Priyadarshi, P.Eng.
17.0	Recovery Methods		Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
18.0	Infrastructure		-	-
	18.1	Site Layout	Tetra Tech	Hassan Ghaffari, P.Eng.
	18.2	Power	Tetra Tech	Hassan Ghaffari, P.Eng.
	18.3	Tailings Management	Knight Piésold	Bruno Borntraeger, P.Eng.
	18.4	Waste Management	Knight Piésold	Bruno Borntraeger, P.Eng.
	18.5	Water Management	Knight Piésold	Bruno Borntraeger, P.Eng.
	18.6	Fresh, Fire, and Potable Water Supply and Sewage Disposal	Tetra Tech	Hassan Ghaffari, P.Eng.
	18.7	Communications	Tetra Tech	Hassan Ghaffari, P.Eng.
19.0	Market Studies and Contracts		Tetra Tech	Hassan Ghaffari, P.Eng.
20.0	Environmental Studies, Permitting and Social or Community Impact		Tetra Tech	Hassan Ghaffari, P.Eng.
21.0	Capital and Operating Costs		Tetra Tech	Hassan Ghaffari, P.Eng.
22.0	Economic Analysis		Tetra Tech	Suraj (Raj) Priyadarshi, P.Eng.
23.0	Adjacent Properties		AC	Matthew Dumala, P.Eng.
24.0	Other Relevant Data and Information		Tetra Tech	Hassan Ghaffari, P.Eng.
25.0	Interpretation and Conclusions		All	Sign-off by Section
26.0	Recommendations		All	Sign-off by Section
27.0	References		All	Sign-off by Section
28.0	Certificates of Qualified Person		All	Sign-off by Section

2.2 SOURCES OF INFORMATION

All sources of information for this study are in Section 27.0.

2.3 UNITS OF MEASUREMENT AND CURRENCY

The International System of Units (SI) is used in this report.

All currency is in Canadian Dollars, unless otherwise noted.

3.0 RELIANCE ON OTHER EXPERTS

Matthew Dumala, P.Eng. of Archer Cathro, relied on Andrew Carne, P. Eng., Project Engineer of Archer Cathro for matters relating to mineral tenure and mining rights permits, surface rights, royalties, agreements, and encumbrances relevant to this report.

Suraj (Raj) Priyadarshi, P.Eng. of Tetra Tech, relied on PricewaterhouseCoopers Ltd. (PwC) concerning tax matters relevant to this PEA and detailed in Section 22.0. The reliance is based on a letter to ATAC titled “Assistance with review of the income and mining tax portions of the economic analysis prepared by Tetra Tech Inc. (“Tetra Tech”) in connection with the Preliminary Economic Assessment (the “Report”) on ATAC Resources Ltd.’s (“ATAC”) Tiger Gold Deposit Project (the “Project”), dated March 06, 2020, in connection with the 2020 Tiger PEA, NI 43-101 Technical Report.

Hassan Ghaffari, P. Eng. of Tetra Tech, relied on Rick Hoos, R. P. Bio. of Tetra Tech, for matters related to environmental, permitting and social or community impact deailed in Section 20.0 of this report.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Tiger Deposit, a part of the Rau Property, is centered at 64.20° north latitude and −134.41° west longitude in east central Yukon. The Property forms the western half of ATAC's 1,756 km^2 Rackla Gold Project and comprises 3,315 contiguous quartz mineral claims. The Property is located on National Topographic System (NTS) map sheets 105M/14, 106D/01, 106D/02, 106D/03, 106D/06, 106D/07, and 106D/08 (Figure 4.1). The Property covers an area of 665.9 km^2 (66,590 ha). The claims are registered with the Mayo Mining Recorder in the name of Archer, Cathro and Associates (1981) Limited, holding them in trust for ATAC. ATAC owns the Property 100%, with no underlying interests. The claims and expiry dates as of February 18, 2020, are tabulated in Table 4.1 and the locations shown on Figure 4.2.

Figure 4.1 Project Location Map

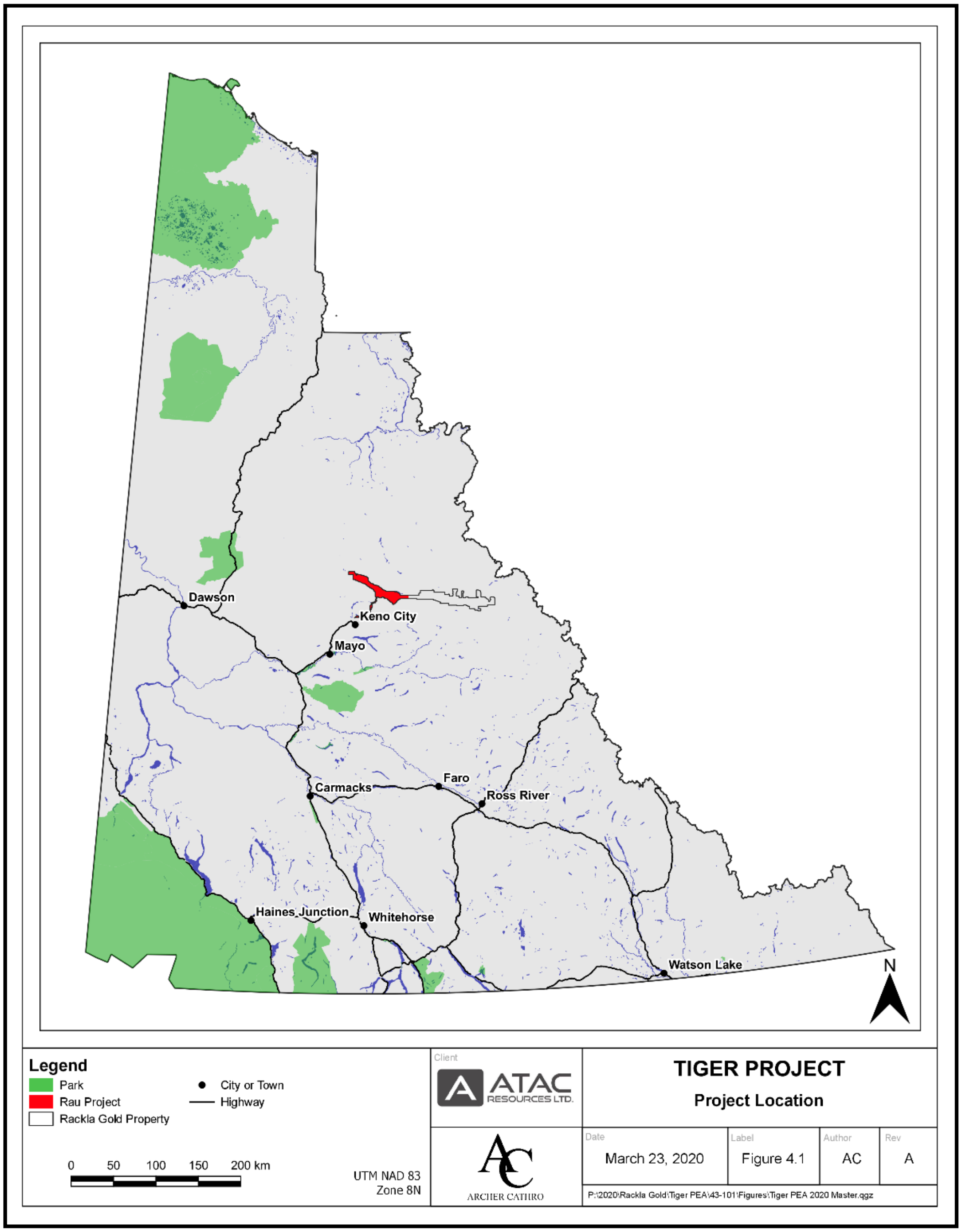


Table 4.1 Claim Data

Claim Name(s)	Grant Number(s)	Expiry Date
ACX 1-234	YD08251-YD08484	April 28, 2031
ACX 1-67	YD08251-YD08317	April 28, 2035
ACX 68	YD08318	April 28, 2033
ACX 69-75	YD08319-YD08325	April 28, 2035
ACX 76	YD08326	April 28, 2033
ACX 77-80	YD08327-YD08330	April 28, 2035
ACX 81-82	YD08331-YD08332	April 28, 2033
ACX 83-112	YD08333-YD08362	April 28, 2035
ACX 113-114	YD08363-YD08364	April 28, 2033
ACX 115-120	YD08365-YD08370	April 28, 2035
ACX 121	YD08371	April 28, 2033
ACX 122-151	YD08372-YD08401	April 28, 2035
ACX 152-153	YD08402-YD08403	April 28, 2033
ACX 154-170	YD08404-YD08420	April 28, 2035
ACX 171-186	YD08421-YD08436	April 28, 2033
ACX 187-206	YD08437-YD08456	April 28, 2035
ACX 207	YD08457	April 28, 2033
ACX 208	YD08458	April 28, 2035
ACX 209-212	YD08459-YD08462	April 28, 2033
ACX 213-218	YD08463-YD08468	April 28, 2035
ACX 219-220	YD08469-YD08470	April 28, 2033
ACX 221-224	YD08471-YD08474	April 28, 2035
ACX 225	YD08475	April 28, 2033
ACX 226	YD08476	April 28, 2035
ACX 227-229	YD08477-YD08479	April 28, 2033
ACX 230	YD08480	April 28, 2035
ACX 231	YD08481	April 28, 2033
ACX 232	YD08482	April 28, 2035
ACX 233	YD08483	April 28, 2033
ACX 234	YD08484	April 28, 2035
ACX 235-238	YD33163-YD33166	April 28, 2035
ACX 239	YD33167	April 28, 2033
ACX 240	YD33168	April 28, 2035
ACX 241	YD33169	April 28, 2033
ACX 242	YD33170	April 28, 2035

ACX 243	YD33171	April 28, 2033
ACX 244	YD33172	April 28, 2035
ACX 245-252	YD33173-YD33180	April 28, 2033
ACX 253-258	YD33181-YD33186	April 28, 2035
ACX 259-260	YD33187-YD33188	April 28, 2033
ACX 261	YD33189	April 28, 2035
ACX 262	YD33190	April 28, 2033
ACX 263	YD33191	April 28, 2035
ACX 264	YD33192	April 28, 2033
ACX 265	YD33193	April 28, 2035
ACX 266	YD33194	April 28, 2033
ACX 267	YD33195	April 28, 2035
ACX 268	YD33196	April 28, 2033
ACX 269-273	YD33197-YD33201	April 28, 2035
ACX 274-282	YD33202-YD33210	April 28, 2033
ACX 283	YD33211	April 28, 2035
ACX 284	YD33212	April 28, 2033
ACX 285	YD33213	April 28, 2035
ACX 286	YD33214	April 28, 2033
ACX 287-292	YD33215-YD33220	April 28, 2035
ACX 293-310	YD33221-YD33238	April 28, 2033
Gam 1-21	YC98437-YC98457	April 28, 2031
Gam 22-32	YE65090-YE65100	May 25, 2026
GF 3-4	YC32305-YC32306	April 28, 2040
Q 1-13	YC92361-YC92373	April 28, 2037
Q 14	YC92470	April 28, 2037
Q 15-24	YC92375-YC92384	April 28, 2037
Q 25-26	YC92385-YC92386	April 28, 2041
Q 27-52	YC92387-YC92412	April 28, 2034
Q 53-58	YC92413-YC92418	April 28, 2038
Q 59-84	YC92419-YC92444	April 28, 2034
Q 85-88	YC92445-YC92448	April 28, 2038
Q 89	YC92449	April 28, 2034
Q 90	YC92450	April 28, 2038
Q 91-109	YC92451-YC92469	April 28, 2034
R 1-6	YC68334-YC68339	April 28, 2043
R 7-10	YC68340-YC68343	April 28, 2040

R 11-12	YC68344-YC68345	April 28, 2036
R 13-20	YC68346-YC68353	April 28, 2043
R 21-24	YC68354-YC68357	April 28, 2040
R 25-54	YC68358-YC68387	April 28, 2036
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R 62	YC68395	April 28, 2043
R 63	YC68396	April 28, 2039
R 64	YC68397	April 28, 2043
R 65	YC68398	April 28, 2039
R 66	YC68399	April 28, 2043
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R 69	YC68402	April 28, 2036
R 70	YC68403	April 28, 2040
R 71	YC68404	April 28, 2034
R 72	YC68405	April 28, 2036
R 73	YC68406	April 28, 2034
R 74	YC68407	April 28, 2036
R 75	YC68408	April 28, 2034
R 76	YC68409	April 28, 2036
R 77	YC68410	April 28, 2034
R 78	YC68411	April 28, 2036
R 79	YC68412	April 28, 2034
R 80	YC68413	April 28, 2036
R 81	YC68414	April 28, 2032
R 82	YC68415	April 28, 2036
R 83	YC68416	April 28, 2032
R 84	YC68417	April 28, 2036
R 85	YC68418	April 28, 2032
R 86	YC68419	April 28, 2036
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R 92	YC68425	April 28, 2036

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R 94	YC68427	April 28, 2036
R 95	YC68428	April 28, 2032
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R 97	YC68430	April 28, 2032
R 98	YC68431	April 28, 2036
R 99	YC68432	April 28, 2032
R 100	YC68433	April 28, 2036
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R 102	YC68435	April 28, 2036
R 103	YC68436	April 28, 2032
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R 106	YC68439	April 28, 2043
R 107	YC68440	April 28, 2039
R 108	YC68441	April 28, 2043
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R 121-124	YC68454-YC68457	April 28, 2036
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R 135-158	YC68468-YC68491	April 28, 2032
R 159-178	YC68492-YC68511	April 28, 2039
R 179-182	YC68512-YC68515	April 28, 2036
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R 193-214	YC68526-YC68547	April 28, 2032
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R 235-238	YC68568-YC68571	April 28, 2036
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R 457-467	YC68790-YC68800	April 28, 2039
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R 490-495	YC68823-YC68828	April 28, 2032
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R 502-508	YC68835-YC68841	April 28, 2043

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R 582-584	YC68915-YC68917	April 28, 2039
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R 730	YC69063	April 28, 2042
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R 766	YC69099	April 28, 2039

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R 982-1003	YC69315-YC69336	April 28, 2032
R 1004	YC69337	April 28, 2043
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R 1006-1009	YC69339-YC69342	April 28, 2043
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R 1052-1053	YC69385-YC69386	April 28, 2039
R 1054-1057	YC69387-YC69390	April 28, 2043
R 1058-1061	YC69391-YC69394	April 28, 2042
R 1062	YC69395	April 28, 2039
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R 1111	YC69444	April 28, 2039
R 1112-1113	YC69445-YC69446	April 28, 2043
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R 1120	YC69453	April 28, 2039
R 1121	YC69454	April 28, 2042
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R 1170-1181	YC69503-YC69514	April 28, 2039
R 1182-1211	YC69515-YC69544	April 28, 2032
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R 1250-1251	YC69583-YC69584	April 28, 2039
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Rau 11	YC50278	April 28, 2047
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Rau 13-16	YC50280-YC50283	April 28, 2047
Rau 17-31	YC50284-YC50298	April 28, 2039
Rau 32	YC50299	April 28, 2047
Rau 33-44	YC50300-YC50311	April 28, 2039
Rau 45-48	YC50312-YC50315	April 28, 2047
Rau 49-53	YC50316-YC50320	April 28, 2039
Rau 54-64	YC50321-YC50331	April 28, 2047
Rau 65-68	YC57529-YC57532	April 28, 2032
Rau 69	YC57533	April 28, 2040
Rau 70	YC57534	April 28, 2042
Rau 71	YC57535	April 28, 2040
Rau 72	YC57536	April 28, 2043
Rau 73	YC57537	April 28, 2039
Rau 74	YC57538	April 28, 2043
Rau 75	YC57539	April 28, 2039

Rau 76	YC57540	April 28, 2043
Rau 77-80	YC57541-YC57544	April 28, 2039
Rau 81-84	YC57545-YC57548	April 28, 2042
Rau 85-86	YC57549-YC57550	April 28, 2043
Rau 87-96	YC57551-YC57560	April 28, 2040
Rau f 97-98	YC69925-YC69926	April 28, 2041
Rau f 99	YC69961	April 28, 2040
Rau f 100	YC69962	April 28, 2033
RR 1-148	YC89701-YC89848	April 28, 2036
S 1-42	YC90801-YC90842	April 28, 2030
S 43-92	YC90843-YC90892	April 28, 2037
S 93-700	YC90893-YC91500	April 28, 2030
S 701-842	YC91901-YC92042	April 28, 2030
S 843	YC92355	April 28, 2030
S 844-860	YC92044-YC92060	April 28, 2030
S 861-878	YC92061-YC92078	April 28, 2029
S 879-896	YC92079-YC92096	April 28, 2030
S 897-912	YC92097-YC92112	April 28, 2029
S 913-932	YC92113-YC92132	April 28, 2030
S 933-946	YC92133-YC92146	April 28, 2029
S 947-968	YC92147-YC92168	April 28, 2030
S 969-980	YC92169-YC92180	April 28, 2029
S 981-1001	YC92181-YC92201	April 28, 2030
S 1002-1013	YC92202-YC92213	April 28, 2029
S 1014-1036	YC92214-YC92236	April 28, 2030
S 1037-1044	YC92237-YC92244	April 28, 2029
S 1045-1070	YC92245-YC92270	April 28, 2030
S 1071-1078	YC92271-YC92278	April 28, 2029
S 1079-1106	YC92279-YC92306	April 28, 2030
S 1107-1114	YC92307-YC92314	April 28, 2029
S 1115-1154	YC92315-YC92354	April 28, 2030
S 1155-1244	YD09635-YD09724	March 1, 2024
S 1245	YD09725	March 1, 2031
S 1246	YD09726	March 1, 2025
S 1247	YD09727	March 1, 2029
S 1248	YD09728	March 1, 2031
S 1249-1250	YD09729-YD09730	March 1, 2024

T 1-28	YD09731-YD09758	March 1, 2036

The gold mineralization that comprises the Tiger Deposit, the main focus of this technical report, is located on quartz mineral claims Rau 56 and 97F. The Tiger Deposit, and other known mineral occurrences documented within the Property, is shown on Figure 4.3.

Figure 4.2 Claim Locations

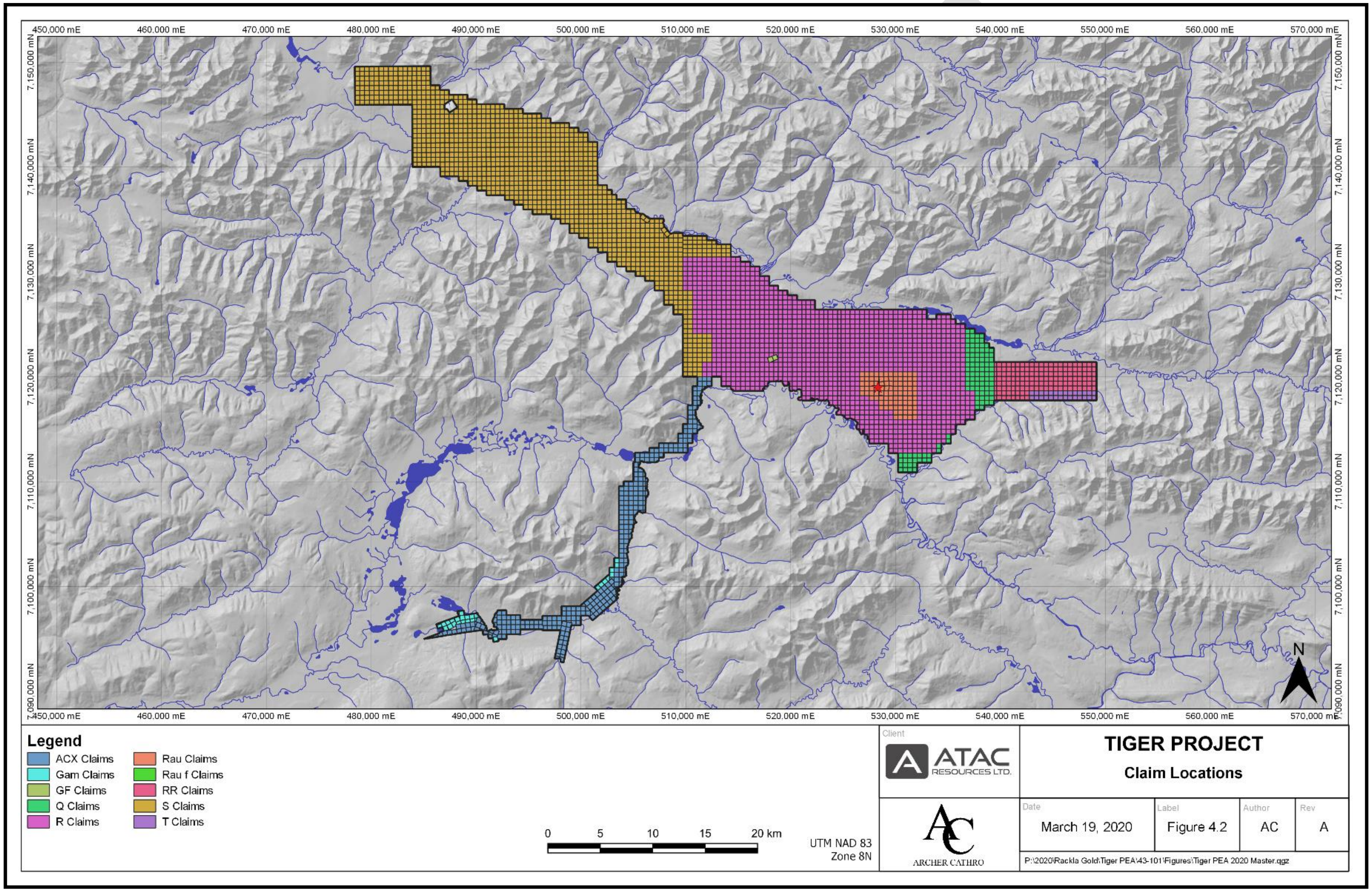
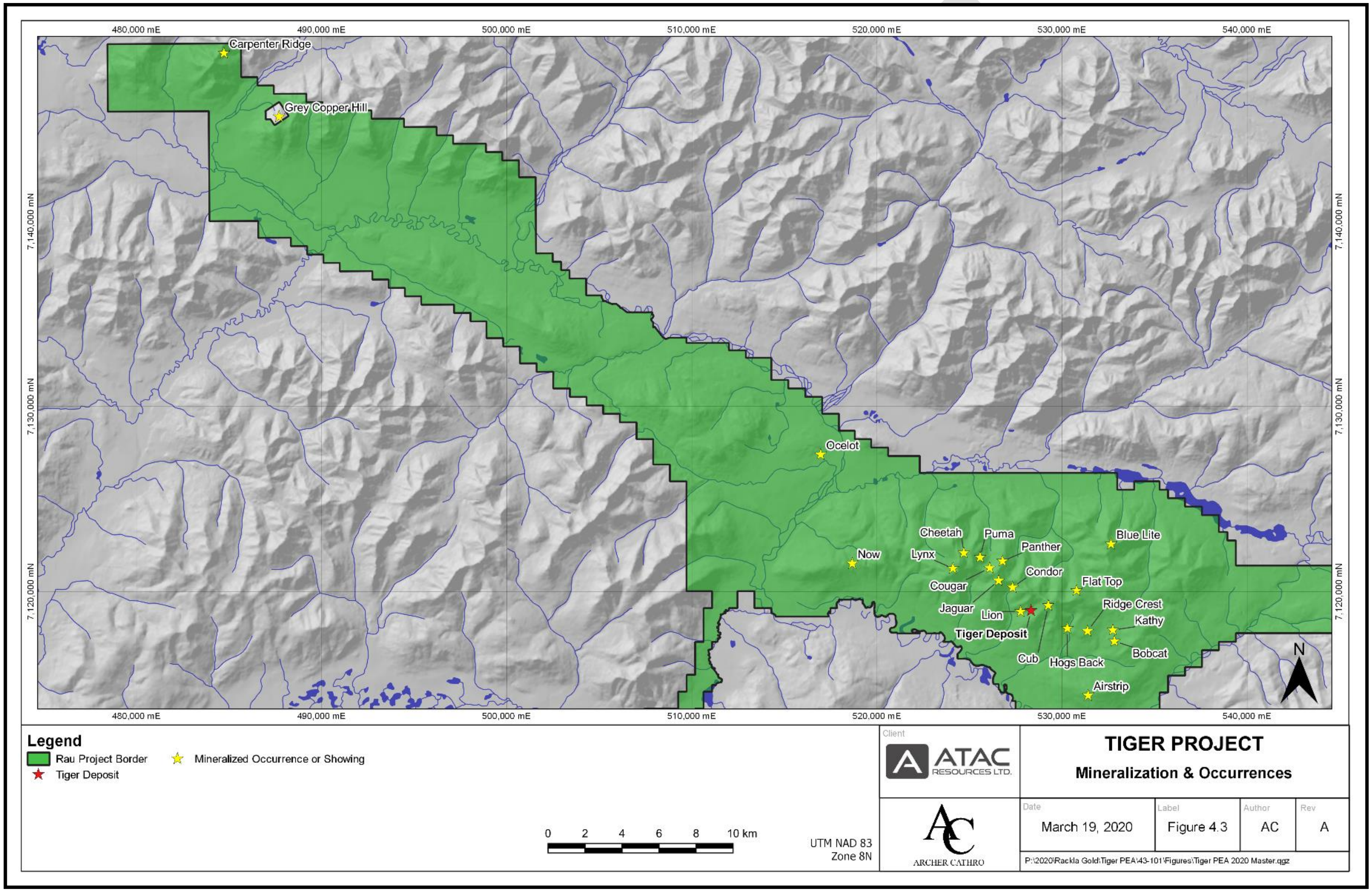


Figure 4.3 Nearby Mineralization and Occurrences



The mineral claims comprising the Property can be maintained in good standing by performing approved exploration work to a dollar value of $100 per claim per year or through payment in lieu. The QP is not aware of any encumbrances associated with lands underlain by the Property.

Exploration is subject to Mining Land Use Regulations of the Yukon *Quartz Mining Act* and the Yukon *Environmental and Socio-economic Assessment Act*. Yukon Environmental and Socio-economic Assessment Board (YESAB) approval must be obtained and a Land Use approval must be issued, before large-scale exploration is conducted. Approval for this scale of exploration has been obtained by ATAC under Class III Mining Land Use Approval LQ00531, which expires August 13, 2024.

Potential mine development on the Property will require YESAB approval, a Yukon Mining License and Lease issued by the Yukon Government, and a permit issued by the Yukon Water Board.

The claim posts on the Property have been located by global positioning system (GPS) using the Universal Transverse Mercator (UTM) coordinate system and North American Datum (NAD) 83.

The Property lies within the traditional territory of the First Nation of Na-Cho Nyak Dun (NNDFN). To the best of the Author's knowledge there are no encumbrances to the Property relating to First Nation Settlement Lands.

Outstanding environmental liabilities relating to the Property are currently limited to progressive reclamation during seasonal exploration activities, and final decommissioning required prior to expiration of the Mining Land Use Approval. Progressive reclamation generally entails backfilling or re-contouring disturbed sites and leaving them in a manner conducive to re-vegetation of native plant species. Back-hauling scrap materials, excess fuel, and other seasonal supplies is also done. Final decommissioning requires that: all vegetated areas disturbed by ATAC's exploration be left in a manner conducive to re-vegetation by native plant species; all petroleum products and hazardous substances be removed from the site; all scrap metal, debris and general waste be completely disposed of; structures be removed; and, the site restored to its previous level of utility. The QP does not know of any other significant factors that may affect access, title, surface rights, or ability of ATAC to perform work on the Property.

5.0 ACCESSIBILITY, CLIMATE, INFRASTRUCTURE, LOCAL RESOURCES AND PHYSIOGRAPHY

The southernmost point on the Property lies 50 km northeast of Mayo, while the Tiger Deposit lies 100 km northeast of Mayo (Figure 5.1). The closest road access is to the community of Keno City, situated 49 km by road northeast of Mayo, the nearest supply center. Mayo and Keno City can be reached in all seasons by two-wheel-drive vehicles using the Yukon highway system from Whitehorse, Yukon. From Whitehorse there is daily jet service to Vancouver, British Columbia and other points south. Whitehorse is a major center of supplies, communications and a source of skilled labor for exploration diamond drilling, construction, and mining operations.

The Wind River Trail, classified as a "winter road", starts at McQuesten Lake near Keno City and crosses the central portion of the Property. The Wind River Trail has been used intermittently by various exploration companies since it was built in the late 1960s and most recently in winter 2007 to bring fuel into the Wernecke Mountains, north of the Property. However, recent warmer average winter temperatures have greatly reduced the period this route is suitable to use as a winter trail.

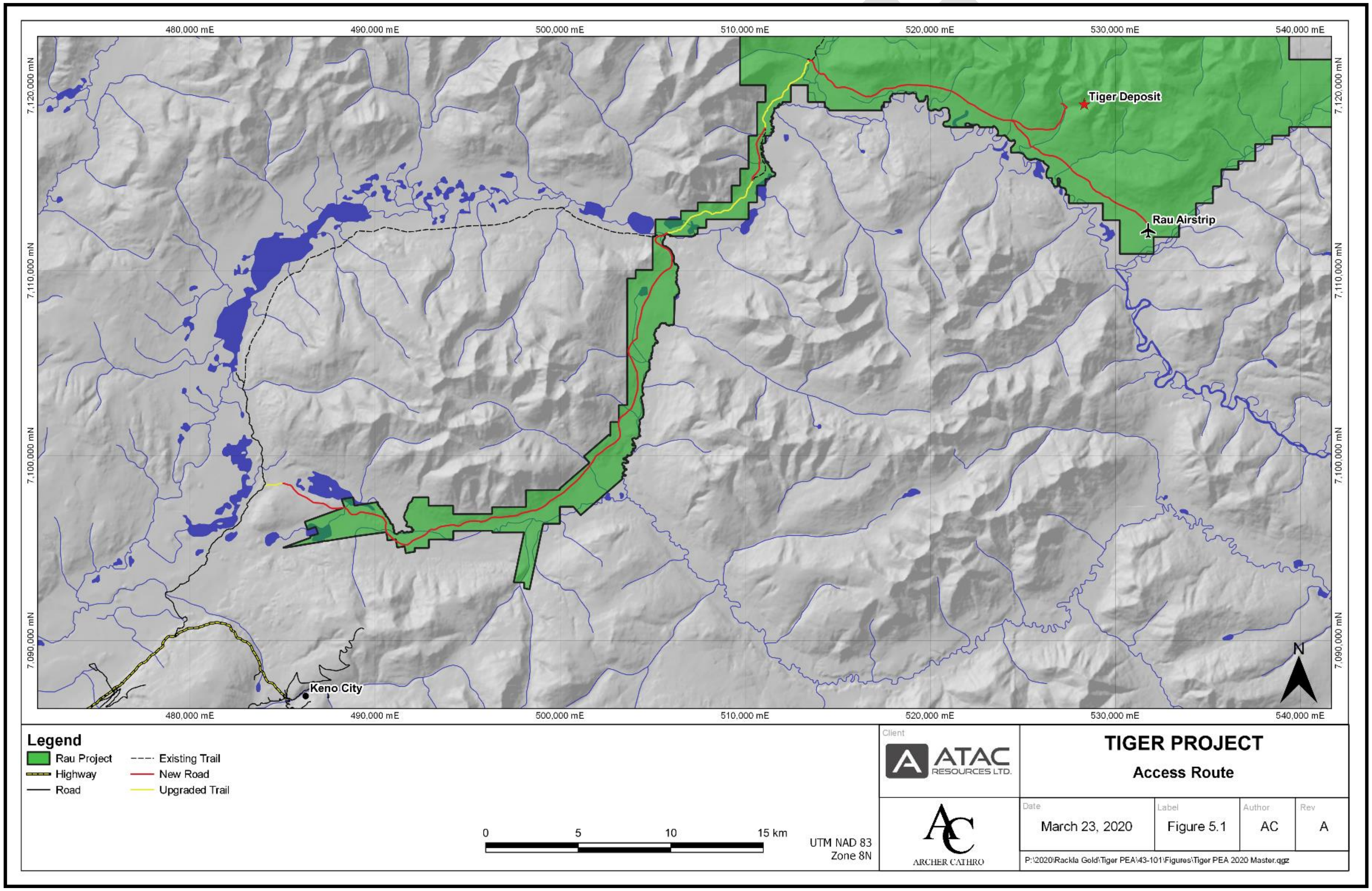
Access to the Property is currently via fixed wing aircraft to ATAC's 900-m-long (2,950-ft) airstrip located 7.5 km southeast of the Tiger Deposit near the southeastern boundary of the Property. Helicopters are used locally to access to exploration targets within the Property. A local trail system provides access within the valley containing the Tiger Deposit. The trail system is accessible with the use of a four-wheel-drive all-terrain vehicle (ATV) from the existing exploration camp to the main deposit area.

Portable electrical generators provide sufficient power for exploration-stage programs and the creeks in the area provide sufficient water for camp and diamond drilling requirements on the Property. Keno City is connected to the Yukon electrical grid and is the nearest source of grid-power to the Property.

The proposed Project infrastructure details are discussed in Section 18.0.

The Property is 70 km long and covers a diverse geomorphological setting. Much of the claim block covers low lying vegetated valley bottom and similarly covered low elevation ridge systems. Abundant, accessible sites for mining, camp facilities, tailings storage areas, waste disposal areas and potential processing plant sites with no conflicting surface rights exist on the Property.

Figure 5.1 Project Access



The majority of the Property is situated within the Nadaleen Range of the Selwyn Mountains and is drained by creeks that flow into the Rackla and Beaver Rivers, which are both part of the Yukon River watershed. Local topography is alpine to sub-alpine and features north and south-trending rocky spurs and valleys that flank a main east-west trending ridge. Elevations range from 725 m along the Beaver River in the center of the Property to 1,800 m atop a peak that is referred to as Monument Hill. Outcrop is most abundant on or near ridge crests and in actively eroding creek beds. Most hillsides are talus covered at higher elevations and are blanketed by glacial till at lower elevations. Soil development is moderate to poor in most areas.

Valley floors are well treed with mature black spruce. The density and size of vegetation gradually decreases with increasing elevation. Undergrowth typically consists of low-lying shrubs and moss. Tree line in the vicinity of the Property is at about 1,500 m. Slopes above that elevation are un-vegetated with the exception of moss and lichen. South facing slopes are typically well drained and are often lightly forested with poplar. Steep, north facing slopes are usually rocky outcrop and talus. Gentler, spruce- and moss-covered terrain, mainly north-facing, exhibits widespread permafrost.

Much of the overburden in the region is associated with the most recent Cordilleran ice sheet, the McConnell glaciation, which is believed to have covered south and central Yukon between 26,500 and 10,000 years ago.

The climate at the Property is typical of northern continental regions with long, cold winters, short fall and spring seasons and mild summers. Snowfall can occur in any month at higher elevations. The Property is mostly snow free from early June to late September, coinciding with the exploration season. According to Environment Canada, Mayo holds the Yukon high-temperature record based on June 14, 1969, when the thermometer peaked at 36.1°C. The lowest temperature in Mayo, recorded on February 3, 1947, is −62.2°C. Mayo holds the Canadian record for the greatest range of absolute temperatures, a difference of 98.3°C between the extreme high and extreme low (Yukon Community Profiles 2016).

Historical weather records over the past three decades show that the average daytime temperature in January in Mayo is −20.5°C, dropping to −31°C at night (Government of Canada 2020). In July the daytime average is close to +23°C while the nighttime temperature drops to about 9°C. Annual precipitation averages 313 mm, as 205 mm of rain and 147 cm of snow.

In May 2011, ATAC established a weather monitoring station at the Rau Airstrip. Data was collected continuously from this station between the time of installation and October 2014. The maximum temperature recorded was 29.5°C on June 23, 2012, while the lowest was −50.0°C on January 29, 2013. Annual average rainfall recorded at this station in this period was 289.5 mm. The average snowpack at a station near the Tiger Deposit, measured in March of each of these years, was 68.5 cm.

6.0 HISTORY

The locations referred to in this section are shown on Figure 4.3.

The earliest reported exploration within the area occurred in 1922 following the discovery of silver mineralization at Keno Hill, when prospectors first identified and staked mineralized float occurrences at Carpenter Ridge north of the far northwest corner of the Property. In 1924, reconnaissance work conducted by the Geological Survey of Canada discovered galena-calcite-siderite float on the southwest end of Carpenter Ridge. A sample of this float assayed 8.75 oz/t silver and 56.0% lead (Cockfield, 1925). However, the source of this mineralization was not found. Hand pits were dug in 1927 and 1928 but little record remains of the work completed during this period. All claims were ultimately dropped.

At Grey Copper Hill, 9 km southeast of Carpenter Ridge, silver-rich tetrahedrite float was discovered in 1923 by an independent prospector. This showing and other nearby prospects were staked later that year. Several exploration adits were dug into the hillside during unsuccessful follow up exploration and eventually all claim holdings lapsed.

Between 1930 and 1974 Grey Copper Hill was staked several times by independent prospectors and exploration companies, including Cypress Resources Limited and United Keno Hill Mines Limited. Little work was reported (Hilker 1969) and all claims ultimately expired.

Hesca Resources Corporation Ltd. (Hesca) staked Grey Copper Hill in 1974 and conducted prospecting, soil sampling, hand trenching and adit maintenance. In addition, two shallow, small diameter diamond drillholes totaling 56.3 m were drilled; however, the results from this drilling are not documented. No further work was done by Hesca and the claims were dropped (Deklerk and Traynor 2005).

In 1978, Prism Resources Limited staked the Grey Copper Hill area and conducted prospecting and geochemical sampling. Soil sampling identified several lead and silver anomalies. Follow up prospecting failed to explain them (Sivertz 1979). A sample collected from an outcrop of dolomite yielded 0.60% lead and 51.43 g/t silver, while a tetrahedrite sample collected near an old adit assayed 7,000 g/t silver (Sivertz 1980). The Prism Joint Venture allowed the claims to lapse.

Grey Copper Hill was again staked in 1983 by a prospector who conducted grid soil sampling later that year. This program delineated silver anomalies coincident with surface lineaments. No further work was completed and the claims expired.

In 1988 Bonventures Limited staked the area and conducted limited blast trenching, prospecting, mapping plus soil and rock sampling. A gossan zone with pyrite and strong fracture filling malachite and azurite was identified between two collapsed adits (Carlyl 1989). These claims eventually lapsed.

The area remained open until August of 2005 when an independent prospector staked four claims over the Grey Copper Hill showing. No work on these claims has been reported and they are now surrounded by the Property.

In the east-central part of the Property, Cominco Limited (Cominco) staked the Beaver claims in 1968 based on results of regional geochemical sampling done the year before. Later that year, L. Elliott staked the nearby Now claims and optioned them to Cominco, who completed mapping and soil sampling in 1968 and 1969 (Johnson and Richardson 1969a and b).

In 1977, the Prism Joint Venture (Asamera Oil Corp, Chieftain Development Company Limited, Prism Resources Ltd, Siebens Oil & Gas Limited and E & B Exploration Limited) staked the area of the Cominco claims as part of a larger block that extended for about 20 km along the north side of the Beaver River. In 1979, Dome Petroleum Ltd. replaced Siebens in the joint venture.

The Prism Joint Venture conducted most of its activities around the original Beaver claims. Soil sampling and mapping were performed in 1977 (Montgomery and Dewonck 1978) and additional soil sampling and trenching were done in 1978 (Prism Joint Venture 1979a). In 1979 the Prism Joint Venture completed six diamond drillholes that totaled 610 m (Dewonck 1980). This work focused primarily on sedimentary exhalative and Mississippi-Valley-type lead-zinc mineralization, but resulted in the discovery of a narrow gold-rich vein (Now Showing).

NDU Resources Ltd. (NDU) staked claims over the Now Showing in 1987 to cover the lead, zinc and silver soil geochemical anomalies identified by Cominco and the Prism Joint Venture. The following year, NDU conducted a geochemical sampling program that focused on the gold vein mineralization at the Now Showing (Cathro 1989).

In 1977, 6.25 km further to the northwest, the Prism Joint Venture conducted mapping, soil sampling and electromagnetic surveys. Numerous samples from that program returned high zinc soil values ranging from 2,100 ppm to 12.2%. One sample collected from a large transported gossan (Ocelot Showing) yielded 3.8 g/t silver, 800 ppm lead and 12.2% zinc (Montgomery and Cavey 1978), suggesting the metals were leached and remobilized in acidic groundwater before being re-precipitated when the fluids were neutralized. These results were not followed up. In 1977, the Prism Joint Venture also performed minor soil sampling near a strong gossan developed along the eastern edge of the Property (Kathy Showing) (Prism Joint Venture 1979b).

In 1979 and 1980, the Prism Joint Venture explored in two areas in the north central part of the Property and conducted prospecting, soil geochemical sampling, and drilled one core hole. This work led to the discovery of scheelite mineralization at the Blue Lite and Flat Top Showings. A tremolite skarn specimen from the Flat Top Showing assayed 8.4% tungsten trioxide, but most material graded below 0.04% (Churchill 1980). No further work was done at either showing.

The GF claims were staked by 39231 Yukon Inc. in August 2004 to cover the Now showing. No work was reported to have been completed on these claims.

In summer 2006, ATAC staked the Rau 1 to 64 claims to cover a drainage where an isolated, high-gold value (150 ppb) was reported by a regional-scale stream sediment geochemical survey, conducted by the Geological Survey of Canada (Hornbrook et al. 1990). This value is in the 99th percentile of gold results from the survey and is supported by a 99th percentile tungsten value (25 ppm). The sample was collected near the Rackla Pluton, 2.5 km east of the Tiger Deposit.

During the staking, a number of rock and soil samples were collected, many of which returned anomalous values for tungsten and a few were notably enriched in gold, lead, zinc, silver, and copper. Cursory prospecting located scheelite-bearing tremolite skarn (Flat Top Showing) and discovered tungsten in diopside-actinolite skarn and highly fractionated intrusive rocks, about 1,500 m to the south.

In 2007, ATAC completed geological mapping, prospecting, grid soil sampling and helicopter-borne variable time-domain electromagnetic (VTEM) surveys (Eaton and Panton 2008). This work partially delineated a large hydrothermal system centered on the largely buried Rackla Pluton. Following that program, ATAC staked the Rau 65 to 96 claims, mostly to improve coverage around a very strong gold-in-soil anomaly outlined on the western edge of the grid overlying what is now the Tiger Deposit.

ATAC and Yankee Hat Minerals Limited signed an option agreement in spring 2008 concerning 40 claims that covered the Rackla Pluton and the tungsten-bearing skarns. During the summer of 2008 Yankee Hat conducted prospecting and a total of 437.4 m of diamond drilling in three holes (Dumala 2008). Several narrow skarn bands with weak to moderate tungsten mineralization were identified within the carbonate host rocks. The option agreement was terminated in late 2008 and the claims were returned to ATAC.

The claims forming the remainder of the Property were staked by ATAC between 2008 and 2010 to cover favourable geology along trend and potential access routes to the south.

On July 2, 2009, ATAC purchased a 100% interest in the GF claims from 39231 Yukon Inc. There are no underlying royalties on these claims.

The 32 Gam claims were staked by ATAC in 2013 and 2016 to cover portions of the final designed tote road access route which fell outside the existing claim block.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The information presented in this section of the report is derived from multiple sources, as cited. The authors have reviewed this information and believe this summary accurately represents the Rau property geology and mineralization as it is presently understood.

7.1 REGIONAL GEOLOGY

The Geological Survey of Canada performed geological mapping in the vicinity of the Rau Property at 1:250,000 scale in the 1960s (Green 1972) and 1970s (Blusson 1978). More recent mapping in the area was completed at 1:50,000 scale by Indian and Northern Affairs Canada (Abbott 1990; Roots 1990) and by Yukon Geological Survey (Colpron et al. 2013).

The Property lies within a band of regional-scale thrust and high-angle reverse faults that imbricate rocks of Selwyn Basin and Mackenzie Platform (Figure 7.1). Selwyn Basin stratigraphy consists of regionally metamorphosed, basinal sediments of Neoproterozoic to Paleozoic age. Mackenzie Platform stratigraphy comprises dominantly shallow-water carbonate and clastic sediments that were deposited from Mid-Proterozoic through Paleozoic times. Both packages of sediments were deposited on the western margin of ancestral North America.

The thrust faults were active during Jurassic to Cretaceous times (160 to 130 Ma), when the area underwent compressional orogenesis related to large-scale plate convergence (Fingler 2005). During Late Cretaceous (94 to 90 Ma), intermediate to felsic plutons of the Tombstone Suite were emplaced (Mortensen et al. 2000). Another compressional orogenic event that occurred about 65 Ma, was accompanied by emplacement of felsic intrusions assigned to the McQuesten Suite.

Figure 7.2 shows regional geology in central Yukon. It is a geological compilation that takes into account recent age dating and new unit correlations that Dr. Charlie Roots prepared for the Yukon Geological Survey (Cathro 2006).

The Tombstone, Dawson, and Robert Service thrust faults, plus a number of lesser thrust faults, affect stratigraphy along the trend of the Rau claim block. All thrusts verge northeasterly and predate emplacement of the Tombstone Suite intrusions. The thrust panel that contains the Property approximately straddles the boundary between Selwyn Basin and Mackenzie Platform and includes units belonging to both tectonic elements.

Figure 7.1 Tectonic Setting

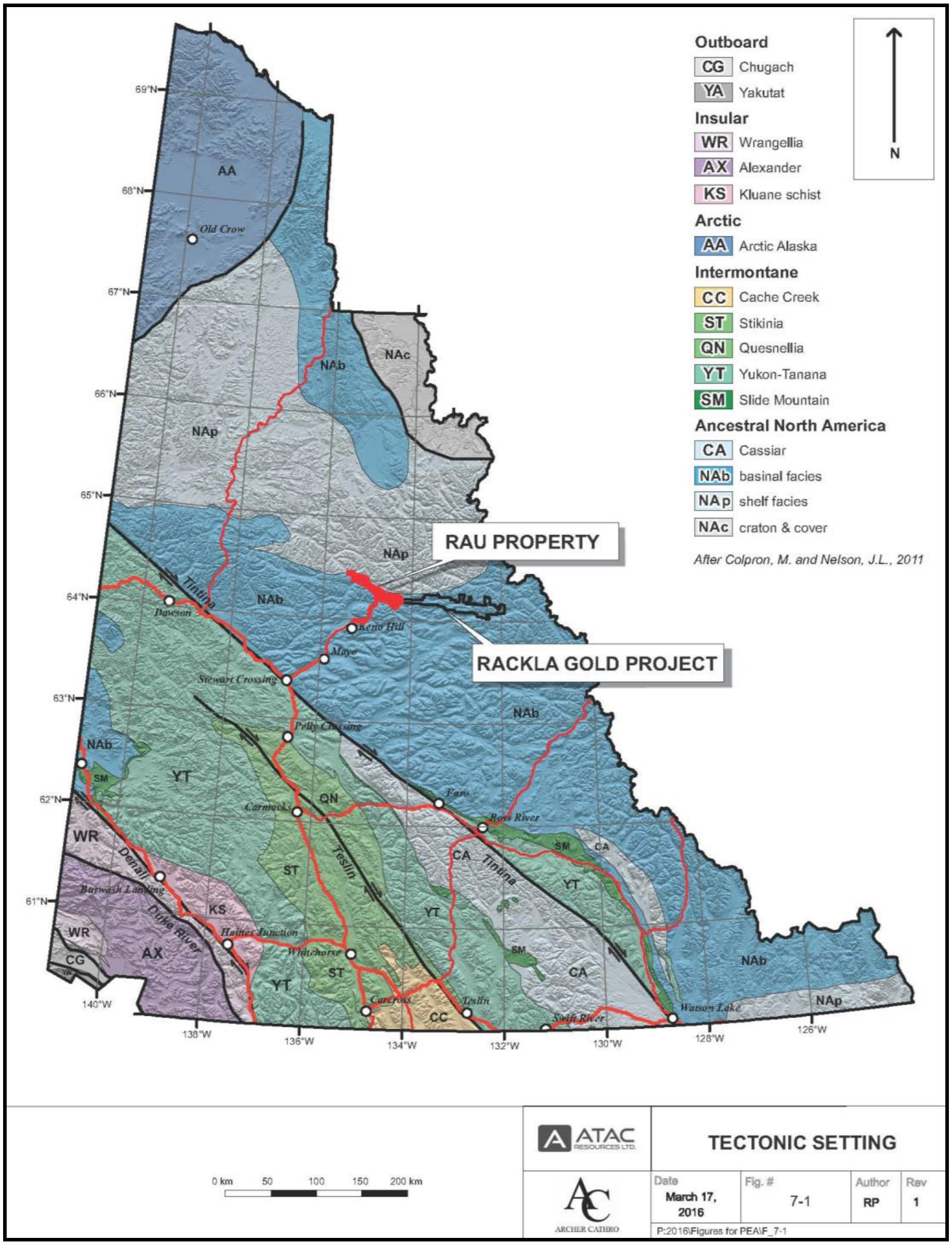


Figure 7.2 Regional Geology

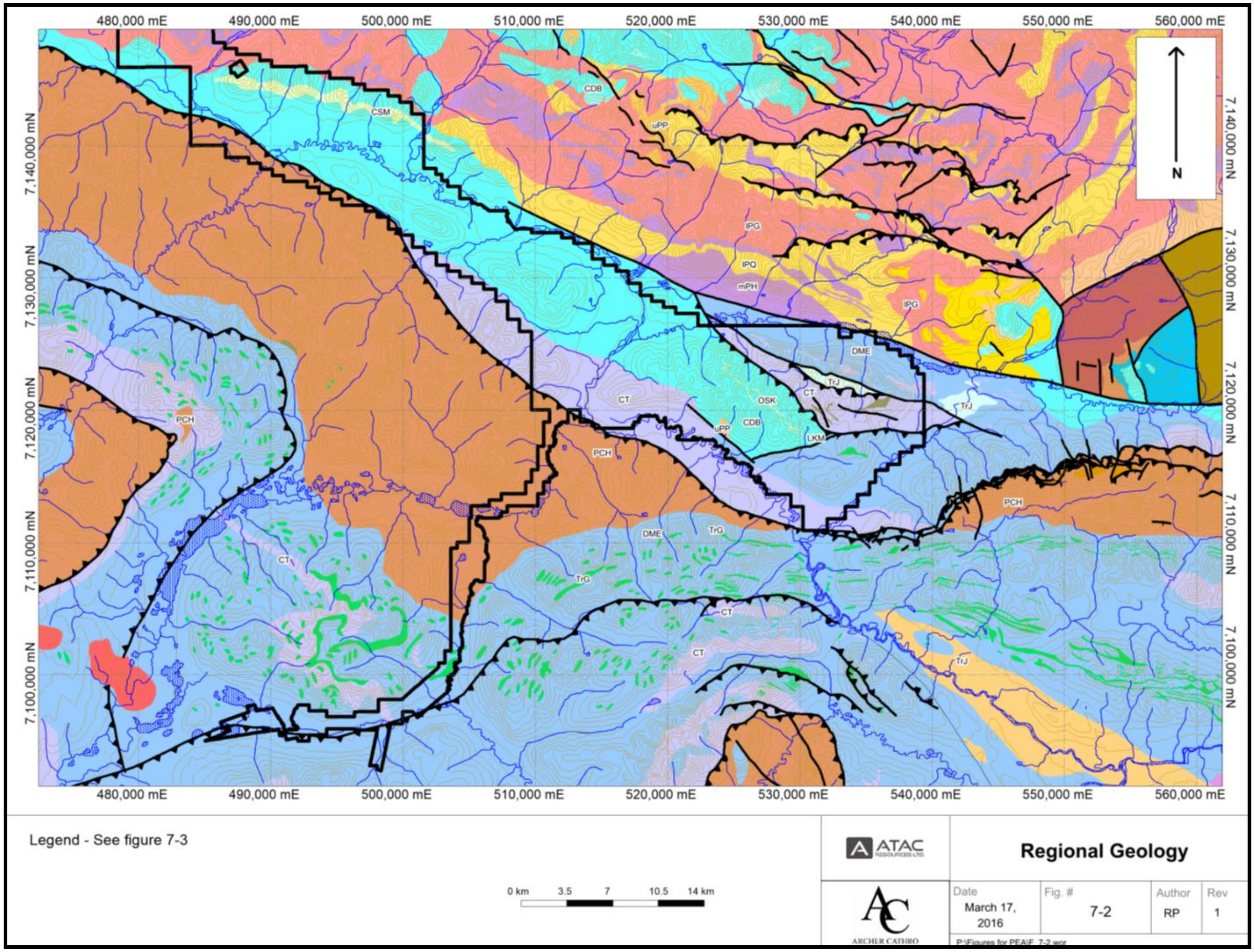


Figure 7.3 Regional Geology Legend

Table 7.1 contains a brief summary of the rock units in the area of the Rau Property.

Table 7.1 Regional Lithological Units (after Roots in Cathro, 2006)

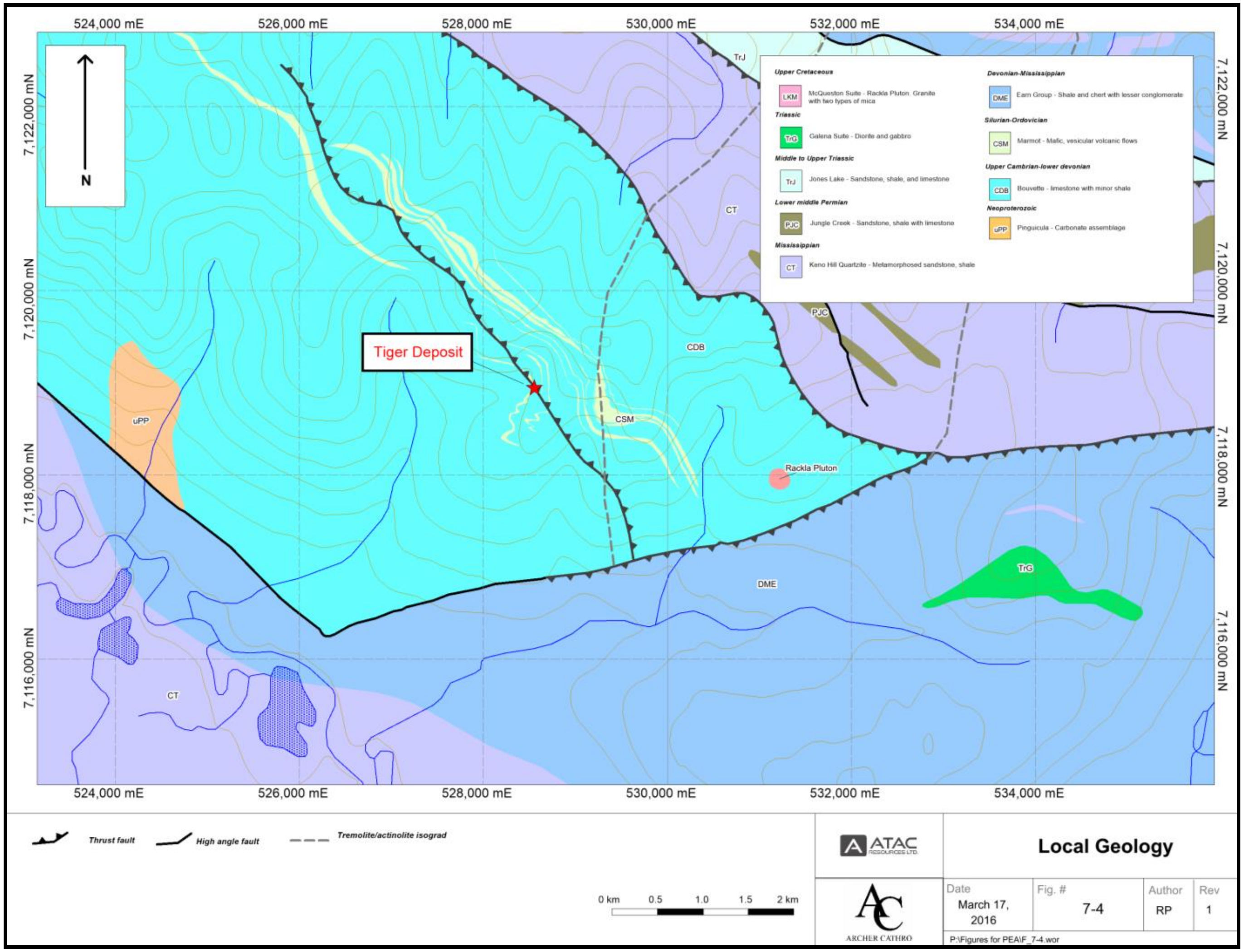
	Unit	Tectonic Element	Age (Ma)	Description
Ancestral North America	Gillespie Lake Group (IPG)	Mackenzie Platform	1700 - 1800	Dolostone and sandstone
	Bouvette Formation (CDB)	Mackenzie Platform	540 - 390	Limestone with rare black shale
	Marmot Formation (CSM)	Mackenzie Platform	540 - 420	Mafic, vesicular and amygdaloidal volcanic flows
	Hyland Group (PCH)	Selwyn Basin	750? - 530	Quartz-mica schist, with rare limestone
	Gull Lake Formation (ICG)	Selwyn Basin	530 - 500	Shale, sandstone, conglomerate and volcanic tuff
	Rabbitkettle Formation (COR)	Selwyn Basin	500 - 480	Silty limestone and limy mica-rich conglomerate
	Road River Group (ODR)	Selwyn Basin	480 - 390	Shale, chert and limy siltstone
Pre-orogenic	Earn Group (DME)	-	390 - 350	Shale and chert with lesser pebble conglomerate, sandstone and grit
	Keno Hill Quartzite (CT)	-	340	Metamorphosed sandstone, minor shale and phyllite
Orogenic	Galena Suite (TG)	-	225	Diorite and gabbro
	Jones Lake and Mt. Christie Formations (TJ)	-	200 - 250	Sandstone, shale and limestone
Post-orogenic	Tombstone Suite (mKT)	-	90 - 94	Granite and granodiorite
	McQuesten Suite (LKM)	-	62 - 67	Granite with two types of mica
-	Overburden (Q)	-	0 - 3	Ice-deposited sand and gravel; river silt

7.2 PROPERTY GEOLOGY

Very little detailed geological mapping has been conducted within the Property boundary, except within the vicinity of the Tiger Deposit. Most work has focused within the favorable Cambrian to Devonian carbonate stratigraphy in close proximity to the Tiger Deposit mineralization. The following descriptions are taken largely from previously documented government and historical mapping.

The Rau Property lies within a northwest trending thrust package bound to the south by the Dawson Thrust and to the north by the Kathleen Lakes Fault (Figure 7.4). Stratigraphy within this package forms open folds that are aligned parallel to the thrusts and plunge gently to the southeast. Several high angle faults that parallel the general structural trend are inferred on the Property and others could be present. One or more of these faults may have acted as a conduit for mineralizing fluids.

Figure 7.4 Local Geology



Early mapping categorized carbonate rocks in the vicinity of the Tiger Deposit as belonging to Ordovician to Devonian Bouvette Formation and was the principal focus of ATAC exploration. More recent mapping has sub-divided this carbonate stratigraphy into three un-named units that young to the northeast. In order from oldest to youngest:

1. Upper Cambrian to Lower Ordovician – massive pale grey dolostone, oncolitic dolostone, minor quartzite and sandy dolostone.
2. Ordovician to Silurian – thin to medium bedded grey and buff weathering silty limestone; massive white limestone, well bedded tan and grey limestone in the upper part of the unit.
3. Silurian to Middle Devonian – thick bedded to massive light grey dolostone and limestone. Dark grey, fetid limestone that contains “two-hole” and “star” crinoids occurs at the top of the unit.

The thickness of this carbonate stratigraphy on the Property is estimated to be at least 1,400 m. The primary focus of mapping has been largely limited to the area around Monument Hill and the Tiger Deposit within the Ordovician-Silurian strata hosting carbonate gold replacement mineralization. Elsewhere the carbonate stratigraphy has not been mapped in detail and remains undifferentiated.

The Marmot Formation consists of thin volcanic horizons that are inter-bedded with the Ordovician and/or Silurian carbonates. The horizons range from a few metres to approximately 20 m thick and comprise dark green to brown weathering mafic, vesicular volcanic flows, carbonate-cemented hyaloclastic breccias and volcanic-derived sandstone, grit and pebble and cobble conglomerate. Locally these horizons are magnetic. Although the Marmot Formation is volumetrically insignificant, it appears to have played an important role in localizing mineralization in the underlying carbonate by acting as an impermeable cap.

Devonian and Mississippian Earn Group rocks are located in the southern half of the Property and bound the carbonate stratigraphy to the south, east and north. This unit is generally recessive weathering and is mostly composed of black shale and chert. To the south a high angle normal fault places Earn Group against carbonates, while a thrust fault marks the southeastern contact. To the north, the Earn Group conformably lies above Cambrian to Devonian shale and limestone, which has been placed against the carbonate stratigraphy by another high angle fault.

The central part of the Property hosts numerous dykes and sills believed to represent a roughly 1,000 m diameter granitic plug referred to as the “Rackla Pluton”. The plug is mostly composed of coarse grained, equigranular, biotite-and muscovite-bearing granite that locally is miarolitic (Panton 2008). The dykes and sills typically range between 30 cm and 7 m in thickness. They are often more fractionated than the plug and include garnet bearing aplite and coarse pegmatite that locally features beryl, amazonite (a green variety of feldspar) and one or more tourmaline minerals (rubellite, indigolite and schorl). The pegmatite bodies comprise mainly orthoclase and quartz but often exhibit abundant lithium-and vanadium-rich micas on their margins.

On surface, the Rackla Pluton is mostly covered by glacial till and only aplite and pegmatite sills and dykes are visible. The pluton is best delineated by its airborne magnetic signature. At the Property scale the pluton is represented by a strong magnetic high. When the data is collapsed to the area immediately surrounding the pluton and a high-pass filter is applied, the signature shows a core magnetic low with a fringing magnetic high.

Analysis of several small bodies of granitic aplite and pegmatite have yielded 40Ar/39Ar muscovite ages of 62.3 ±0.7 Ma, 62.4 ±1.8 Ma and 59.1 ±2.0 Ma (Kingston 2009; Kingston et al. 2010). Based on this data and the composition of the intrusion, Kingston concluded that the Rackla Pluton is younger than the McQuesten Suite (65.2 ±2.0 Ma) intrusive bodies.

Skarn and minor hornfels are developed locally within the carbonates proximal to the intrusions. Skarn grades from distal tremolite-rich (iron-deficient) facies, which are most abundant near the Flat Top Showing (approximately 1,000 m northwest of the pluton), to proximal actinolite-diopside ± garnet ± pyrrhotite (iron-rich) facies, which are found closer to the pluton and on the margins of some dykes and sills. Massive skarns are mostly developed at contacts between limestone and volcaniclastic horizons. Hornfels is restricted to thin volcaniclastic layers within the Marmot Formation. It is normally rusty weathering and often contains disseminated to semi-massive pyrrhotite. Limestone and dolomite are locally altered to marble and often contain disseminated, light grey scapolite crystals. The scapolite is difficult to recognize on freshly broken surfaces but stands out on weathered surfaces as prismatic randomly orientated crystals.

7.3 DEPOSIT SCALE LITHOLOGY

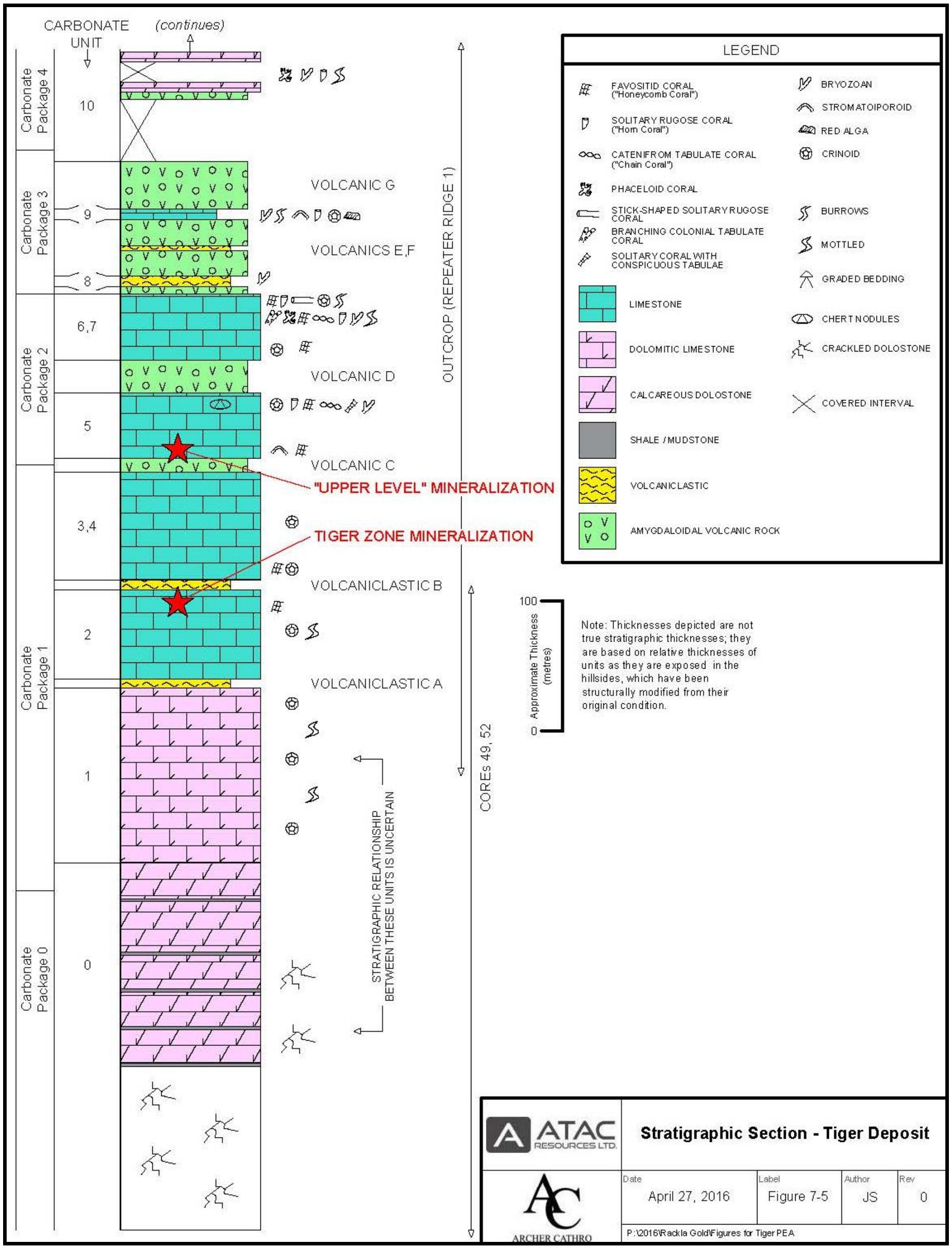
During 2010, detailed deposit-scale mapping was conducted within the Ordovician to Silurian carbonate sequence in the vicinity of the Tiger Deposit mineralization. This work was conducted at different periods during the field season by Dr. Elizabeth Turner (Laurentian University), Dr. Harry E. Cook (Nevada consultant and formerly with the United States Geological Survey), and Archer Cathro personnel. The following descriptions are largely based upon the observations made by Drs. Turner and Cook.

The stratigraphy of the carbonate sequence at the Tiger Deposit was established using rock texture, fossil composition, and relationships with the inter-layered non-carbonate material. Most carbonate, volcanic flow and volcaniclastic lithostratigraphic units exposed at surface are relatively laterally continuous but differences in structural thinning of individual units are evidenced.

The stratigraphic succession exposed at surface above the Tiger Deposit mineralization consists of ten carbonate units (0 to 10) and seven intercalated non-carbonate units (A to G). Carbonate units 1 through 10 are identified based largely on their fossil composition, textures and relationships with associated volcanic and volcaniclastic rocks. These carbonate units are grouped into four subtly distinct packages based on fossil content and rock texture. Non-carbonate units A through G consist of volcanic flows and associated reworked volcanic material.

The relationship of the map units and descriptions are illustrated in Figure 7.5.

Figure 7.5 Stratigraphic Section – Tiger Zone



Mineralization at the Tiger Deposit is hosted by carbonate rocks in the middle of the succession near the contact of Carbonate Unit 2 and Volcaniclastic Unit B. Additional mineralization in an upper horizon occurs within carbonate rocks of the lower Carbonate

Unit 5 immediately above the lowest amygdaloidal volcanic unit. A brief description of all pertinent stratigraphy comprising the stratigraphic column is contained below.

Carbonate Unit 0 consists of graded beds of dark grey to black, variably calcareous mudstone or argillaceous carbonate mudstone, interlayered with paler layers of coarser particles. Original layering is generally obliterated by collapse brecciation, such that dark and light breccia clasts are intermingled. The original layering, where preserved, represents turbidites. This lithofacies passes downward into crystalline dolostone with no hint of original rock texture. The exact relationship with overlying units 1 to 10 is uncertain because the core crosses a presumed fault zone.

Carbonate Unit 1 is dominated by crinoid wackestone and lime mudstone, with rare favositid and halysitid coral fragments and solitary rugose corals. Yellow dolomitic mottles are locally conspicuous.

Volcaniclastic Unit A consists of sericitized silt- to mud-grade clastic material presumed to be of volcanic origin. It may be laterally equivalent to a thin pyroclastic flow exposed on the knoll of the Puma showing.

Carbonate Unit 2 is dominated by lime mudstone with rare crinoid fragments. Layering and sedimentary structures are generally absent.

Volcaniclastic Unit B consists of sericitized silt- to mud-grade clastic material and local granule-grade particles, presumably of volcanic origin. It may be laterally equivalent to a thick pyroclastic flow exposed on the knoll of the Puma showing.

Carbonate Units 3 and 4 are dominated by crinoid wackestone and lime mudstone, with no layering or sedimentary structures.

Volcanic Unit C consists of brownish-green-weathering variably amygdaloidal volcanic flows and associated volcaniclastic material.

Carbonate Units 5 to 7 consist of lime mudstone to crinoid wackestone with rare large fossils that are dominated by a range of tabulate and rugose corals and distributed both as isolated specimens and in conspicuous fossil-rich rudstone to floatstone layers. Carbonates 5 and 6 are separated by a green-weathering volcanic flow unit (Volcanic D). Carbonate 7 is overlain by volcanic flow unit E.

Volcanic Unit D is a conspicuously green-weathering volcanic flow unit that generally lacks vesicular textures. It lies between carbonate units 5 and 6.

Volcanic Unit E is a very thin (several metres), green-weathering amygdaloidal flow unit.

Carbonate Unit 8 is very thin and lies between volcanic flow units E and F. It consists of bryozoan floatstone in a matrix of mixed carbonate mudstone and volcaniclastic fines.

Carbonate Unit 9 thinly separates volcanic flow units F and G, and consists of lime mudstone and skeletal wackestone with a characteristic fauna of bryozoans, rugose corals and crinoid fragments.

Volcanic Unit F is a green-grey-weathering amygdaloidal flow unit.

Carbonate Unit 10 is a group of different rock types, including distinctly mottled carbonates, lenses of amygdaloidal volcanic rock and bright orange marker dolostone layers. The biota is dominated by large phaceloid tabulate corals, bryozoans, and rugose corals concentrated in certain beds only. This part of the succession was not examined in detail. Its contact with the underlying volcanic succession (Volcanic Units E-G) may be structurally modified.

7.4 Mineralization

Several types of mineralization are known to occur on the Property including:

1. sediment-hosted replacement-style gold
2. zinc ± silver ± lead ± gold ± bismuth ± tungsten in limonite-rich veins and replacement bodies
3. scheelite in tremolite skarns
4. pyrrhotite ± scheelite ± chalcopyrite in actinolite-diopside ± garnet skarns
5. wolframite ± tantalite in granite
6. gold bearing quartz-boulangerite veins
7. pyrite-sphalerite-galena in carbonate replacement deposits.

Sediment-hosted replacement style gold mineralization is the most significant economic mineralization explored on the Property to date. Known showings of this type include the Tiger Deposit.

7.4.1 Replacement Style Gold Mineralization – Tiger Deposit

Replacement style gold mineralization has been the primary focus of exploration on the Property. The Tiger Deposit is the best understood. It has been the most aggressively explored occurrence of this type identified to date and is the focus of this technical report.

The Tiger Deposit is located 3 km west-northwest of the Rackla Pluton in a moderate to steep walled valley. It consists of a thick northwesterly trending body of carbonate replacement style gold mineralization hosted by a variably northeast dipping horizon. It has been delineated for about 900m along strike and about 400m in dip extent. The mineralization occurs in multiple zones over an aggregate thickness of up to about 150m. Mineralization is developed within and adjacent to a regionally extensive corridor of highly strained rocks that are manifested as a 40 to 150 m wide zone of small-scale folding and shearing. The geometry of the mineralized system is defined by a series of stacked and folded carbonate horizons intercalated with locally extensive mafic flows and volcaniclastic units.

Most of the exploration at the Tiger Deposit has been directed toward the Discovery Horizon, although there is evidence of at least one additional stratabound interval of gold mineralization above the Discovery Horizon.

Due to a combination of topography, overburden and stratigraphic orientation, the Discovery Horizon is the only mineralized horizon observed at surface. It is exposed over a 75 m long by 10 m wide area on the east side of Tiger Creek. At the northeast end of this exposure, a hand trench dug in 2009 uncovered moderately oxidized limonite boxwork with remnant sulphide mineralization, capped by highly sericite altered volcaniclastic unit. Two samples of sub crop collected in 2008 from near this trench returned 22.5 g/t gold, greater than 1% arsenic, 415 ppm bismuth, and 116 ppm tungsten; and 13.6 g/t gold, greater than 1% arsenic, 410 ppm bismuth, and 51.9 ppm tungsten.

Gold occurs in both sulphide and oxide facies mineralization at the Tiger Deposit. Sulphide mineralization is accompanied by, and developed within, limestone that is replaced by ferruginous dolomite and iron carbonate minerals. Sulphide species consist of disseminated to banded pyrite, with subordinate arsenopyrite and pyrrhotite and minor bismuthinite and sphalerite. Variable amounts of disseminated scheelite are also present. The main sulphide minerals exhibit at least three stages of mineralization.

Oxide mineralization contains weakly disseminated scheelite and is completely devoid of all sulphide minerals. The oxidized rock ranges from very competent, weakly porous limonitic mud to rubbly porous limonitic grit. It appears texturally amorphous within most intersections but occasionally exhibits residual color banding that may represent relict sulphide textures. Complete oxidation extends up to 150 m from surface. The highest-grade and deepest oxidation occurs where northerly trending extensional faults intersect the northwest trending regional shear structure. Detailed observations predominantly collected from drill core on site are described below with respect to pre-mineralizing ground preparation and sulphide/oxide paragenesis. Much of this work is based on paragenetic studies conducted by Eric Theissen for his Master's thesis (Theissen 2013).

Carbonate Ground Preparation

The favorable carbonate lithological horizons consist of Carbonate Units 2, 3, and 4. Carbonate Unit 2 is expressed in drill core as mineralized and non-mineralized Tiger Deposit equivalent stratigraphy and Carbonate Units 3 and 4 occur stratigraphically above the Tiger Deposit mineralization.

Ground prepared units are characterized by grey fluid-assisted to solution collapse brecciated lime-mudstone to dolo-mudstone. Clasts average 3 to 10 cm and have sub-angular to sub-rounded to corrosive-irregular margins with many re-entrants due to dissolution and subsequent clast formation. These primary carbonate clasts average a homogenous to slightly mottled medium-grey colour. Mottling is due to bioturbation as well as irregular anastomosing stylo-mottling. Fossils and clastic textures are rare. Single-seamed serrated stylolites are common and may be as abundant as several hundred per metre in places. Polyphase carbonate and silicate fluid events establish what is observed today as classic karst dissolution and phreatic zone precipitation followed by subsequent open-space filling fluids.

The delicate and irregular margins of the carbonate clasts as well as the rarely preserved speleothems are the product of meteoric karsts that differ from the “puzzle piece” angular fragments commonly produced from tectonic brecciation. The clasts and open space margins are lined by “dog-tooth” calcite spar, white sub-centimetre size angular calcite crystals which are in turn often rimmed by a thin veneer of sub-mm size tabular pyrobitumen. The “dog-tooth” calcite spar cement, which lines open spaces and clasts, is a product of calcite crystallization in the phreatic zone of the meteoric realm, meaning that the open spaces were fully saturated with meteoric fluids allowing isopachous crystallization on all clast faces and open spaces. This calcite spar is also differentiated from marine cement since it exhibits a low-magnesium calcite.

Regionally thrusted and compressed basinal shales may be the source of pyrobitumen that channel through platformal porosities. It is likely that petroleum residues once resided in the karst produced openings, later remobilized and are now lining the calcite spar. Open space filling is primarily composed of a clear anhedral pyrite-bearing quartz-calcite phase with other phases of grey coarse-grained calcite filling voids.

This carbonate unit is interpreted to have acted as a fluid pathway or ground preparation for the mineralization present today. The favorable mineralized horizon, Carbonate Unit 2, bounded by two volcanic packages, is exposed at surface.

Tiger Deposit Sulphide Paragenesis

The primary Tiger Deposit mineralization occurred in at least three distinct events with potentially more cryptic events. The earliest recognizable event in the Tiger Deposit, phase one, is a pervasive, fabric destructive, re-crystallized hydrothermal ankerite phase associated with arsenopyrite. The ankerite occurs as medium to coarse-grained euhedral to anhedral angular crystals that have no distinct sign of strain. Some mineralized open spaces, postulated to be equivalent to the karstification vugs in non-mineralized units, display hydrothermal saddle ankerite with curviplanar crystal faces. The ankerite varies in colour from a deep peach-salmon colour to a white-buff colour.

Ankerite in the Tiger Deposit occurs in these two seemingly distinct forms, the white version is dolomite as confirmed using the carbonate staining technique ‘Alizarin Red and Potassium Ferricyanide’. The arsenopyrite occurs commonly as disseminations to weakly bedding/cleavage plane parallel and crystals are medium to coarse-grained and commonly euhedral. Arsenopyrite crystals appear to be in equilibrium with the initial carbonate phases as well as being unstrained. The arsenopyrite and ankerite are thought to have been precipitated by a common arsenian and iron rich hydrothermal fluid.

Phase two of the sulphide mineralization is characterized by pyrite precipitated in a strained environment to produce parallel bands (stripes) that give rise to the "Tiger" stripe nature of sulphide mineralization.

This pyrite is referred to as pyrite-1 and occurs as medium to fine-grained commonly cubic euhedral grains that cut between ankerite grain boundaries and across arsenopyrite in a non-destructive manner. Moreover, pyrite 1 overprints the ankerite and arsenopyrite yet appears in equilibrium with these phases as they remain euhedral and in their primary form. The pyrite-1 banding is parallel to the banding developed within the volcanic packages bounding the Tiger Deposit stratigraphy, and thus thought to be coeval. Although no obvious ductile structures occur within the Tiger Deposit sulphide mineralization, sigmoidal shear bands amongst other shear sense indicators occur in the foliation parallel fabric of the bounding volcanic units (Fedorowich 2011). These observations as well as the lack of brittle features within the Tiger Deposit sulphide mineralization indicate that pyrite-1 was likely precipitated during a ductile stress regime. This shearing is also likely responsible for the alignment of the micaceous cleavage in the bounding volcanic packages and has aided in the broken nature of those units along their sericitized micaceous foliations.

A grey coarse-grained euhedral and often zoned ferro-dolomite occurs as a late stage mineral phase within phase two that hosts pyrite-1 mineralization. This ferro-dolomite occurs in bands parallel to pyrite-1, and may exist as thin millimetre-scale or thick centimetre-scale units. The ferro-dolomite often occurs along the same foliation plane as pyrite-1, surrounding the sulphide and giving the appearance of coeval precipitation. This ferro-dolomite is suggested to be a late stage mineral in the same phase because it also intrudes parallel and between pyrite-1 bands plastically deforming the once planar pyrite-1 fabric.

The next stage in mineralization, phase three, consists of an intruding fluid phase of quartz-ferro-calcite-talc +- pyrite-pyrrhotite-bismuthinite-sphalerite and is potentially associated with secondary magnetite and biotite. This fluid phase is destructive and overprints all minerals in phases one and two. A second pyrite (pyrite-2) occurs and overprints all mineral phases in phase one and two, but is overprinted or destroyed by the intruding fluids of phase three.

Pyrite-2 occurs as fine to very-fine anhedral grains commonly with a dull green hue and pervasively overprints all previous phases.

Its most common occurrence is as anhedral diffuse masses but also occurs parallel to pyrite-1 bands and cuts obliquely across foliation to pyrite-1. Rather than placing pyrite-2 into its own class or fluid phase, it is suggested that pyrite-2 is a product of pyrite-1 from phase two interacting with the fluids and stress regime of phase three. Because pyrite-2 is cut by the mineral phases in phase three, phase three is by inference a later phase. Pyrite-2 often forms parallel to pyrite-1 bands, and is commonly rimmed on its margins by a medium to fine-grained pyrite diagnostic of pyrite-1. This infers that pyrite-2 is not necessarily a newly precipitated mineral but that it could be the product of re-crystallization of pyrite-1 in a slightly different stress regime. The fine-grained dull nature of pyrite-2 thus may be characteristic of sub-grain formation via dynamic recovery mechanisms or thermally induced grain boundary migration.

The notion of phase three occurring in a different stress regime is due to oblique overprinting of pyrite-2 over pyrite-1, as well as phase three fluids that commonly crosscut earlier phases oblique to the pyrite-1 bands.

Phase three minerals consist of quartz-ferro-calcite-talc ± pyrite-pyrrhotite-bismuthinite.

This mineral phase is not always observed together but when they do they exhibit a distinct relationship. Rimming phase three is a grey to white coloured fibrous ferro-calcite, the fibrous nature is accompanied by talc crystals and occur perpendicular to the phase's contacts. Quartz commonly is central in the intruding phase as coarse-grained sub-angular grains. Pyrite-3 occurs as medium to coarse-grained angular to euhedral grains disseminated within the fluid phase as well as disseminated overprinting phase one and two minerals in close proximity to phase three.

The pyrrhotite occurs mostly as fine-grained anhedral masses within the ferro-calcite or interstitial to the quartz, and more rarely interstitial to the quartz and more rarely interstitial to the phase one and two minerals. Pyrrhotite rarely occurs as 1 to 2 m intervals of massive sulphide where phase three fluids have had profound influence. Red sphalerite occurs in very small amounts as anhedral fine- to medium- sized grains within late calcite veins that cut all phases. Phase three is associated with the destruction of previous mineralization phases including arsenopyrite.

Tiger Deposit Oxide Mineralization

The overall character of the oxide zone is partial to complete destruction of primary features and rarely preserved secondary features. The oxide is a combination of siderite, goethite and limonite (potentially more phases) that vary from moderately hard competent sections to gritty-clay to silt rich rubble. Oxide colour varies from deep red to bright orange-rust to dark brown in color.

Transition zones of oxide to sulphide where the rock has not undergone complete destruction, support first order observations that can be made on general paragenesis. Non-oxidized rock is often equivalent to Tiger Deposit sulphide mineralization with minor but important differences. Typically the ankerite and phase three minerals are present; however there is usually a depletion of arsenopyrite accompanied by strong iron staining throughout. Strongly oxidized portions may show a fine-grained diffuse pyrite (pyrite-2?) that is resistant to the oxidation. Brittle core axis parallel fractures occur more frequently in these sulphide-oxide transition zones and are thought to be attributed to a higher fracture density proximal to the late north trending structures.

Alteration Phases

Sericite alteration is light brown to pale yellow and often occurs within the volcanic horizons proximal to the Tiger Deposit. The sericite is best developed at the upper and lower contacts of the volcanic packages bounding the Tiger Deposit sulphide mineralization. The sericitized volcanics have a preferred banding/cleavage developed parallel to pyrite-1 and is thought to be coeval with pyrite-1. Pyrite-2 is observed to overprint sericitized volcanic units thus sericitization occurred before pyrite-2 and phase three mineralization.

Talc forms white-to-grey fibrous-to-radiating crystals associated with the ferro-calcite minerals of phase three. The talc is most often rimming a phase three fluid intrusion with ferro-calcite surrounding a central quartz phase.

Potassic alteration occurs as an overprinting biotite-magnetite-calcite phase mostly within the volcanic packages, and more specifically within the sericitized units. This phase commonly intrudes and occurs as bleb-like clear calcite masses with rounded boundaries connected to one another by an irregular stockwork pattern.

These intrusions bend and warp the cleaved and wispy sericitized ash within the volcanics. Rimming the calcite phase is a fine-grained biotite that lines the calcite contacts by a thin brown demarcation. Magnetite often occurs as fine-grained euhedral disseminations throughout the volcanic units and may occur in more concentrated masses. This late phase fluid also precipitates an anhedral fine-grained pyrite and pyrrhotite usually within the calcite phase. Relationships of this phase and phase three mineralization are unknown.

Lower Pyrite Zone

The Lower Massive Pyrite Zone is only observed in diamond drill hole Rau-08-18. The host lithology is a heavily altered cryptic carbonate unit that occurs stratigraphically beneath Tiger Deposit mineralization. Pyrite mineralization occurs in intervals tens of metres in length and is closely associated with quartz. The pyrite is coarse-grained, generally massive, angular to euhedral textured with variable amounts of clear to white anhedral quartz within interstices. Pyrite often exhibits a brittle fracturing habit of coarse grains which are subsequently annealed around grain margins. These pyrite grain margins exhibit no fracturing and appear to be a recrystallization of the fractured pyrite. All primary textures of the limestone have been destroyed by the pervasive silicification.

This late quartz-base metal mineralization is also observed as a late phase in the Tiger Deposit. Overall, this style of mineralization does not appear to be associated with the earlier gold mineralizing events.

East Zone Mineralization

The East Zone occurs in a litho-stratigraphical unit equivalent to the Tiger Deposit mineralization down dip to the southeast and structurally down dropped. Key distinctions that differentiate the East Zone from the Tiger Deposit are a decrease in hydrothermal ankerite and arsenopyrite and an increase in phase three mineralization, in particular pyrite-3 accompanied by pyrrhotite and talc.

The East Zone horizon, where unaltered, displays karst solution brecciation and subsequent dogtooth spar, bitumen, and quartz infilling. This carbonate package is bound by traceable volcanic units that correlate to the equivalent litho-stratigraphy as the Tiger Deposit horizon. Typical Tiger Deposit mineralization, ankeritization and foliated pyrite (phase 1 and phase 2), occur replacing carbonate textures in only a few of the East Zone intersections. Tiger Deposit equivalent style mineralization within the East Zone occurs in small discrete intervals, has low amounts of arsenopyrite, and becomes non-existent down dip to the southeast. Massive pyrite and quartz occur in discrete intervals separated by a light-grey 'bleached' silicified limestone.

Phase three mineralization (quartz-calcite-talc-pyrrhotite-pyrite-3-bismuthinite) occurs in much greater abundances in the East Zone overprinting mineralized units as well as overprinting unmineralized karsted limestone. Mineralization ranges from long intervals of coarse to fine-grained pyrite, to massive pyrrhotite and is commonly associated with extensive talc alteration. The pyrite occurs either as "splashy" medium-grained disseminations, massive coarse-grains or massive fine-grains, and is often associated with quartz. All of these pyrite types overprint all previous phases including brecciated limestone and Tiger Deposit style mineralization.

Pyrrhotite-talc and lesser bismuthinite also occur in the East Zone in much higher proportions than in the Tiger Deposit and do not appear to be positively correlated to gold grade.

East Zone Alteration

The volcanic packages bounding the East Zone are traceable and believed to be equivalent with the Tiger Deposit volcanic horizons. However, the volcanic horizons in the East Zone show a much stronger sericite alteration being very pale yellow and are often strongly cleaved. The sericite is mostly localized within what appears to be fine-grained volcanic ash and pumice fragments that occur as weak bands.

The East Zone volcanics are strongly altered by a calcite-biotite-pyrite-pyrrhotite-arsenopyrite phase. This alteration is much more pervasive and extensive compared to the Tiger Deposit "potassic" alteration and in particular has much more biotite and pyrrhotite throughout. The calcite intrudes in rounded irregular masses connected by a stockwork calcite matrix. This phase bends and warps the sericitized ash fragments as it intrudes. The calcite is rimmed with fine-grained brown biotite crystals and has fine-grained anhedral pyrite and pyrrhotite within the phase. Arsenopyrite occurs as medium-grained euhedral crystals within this phase in the volcanic horizons above the East Zone.

Upper Tiger Zone

A mineralized zone, known as the Upper Tiger Zone, was discovered by drilling above the East Zone mineralization in 2010 above the amygdaloidal Volcanic Unit C (Dumala and Lane 2010). This zone is between 4 and 11 m thick and is almost identical to typical Tiger Deposit sulphide mineralization on the west side of Tiger Creek. The ankerite in the Upper Tiger Zone is white and pyrite-1 is difficult to distinguish. The arsenopyrite is coarse-grained and very prevalent throughout the unit and appears to be in equilibrium with all other phases present. The upper and lower contacts of the Upper Tiger Zone are very sharp and consist of a white marble. Phase three mineralization is observed in small amounts in the Upper Tiger horizon.

Peripheral Occurrences

Several other showings containing mineralization similar to that found at the Tiger Deposit have been identified on the Property. These include the Bengal, Cheetah, Condor, Cougar, Cub, Jaguar, Kitty, Lion, Lynx, Panther, Puma and Serval Showings (Figure 4.3). All of the showings are occurrences of mineralized float found on grassy or talus covered slopes and ridges. Of these, only the Cheetah, Condor, Kitty and Puma have received limited diamond drilling.

8.0 DEPOSIT TYPE

The Tiger Deposit represents a distal variety of the reduced intrusion-related gold system ("RIRGS") model. The following is based on a paper completed by Craig Hart (Hart 2007) and work completed by Eric Theissen for his Master's thesis (Theissen 2013).

Intrusion-related gold systems have been divided into two categories, "reduced" and "oxidized" (Hart 2007). Reduced systems are a distinct class that lacks anomalous copper, has associated tungsten, contains low volumes of sulphide minerals, has a reduced sulphide mineral assemblage, and is associated with felsic, moderately reduced plutons. Oxide systems are mostly gold-rich (or relatively tungsten-poor) variants of the porphyry tungsten deposit model, associated with more mafic, oxidized, magnetite-series plutons.

RIRGS are best developed around small, cylindrical-shaped plutons that have intruded sedimentary or metasedimentary host rocks. Plutons are typically emplaced between 5 and 9 km depth. Fluid flow and mineralization is largely controlled by structural features.

The dominant structural control on RIRGS is weak extension that forms arrays of parallel fractures in the brittle carapace. These are filled with auriferous, low-sulphide quartz veins that form extensive, intrusion-hosted sheeted arrays.

Mineralization often extends beyond the limits of the intrusion and locally beyond the limits of the thermal aureole. RIRGS exhibit predictable zonation and differing deposits styles outward from the central, mineralizing pluton, with increasing structural control on the more distal mineralization. Tiger is located ~3 km west-northwest of the causative Rackla pluton and exhibits typical carbonate replacement style mineralization in this distal part of the system.

Low-sulphide contents (0.1 to 2.0%) are common in intrusion-hosted systems, and the sulphides are dominated by pyrite, pyrrhotite and arsenopyrite, with accessory scheelite and bismuthinite. Arsenopyrite is more abundant in veins outside of the intrusion and is commonly associated with pyrrhotite in replacement-style mineralization.

Dublin Gulch (Yukon) and Fort Knox (Alaska) are examples of RIRGS, both representative of the more characteristic sheeted vein deposit style, whereas Brewery Creek represents a more shallow-level, distal variety. These deposits are all associated with Tombstone Plutonic Suite intrusions (90 to 100 Ma). Dublin Gulch is located 72 km west-southwest of the Tiger Deposit.

The Tiger Deposit has many characteristics that resemble the distal expression of a RIRGS in the northern Cordillera. However, it is associated with the much younger Rackla pluton (62.9 ±0.5 Ma), which belongs to the McQuesten Plutonic Suite. This plutonic suite is poorly studied and was not previously recognized as a prospective target for gold mineralization.

9.0 EXPLORATION

Exploration activities on the Property prior to 2007 are referenced in this report as historical activities and are described in Section 6.0.

The diamond drilling programs, conducted between 2008 and 2019, that are the basis for the Mineral Resource estimation reported herein, are discussed in Section 10.0.

9.1 SOIL SAMPLING

Soil geochemical sampling was conducted between 2007 and 2019 on the Property. The majority of the samples were taken from grids and contour lines within an 8 km wide by 20 km long area along the Rau Ridge System. A total of 28,163 soil samples have been collected on the Property.

Grid samples were collected at 50 m intervals along lines spaced 50 m apart over most of the known showings. In areas where there were no known showings, samples were collected along contour lines at 50 m intervals. The relative sample and line positions were established using a handheld GPS. Prior to 2010, sample spacing along lines was maintained using compass and topofil chain. Sample sites are marked by wooden lath bearing aluminum tags inscribed with the corresponding sample number and the grid coordinates, where appropriate.

All soil samples were collected from holes that were dug with a mattock or hand auger to depths of 20 to 50 cm below surface. Soil was taken from the bottom of the holes and placed in pre-numbered Kraft paper bags. Above tree line, the samples consisted of poorly developed soils mixed with talus fines. At lower elevations, the sampled material mostly comprised residual soils mixed with glacially transported material.

Background and anomalous values for gold, arsenic and bismuth are summarized in Table 9.1. Background averages, weak, moderate, strong and very strong anomalous thresholds approximately correspond to the 50th, 90th, 98th, 99th, and 99.9th percentiles.

Table 9.1 Geochemical Characteristics for Soil Samples, Rau Property

Level	Gold (ppb)	Arsenic (ppm)	Bismuth (ppm)
Background	3	17	1
Weak	15	50	5
Moderate	50	150	10
Strong	100	250	25
Very Strong	500	1,500	200
Peak	11.65 g/t	>1%	1,300

Integrated soil geochemical results for gold and arsenic are illustrated on Figure 9.1 and Figure 9.2, respectively.

Figure 9.1 Gold-in-soil Geochemistry

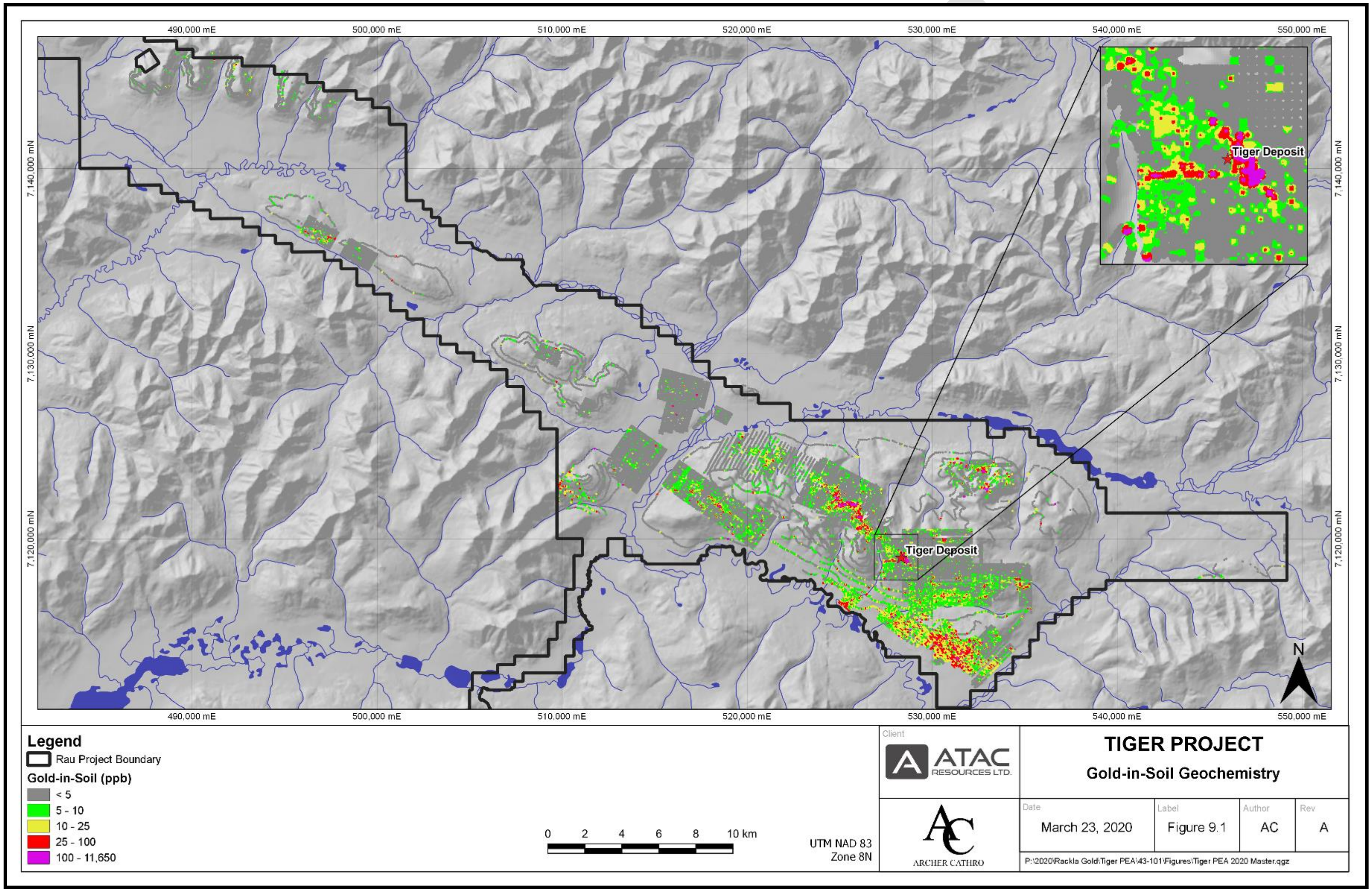
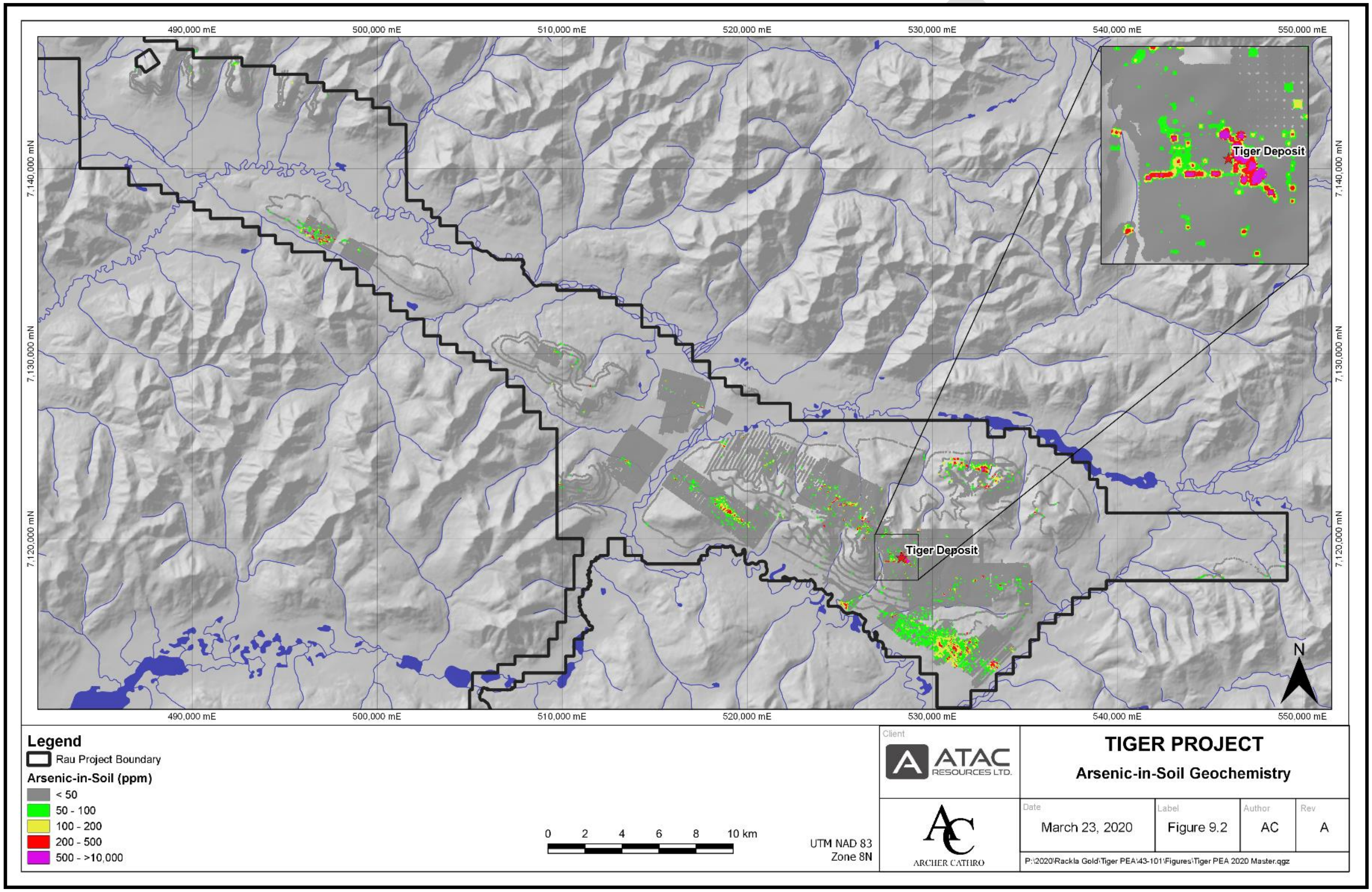


Figure 9.2 Arsenic-in-soil Geochemistry



9.2 SURFACE ROCK SAMPLING

A total of 2,344 rock samples were collected from various targets along the Rau Trend between 2007 and 2019.

Rock samples included measured chip samples across mineralized zones, grab samples collected from selected mineralized intervals, and mineralized float samples. Rock sample sites were marked with orange flagging tape labeled with the sample number. The location of each sample was determined using a handheld GPS unit. Rock samples taken at surface are commonly strongly weathered with sulphides oxidized to limonite. These samples often yields higher metal assays than unweathered mineralized samples.

During 2007, localized prospecting was done around the periphery of the Rackla Pluton in close proximity to the surrounding carbonates focusing on proximal tungsten-gold skarn potential. Samples of mineralized actinolite-diopside skarn generally returned low values but some material returned up to 3.23% tungsten trioxide and one sample yielded a gold value of 1.24 g/t (Eaton and Panton 2008).

Additional prospecting ensued in 2008 following up very strongly anomalous gold-in-soil geochemistry from grid soil sampling completed in 2007. Grab samples were collected mostly from an area of strong gold-in-soil geochemical response which resulted in the discovery of oxide gold mineralization at the Tiger Showing in Tiger Creek. Samples collected from the exposed Discovery Horizon yielded 22.5 g/t gold, greater than 1% arsenic, 415 ppm bismuth, and 116 ppm tungsten and 13.6 g/t gold, greater than 1% arsenic, 410 ppm bismuth, and 51.9 ppm tungsten. A limonitic float sample taken 90 m downstream contained 241 g/t silver, 3,730 ppm arsenic, 388 ppm bismuth, 3.27% lead, and 2.09% zinc (Dumala 2009).

Prospecting between 2009 and 2019 focused primarily along strike from the Tiger Deposit, following up soil geochemical anomalies defined at intermittent locales. A number of new showings were identified by this work. These include the Lion, Cub, Jaguar, Panther, Cougar, Puma, Cheetah, Lynx, Ocelot, Caracal, Bengal, Condor, Serval, Spotlight, Bobcat, and Airstrip.

Mineralized float (Cub Showing) was found 575 m to the east-northeast of the Tiger Showing within a 110 m wide by 250 m long area on a south facing talus covered slope. Samples from the Cub Showing returned peak gold values of 1.15 g/t and 1.08 g/t. Neither of these samples had any noteworthy values for other metals. A sample taken from the southeastern edge of this showing contained 18.15 g/t silver, 4,630 ppm lead, and greater than 1% zinc. Nine other samples collected from the showing returned between 0.98 to 6.54 g/t silver.

Samples collected 350 m east of the Tiger deposit, and 230m downslope to the southwest of the Cub Showing contain elevated silver, copper, lead, and zinc. With peak values of 18.15 g/t, 4.63%, 1.14%, and 4.36% respectively, from separate samples.

The Bengal Showing was discovered in 2012, 3.2 km south of the Tiger Deposit, while following up an intermittent gold-in-soil anomaly. Hand trenching exposed highly friable argillites interbedded with fossiliferous carbonate debrite layers. Samples of oxidized carbonate returned elevated values ranging from 0.3 to 1.61 g/t gold, while samples of the argillite ranged from 0.25 to 4.59 g/t gold.

The Puma Showing is located 4.3 km northwest of the Tiger Deposit and is has a similar geochemical signature to the Tiger Deposit. Goethite-rich limonite grab samples collected in 2009 yielded anomalous values. Six samples returning greater than 1.0 g/t gold, including a peak value of 18.45 g/t.

The Bobcat Showing was discovered in 2018 and is located 4.5 km to the southeast of the Tiger Deposit. Grab samples of distal, retrograde skarn mineralization was collected from an area of altered limestone and marble bedrock. Of the 168 rock samples collected at the Bobcat showing, 34 yielded greater than 1.0 g/t gold, 9 of which were greater than 10.0 g/t, with a peak value of 50.8 g/t. Select, highlight grab samples include 41.90 g/t gold and 10.55% copper, and 16.15 g/t gold and 7.24% copper.

9.3 GEOPHYSICS

A variety of airborne and ground geophysical work was carried out over the Rau Ridge system hosting the Tiger Deposit mineralization between 2007 and 2019. Airborne surveys consisted of Vertical Time Domain Electromagnetic (VTEM) and Z-axis Tipper Electromagnetic (ZTEM) surveys while ground surveys consisted of Induced Polarization (IP)/Resistivity and Gravity. The airborne surveys covered the entire Rau Ridge system while the ground based surveys were conducted specifically over the Tiger Deposit, select known showings, and isolated VTEM/ZTEM anomalies.

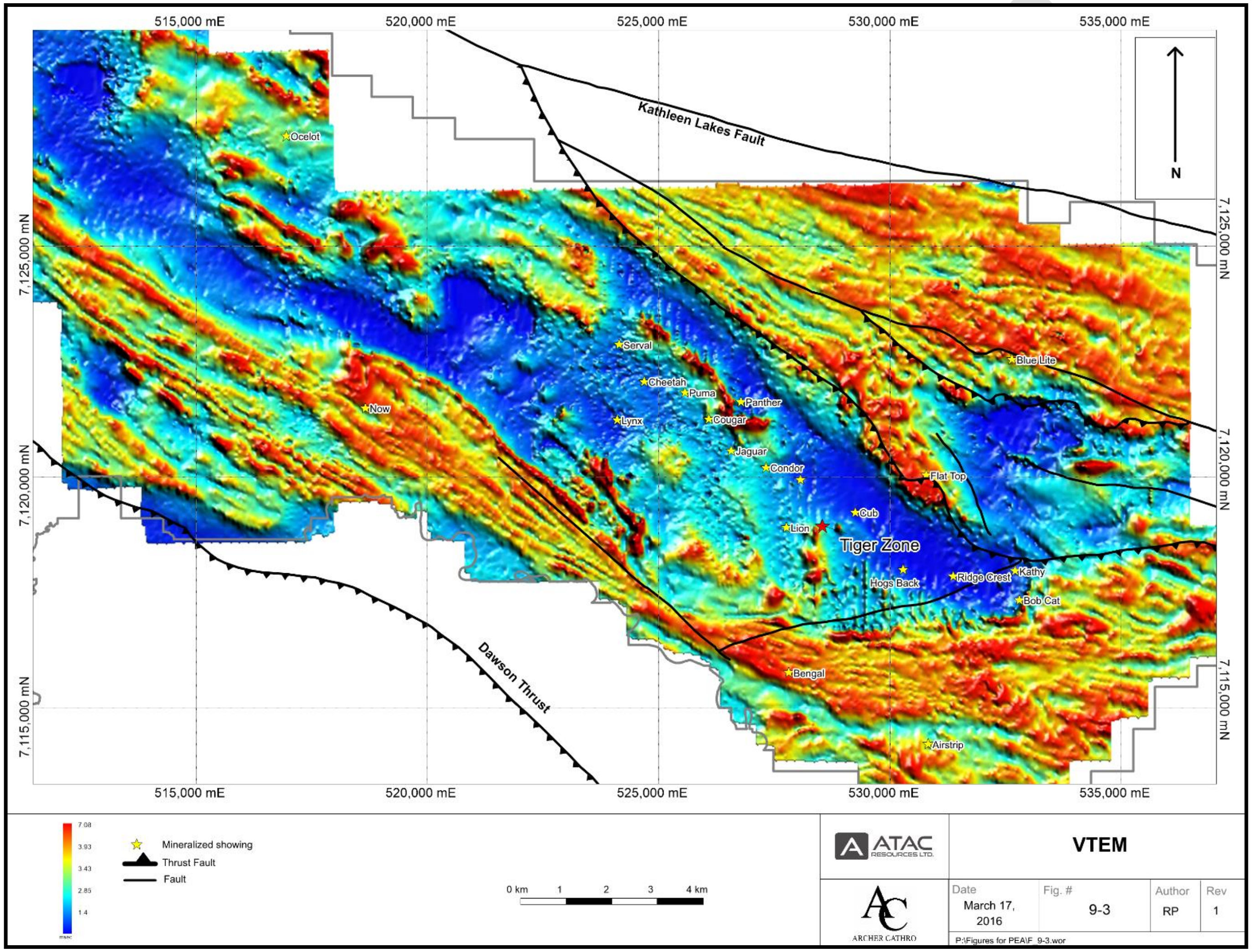
In addition to these airborne and ground based surveys, Scintrex Limited was contracted to conduct a specialized borehole gravity meter survey in a number of drillholes that intersected significant oxide mineralization for purposes of determining in situ specific gravity. This survey is discussed in more detail in Section 12.4.4.

Helicopter-borne magnetic and VTEM surveys were flown over the 64 claims that comprise the core of the Property on August 12, 2008 by Geotech Ltd. of Aurora, Ontario. A total of 135.09 line-kilometers were flown on north-south lines spaced 100 m apart. The total magnetic data outlined an area of high susceptibility directly over the Rackla Pluton. This high gradually weakens to the west but continues into the area of dykes and sills, suggesting that these tabular intrusions may coalesce with the Rackla Pluton to form a larger body at depth. Strings of weaker magnetic highs in the western part of the survey area are closely correlated magnetic volcanic horizons correlative with Units E, F and G.

Additional VTEM surveys were completed over the rest of the Rau Ridge system in two phases on July 14 and 15 and between August 13 and September 23, 2008 by Geotech Ltd. of Aurora, Ontario. In total, 2,994 line-kilometers were flown on north-south lines spaced 100 m apart. The preliminary total magnetic and electromagnetic data outlined a strong, 23.5 km long, northwest trending linear feature originating near the Rackla Pluton.

In spring 2009, Condor Consulting Inc., was retained to complete processing, analysis, and interpretation of electromagnetic and magnetic data obtained from VTEM surveys completed in 2007 and 2008. Condor's work outlined a series of conductive units originating from the Tiger Deposit and extending approximately 15 km northwest along trend (Figure 9.3). Many of these conductors parallel the property extensive shear zone believed to be associated with the fluid conduit localizing gold at the Tiger Deposit. An approximately 25 km long linear magnetic high, corresponding to the Marmot Formation volcanic units can be traced through the center of the Property. Typically, conductors can be found along the south side of this magnetic feature.

Figure 9.3 Modelled VTEM Anomalies



In late spring 2010, Geotech Ltd. was contracted to complete two helicopter-borne ZTEM and aeromagnetic geophysical surveys over the Property. The initial survey was conducted between May 27 and June 6, and comprised 331 line-kilometers, while the second survey was completed between June 14 and 22 and totaled 3,018.6 line-kilometers which included the area covering the Rau Ridge system.

Strings of ZTEM conductors were identified throughout the Property, many of which correspond with the VTEM conductors. A number of new anomalies were also identified a short distance from the Tiger Deposit with similar geophysical signatures which were the focus of some of the 2010 exploration. Figure 9.4 illustrates the extent of the ZTEM survey and the associated anomalies identified within the vicinity of the Rau Ridge system.

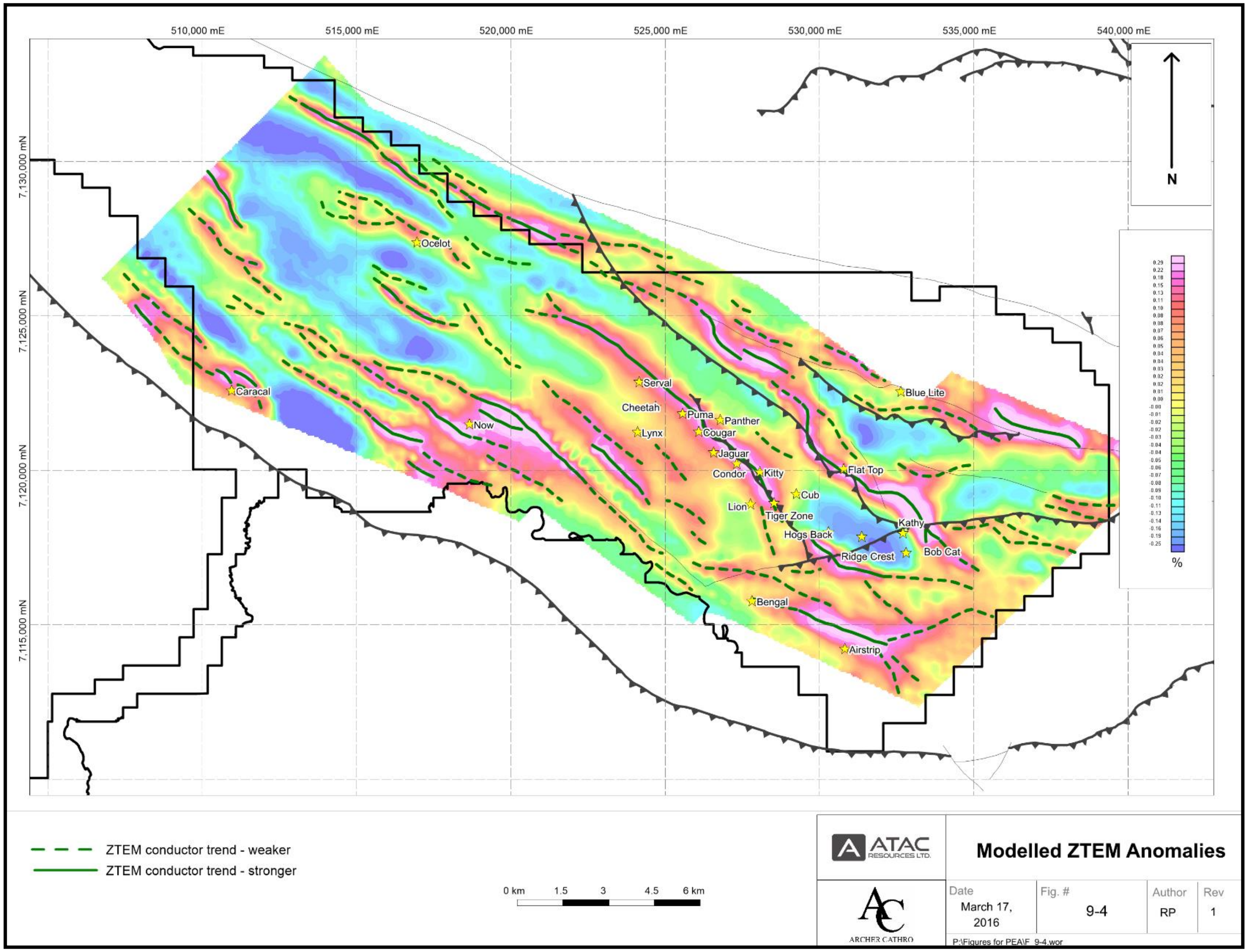
A modified ground pole-dipole IP survey was completed at the Tiger Deposit by Aurora Geoscience of Whitehorse between August 4 and 13, 2009. Three lines, totaling 4.2 line-kilometers corresponding to section lines 10+400NW, 10+120NW and 09+650NW were tested by this survey.

On section line 10+400NW, a well-defined chargeability high is located to the southwest of the baseline. The lower portion of this anomaly would have been pierced by hole Rau-09-52, which intersected unmineralized, brecciated limestone in this area. Unfortunately noise on section line 10+120NW made interpretation at depth near the massive pyrite intersection in hole Rau-08-18 unusable. Only the near surface data (less than 100 m) was usable in the area of interest. On section line 09+650NW, the chargeability high is located at surface at 10+1000NE and dips to the northeast at approximately 45°.

In June 2011, MWH Geo-Surveys, Inc. completed ground gravity surveys over three grids along the Rau Ridge system. These grids were located over the Tiger Deposit, Condor, and Puma showings. At the Puma Showing, a linear, north trending gravity high is defined along the western edge of the grid, near the ridgeline. No significant anomalies were identified at the Tiger Deposit or Condor Showing.

In June 2019, Scott Geophysics Ltd., conducted ground based IP and magnetic surveys over a 12 km^2 area covering the Bobcat Showing and the Rackla Pluton. These surveys have outlined four potential target areas for skarn mineralization with magnetic and conductivity anomalies.

Figure 9.4 Modelled ZTEM Anomalies



10.0 DRILLING

10.1 DIAMOND DRILLING SUMMARY

The Mineral Resource discussed in this report was estimated using the diamond drilling data collected between 2008 and 2019, which was provided by ATAC. No diamond drilling was conducted at the Tiger Deposit between 2011 and 2014, or in 2018. Figure 10.1 illustrates the drillhole locations utilized for the Mineral Resource estimation.

Drilling at the Tiger Deposit has delineated cohesive oxide and sulphide zones from surface to 250 m depth. The main part of the mineralization is confined to a single horizon, known as the Discovery Horizon, which has been structurally displaced near Tiger Creek. The structural displacement also defines the boundary between oxide dominant and sulphide dominant portions of the mineralized system. The limits of the mineralization are not fully delineated along strike or down dip and there is evidence that suggests that there are potential mineralized horizons above and beneath the Discovery Horizon.

A total of 166 (28,610 m) exploration and definition drillholes were completed through 2019 and evaluated for use in the Tiger Deposit Mineral Resource estimation (Table 10.1). Down hole drill depths range from 5 to 593.45 m with an average depth of 172.4 m. This drilling was completed on a nominal 50 m spaced grid over the main area of interest with portions of the oxide mineralization being drilled at 25 to 30 m spacings. The drill sections are all oriented northeast-southwest.

Figure 10.1 Tiger Zone Drillhole Locations

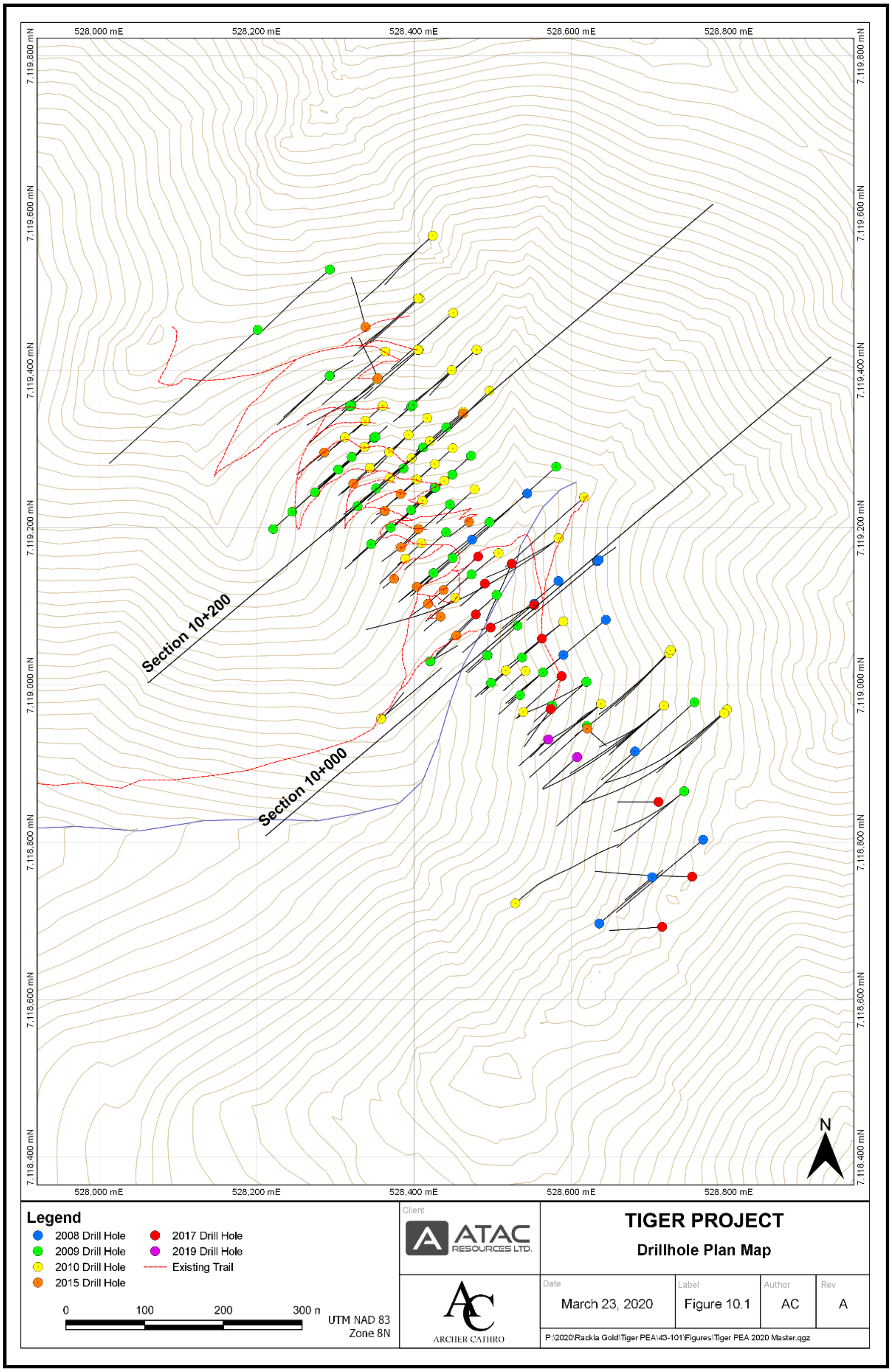


Table 10.1 Tiger Deposit Mineral Resource Database Summary

Year	Holes Drilled	Total Drilled (m)
2008	18	3,375.89
2009	53	8,651.26
2010	61	13,398.02
2015	18	1,418.60
2017	12	1,371.29
2019	4	395.02
Total	**166**	**28,610.08**

Some of the 2015 drilling was done primarily for geotechnical and environmental purposes, with all other drilling focused on exploration or resource definition. To monitor water levels, vibrating wireline piezometers (VWPs) were installed in two holes in 2015. Drilling in 2015 and 2019 used oriented core tooling to provide detailed orientation of structures and other features.

The Tiger Deposit is a thick northwesterly trending body of carbonate replacement style gold mineralization hosted by a moderately northeast dipping horizon. It is 700 m long, 100 to 200 m wide, and up to 96 m thick. Mineralization is developed within and adjacent to a regionally extensive corridor of highly strained rocks that are manifested as a 40 to 150 m wide zone of small scale folding and shearing. The geometry of the mineralized system is defined by a series of stacked and folded limestone horizons intercalated with locally extensive mafic flows and volcaniclastic units. Examples of this geometry are illustrated within sections in Figure 10.2 and Figure 10.3.

Gold is the primary commodity of interest, however elevated tungsten is also variably present in areas. Gold and tungsten occur in both sulphide and oxide facies mineralization. Sulphide mineralization is accompanied by, and developed within, limestone that is replaced by ferruginous dolomite and iron carbonate minerals. Sulphide species consist of disseminated to banded pyrite, with subordinate arsenopyrite and pyrrhotite and minor bismuthinite and sphalerite. Small amounts of disseminated scheelite are also present. The main sulphide minerals exhibit at least three stages of mineralization.

Figure 10.2 Tiger Section 10+000

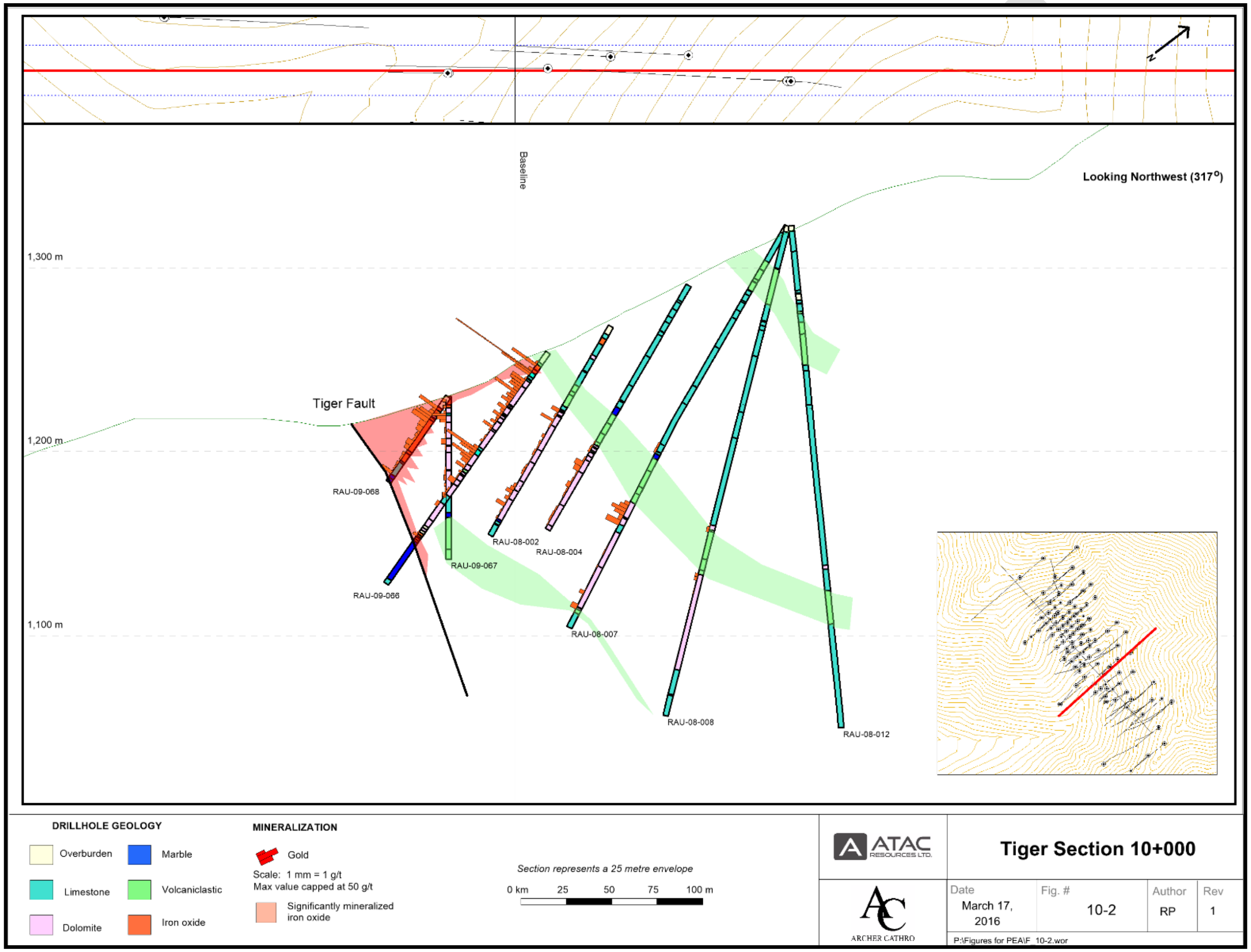
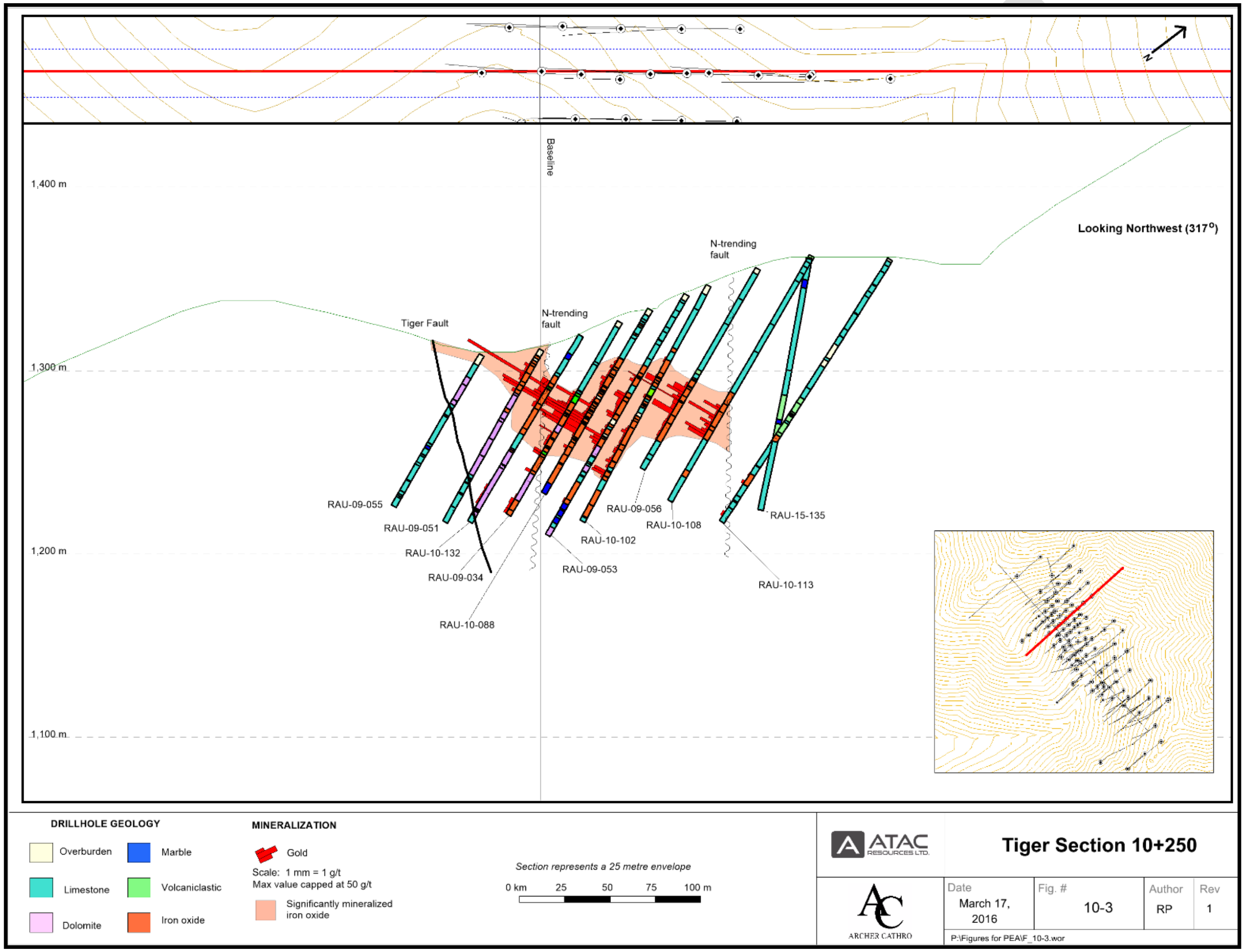


Figure 10.3 Tiger Section 10+250



Oxide mineralization is partially-to-completely devoid of sulphide minerals and ranges from very competent, weakly porous limonitic mud to rubbly porous limonitic grit. The oxide appears texturally amorphous within most intersections but occasionally exhibits residual color banding that may represent relict sulphide textures. Complete oxidation extends up to 150 m from surface. The best oxide gold grades and deepest oxidation occur where northerly trending extensional faults intersect the regional structure. The nature of the contacts between the oxide and sulphide facies is not well understood, nor is the gold distribution within the mineralized horizon.

10.2 Diamond Drilling Specifications

All drill campaigns were contracted to Superior Diamond Drilling of Kelowna, BC, which conducted work within the deposit on behalf of ATAC.

During 2008, diamond drilling at the Property was completed with a Mandrill 1200 diesel-powered drill using BTW equipment yielding core diameters of 4.17 cm. In 2009, drilling was performed with a Mandrill 1200 and two B-15 diesel-powered drills using BTW (4.17 cm core diameter), NQ2 (5.06 cm core diameter) and HQ3 (6.11 cm core diameter) equipment. The same equipment was used among the three drills in 2010 with the addition of a track mounted mobilization system on one of the B-15 drills and the ability to utilize PQ (8.50 cm core diameter) tooling. Only one B-15 drill using HQ3 equipment was used in 2015, 2017, and 2019.

Different diameter tooling was used in certain parts of the mineralizing system as the effectiveness of certain diameters was determined throughout the evolution of the exploration campaigns. BTW tooling was considered efficient from a recovery perspective within the sulphide portion of the deposit while NQ2 and HQ3 were the best means of properly testing the oxide parts of the mineralizing system to maximize recovery. PQ size holes were drilled as infill and twinned holes from earlier campaigns to evaluate the effects of larger diameter core diameter with respect to recovery and continuity of gold grade.

10.3 Core Logging Procedures

Core logging in 2008 was conducted using a generic logging form that was filled out in hardcopy form during the day and entered into Microsoft® Excel spreadsheets during the evening. As the project evolved a site-specific core assessment manual with a project-specific drill log was created for the Property and included fields for rock type and modifiers for lithology, minerals, alteration, textures, structure, hardness, weathering and concentrations. Where oxide was logged, a Munsel color chart was utilized to characterize colour for maximum continuity.

In 2015 and 2017, logging was recorded as hardcopy and then entered into a Microsoft-SQL Server® database (the Database). All of the pre-2015 data was transferred to the Database. In 2019, logging was done directly into the Database.

Drill core samples were collected using the following procedures:

1. Core was reassembled, lightly washed, and measured.
2. Core was wet photographed.
3. Core was geotechnically logged.
4. Magnetic susceptibility measurements were taken at 1 m intervals along the core.
5. Core was geologically logged and sample intervals were designated. Sample intervals were set at geological boundaries, drill blocks or sharp changes in sulphide/oxide content.
6. If oriented core was used, the orientation line was drawn on core between marks, and the orientation of fractures, structures and other key features was recorded.
7. Core recovery was calculated for each sample interval.
8. From 2015 to 2019, samples of competent rock types were selected for point load testing.
9. Core was sawn or split in half depending on the type of mineralization encountered. Oxide was generally split with the impact splitter or putty knife to avoid washing away potential gold bearing material. One-half was sent for analyses and one-half returned to the core box.
10. Samples were double bagged in 6 mm plastic bags, a sample tag was placed in each sample bag, then two or three samples were placed in a fiber glass bag sealed with a metal clasp and sample numbers were written on the outside of that bag with permanent felt pen.
11. QA/QC samples including standards, blanks and duplicates were included in every batch of core samples. See Section 12 for details on QA/QC procedures.

A geotechnical log was filled out prior to geological logging of drill core and included the conversion of drill marker blocks from imperial to metric plus determinations of recovery, rock quality designations (RQD), hardness and weathering. Starting in 2015, fracture frequency, joint sets, and joint set roughness, shape and infill were also recorded.

Within oriented intervals, alpha and beta angles were recorded for each joint along with the roughness, shape and infill material and thickness. In 2015 measurements were taken using a goniometer. In 2019 a REFLEX IQ-LOGGER™ was used to take orientation measurements.

A total of 335 point load measurements were taken on core in 2015, 2017 and 2019 using an ELE International digital point load test apparatus (Model 77-0115). Both axial and diametral measurements were taken intermittently on all competent rock types.

10.4 DENSITY MEASUREMENTS

Density measurements were systematically taken on core, throughout the drill 2008 thru 2010 drill programs. Measurements were taken over the course of the drill programs from a variety of holes and lithologies. Three methods were used to calculate densities from drill core, while a fourth was used to determine density in situ. The first three methods used the following formula:

$$\textit{Density} = \textit{weight in air} / [\textit{Pi} * (\textit{diameter of core} / 2)^2 * \textit{length of core}]$$

The first method determined sample densities by cutting a 10 cm long section of core and then determining its weight dry and its weight immersed in water. These measurements were used in the formula above to calculate the density of each interval. Specific gravity was also calculated for these intervals as a quality control check.

The second method was used exclusively for oxidized intervals that were too fragile to weigh using other procedures. Densities were determined by using a bathroom scale to weigh full core boxes containing homogenous intervals of oxide material. The average weight of an empty core box was determined at the start of the program and subtracted from the measurement. The volume was then calculated using the above formula. To do this calculation, the length of core within each box was measured. Due to the many variables and assumptions inherent in this method, these calculations are only considered an approximation.

The third method was used to calculate density of oxide material in the later programs, when drill crews were consistently achieving high recoveries. Oxide intervals with 100% recovery were selected and a 10 cm length extracted and placed into a metal pan. The interval was dried in a small oven to remove any excess water before being weighed. The weight of the pan was subtracted from this measurement. Density was then calculated using the formula mentioned above.

A fourth method, using the Scintrex Gravilog Borehole Gravity system was used to determine bulk density of formations intersected by the drill hole. The Gravilog sensor is based upon the fused quartz technology and has been miniaturized to fit into a narrow-diameter borehole. It measures the change in gravity between two locations within a borehole, which is directly proportional to the density of the formation between the measurement points. In 2010, bulk densities of oxide zones within the Tiger Deposit were determined using measurements collected in eight drillholes.

10.5 DRILL COLLAR AND DOWN HOLE SURVEYS

All drillhole collars were surveyed by Archer Cathro employees using a Real Time Kinematic (RTK) GPS system. The collars are marked by individual lengths of drill rod that are securely cemented into holes. A metal tag identifying the drillhole number is affixed to each drillhole marker.

Between 2008 and 2010, drill collars were aligned at surface using a Brunton compass. In 2015, 2017, and 2019, a Reflex North Finder azimuth pointing system (APS), a GPS based compass, was used to align the drillholes.

To determine the deflection of each drillhole, the orientation was measured at various intervals down the hole using a magnetic multi-shot survey tool. Survey tools were provided by Icefield Tools (2008), Ranger Survey Systems (2009 and 2010), and Reflex (2015, 2017, and 2019).

Shots were every 50 feet (15 m). Measurements taken and recorded were azimuth, inclination, temperature, roll angle (gravity and magnetic) plus magnetic intensity, magnetic dip and gravity intensity (for quality assurance). All readings were reviewed and erroneous data were not used when plotting the final drillhole traces.

10.6 Oriented Core Surveys

A Reflex ACT II downhole digital core orientation system was used in 2015 to orient the core in a total of five holes.

All of the oriented drill holes were drilled using split tubes. The use of split tubes allowed orientation measurements to be collected across incompetent intervals or intervals with poor recovery.

Split tube intervals were oriented by Archer Cathro employees at the drill site. The core tube was first aligned by the driller's helper using the ACT II tool before the split tube was extracted from the core tube. Care was taken to not shift the core during this process. A line representing the bottom of the hole was marked down the length of the core by the Archer Cathro employees. Structural orientation measurements within the interval were taken prior to the core being transferred to core boxes.

A Reflex ACT III downhole digital core orientations system was used in 2019 to orient the core in all four drill holes. Split tubes were not used in 2019. Core was otherwise treated similarly to the procedure for 2015 discussed above. In 2015, a goniometer was used for angle measurements, and in 2019, a Reflex IQ-LOGGER was used to measure feature orientations.

11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

This section describes the sample procedures followed during the diamond drilling exploration programs supervised by Archer Cathro for ATAC. Also described are sample handling and analysis procedures followed during the exploration programs. A project-specific sample handling manual was designed in conjunction with the field operations manual specific to core processing.

11.1 SAMPLE SHIPMENT AND SECURITY

All drill core was flown by helicopter to a processing facility on the Property where the core was logged and sawn or split. Between 2008 and 2010, surface rock and core samples were flown by helicopter from the Property to a staging area at McQuesten Lake, then transported to Whitehorse by truck. During programs conducted between 2015 and 2019, samples were flown by helicopter to the Rau Airstrip and transferred to a fixed wing aircraft and flown to Mayo. From here there were loaded onto a truck and transported to ALS Minerals' Whitehorse preparation facility.

ALS Minerals was responsible for shipping the prepared sample splits from Whitehorse to its North Vancouver laboratory, where they were analyzed. All samples were controlled by employees of Archer Cathro until they were delivered to a commercial courier or directly to ALS Minerals in Whitehorse.

For all programs after and including 2010, Archer Cathro ensured that a chain of custody form accompanied all batches of drill core during transportation from the Property to the preparation facility. A unique security tag was attached to each individual fiberglass bag when the bag was sealed. The bags and security tags had to be intact in order to be delivered to ALS Minerals. If a security tag or bag arrived at the laboratory damaged, an investigation into the transportation and handling of that sample bag was undertaken by ALS Minerals and Archer Cathro and affected samples were not processed until a resolution was reached regarding the security of the samples.

Starting in 2010, and continuing through all subsequent programs, individual samples were weighed prior to shipping. These weights were compared to weights recorded by ALS Minerals upon receiving the samples. Any discrepancies between the two weights were investigated.

11.2 Sample Preparation and Analysis

All samples were sent to ALS Minerals for preparation and analysis. ALS Minerals, a wholly owned subsidiary of ALS Limited, is an independent commercial laboratory specializing in analytical geochemistry services. Between 2008 and 2010, samples were sent to ALS Minerals' laboratory in North Vancouver for preparation and analysis. Samples collected between 2015 and 2019 were prepared at ALS Minerals' laboratory in Whitehorse before beings sent to North Vancouver for analysis. Both ALS Minerals' Whitehorse and North Vancouver laboratories are individually certified to standards within International Organization for Standardization (ISO) 9001:2008.

Soil samples were dried and screened to -35 mesh to produce a fine fraction, which was then pulverized to 85% passing 75 µm. Splits of the pulverized fraction were routinely dissolved in aqua regia and analyzed for 35 elements using the inductively coupled plasma (ICP)-atomic emission spectroscopy (AES) technique (ME-ICP41). All samples were also analyzed for gold using fire assay and ICP-AES (Au-ICP21).

Core and rock samples were dried and crushed to 70% -2 mm, before a 250 g split was taken and pulverized to better than 85% -75 µm. To reduce cross contamination between core samples during preparation, the equipment was washed twice with quartz silica sand. Splits of the pulverized fraction were dissolved in aqua regia and analyzed for 48 elements using technique ME-MS61, which combined ICP with mass spectroscopy (MS) and AES. Samples were analyzed for gold by fire assay finished with atomic absorption spectroscopy (Au-AA26).

All mineralized drill core was split/sawn for assay. The mineralization is readily recognizable by sulphide /oxide content in the core.

11.3 Quality Assurance and Quality Control

ATAC has implemented a QA/QC program to monitor the quality of assays in every drilling campaign at the Tiger Project. In all campaigns standards and blanks were used. Starting in 2009 and in every year since, field duplicates were also collected. In 2015, 2017 and 2019, preparation duplicates were prepared and analyzed at the lab. The numbers and types of QA/QC samples collected each year are summarized in Table 11.1.

Oxide standards used in the 2009, 2010, and 2015 programs were purchased from Geostats Pty Ltd. (Geostats). Sulphide standard samples used in the 2009 and later drill programs were prepared from coarse reject material from 2008 and 2009 core samples. In 2008, standards were purchased from CDN Resource Laboratories Ltd. (CDN Resource) of Delta, British Columbia. These assay standards were prepared, homogenized and packaged by CDN Resource. Geostats certifies its own standards. The CDN Resource and project-specific standards were certified by Smee & Associates Consulting Ltd. of North Vancouver, British Columbia.

For 2008 all the samples are recorded in the database using a single "batch" identifier, but they were in fact shipped to the lab in multiple batches and analyzed by the lab in 32 "jobs". Approximately one standard and one blank were submitted with each job. In subsequent years samples were submitted to the lab in batches each of which contained 30 to 32 core samples, with each batch corresponding to one laboratory job. In those years most batches contained two standards, two blanks, one field duplicate and starting in 2015, one preparation duplicate.

Table 11.1 Numbers and Types of QA/QC Samples Collected Each Year

Year	Batches	All Samples	Core Samples	Standards	Blanks	Quarter-core Duplicates	Coarse Reject Duplicates
2008	1	783	706	34	43	--	--
2009	77	2,620	2,258	147	144	71	--
2010	104	3,383	2,917	186	185	95	--
2015	12	407	340	22	23	11	11
2017	16	555	462	31	31	15	16
2019	6	207	177	12	6	6	6
Total	**216**	**7,955**	**6860**	**432**	**432**	**198**	**33**

MDA has reviewed the results obtained for ATAC's Tiger QA/QC samples. MDA's review and opinion of the QA/QC results are described in Section 12.5.

11.4 SUMMARY STATEMENT

In the opinions of Mr. Ronning and Mr. Ristorcelli the sample preparation, security and analytical procedures employed by ATAC on the Tiger Project are adequate and the project's data are acceptable for use in the Resource Estimate described in this report.

12.0 DATA VERIFICATION

Data verification, as defined in NI 43-101, is the process of confirming that data has been generated with proper procedures, has been accurately transcribed from the original source and is suitable to be used. There were no limitations on, or failure to conduct, the data verification for this report. Additional confirmation on the drill data's suitability for use are the analyses of the Tiger project QA/QC procedures and results as described in Section 12.5.

12.1 SITE VISIT

Please refer to Section 2.1 of this PEA report.

12.2 DATABASE AUDIT

Mr. Ronning, on behalf of MDA, did the audit of the drill database described in this section.

12.2.1 DRILL-COLLAR LOCATION AUDIT

Since the inception of drilling at Tiger in 2008 ATAC has used several methods to establish drill-hole collar locations. In 2008 and 2009 these methods included hand-held GPS and Total Station surveys. Between 2010 thru 2019 ATAC used an RTK[1] system.

During the 2017 field season ATAC realized that many drill-hole locations in the database obtained prior to 2015 were offset relative to positions obtained using more recent surveys or to positions evident on an orthophoto and/or LIDAR image. In 2017 many collars from earlier years were re-surveyed using an RTK system and/or their locations were determined from an orthophoto or a LIDAR surface.

To verify that collar locations in the database accurately reflect the true field locations of drill hole collars MDA used copies of field records in the form of digital data files, and in cases where corresponding field records are not available, the LIDAR surface. The results are listed in Table 12.1. Explanations of the results in the table follow in the "Notes".

[1] Real-time kinematic (RTK) positioning is a satellite navigation technique used to enhance the precision of position data derived from satellite-based positioning systems (global navigation satellite systems, GNSS) such as GPS, GLONASS, Galileo, and BeiDou. (from Wikipedia, "https://en.wikipedia.org/wiki/Real-time_kinematic", Nov. 7, 2019.)

Table 12.1 Summary of Drill Hole Location Verification Status

A Year	B Holes Drilled	C Survey Exact Match	D Survey Within 3 Meters	E Identifiable Drill Pad	F No Location Verification
2008	18	3(L)	nil	15	nil
2009	53	25(L)	5(L)	23	nil
2010	61	14(L)	2(L)	45	nil
2015	18	5	7	n/a	6
2017	12	12	n/a	n/a	nil
2019	4	4	n/a	n/a	nil
Total	**166**	**63**	**14**	**83**	**6**
Percent	**100**	**38**	**8**	**50**	**4**

Notes:

Column C: A survey location is deemed to be an exact match if it is within 1 centimeter of the database collar location.

Column D: A survey location within 3 meters of the database collar location is deemed to confirm the database location.

Column E: Where a collar location cannot be confirmed by a survey location, it is deemed to be correct if it is on or adjacent to a drill pad identifiable on the LIDAR surface. This criterion cannot be applied to holes from 2015, 2017 and 2019 as these post-date the LIDAR survey.

Columns D,E: (L) in these columns indicates that the locations are also confirmed by visible drill pads on the LIDAR surface.

Column F: MDA was unable to confirm the locations of 6 drill holes from the 2015 campaign.

MDA was able to check the locations of 77 drill holes against field survey records. In Table 12.1 this number is obtained by adding the totals of Column C and Column D. While MDA deemed RTK survey locations within 3 meters of the database locations to be adequate confirmation, the large majority, 63 of 77 or 82%, of the collar locations confirmed from survey locations are exact matches.

Field Checks of Collar Locations

In early October of 2019 Mr. Suraj Priyadarshi of Tetra Tech was on the Tiger site, and at MDA's request checked the locations of four drill-hole collars. The checks were done using a consumer-level hand-held GPS without any differential corrections. The results are set out in Table 12.2.

Table 12.2 Field Checks of Drill Hole Collar Locations

Hole ID	db East	check East	East Diff	db North	check North	North Diff	db Elev	check Elev	Elev Diff
RAU-15-136	528620.06	528622	-1.94	7118944.56	7118945	-0.44	1295.71	1321	-25.29
RAU-15-146	528403.47	528402	1.47	7119124.99	7119124	0.99	1258.78	1269	-10.22
RAU-17-151	528481.45	528483	-1.55	7119163.472	7119164	-0.53	1265.36	1267	-1.64
RAU-17-158	528587.62	528591	-3.38	7119011.431	7119014	-2.57	1275.84	1277	-1.16

Notes:
1. "db" in column headers indicates data from ATAC's database.
2. "check" in column headers indicates data from the field checks.
3. "Diff" in column headers indicates the difference between the database data and field checks, calculated as *database - check*.

In MDA's opinion, within the limits of the accuracy of the instrument used, the field checks confirm the east and north coordinates of the collars. The elevation differences for RAU-15-136 and RAU-15-146 are significant, but the accuracies of uncorrected elevations measured with a consumer-level GPS are often poor. The collar elevations used in ATAC's database have been checked against model surfaces generated photogrammetrically and using LIDAR data and in MDA's opinion are more reliable than the uncorrected GPS elevations.

12.2.2 Downhole Survey Audit

In this report "downhole survey" refers to surveys done for the purpose of measuring the orientation (direction) of drill holes at several points down the hole.

ATAC began doing downhole surveys in 2008. The first hole to be surveyed was RAU-08-010. Downhole surveys were not done in earlier drill holes, although acid tests were done at the ends of each of the first nine holes. Different instruments were used in different years, as listed below in Table 12.3. All of them use a magnetic compass to determine azimuths.

Table 12.3 Instruments Used for Downhole Surveys

2008:	Icefield instrument, capable of doing magnetic declination corrections if an appropriate correction is input by the operator.
2009 & 2010:	Ranger instrument, not capable of declination corrections.
2015, 2017 & 2019:	Reflex instrument, not capable of declination corrections.

Magnetic declination has been changing rapidly in recent years. MDA used an online resource at "http://www.geomag.nrcan.gc.ca/calc/mdcal-en.php" (the "online declination calculator") to calculate the declination for a date during the field season of each drill campaign. These are compared to the declinations used in the Tiger Project database in Table 12.4.

Table 12.4 Magnetic Declinations

Latitude N	Longitude W	Date	Magnetic Declination	Grid Declination	Convergence Angle	Declination Used in Database
decimal degrees		yyyy mm dd	decimal degrees east			
64.2 N	134.41 W	2008 09 01	24.83	24.30	0.53	24.73
64.2 N	134.41 W	2009 07 01	24.45	23.92	0.53	24.00
64.2 N	134.41 W	2010 07 01	23.99	23.46	0.53	24.00
64.2 N	134.41 W	2015 09 01	21.59	21.06	0.53	21.8
64.2 N	134.41 W	2017 07 01	20.81	20.28	0.53	20.77
64.2 N	134.41 W	2019 09 20	19.87	19.34	0.53	19.90

Note: Data in shaded columns were obtained from "http://www.geomag.nrcan.gc.ca/calc/mdcal-en.php" in early October of 2019.

The declinations used to correct the azimuths in the database are acceptably accurate.

Downhole surveys were not done in every drill hole, and not all measurements obtained from downhole surveys were used. Most of the downhole data that was not used consists of azimuths that ATAC's personnel deemed to be erratic readings. This is a common problem when magnetic instruments are used to determine downhole azimuths in lithologic packages containing some components with magnetic minerals. Erratic readings can also occur if the tool is too close to drill rods when the measurement is taken.

MDA obtained original field records from ATAC, for the majority of downhole surveys, and used those to check the drill-hole orientations as recorded in the database. MDA has field records for 69% of the downhole survey measurements in the database, representing approximately half of the drill holes and drilled meters in the database.

The field records consist of human-readable digital data processed by software provided by the manufacturers of the instruments. MDA does not have this software and relied on ATAC to obtain the human-readable data. Table 12.5 and Table 12.6 summarize the checks.

Table 12.5 Downhole Survey Record Checks

	Count	Percent of Total	Comment
Records in Database[1]	1,375	100	
Dips Verified[2]	947	69	No dip measurements were rejected
Azimuths Verified[2]	640	47	307 azimuth measurements were deemed by ATAC to be unreliable

Notes: 1. Excluding records at zero depth, where orientations were measured using means other than the downhole instruments.

2"verified" means that MDA has confirmed that the measurement is derived from a downhole-survey instrument and has been accurately entered in the database.

Table 12.6 Summary of Drill Holes with Downhole Measurements

	Count	Percent of Total	Meters	Percent of Total m
Drill Holes in Database	166	100%	28,610	100%
Holes with Downhole Dip Measurements	80	48%	15,014	52%
Holes with Downhole Azimuth Measurements	58	35%	11,057	39%

The downhole survey data in the database accurately reflect the available downhole survey data, omitting data that ATAC rejected on grounds of suspect quality. MDA concurs with ATAC's decisions as to which data are usable. The absence of usable downhole survey data for approximately half the drill holes was a factor in decisions about resource classification, but it is mitigated by the fact that most of the drill holes are shallow.

12.2.3 Assay Database Audit

The assay table that MDA used for the resource estimate was given to MDA by ATAC on 30 October, 2019. It contains 6,860 gold assays and 6,810 tungsten assays.

MDA checked 6,809 gold assays against digital data files obtained directly from ALS Canada Ltd. The remaining 51 gold assays were checked against a table from SGS Minerals Services, 2012. No errors were identified.

MDA checked all of the 6,810 tungsten assays against digital data files obtained directly from ALS Canada Ltd. No errors were identified.

12.3 Discussion of Geological Data

MDA did not do a formal audit of the geological data, such as lithology, alteration, minerals present and structure. However, MDA's modeling procedures involve frequent reference to core photos and logged geologic data, while working on sections with the information in the drill database. This results in extensive cross-checking of information in the database against the photos and the field data. MDA found that the geological information in the database is a reasonable reflection of the field data.

12.4 Review of Density and Specific Gravity Data

During its several drill campaigns, ATAC obtained density data for the rock within the Tiger Deposit using four different methods, as described in the following sections 12.4.1 through 12.4.4. For the densities used in the resource estimate, MDA relied on the density measurements using the method described in section 12.4.1, which are reasonably corroborated by the method described in section 12.4.4.

12.4.1 ROUTINE MEASUREMENTS IN 2008 - 2010

During the 2008, 2009 and 2010 field seasons, ATAC measured density and specific gravity using selected intervals of core. Calculations were done as follows:

Equation 12-1 $$Density = \frac{(weight\ in\ air)}{(volume\ of\ specimen)}$$

and

Equation 12-2 $$Specific\ Gravity = \frac{(weight\ in\ air)}{(weight\ in\ air) - (weight\ in\ water)}$$

The volume in Equation 12-1 was obtained using the measured length of the specimen and the nominal diameter of the core based on the size of the drill bit. ATAC also measured a number of actual core diameters in the field to check against the nominal core diameters. The average measured diameters in the field were slightly less than the nominal diameters, but MDA and ATAC elected to use the nominal core diameters. Lengths of specimens ranged from 7.1 cm to 20.0 cm, with an average of 14.3 cm and a median value of 14.8 cm.

Except as described in section 12.4.3 no formal drying tests were done on the specimens used for density and specific gravity measurements, so the in-situ moisture content of the specimens is unknown. During the immersion tests, the specimens were not sealed to prevent absorption of water.

The Tiger database contains 825 density measurements and 785 specific gravity measurements. The difference exists because the immersion method for measuring specific gravity relies on having competent specimens that can be submerged in water without disintegrating. This precludes measurements of highly oxidized and other very friable rocks.

In order to estimate the density of the different types of material included in the resource estimate, MDA elected to use the density measurements (Equation 12-1) so as to have as many measurements as are available for the oxidized material. These measurements were merged into sample intervals; for cases in which more than one density measurement falls within a sample interval, the density measurements were averaged. There are 461 assay intervals with corresponding density values.

12.4.2 DENSITY ESTIMATES USING FULL CORE BOXES

This was used exclusively for oxidized intervals that were too fragile to weigh using other procedures. Densities were determined by using a bathroom scale to weigh full core boxes containing homogenous intervals of oxide material. The average weight of an empty core box was determined at the start of the program and subtracted from the measurement. The volume was then calculated using the core diameter and the length of core within each box. Due to the many variables and assumptions inherent in this method, these calculations are only considered an approximation. (*description adapted from Ghaffari et al., 2016*)

MDA has not made use of the density measurements obtained using this method.

12.4.3 Density Measurements Using Oven-Dried Core

This method was used to calculate density of oxide material in the later programs, when drill crews were consistently achieving high recoveries. Oxide intervals with 100% core recovery were selected and a 10 cm length extracted and placed into a metal pan. The interval was dried in a small oven to remove any excess water before being weighed. The weight of the pan was subtracted from this measurement. Density was then calculated using Equation 12-1. (*description adapted from Ghaffari et al., 2016*)

There is concern that the drying temperatures used were too high and may have resulted in the breakdown of some hydroxyl-bearing minerals in the oxide material, so MDA has not made use of the data obtained using this method.

12.4.4 Densities Calculated from Scintrex Gravilog Borehole Gravity System

In eight drill holes in 2010 the Scintrex Gravilog Borehole Gravity system was used to determine bulk density of formations intersected by the drill hole. The Gravilog sensor is based upon the fused quartz technology and has been miniaturized to fit into a narrow-diameter borehole. It measures the change in gravity between two locations within a borehole, which is directly proportional to the density of the formation between the measurement points. (*description adapted from Ghaffari et al., 2016*)

ATAC and MDA have reviewed the densities calculated using the Scintrex system and found that they provide a reasonable corroboration of the direct measurements described in section 12.4.1.

12.5 Quality Control and Quality Assurance

Mr. Ronning, on behalf of MDA, evaluated the Quality Control and Quality Assurance ("QA/QC") data provided by ATAC.

ATAC implemented a QA/QC program to monitor the quality of assays in every drilling campaign at the Tiger Project. The QA/QC program is described in Section 11.3. The present section describes MDA's evaluation of the results. Table 12.7 sets out the numbers and types of QA/QC samples collected each year. It is the same as Table 11.1, repeated here for the convenience of the reader.

Table 12.7 Numbers and Types of QA/QC Samples Collected Each Year

Year	Counts						
	Batches	All Samples	Core Samples	Standards	Blanks	Quarter-Core Duplicates	Coarse Reject Duplicates
2008	1	783	706	34	43	--	--
2009	77	2,620	2,258	147	144	71	--
2010	104	3,383	2,917	186	185	95	--
2015	12	407	340	22	23	11	11
2017	16	555	462	31	31	15	16
2019	6	207	177	12	6	6	6
Totals	**216**	**7,955**	**6,860**	**432**	**432**	**198**	**33**

The Tiger database contains no data for any QA/QC samples from two batches, numbers 10-062 and 10-071. Although these batches originally contained 35 and 36 samples respectively, only 6 samples from each batch are used in the database. All six samples from batch 10-071 were reported to contain no detectable gold and the six samples from 10-062 contained low gold grades ranging from 0.02 to 1.95 ppm Au.

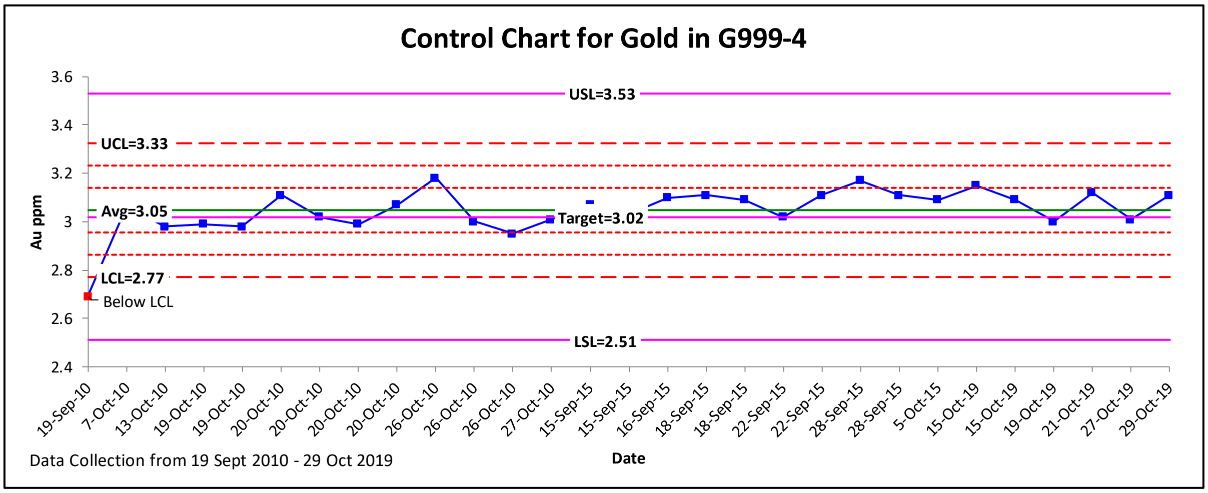
ATAC monitors the results of QA/QC analyses for each batch as they are received, and when failures are deemed to have occurred in analytical batches that are material to the project, ATAC instructs the lab to re-analyze the affected samples. In cases for which re-analyses were requested, the assay table that MDA has worked with contains only the final accepted assays for core and QA/QC samples, not the earlier rejected ones.

12.5.1 STANDARDS

Since the beginning of drilling at Tiger in 2008, ATAC has made use of standards to monitor the quality of assays. Standards are “Certified Reference Materials” consisting of powder containing known “certified” grades of the element of interest. Fourteen different standards have been used during drill programs at Tiger. Seven are commercial standards from reputable suppliers and seven are project-specific standards prepared by CDN Labs of Langley B.C. using material from Tiger supplied by ATAC. All of the standards are certified for gold, and one of the project-specific standards is also certified for tungsten.

ATAC provided MDA with records of the results for standards in the form of a digital assay table. MDA evaluated the results of the analyses of standards using a variant of the conventional Shewhart chart. Figure 12.1 is an example, presented to illustrate the method.

Figure 12.1 Control Chart for Gold in Standard G999-4



Notes: 1. "Target" is the expected or certified value for gold in the standard.

2. "LSL" and "USL" are the lower and upper specification limits, respectively. They are the Target value ± 3 standard deviations based on the statistics provided by the supplier of the standard.

3. "Avg" is the average obtained in the analyses of the standard done for ATAC.

4. "LCL" and "UCL" are the lower and upper control limits, respectively. They are the average value ± 3 standard deviations based on statistics derived from the analyses done for ATAC.

MDA considers analyses of the standards to be acceptable if they fall within the specification limits.

GOLD IN THE TIGER STANDARDS

Table 12.8 summarizes the results obtained for gold in the Tiger standards.

Table 12.8 Summary of Results for Gold in the Tiger Standards

Standard ID	Gold Grades					Count	Dates		Fail Counts		Bias pct
	Target	Avg.	Max.	Min.	Units		Start	End	High	Low	
CDN-GS-15A	14.83	14.49	15.45	13.25	ppm	32	26-Jul-08	23-Oct-08	0	6	-2.29
G306-3	8.66	8.62	9.07	7.15	ppm	34	08-Jul-10	15-Oct-19	0	1	-0.46
G912-7	0.42	0.41	0.43	0.37	ppm	10	16-Sep-15	29-Oct-19	0	0	-2.38
G999-4	3.02	3.05	3.18	2.69	ppm	29	19-Sep-10	29-Oct-19	0	0	0.99
G306-1	0.41	0.39	0.42	0.34	ppm	35	30-Jun-10	28-Oct-10	0	0	-4.88
G399-2	1.46	1.4	1.49	1.27	ppm	30	01-Jul-10	26-Oct-10	0	0	-4.11
2010-A	0.437	0.44	0.51	0.36	ppm	31	30-Jun-10	20-Sep-17	1	1	0.69
2010-B	1.83	1.83	2.06	1.57	ppm	27	03-Jul-10	20-Sep-17	0	0	0
2010-C	3.83	3.87	4.27	3.34	ppm	23	05-Jul-10	20-Sep-17	0	0	1.04
Rau Standard 1	0.514	0.5	0.61	0.41	ppm	49	03-Jul-09	23-Oct-09	0	0	-2.72

table continues...

Rau Standard 2	1.527	1.52	1.72	1.3	ppm	52	03-Jul-09	29-Oct-09	0	0	-0.46
Rau Standard 3	5.705	5.51	6.02	4.87	ppm	49	06-Jul-09	27-Oct-09	0	0	-3.42
CDN-GS-3Q	3.3	3.36	3.48	3.08	ppm	8	24-Aug-17	20-Sep-17	0	0	1.82
2010-D	2.72	2.7	3.2	2.26	ppm	25	24-Jun-10	19-Aug-10	0	0	-0.74

Notes: 1. The two failures listed for standard 2010-A fall almost exactly on the specification limits.

2. Standards whose IDs appear in **bold** text are project-specific standards. The others are commercial standards.

3. The "Bias pct" listed in the last column of Table 12.8 is calculated as:

$$100\ x\ \frac{Avg - Target}{Target}$$

There are few failures in the gold analyses and all but one fall on the low side. The single high-side failure in standard 2010-A is almost exactly on the upper specification limit and is not of concern.

Any set of analyses of a standard obtained from a single lab is expected to exhibit some bias relative to the target value. The biases listed in the last column of Table 12.8 are within the range of typical biases that MDA has seen in many such data sets and are considered acceptable.

In MDA's opinion the results obtained for gold in the Tiger standards are acceptable.

Tungsten in the Tiger Standards

Only one of the Tiger standards, 2010-D, has a certified value for tungsten. It is nevertheless useful to consider the results obtained for tungsten from all of the standards, as it gives an indication of the ability of the analytical method to produce precise tungsten analyses. For the standards for which tungsten specifications are not available, the upper and lower control limits are substituted for the specification limits in order to identify "failures". The results obtained for tungsten are summarized in Table 12.9.

Table 12.9 Summary of Results for Tungsten in the Tiger Standards

Standard ID	Tungsten Grades					Count	Dates		Fail Counts		Bias pct
	Target	Avg.	Max.	Min.	Units		Start	End	High	Low	
2010-D	0.214	0.21	0.226	0.195	pct	25	24-Jun-10	19-Aug-10	0	0	-1.87
2010-A	n/a	132.55	149	117.5	ppm	31	30-Jun-10	20-Sep-17	0	0	n/a
2010-B	n/a	59.89	65.1	53.2	ppm	27	03-Jul-10	20-Sep-17	0	0	n/a
2010-C	n/a	276.65	291	258	ppm	23	05-Jul-10	20-Sep-17	0	0	n/a
CDN-GS-3Q	n/a	2.08	2.2	1.9	ppm	8	24-Aug-17	20-Sep-17	0	0	n/a
CDN-GS-15A	n/a	9.54	24.2	8.7	ppm	33	26-Jul-08	23-Oct-08	3	0	n/a
G306-1	n/a	0.43	1	0.3	ppm	35	30-Jun-10	28-Oct-10	3	0	n/a

table continues…

G306-3	n/a	0.47	1.8	0.3	ppm	34	08-Jul-10	15-Oct-19	3	0	n/a
G399-2	n/a	8.78	10.1	7.7	ppm	30	01-Jul-10	26-Oct-10	0	0	n/a
G912-7	n/a	0.83	0.9	0.8	ppm	10	16-Sep-15	29-Oct-19	0	0	n/a
G999-4	n/a	40.76	43.9	36.3	ppm	29	19-Sep-10	29-Oct-19	0	0	n/a
Rau Standard 1	n/a	62.64	69	53.9	ppm	49	03-Jul-09	23-Oct-09	0	0	n/a
Rau Standard 2	n/a	199.45	250	2	ppm	52	03-Jul-09	29-Oct-09	1	1	n/a
Rau Standard 3	n/a	36.39	40.9	31.1	ppm	50	06-Jul-09	27-Oct-09	0	0	n/a

Notes: 1. The two failures listed below may be due to mis-identification of the standard:
a) Two of the three high failures in CDN-GS-15A.
b) Low failure in Rau Standard 2.
2. Standards whose IDs appear in **bold** text are project-specific standards. The others are commercial standards.
3. The "Bias pct" listed in the last column of Table 12.9 is calculated as:

$$100\ x\ \frac{Avg - Target}{Target}$$

4. No bias can be calculated for standards that do not have a certified ("Target") value for tungsten.

No causes for concern are revealed in the analyses of the standards for tungsten.

12.5.2 Duplicates

Beginning in 2009 ATAC has collected quarter-core field duplicates, and beginning in 2015 has instructed the lab to take coarse crush duplicates, at a rate of one each per 30 core sample batch. The assay table of November 17, 2017 contains records for 198 field duplicates (labelled as "duplicate" in the database), and 33 coarse crush duplicates (labelled as "coarse duplicate").

It does not appear that ATAC obtains assays of pulp duplicates (sometimes called replicates), as part of the initial set of analyses in the primary lab. It would be useful to obtain the laboratory's in-house QA/QC data, which likely does contain analyses of replicates.

For each of the two sets of duplicates, MDA prepared three types of charts:

- A scatterplot, showing an RMA regression
- A quantile/quantile plot
- Several relative difference plots (see explanation, below).

MDA uses a relative difference expressed as a percentage for each duplicate pair calculated as follows:

Equation 12-3 $$100\ x\ \frac{(Duplicate - Original)}{Lesser\ of\ (Duplicate, Original)}$$

An alternative calculation, which MDA also uses, but whose results are not listed in Table 12.10, is:

Equation 12-4 $$100\ x\ \frac{(Duplicate - Original)}{Mean\ of\ (Duplicate, Original)}$$

Table 12.10 summarizes the results for the field duplicates and the coarse duplicates. The averages of the relative differences listed in the Table 12.10 are based on Equation 12-3 and are indications of the biases between the duplicates and the originals. The “Abs Rel Pct Diff” is the average of the absolute relative differences and gives an indication of the degree of variability between the duplicates and originals.

Table 12.10 Summary of Results for Duplicates

Type	Period	Corr. Coeff.*	Counts			RMA Regression	Averages as Percent	
			All	Used	Outliers	(y = dup, x = orig)	Rel Pct Diff	Abs Rel Pct Dif
Gold								
Field Dup	2009 - 19	0.10	198	146	8	y = 0.954x - 0.011	-3.5	39.0
Coarse Dup	2015 - 19	0.97	33	22	2	y = 1.004x + 0.023	10.1	14.3
Tungsten								
Field Dup	2009 - 19	0.69	198	179	17	y = 1.031x - 7.271	-9.3	33.6
Coarse Dup	2015 - 19	0.88	33	31	2	y = 1.030x - 0.637	-1.7	16.8

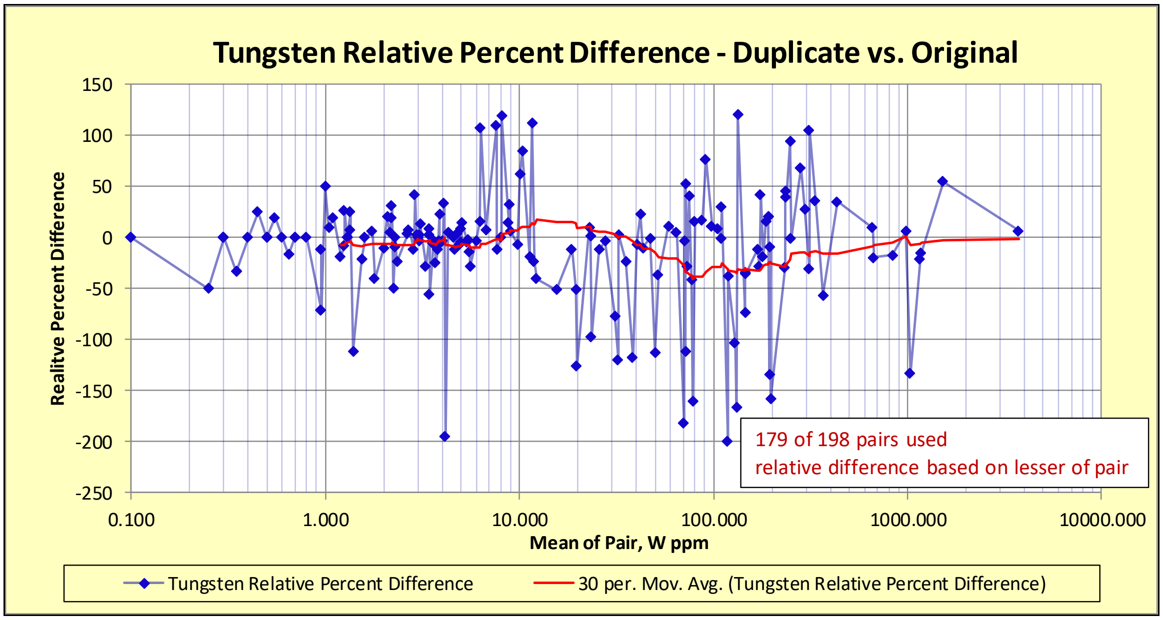
Note: *Correlation coefficients are calculated using all the data, not the subset in the “Used” column. Relative differences in this table are averages of those calculated using Equation 12-3.

The disparity in Table 12.10 between the total numbers of pairs (“All”) and the numbers of pairs used (“Used”) exists because MDA did not include in calculations those pairs in which one or both analyses fell below the analytical detection limit. In addition, “outlier” pairs were also excluded because their differences were so great as to skew the statistics of the data set.

The averages reported in the table are for all grades above the detection limit and excluding the outliers. Reporting single averages for each data set masks different responses in different grade ranges. See the chart in Figure 12.2 for an example of a more complete view of the field duplicate data for tungsten.

As expected, the absolute relative differences listed in Table 12.10 are significantly greater in the field duplicates than in the coarse duplicates. The absolute relative differences in the field duplicates reflect natural geological heterogeneity plus any field “sampling error” that may exist.

Figure 12.2 Tungsten Field Duplicates - Relative Percent difference



Notes: 1. Each point on the red moving average line represents the preceding 30 points, so features in the moving average line are offset to the right by 15 points from the central point of the data they represent.
2. The relative differences in Figure 12.2 are calculated using Equation 12-3.

In Figure 12.2, variability of the average relative difference between grade ranges is evident. For example, between about 6 and 12 ppm tungsten the duplicates tend to have higher grades than the originals. From about 12 ppm to 200 ppm tungsten the duplicates tend to have lower grades than the originals. It is this latter range that contributes most to the negative average relative difference of -9.3% for tungsten in field duplicates, reported in Table 12.10.

Tungsten was chosen as the example to use in Figure 12.2, even though gold is much more important, because the relative difference chart for gold does not show such marked trends in different grade ranges as does the one for tungsten.

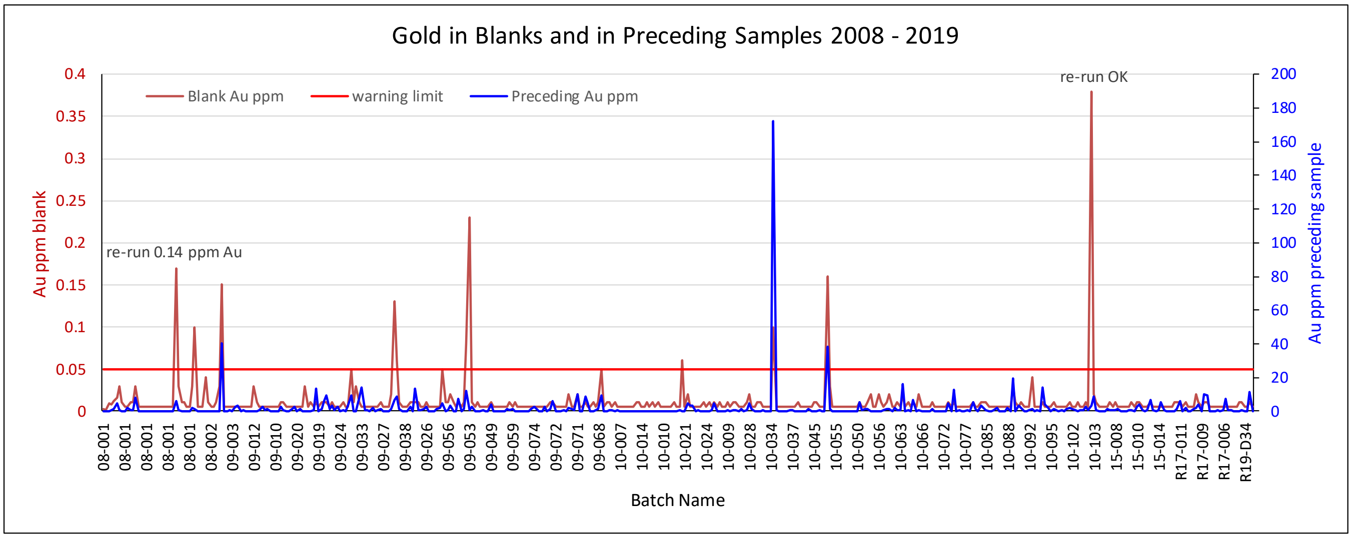
12.5.3 Blanks

The Tiger QA/QC assay table of Oct. 31, 2019 contains 432 analyses of barren landscaping marble, which ATAC uses as a coarse blank. Coarse blanks consist of rock known to be devoid of gold or other substances of interest, which are submitted to the lab among other samples and go through the entire crushing, pulverizing and analytical procedures as the real samples. Their purpose is to detect any contamination that might occur in the lab.

Gold in Blanks

MDA prepared a run chart for gold analyses of the blanks for all the drilling. On the same chart, MDA plotted the gold analyses of the samples immediately preceding each blank in the numerical sequence. The chart appears in Figure 12.3.

Figure 12.3 Gold in Blanks and in Preceding Samples 2008 - 2019



The "warning limit" in Figure 12.3 is set at five times the lower detection limit for gold. Practitioners commonly set the warning limits for analyses of blanks at somewhere between three and six times the lower detection limit. In the years 2008 through 2010, there were eleven instances of gold analyses in blanks exceeding 0.05 ppm Au. In the years 2015 through 2019 there were no such instances.

The purpose of plotting the gold analyses for the preceding samples on the run chart for gold in blanks is to gain a visual impression as to whether a blank that immediately follows a higher-grade sample through the sample preparation process tends to have a higher reported gold grade than blanks that follow low-grade samples. If there is such a tendency, it may imply that equipment in the lab is not adequately cleaned between samples[2]. This is a useful test only if the samples in a batch are processed in numerical sequence through the same crushing and grinding circuit. ATAC advised MDA that this is the case for most of their samples.

In Figure 12.3 there is a visual impression of a weak tendency for blanks to have slightly higher gold analyses if they follow a high-grade sample in numerical sequence. In three of the eleven cases in which gold in the blank exceeded the warning limit the blank numerically followed a sample containing high grade gold. A calculated correlation coefficient between gold in the blanks and in the preceding samples using all 432 pairs is 0.3 (a perfect correlation would have a coefficient of 1).

The frequency and magnitudes of the high gold analyses in blanks are not sufficient to be a material concern in the resource estimate.

[2] Early in the 2009 program, which focused on oxide gold there was concern and it was noted that there may potentially have been some carry over. For this reason, ATAC requested a second cleaning step with silica sand between all samples.

Tungsten in Blanks

MDA prepared a chart similar to Figure 12.3, but for tungsten. MDA found, however, that more than 30% of the tungsten analyses in the blanks exceed the warning level. MDA does not know the reason for this, but one possibility could be that the material used as blanks contains measurable but erratic quantities of tungsten, making it unsuitable for use as a blank when analyzing for tungsten. Some basic descriptive statistics for tungsten in blanks are shown in Table 12.11.

Table 12.11 Statistics for Tungsten in Blanks

Statistic	Value
Count	428
Max ppm	87.4
Min ppm	0.05
Average ppm	1.04
Median ppm	0.3
Mode ppm	0.2

MDA concludes that, while some of the material used for blanks is probably unsuitable for monitoring tungsten assays, the issue is not material to the resource reported herein.

12.5.4 Check Assays

In January of 2010 ATAC sent a set of pulps and a set of coarse rejects to ACME Analytical laboratories in Vancouver, B.C., for check assays. MDA matched 127 check assays of pulps and 131 check assays of coarse reject material to samples in the current Tiger assay table. MDA evaluated these using the same types of charts and statistical methods described for duplicates in section 12.5.2.

Table 12.12 Summary of Results for 2009 Check Assays

Type	Period	Corr. Coeff.*	Counts			RMA Regression	Averages as Percent	
			All	Used	Outliers	(y = dup, x = orig)	Rel Pct Diff	Abs Rel Pct Diff
Gold								
Pulp Check	2009	0.747	127	66	2	y = 0.974x + 0.244	-2.0	12.5
Crs. Reject	2009	0.953	131	64	6	y = 0.99x - 0.104	-8.6	21.8
Tungsten								
Pulp Check	2009	0.685	127	55	1	y = 1.107x - 12.577	6.2	11.4
Crs. Reject	2009	0.995	131	55	0	y = 1.081x + 13.27	21.6	25.4

Note: *Correlation coefficients are calculated using all the data, not the subset in the "Used" column. Relative differences in this table are averages of those calculated using Equation 12-3.

In Table 12.12, 59 pulp checks assays and 61 coarse reject check assays are not used because they have gold values less than 0.1 ppm Au. At those low grades the differences between the two labs are significantly greater than they are at higher grades. Eliminating the lower grades from statistical calculations yields comparisons that are more relevant to potentially economic mineralization.

A comparison between any two labs will show some bias. In the pulp check assays for gold, ACME is biased -2% relative to ALS, which is well within the range of typical biases between labs.

The relative difference for gold in the coarse rejects in Table 12.12, at -8.6%, is a moderately strong negative bias, that is, ACME's check assays using coarse rejects, at grades above 0.1 ppm Au, tend to be lower than those of ALS. If grades below 0.1 ppm Au are included, then overall ACME has a positive bias of +12.5% relative to ALS. MDA does not know the reasons for these biases in the coarse reject checks.

The method used by ACME for analyzing tungsten has a lower detection limit of 0.005% W or 50 ppm W, whereas the method used in the original assays done by ALS has a lower detection limit of 0.1 ppm W. Thus, any comparisons between ALS' and ACME's results at grades of less than 50 ppm W are meaningless. This forced MDA to eliminate 71 pairs of pulp assays and 76 pairs of coarse reject assays for tungsten from use in the pulp check comparison.

The check assays for tungsten show a significant positive bias (ACME > ALS) for both the pulp and the reject checks. MDA does not know the reason for this.

12.5.5 Verification Sampling for MDA

MDA has not visited the Tiger project. In order to obtain some verification assays that are not part of ATAC's in-house QA/QC, MDA selected six intervals of drill core for verification sampling. An individual who is not an employee and is otherwise independent of ATAC was onsite and verified that the intervals selected by MDA were the intervals that were sampled. This individual also witnessed the cutting of the first sample.

The verification sampling was done by ATAC personnel. They collected quarter-core samples using ATAC's normal core-sampling procedures. The verification samples were shipped to ALS as part of a routine shipment of ATAC's samples and analyzed by the same methods normally used for ATAC's samples. The results of the verification sampling are set out in Table 12.13.

Table 12.13 Results of Verification Sampling, Gold and Tungsten

Hole ID	Original Sample	Check Sample	Analyses in ppm			
			Original Gold	Verification Gold	Original Tungsten	Verification Tungsten
RAU-09-044	H884773	W843430	14.05	10.60	1110	335
RAU-09-046	H884821	W843427	9.39	5.90	271	275
RAU-10-115	I077434	W843428	0.35	0.30	4.5	2.5
RAU-10-131	I078180	W843425	0.39	0.23	268	316
RAU-15-139	R608660	W843426	1.17	0.95	12.7	12.8
RAU-17-151	W420019	W843429	0.97	1.94	354	387

The verification assays in Table 12.13 compare well to the original assays for the same intervals. The differences that exist are likely due to natural heterogeneity of the mineralization, unavoidable sampling variability, and the compromise inherent in comparing quarter-core to half-core samples.

12.5.6 Discussion of QA/QC Results

In the opinion of Mr. Ronning, the QA/QC results for gold support the use of the assay data in the Tiger database for the Resource estimate described in this report.

During all but the most recent drill campaign, tungsten was not contemplated as a possible economic contributor to the Tiger resource. Consequently, the QA/QC protocols were not designed with tungsten in mind. The data for the standards can be used to evaluate the primary laboratory's precision on tungsten analyses, with the caveat that there are no certified target values for tungsten. The data for duplicates are useful, and the 2010 check assays show that two labs give reasonably comparable results for tungsten. The tungsten data for the coarse blanks suggest that some of the material used for blanks contains variable low but measurable quantities of tungsten, making the material unsuitable for use in monitoring tungsten assays.

Despite the shortcomings of the QA/QC data with respect to tungsten, they are sufficient to give an acceptable level of confidence in the tungsten assays used in the resource estimate, considering that the overall contribution of the tungsten to the potential economics of the deposit is greatly subordinate to the contribution of the gold.

12.6 SUMMARY STATEMENT ON DATA VERIFICATION

The author responsible for MDA's data verification for the Tiger Project is Mr. Ronning. ATAC has provided all of the available information that MDA has requested and has been forthcoming in all respects relating to the data.

In Mr. Ronning's opinion the data provided to MDA by ATAC for use in the Resource Estimate described in this report are of good quality and form a sufficient basis for the Resource Estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Preliminary metallurgical testwork has been conducted on samples from the Tiger Deposit through multiple different studies, initially by G&T Metallurgical in 2010 and later by SGS Canada in Lakefield and Vancouver in 2011 and 2012. Heap leach amenability studies were conducted in 2014 by Kappes Cassidy and Associates, and in 2016 Blue Coast Research conducted additional testing to start building a picture of variability in leach recovery in both sulphide and oxide zones. These data were used to create the metallurgical forecast used in the 2016 PEA.

Subsequent to the 2016 study, mostly in 2018, Blue Coast Research conducted additional work focusing on samples from the sulphide zone.

The following sections primarily focus on testwork conducted subsequent to 2016, and metallurgical projections used in this updated Technical Report. For details additional to those described in this section, on testwork conducted prior to 2016, the reader is referred to the NI 43-101 technical report entitled "Technical Report and Preliminary Economic Assessment of the Tiger Deposit, Rackla Gold Project, Yukon, Canada" and dated May 31st, 2016.

13.1 OXIDES

Limited mineralogy work has been done to date on Tiger oxide material. Petrographic and XRD work has shown it to comprise quartz, dolomite and goethite with lesser limonite, rutile, hematite, calcite, various altered silicates and pyrite (Figure 13.1).

Figure 13.1 Photomicrographs of Tiger Oxide Textures

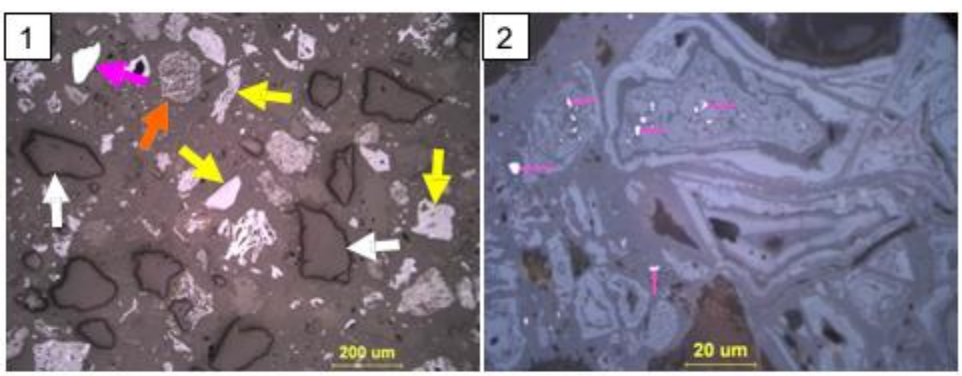


Plate 1 - Pink Arrow: Pyrite, Yellow Arrows: goethite, Orange arrow: rutile, White arrow: Non opaque. Plate 2 - Pink Arrows: Fine pyrite inclusions.

SGS studied a composite representing material from 51 core samples and assaying 4.8 g/t. Gravity testwork at SGS recovered 18% of the gold suggesting the presence of a small amount of coarse gold. Cyanidation was conducted on whole mineralized material at four different grind sizes, with leach extractions being similar in all cases at around 90%. Similar leaches were run on the gravity tails with a similar final residue assay. There is no evidence of enhanced overall recoveries from the use of gravity in conjunction with cyanidation.

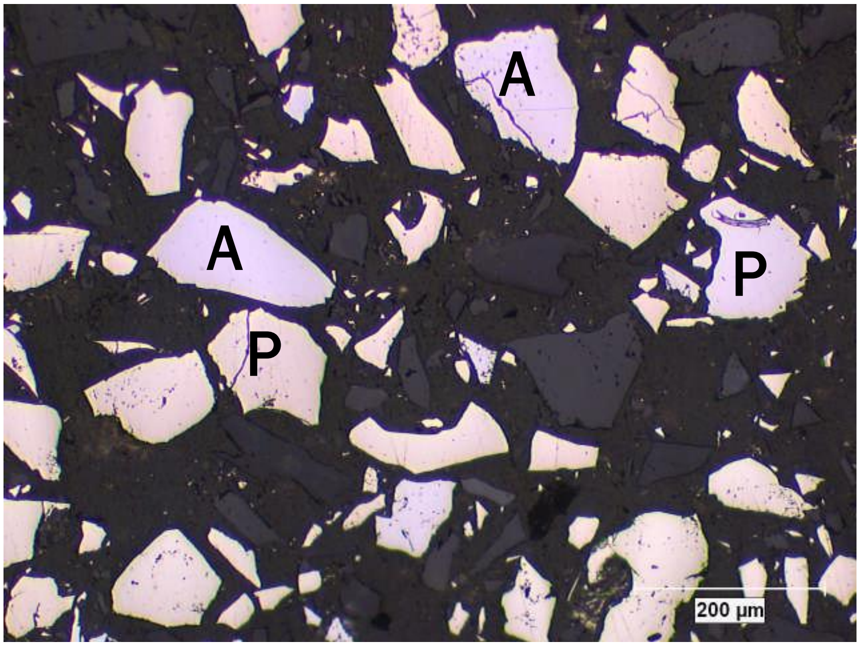
In 2016, Blue Coast Research studied a composite and a suite of variability samples created from 88 different intervals, taken from 8 different drill holes. The material from each different drill hole was used to create a separate variability composite. These composites varied in head grade from 1.4 g/t to 13 g/t. A master composite was created from all 88 samples. This was leached at 172 and 65 microns, with the leach recovery being 90% and 92%, respectively, indicating a small benefit can be gained from finer grinding.

The arithmetic average leach extraction from the eight variability composites was 88%. Five of the eight samples leached well, on average yielding over 90% recovery while the other three responded relatively poorly with an average extraction of less than 80%. This bimodal distribution in response warrants closer investigation in the future. Average cyanide consumption was 0.29 kg/tonne and lime consumption 5 kg/tonne. Kinetics data show the leach to be close to completion after 24 hours, with very little additional recovery being achieved from another 24 hours of leaching.

13.2 SULPHIDES

The sulphide mineralogy of this material is comprised mostly of pyrite and arsenopyrite, The average abundance from 13 samples has been 34% pyrite and 7% arsenopyrite. Non-sulphides mostly comprise dolomite (26%), ankerite (17%) and quartz (4%). The sulphides are commonly coarse and when ground to 80% passing 65 microns are mostly liberated, as shown in Figure 13.2.

Figure 13.2 Photomicrographs of Tiger Oxide Textures



Note: A–Arsenopyrite, P–Pyrite.

In 2018, eight samples were submitted to Surface Science Western in London, Ontario to establish the gold content contained in solid solution in the key sulphides. The test results are presented in Table 13.1. This is important as it has a direct bearing on the type of process needed to recover the gold. As it was assumed most of the refractory gold would be in arsenopyrite, this was the primary host mineral of interest. On average across the eight samples, 16% of the gold was contained in solid solution in the sulphides, though this varied widely, from 3% to 33%. Pyrite was not extensively studied but based on available data the gold content is probably close to 10%, ranging from as little as 5%, up to 36%.

The grades of gold in the sulphides were low, averaging 9 g/t in the arsenopyrite and 0.9 g/t in pyrite. Both grades are too low for economic treatment by roasting, autoclaving or bioleaching, the typical treatment routes required to recover this gold. They are also too low to consider the production of a saleable gold concentrate by flotation. Accordingly, gold recovery by gravity and/or direct cyanidation is probably the only viable process route for recovery of gold from the Tiger sulphide material.

The individual samples were combined to create four composites as shown below. Two of the composites, the most free-milling and most refractory, were tested for gravity recovery. The free milling sample recovered 42% of the gold to a gravity concentrate, the refractory sample 10%. This suggests that the sulphide mineralized materials may contain slightly coarser discrete gold than the oxides.

Flotation is sometimes used as a pre-concentration step ahead of cyanidation where this allows for a significant reduction in the size of the leach circuit without significant loss in

P

gold recovery. To explore this, the same two samples were floated, with 95-97% gold recovered to a concentrate, albeit at about 40% mass pull in each case. Sulphur recoveries were similar to the gold. While recoveries are high, the high mass pull rates negate from any clear process advantage being gained, so flotation does not appear to be a useful process for Tiger's sulphide mineralized materials.

Finally, composites 1B, 2 and 3, as well as selected samples were subjected to bottle roll leach testing. A brief leach optimization program was conducted on Composite 1B. This work indicated that the sulphides tend to consume cyanide so doses need to be carefully maintained early in the leach. Lead nitrate was added in one test which seemed to have a beneficial effect on leach performance. Pre-oxidation was not tested, as this may have a similar effect of passivating sulphide surfaces. Sulphide oxidation is likely to be impeding leaching so may be best completed before cyanide is added.

Grind size was also tested, within a range of 80 percent passing 50 to 100 microns. The finer grind dropped residue assays, however, variability in the calculated head assays masked any real benefit in recovery so the effect of primary grind size remains unclear.

From Composites 1B and 3 recoveries were considerably higher than had been predicted from the AuCN analyses, but similar to those predicted from the mineralogy. Similarly, two of the more free-milling samples, namely S-1 and S-6 were leached and yielded recoveries of 91% and 87% respectively. Again, these recoveries were close to those indicated from the mineralogy work.

The more refractory composite, Composite 2, yielded just 18% leach recovery. This is far lower than predicted by the mineralogy work, but still somewhat higher than predicted from the AuCN assays. The refractory sample S-2 was tested in the same way, yielding similar results. The reason for the apparent difference in leach extraction and gold mineralogical balance has so far not been identified. Sometimes this points to the presence of a preg-robbing component in the feed, but preg-robbing tests conducted to date on Tiger samples have indicated an absence of preg-robbing characteristics so this can probably be discounted. A more likely reason lies in the statistics behind the mineralogy work, and the presence of sulphide grains with much higher gold contents not being adequately represented in the probe work.

Table 13.1 Testing of Sulphide Samples at Blue Coast Research in 2018

Sample Name	Sample g/t Au	Sample % AuCN	Composite Name	Composite geochem % AuCN	Process response	mass pull	% Au recovery
S-1	3.56	71.4%			Sample S-1		
					Bottle roll leach		91%
					Projected leachable by mineralogy		94%
S-3	13.2	69.2%	Comp 1	66.8%	Comp 1 (free milling)		
						mass pull	
					Gravity concentration	1.2%	41%
S-7	2.88	50.3%			Flotation	37.5%	97%
					Projected leachable by mineralogy		93%
			Comp 1B	68.4%	Comp 1B (free milling)		
G-1	6.82	n/a			Bottle roll leach		88%
					Projected leachable by mineralogy		92%
G-2	4.32	n/a					
S-2	2.95	16.6%			Sample S-2		
					Bottle roll leach		18%
					Projected leachable by mineralogy		57%
S-4	6.67	13.7%	Comp 2	13.2%	Comp 2 (refractory)		
					Bottle roll leach		17%
						mass pull	
					Gravity concentration	1.3%	10%
S-5	3.33	9.3%			Flotation	42.0%	95%
					Projected leachable by mineralogy		16%
S-6	1.72	52.3%			Sample S-6		
					Bottle roll leach		87%
					Projected leachable by mineralogy		85%
S-7	2.88	50.3%	Comp 3	49.0%	Comp 3 (transition)		
					Bottle roll leach		79%
					Projected leachable by mineralogy		77%
S-8	1.88	44.1%					

13.3 TUNGSTEN RECOVERY

Tungsten recovery was tested on an oxide sample, using a Knelson concentrator as a roughing device, the Knelson concentrate being cleaned by tabling. The small sample, coupled with the low tungsten head grade (0.03% W), meant that demonstrating the production of a high-grade concentrate was impractical, however, a 0.25g concentrate sample was created. This concentrate assayed 5.3% tungsten and contained 37% of the tungsten. This suggests that, where present, at least some of the tungsten is amenable to gravity recovery.

Tungsten recovery was also tested on a sulphide sample, but the presence of the dense sulphides interfered with the test. In practice, the sulphides would need to be floated ahead of gravity, which was not the case in the test and would not be a practical process given the cost of such a flotation circuit and the tungsten content in the mineralized material. It is therefore reasonable to assume that tungsten recovery from sulphide material would not be economic.

13.4 METALLURGICAL FORECAST

13.4.1 OXIDES

Gold leach recovery from oxide samples has been regressed against a host of candidate geochemical parameters, without exposing any useful algorithms for predicting gold extraction. Significantly, evidence from this data of a head grade effect on recovery is weak so a fixed gold recovery has been used for the 2020 PEA (Table 13.2).

Table 13.2 Gold Extraction from Leaching of Nine Oxide Samples

	p80	Au Head Grade, g/t	24 hr Extraction, %	Feed Gold Units	Recovered Gold Units
SGS Oxide	69	4.93	91.0	4.93	4.49
Comp 01	65	3.26	88.2	3.26	2.88
Comp 02	65	5.11	97.1	5.11	4.96
Comp 03	65	1.58	92.7	1.58	1.47
Comp 04	65	4.38	76.5	4.38	3.35
Comp 05	65	13.18	97.7	13.18	12.88
Comp 06	65	1.73	76.9	1.73	1.33
Comp 07	65	2.94	80.9	2.94	2.38
Comp 08	65	1.48	91.0	1.48	1.35
Average Grade, g/t		4.29		4.29	3.90
Gold Recovery, 24 hr Leach					90.9%
Projected Gold Recovery, 36 hr Leach (Based on SGS 48 hr Kinetic Test Data)					91.4%

The average grade of gold in the nine tests was 4.29 g/t, which is quite close to the projected LOM feed grade. After 24 hours of cyanide leach, 90.9% of the gold in the nine samples had been leached. The designed leach residence time is 36 hours, which based on SGS kinetic data should add another 0.5% recovery (to 91.4%). Assuming a 1% drop in gold recovery as result of solution and carbon losses, the recovery to doré becomes 90.5%.

There is no supportable evidence of a gold head grade/leach recovery relationship, so a fixed recovery has been assumed for this PEA.

Gold recovery from oxides = 90.5%

13.4.2 Sulphides

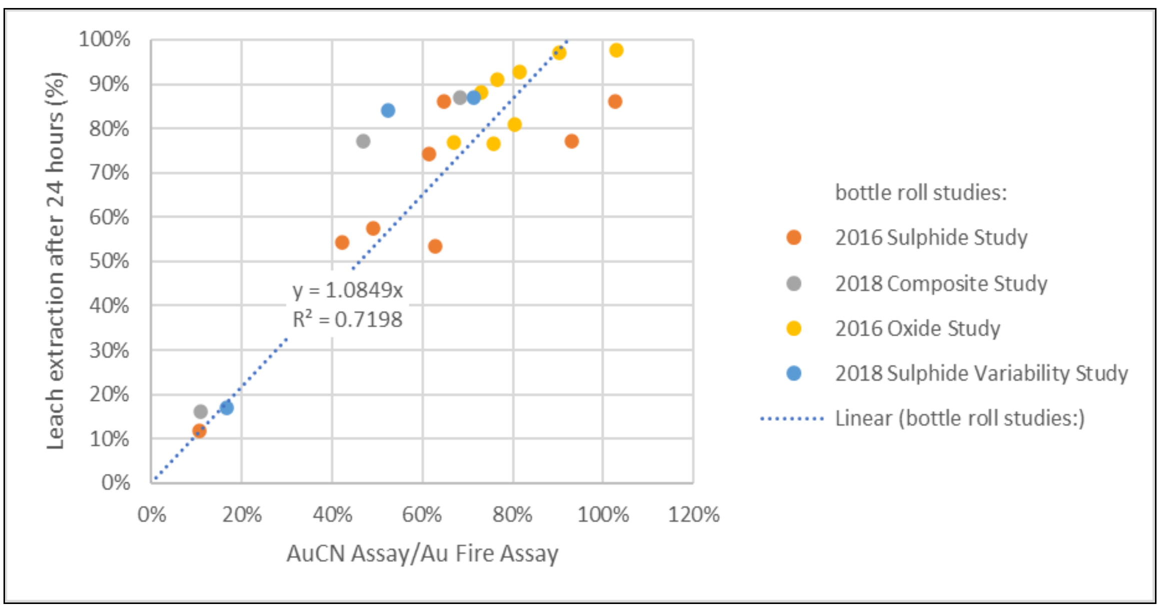
Unlike for the oxides, there are enough available AuCN geochem assays for sulphide material to allow for creation of an AuCN-enabled resource model.

Gold extraction rates by bottle roll leaching are plotted below for 22 samples vs. AuCN estimates (Figure 13.3). These include 14 sulphide samples and 8 oxide samples. On average, leach extractions after 24 hours are 8.5% higher than the AuCN/AuFA assay ratio.

As 1% of the gold extracted to solution is lost in the process plant to solution and carbon losses, the recovery of gold from sulphide samples should be assumed to be:

Gold recovery from sulphides: 1.074 x the AuCN/AuFA assay ratio.

Figure 13.3 Relationship Between AuCN/AuFA Assay Ratio and 24 hr Bottle Roll Extraction



There may be small pockets of non-dolomitic sulphide material where insufficient AuCN data are available for modelling purposes. For these materials, the average bottle roll leach recovery for all sulphide samples tested so far has been used (63%).

13.4.3 Tungsten Recovery

From the two gravity tests have been run to explore the recovery of tungsten, there is evidence of the potential for limited tungsten recovery from oxide materials where there is enough tungsten in the feed to warrant its recovery. Based on the result from the oxide test, there is potential for recovery of 30% of the tungsten to a saleable concentrate from such materials.

14.0 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

The mineral resource estimation for the Tiger project was completed in accordance with the guidelines of Canadian National Instrument 43-101 ("NI 43-101"). The modeling and estimation of the mineral resources were completed on January 3, 2020 under the supervision of Mr. Steven J. Ristorcelli, a qualified person with respect to mineral resource estimations under NI 43-101. The Effective Date of the resource estimate is January 3, 2020. The Effective Date of the Tiger database on which this Resource estimate is based is November 23, 2019.

Mr. Ristorcelli is independent of ATAC by the definitions and criteria set forth in NI 43-101; there is no affiliation between Mr. Ristorcelli and ATAC except that of independent consultant/client relationships. Mr. Ristorcelli is not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the Tiger project mineral resources as of the date of this report.

The Tiger project mineral resources are classified in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories in accordance with the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" (2014) and therefore NI 43-101. CIM mineral resource definitions are given below, with CIM's explanatory text shown in italics:

MINERAL RESOURCE

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

INFERRED MINERAL RESOURCE

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and *quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.*

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This *category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.*

Mr. Ristorcelli reports resources at cutoffs that are reasonable for deposits of this nature given anticipated mining methods and plant processing costs, while also considering economic conditions, because of the regulatory requirements that a resource exists “in such form and quantity and of such a grade or quality that it has reasonable prospects for eventual economic extraction.”

The resource block model is rotated 42^{o} counterclockwise (to the northwest) and the blocks are 5m by 5m by 5m. Gold and tungsten resources are reported based on gold cutoff grades.

14.2 DATABASE

There are 166 holes totaling 28,610m of drilling in the resource database. All of the drill holes are core holes. Table 14.1 presents descriptive statistics for the ATAC data in the Tiger database that was audited and imported into MineSight by MDA. The database contains 6,861 gold assay records and 6,811 tungsten assay records. The average drill spacing is presently around 30m within the mineralized zones.

Table 14.1 Descriptive Statistics - Resource Drill-Hole Database

Resource Database								
	Valid	Median	Mean	Std. Dev.	CV	Minimum	Maximum	Units
From	7,899					0.0	590.4	m
To	7,899					0.1	593.5	m
Length	7,899	2.7	3.6			0.01	83.1	m
Au	6,861	0.069	1.013	5.284	5.2	0.002	175.0	g/t
W	6,811	10.1	112.6	376.4	3.3	0.05	10001	ppm
AuCN	328	0.960	1.902	2.563	1.3	0.060	19.7	g/t
AuCN/AuFA Ratio*	328	67.0	62.0	27.7	0.5	3.00	168.0	%
Core recovery	6,864	94.0	83.6	22.4	0.3	0.0	100.0	%
RQD	6,864	55.6	49.3	37.0	0.8	0.0	100.0	%
Density	429	2.86	2.93	0.70	0.24	0.26	9.95	g/cm^3

*Note: Those ratios >100% result from imprecision in analytical data at low grades

Logged core recovery and RQD were loaded into the database but were not audited. The database also contains logged geologic features, including rock type and unit, magnetic susceptibility, texture, and alteration.

14.3 MODELS

ATAC built 3D interpretations for two dolomite units (both unoxidized), four volcanic units including the leopard spot unit, three oxide units, and overburden. ATAC also modeled a surface that separates generally higher-cyanide-soluble-gold from generally lower-cyanide-soluble-gold zones in the dolomite. Faults were modeled by ATAC, but those faults are more conceptual than explicit. MDA used each of these solids and surfaces to refine the interpretations slightly on sections (oriented N48° E and looking 42° northwest) using logged lithologic data. Aside from grouping the individual units, i.e., two dolomite units were modeled as one, the units were substantially the same as those interpreted by ATAC. After modeling the units on sections, those 2D interpretations were snapped to drill holes and then sliced on sections perpendicular to the original sections on five-meter intervals along the block model columns. These slices were then re-interpreted so each block-model column was coded by interpretations on long-sections. This produces pseudo-solids, which MDA used to code the model. Detailed stratigraphic descriptions are given in Section 7.3.

MDA used these unit-interpretations to guide the explicit modeling of gold and tungsten domains on cross-sections with the same N48° E orientation as the lithologic sections, looking northwest and spaced 25m apart. The dominant host rock is the oxide unit (protolith may be dolomite prior to complete destruction) which for the most part appears texturally amorphous within most drill intersections, but occasionally exhibits residual colour banding that may represent relict sulphide textures or bedding. Within the oxide zone sulphide minerals range from being completely absent to being present as relicts.

Sulphide mineralization is accompanied by, and developed within, limestone that is replaced by ferruginous dolomite and iron carbonate minerals. Sulphide species consist of disseminated to banded pyrite, with subordinate arsenopyrite and pyrrhotite and minor bismuthinite and sphalerite. Small amounts of disseminated scheelite are also present.

Gold and tungsten domains were defined based on population breaks on cumulative probability plots and grade changes in the drill-hole samples. Core photos were reviewed and used to support the metal-domain interpretations. Clear geologic breaks are generally absent because of the extreme oxidation, and even in the carbonate-hosted mineralization distinctive changes in geologic features are not common. After sectional interpretations were completed, the gold and tungsten domains were snapped to the drill holes and sliced for modeling on vertical long-section planes at 5m spacing.

14.3.1 GOLD MODEL

Three gold domains with the following characteristics were defined based on cumulative probability plots ("CPPs"; Figure 14.1):

- Low-grade halo from ~0.01 to ~0.9g Au/t,
- Mid-grade domain from ~0.9 to ~6.5g Au/t, and
- High-grade domain above ~6.5g Au/t

Descriptive statistics are presented in Table 14.2. The gold mineralization largely occurs grossly parallel to the stratigraphy but is much better developed in the oxide zone. A representative cross section is given in Figure 14.2. Overall the continuity seems to be grossly predictable.

Figure 14.1 Cumulative Probability Plot of Gold Assays

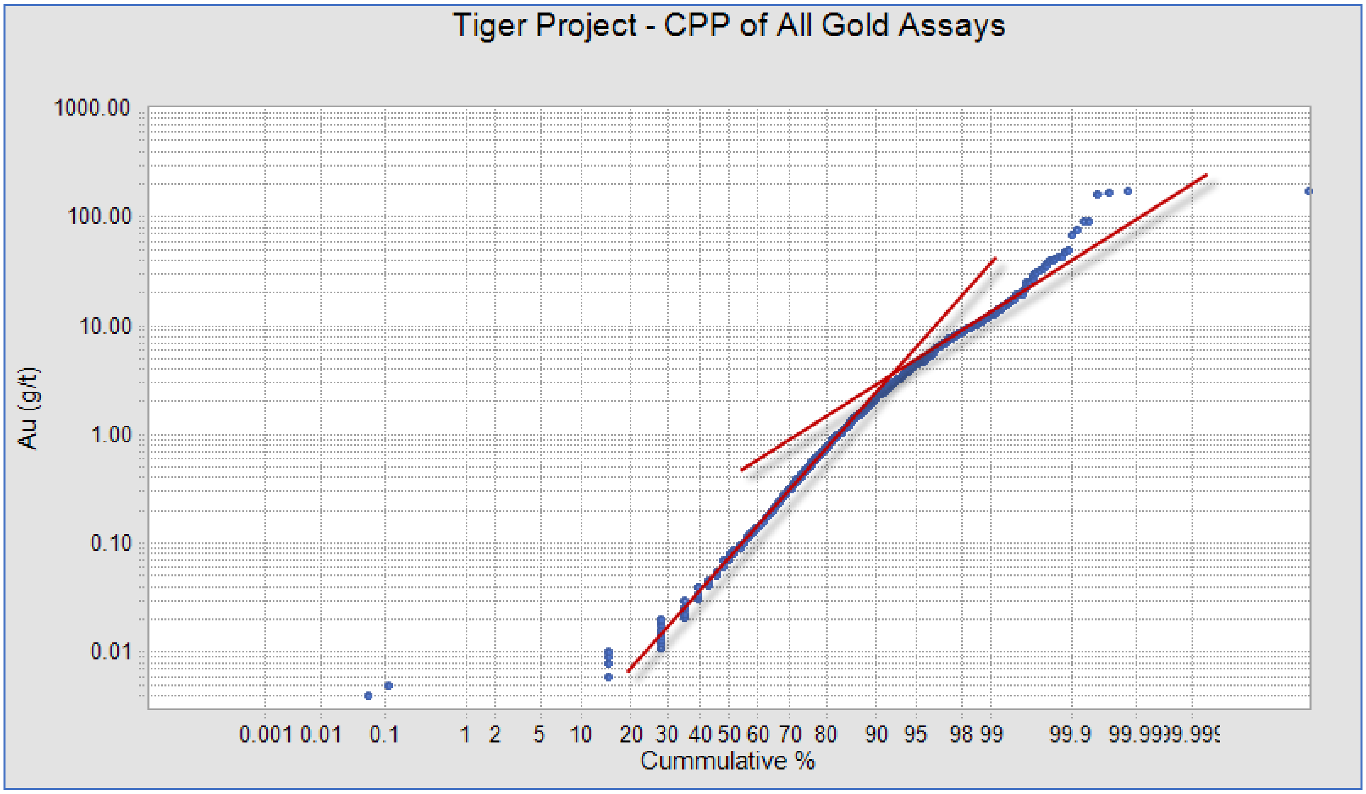


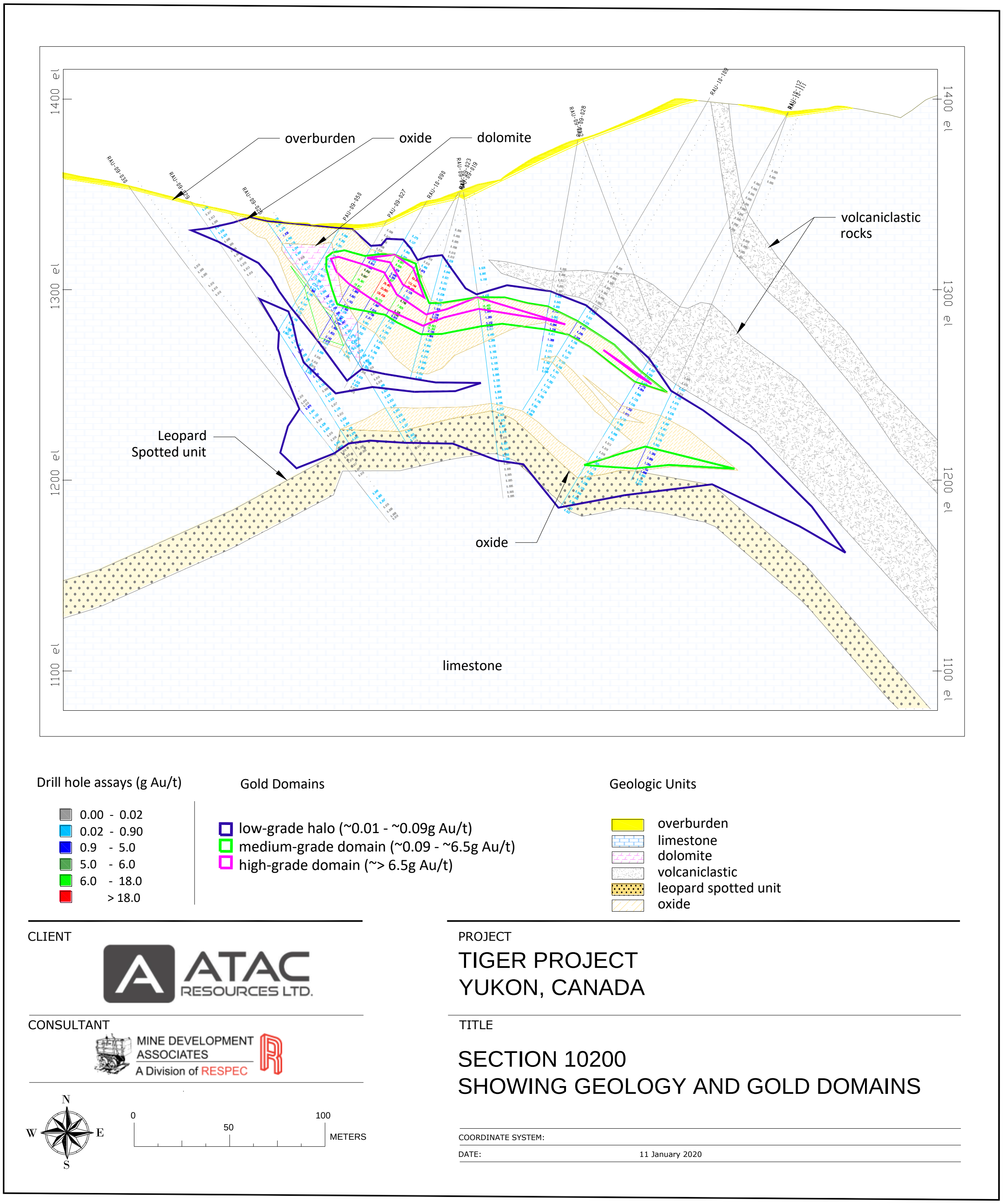
Table 14.2 Descriptive Statistics by Gold Domain

Low-Grade Gold Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
Length	3,964	2.2	2.2			0.01	10.7	m
Au	3,900	0.131	0.394	0.790	2.00	0.005	14.3	g/t
Capped Au	3,900	0.130	0.392	0.756	1.93	0.005	8.5	g/t
AuCN	177	0.491	0.790	0.926	1.17	0.040	7.9	g/t
AuCN/AuFA Ratio	177	66.0	63.2	26.8	0.4	3.0	168.0	%
Core recovery	3,902	92.7	81.5	23.7	0.3	0.0	100.0	%
RQD	3,902	48.3	46.3	36.9	0.8	0.0	100.0	%
Density	260	2.87	2.99	0.54	0.18	2.03	5.06	g/cm3
Mid-Grade Gold Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
Length	620	1.6	2.1			0.31	6.1	m
Au	613	2.131	2.745	2.191	0.80	0.040	15.2	g/t
Capped Au	613	2.130	2.745	2.191	0.80	0.040	15.2	g/t
AuCN	162	1.711	2.270	1.924	0.85	0.080	11.1	g/t
AuCN/AuFA Ratio	162	73.0	68.5	28.5	0.4	3.0	224.0	%
Core recovery	613	86.8	76.7	25.1	0.3	0.0	100.0	%
RQD	613	0.0	28.7	38.8	1.4	0.0	100.0	%
Density	27	2.88	2.80	0.57	0.20	1.95	3.85	g/cm3
High-Grade Gold Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
Length	256	1.5	2.0			0.44	6.1	m
Au	253	8.590	14.541	23.092	1.59	0.210	175.0	g/t
Capped Au	253	8.590	13.297	15.660	1.18	0.210	90.0	g/t
AuCN	68	5.809	16.340	36.878	2.26	0.200	196.5	g/t
AuCN/AuFA Ratio	68	68.0	62.8	31.3	0.5	7.0	120.0	%
Core recovery	253	84.2	74.5	25.3	0.3	0.0	100.0	%
RQD	253	0.0	26.5	38.6	1.5	0.0	100.0	%
Density	9	2.33	2.85	1.06	0.37	1.90	5.13	g/cm3
Outside Modeled Gold Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
Length	3,059	3.0	5.9			0.01	83.1	m
Au	2,095	0.006	0.026	0.210	8.22	0.002	8.1	g/t
Capped Au	2,095	0.007	0.022	0.097	4.45	0.002	2.0	g/t
AuCN	1	0.200	0.200	0.000	0.00	0.200	0.2	g/t
AuCN/AuFA Ratio	1	26.0	26.0	0.0	0.0	26.0	26.0	%

table continues...

Core recovery	2,096	96.7	90.6	15.9	0.2	0.0	100.0	%
RQD	2,096.0	72.8	63.5	30.4	0.5	0.0	100.0	%
Density	165	2.88	2.94	0.33	0.11	2.31	4.62	g/cm3

Figure 14.2 Tiger Gold Domains– Section 10200



14.3.2 TUNGSTEN MODEL

As with the gold domains, the geologic model guided the explicitly modeled tungsten domains. The tungsten domains have similar geometry to the gold domains, but not the same. Three tungsten domains were defined based on population breaks on cumulative probability plots (Figure 14.3):

- Low-grade halo from ~2 to ~10ppm W
- Mid-grade domain from ~10 to ~170ppm W
- High-grade domain >~170ppm W

Descriptive statistics are presented in Table 14.3. Like the gold mineralization, the tungsten mineralization mostly occurs grossly parallel to stratigraphy. A representative cross section is given in Figure 14.4. Overall the continuity is predictable, much like gold.

Figure 14.3 Cumulative Probability Plot of Tungsten Assays

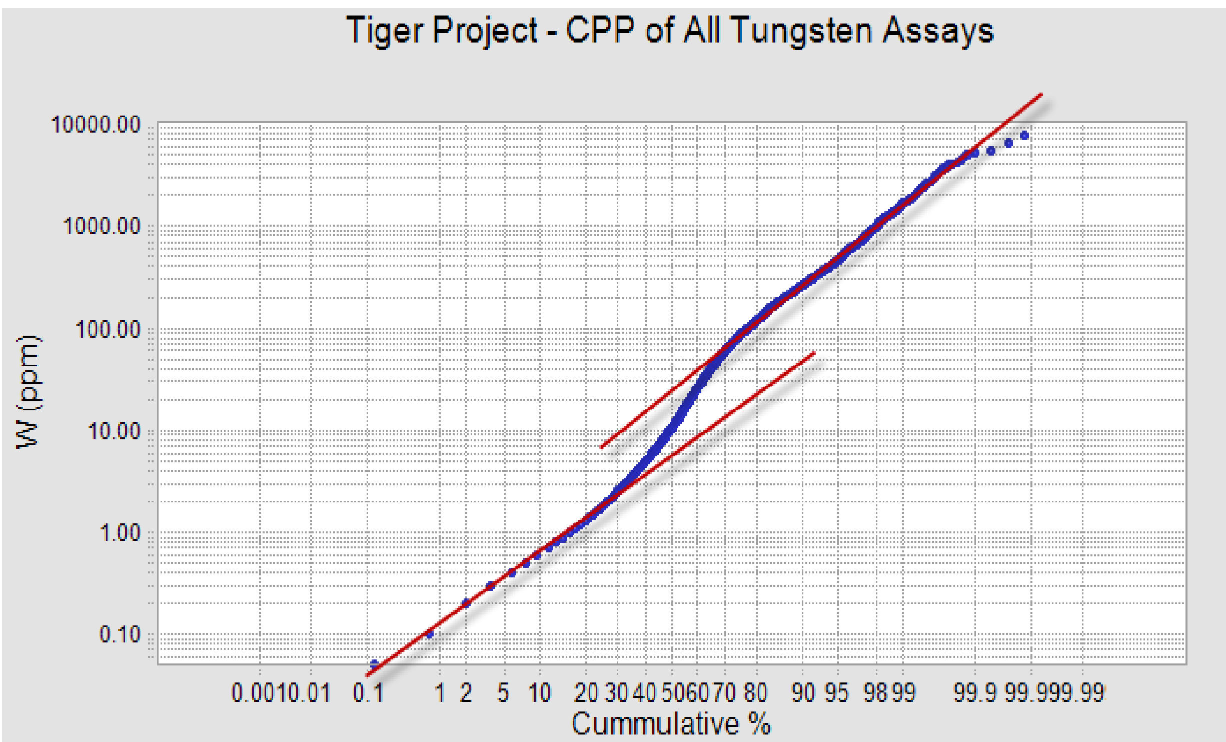


Table 14.3 Descriptive Statistics by Tungsten Domain

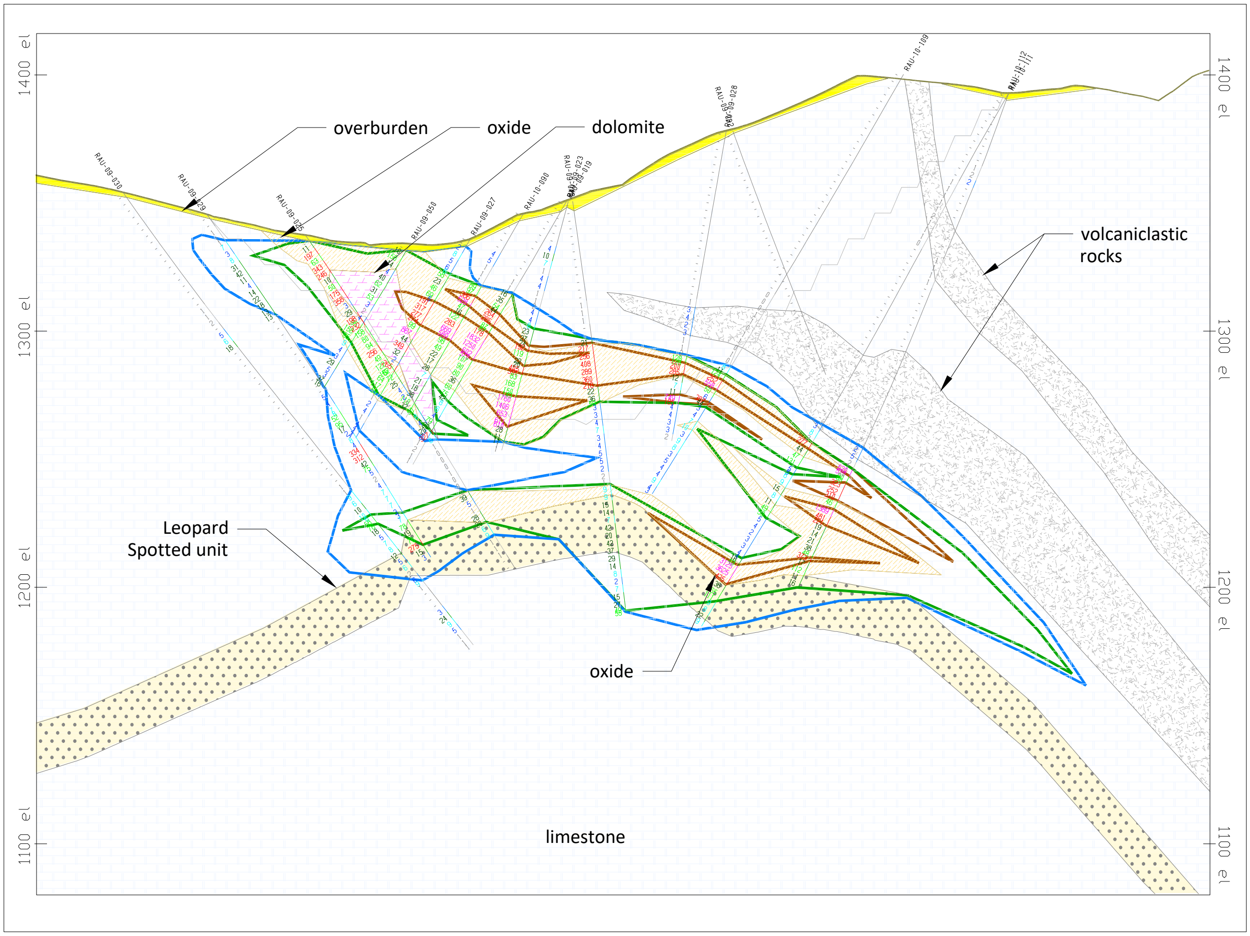
Low-Grade Tungsten Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
Length	1,282	2.4	2.3			0.01	9.2	m
W	1,258	4	10	65	6.4	0.1	1820	ppm
Capped W	1,258	4	8	15	1.9	0.1	150	ppm
Core recovery	1,260	95	87	19	0.2	0.0	100	%
RQD	1,260	59	53	33	0.6	0.0	100	%
Density	119	2.86	2.91	0.42	0.1	1.90	5.03	g/cm3

Mid-Grade Tungsten Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
Length	2,455	2.0	2.1			0.01	10.7	m
W	2,398	45	73	142	1.9	0.5	3200	ppm
Capped W	2,398	45	70	98	1.4	0.5	1000	ppm
Core recovery	2,428	90	78	26	0.3	0.0	100	%
RQD	2,428	30	40	38	1.0	0.0	100	%
Density	125	2.93	2.99	0.61	0.2	1.95	5.13	g/cm3

High-Grade Tungsten Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
Length	1,062	2.0	2.2			0.30	6.1	m
W	1,032	288	542	798	1.5	2.0	10001	ppm
Capped W	1,032	288	535	733	1.4	2.0	5500	ppm
Core recovery	1,053	88	78	24	0.3	0.0	100	%
RQD	1,053	13	36	40	1.1	0.0	100	%
Density	48	2.91	3.08	0.66	0.2	2.03	4.64	g/cm3

Outside Modeled Tungsten Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
Length	3,100	3.0	5.9			0.01	83.1	m
W	2,123	1	9	109	12.1	0.1	3670	ppm
Capped W	2,123	1	4	13	3.5	0.1	100	ppm
Core recovery	2,123	97	91	16	0.2	0.0	100	%
RQD	2,123	74	65	30	0.5	0.0	100	%
Density	169	2.87	2.93	0.37	0.1	2.20	4.85	g/cm3

Figure 14.4 Tiger Tungsten Domains– Section 10200



Drill hole assays (ppm W)

- 0 - 2
- 2 - 6
- 6 - 10
- 10 - 45
- 45 - 170
- 170 - 500
- 500 - 40000
- > 40000

Tungsten Domains

- low-grade halo (~2 - 10ppm W)
- medium-grade domain (~10 - 170ppm W)
- high-grade domain (~>170ppm W)

Geologic Units

- overburden
- limestone
- dolomite
- volcaniclastic
- leopard spotted unit
- oxide

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PROJECT
TIGER PROJECT
YUKON, CANADA

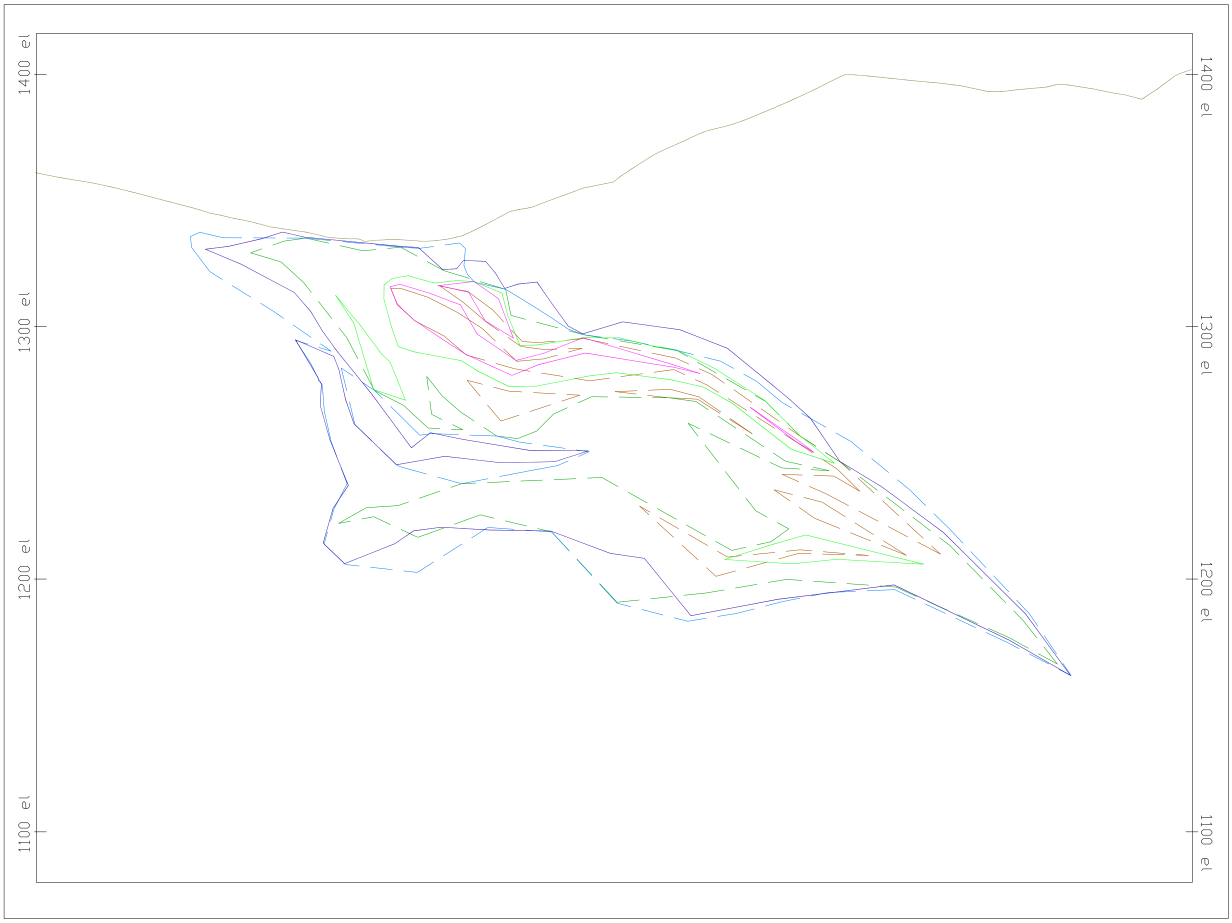
TITLE
SECTION 10200
SHOWING GEOLOGY AND TUNGSTEN DOMAINS

COORDINATE SYSTEM:
DATE: 11 January 2020

N S E W
0 50 100 METERS

Gold and tungsten are individually zoned and occur in similar form, often overlapping, sometimes not. Figure 14.5 presents the tungsten and gold domains as interpreted and explicitly modeled. In general, the tungsten domains are larger and more common than the gold domains.

Figure 14.5 Tiger Gold and Tungsten Domains– Section 10200



Gold Domains

- low-grade halo (~0.01 - ~0.09g Au/t)
- medium-grade domain (~0.09 - ~6.5g Au/t)
- high-grade domain (~> 6.5g Au/t)

Tungsten Domains

- low-grade halo (~2 - 10ppm W)
- medium-grade domain (~10 - 170ppm W)
- high-grade domain (~>170ppm W)

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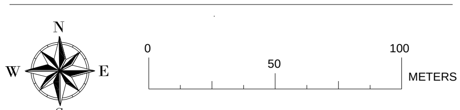


PROJECT

TIGER PROJECT
YUKON, CANADA

TITLE

SECTION 10200
SHOWING GOLD AND TUNGSTEN DOMAINS

COORDINATE SYSTEM:

DATE: 11 January 2020

14.3.3 Cyanide Soluble Gold Model

The amount of gold that is soluble by cyanide in the oxide is high and consistent. That is not the case in the dolomite which has high variability in gold solubility, but it does seem to have higher recoveries in the upper half of the unit than the lower half of the unit. Consequently, MDA divided the dolomite into an upper domain and a lower domain and modeled each area independently.

There are not many samples with CN gold assays. Considering the few CN gold assays along with the high variability in recoveries, this model is not considered reliable for final mine planning if CN solubility variability materially affects the economics of the gold mineralization in the dolomite.

14.4 Density

There are 458 density measurements in the Tiger resource database within assayed intervals. The average density values, and the values assigned to the units in the model, are summarized in Table 14.4. Density measurements were made by ATAC geologists. Measured samples are well-distributed throughout the entire deposit. ATAC took special care because of the complications of having highly oxidized material. They took measurements using a multiple variety of methods (see Section 12.4), but after detailed reviews, MDA chose to use the conventional density measurements.

Table 14.4 Density Values Applied to the Tiger Block Model

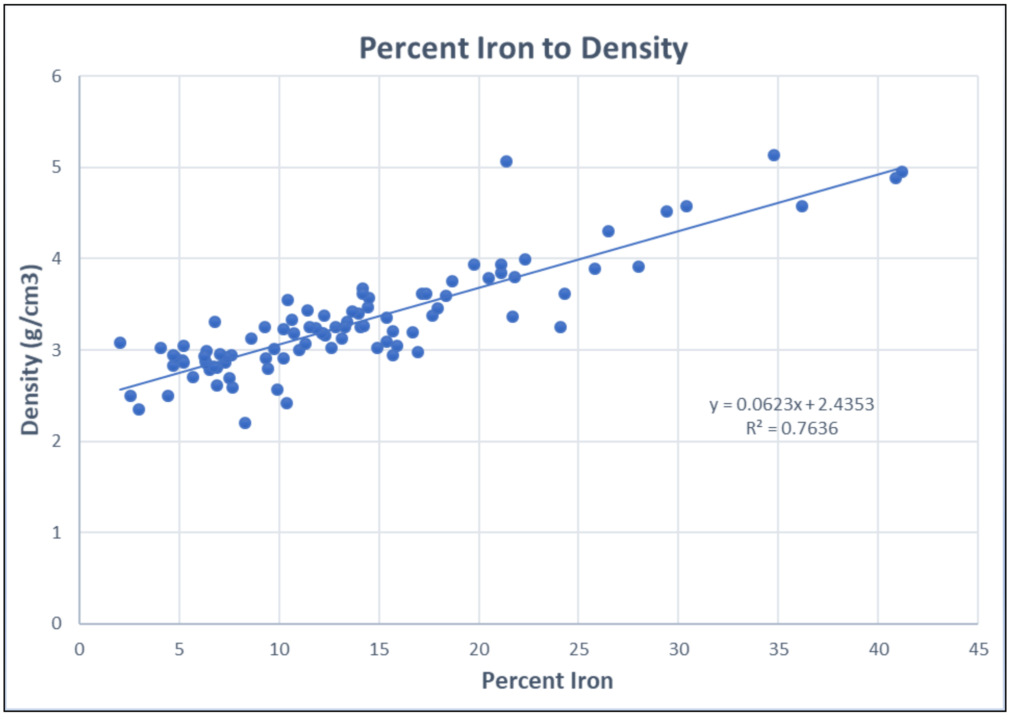
Description	Count	Mean (g/cm3)	Proposed Assigned Density
Upper Dolomite	95	3.31	(0.0623 * FEIT) +2.4353
All Oxide Zones	41	2.40	2.30
Volcanic 1a	52	2.97	2.95
Volcanic 1b		2.86	
Volcanic 2		2.94	
Leopard Spotted Unit	55	2.98	3.00
Default (mostly Limestone)	215	2.90	2.85
Overburden	0	NA	1.80

*FEIT = estimated iron grade

The rock, except for the oxide, has low porosity and permeability so the immersion method is deemed appropriate and was used without sample coating. ATAC did not check for moisture content of the rock but did at least air dry the samples. The lack of active oven-drying likely does not have a material impact on measured density for rocks other than the oxide. However, the lack of active oven-drying may well have an impact on the oxide rock since it has high clay content. That was a factor in lowering the applied density of 2.30g/cm^3 from the average measured density of 2.40g/cm^3.

Because the sulphide minerals are variable and sometimes occur in important concentrations, MDA evaluated the relationship between analyzed iron and sulfur. The overlimits in sulphide rendered that relationship not useful, although there is a reasonable relationship. The relationship of density to iron is good (Figure 14.6). MDA used the database analyses to estimate iron into the model and then used the regression shown in Table 14.4 and Figure 14.6 to determine a variable density within the modeled dolomite.

Figure 14.6 Scatterplot of Analyzed Fe% to Measured Density in the Dolomite



Note: The blue line is a best-fit line.

14.5 Sample and Composite Statistics

Once the mineral domains were defined and modeled on the 25m-spaced cross sections, the samples were coded to the gold and tungsten domains by the polygon interpretations on those sections. Quantile plots were made of the coded assays. Capping for each domain was determined by first assessing the grade above which the outliers occur and then the outlier grades were reviewed on screen to determine materiality, grade and proximity of the closest samples, and general location. Capping levels and number of samples capped are presented in Table 14.5 and Table 14.6 for gold and tungsten, respectively. Descriptive statistics of each domain were generated and then considered when deciding capping levels. Capping values were determined for each of the gold and tungsten domains separately.

Table 14.5 Capping Levels for Gold by Domain

Domain	Number	Capped g Au/t	Max g Au/t
Low-grade	3	8.50	14.3
Mid-grade	0	15.15	15.15
High-grade	4	90.00	175.0
Outside	2	2.00	8.1

Table 14.6 Capping Levels for Tungsten by Domain

Domain	Number	Capped ppm W	Max ppm W
Low-grade	4	150	1820
Mid-grade	10	1000	3200
High-grade	3	5500	10000
Outside	27	100	3670

Once the capping was completed, the drill holes were down-hole composited to 3m intervals honoring domain boundaries. Three meters was chosen because the most samples are 1.5m or 3m in length. Descriptive statistics of the composite database are given in Table 14.7 and Table 14.8 for gold and tungsten, respectively.

Table 14.7 Descriptive Composite Statistics by Gold Domain

Low-grade Gold Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
LNGTH	3145	3.00	2.76			0.02	3.0	m
AU	3145	0.153	0.386	0.664	1.7	0.006	9.3	g/t
AUC	3145	0.153	0.384	0.640	1.7	0.006	7.6	g/t
CAUCN	229	0.460	0.760	0.921	1.2	0.058	7.9	g/t
CAUR	229	64.0	62.2	25.8	0.4	3.0	168.0	%

Mid-grade Gold Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
LNGTH	577	3.00	2.26			0.01	3.00	m
AU	577	2.233	2.820	2.052	0.728	0.100	14.050	g/t
AUC	577	2.232	2.820	2.052	0.728	0.100	14.050	g/t
CAUCN	174	1.751	2.250	1.882	0.836	0.080	10.956	g/t
CAUR	174	69.9	65.2	27.0	0.4	3.0	164.3	%

High-grade Gold Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
LNGTH	228	3.00	2.20			0.04	3.00	m
AU	228	8.510	13.292	17.732	1.334	1.075	164.000	g/t
AUC	228	8.510	12.288	12.291	1.000	1.075	90.000	g/t
CAUCN	66	5.401	13.417	31.217	2.327	0.200	196.500	g/t
CAUR	66	60.0	58.7	30.0	0.5	7.0	110.0	%

Outside Modeled Gold Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
LNGTH	1970	3.00	2.58			0.01	3.00	m
AU	1970	0.009	0.025	0.222	8.849	0.003	8.130	g/t
AUC	1970	0.008	0.020	0.079	4.015	0.003	2.000	g/t
CAUCN	None							g/t
CAUR	None							%

Codes: LNGTH length in meters
AU gold grade
AUC capped gold grade
CAUCN cyanide soluble gold grade
CAUR ratio of cyanide-soluble gold divided by total gold

Table 14.8 Descriptive Composite Statistics by Tungsten Domain

Low-grade Tungsten Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
LNGTH	1,129	3.00	2.51			0.01	3.00	m
W	1,129	4.7	8.3	32.5	3.9	0.1	992.5	ppm
WC	1,129	4.7	7.2	10.9	1.5	0.1	137.0	ppm

Mid-grade Tungsten Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
LNGTH	2,037	3.00	2.54			0.02	3.00	m
W	2,037	51.8	72.8	114.7	1.6	0.8	3083.3	ppm
WC	2,037	51.8	69.8	74.8	1.1	0.8	963.9	ppm

High-grade Tungsten Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
LNGTH	936	3.00	2.40			0.01	3.00	m
W	936	317.0	532.3	705.9	1.3	4.7	10000.5	ppm
WC	936	317.0	524.8	638.2	1.2	4.7	5500.0	ppm

Outside Modeled Tungsten Domains								
	Valid	Median	Mean	Std. Devn.	Co. of Variation	Minimum	Maximum	Units
LNGTH	2,018	3.00	2.59			0.01	3.00	m
W	2,018	1.1	6.3	48.2	7.7	0.1	1542.2	ppm
WC	2,018	1.1	3.3	10.7	3.2	0.1	100.0	ppm

Codes: LNGTH length in meters
W tungsten grade
WC capped tungsten grade

Correlograms were built from the composited gold grades in order to evaluate grade continuity. Correlogram parameters were used in the kriged estimate, which was used as a check on the reported inverse distance estimate, and also to give guidance to classification of resources. The correlogram results by unit and domain are summarized as follows:

OXIDE

Low-grade gold domain - The nugget is 30% of the total sill. The first sill is 80% of the total sill with a range of 10m to 15m depending on direction but little anisotropy. The remaining sill (20%) has a range of around 40m to 70m depending on direction, with the shortest being along strike.

Mid- and high-grade gold domains - The nugget is 5% of the total sill. The first sill is 85% of the total sill with a range of 5m to 25m depending on direction. The remaining sill (15%) has a range of around 30m to 55m depending on direction. The longest range is along dip; the shortest is perpendicular to bedding.

Low-grade tungsten domain - The nugget is 70% of the total sill. The single sill (30%) has a range of around 15m to 30m, with little longest range downdip.

Mid- and high-grade tungsten domains - The nugget is 80% of the total sill. The single sill is the remaining 20% of the total sill with a range of 5m to 20m with the longest range in the dip direction.

SULPHIDE

Low-grade gold domain - The nugget is 50% of the total sill. The first sill is 90% of the total sill with a range of 15m to 35m depending on direction. The remaining sill (10%) has a range of around 110m to 150m depending on direction, with the longest direction along strike.

Mid- and high-grade gold domains - The nugget is 60% of the total sill. The first sill is 85% of the total sill with a range of 10m to 70m depending on direction. The remaining sill (20%) has a range of around 45m to 105m depending on direction. The longest range is along strike; the shortest is along strike.

Low-grade tungsten domain - The nugget is 75% of the total sill. The first sill is 85% of the total sill with a range of 5m to 15m depending on direction. The remaining sill (15%) has a range of 25m to 38m, with little anisotropy evident.

Mid- and high-grade tungsten domains - The nugget is 80% of the total sill. The first sill is 95% of the total sill with a range of 10m to 14m depending on direction. The remaining sill (5%) has a range of around 80m to 100m depending on direction.

14.6 ESTIMATION

A polygonal estimate was done first in order to get a sense of reasonableness and those results were used to check that the global resources were reasonable at a cutoff of zero. Three types of block-model estimates were completed: nearest neighbor, inverse distance, and kriged, with the inverse-distance estimate being reported. The nearest neighbor, inverse distance and kriged estimates were run several times in order to determine sensitivity to estimation parameters, and to evaluate and optimize results. The inverse distance power was three ("ID^3") for all domains.

Four different estimation areas were defined to best match the search to the domain orientations (Table 14.9). Most of the longest search ranges were 100m, but a few needed to be pushed out to 200m to estimate a few distal blocks. Those 200m searches were then overwritten with 100m passes so that the Measured and Indicated blocks would have a more local search. The estimate was done respecting domains but also respecting rock units in that the oxide was estimated apart from all other units which were estimated together, but again still respecting domains. All estimates and estimation runs weighted the composites by the sample lengths. Estimation parameters for gold and tungsten are given in Table 14.10 and Table 14.11, respectively.

Table 14.9 Estimation Areas

Area	Azimuth	Dip	Rotation
1	345°	0°	-50°
2	345°	0°	-65°
3	345°	0°	-20°
4	345°	0°	-45°

Table 14.10 Estimation Parameters for Gold

Description	Parameter
Low-grade Gold Domain	
Composites: minimum/maximum/maximum per hole	1 / 9 / 2
Search orientation (see Table 14.9):	variable
Search distances: major/semimajor/minor (m): first pass; second pass	150 / 150 / 50 (oxide and non-oxide)
Inverse distance power	3
High-grade restrictions (grade in g Au/t and distance in m)	4g Au/t – 25m in oxide 4.5g Au/t – 25m in non-oxide
Mid-grade Gold Domain	
Composites: minimum/maximum/maximum per hole	1 / 9 / 2
Search orientation (see Table 14.9):	variable
Search distances: major/semimajor/minor (m): first pass; second pass	200 / 200 / 100 (non-oxide only) 100 / 100 / 33 (oxide and non-oxide)
Inverse distance power	3
High-grade restrictions (grade in g Au/t and distance in m)	None in oxide 9g Au/t – 50m in non-oxide
High-grade Gold Domain	
Composites: minimum/maximum/maximum per hole	1 / 9 / 2
Search orientation (see Table 14.9):	variable
Search distances: major/semimajor/minor (m): first pass; second pass	200 / 200 / 100 (non-oxide only) 100 / 100 / 33 (oxide and non-oxide)
Inverse distance power	3
High-grade restrictions (grade in g Au/t and distance in m)	45g Au/t – 50m in oxide None in non-oxide
Outside Modeled Gold Domains	
Composites: minimum/maximum/maximum per hole	2 / 9 / 2

table continues…

Search orientation (see Table 14.9):	variable
Search distances: major/semimajor/minor (m)	60 / 60 / 20
Inverse distance power	2
High-grade restrictions (grade in g Au/t and distance in m)	0.1g Au/t / 5m

Table 14.11 Estimation Parameters for Tungsten

Description	Parameter
Low-grade Tungsten Domain	
Composites: minimum/maximum/maximum per hole	1 / 9 / 2
Search orientation (see Table 14.9):	variable
Search distances: major/semimajor/minor (m): first pass; second pass	150 / 150 / 50 (oxide and non-oxide)
Inverse distance power	3
High-grade restrictions (grade in ppm W and distance in m)	12ppm W – 25m in oxide 25ppm W – 25m in non-oxide
Mid-grade Tungsten Domain	
Composites: minimum/maximum/maximum per hole	1 / 9 / 2
Search orientation (see Table 14.9):	variable
Search distances: major/semimajor/minor (m): first pass; second pass	200 / 200 / 100 (non-oxide only) 100 / 100 / 33 (oxide and non-oxide)
Inverse distance power	3
High-grade restrictions (grade in ppm W and distance in m)	500ppm W – 50m in oxide
High-grade Tungsten Domain	
Composites: minimum/maximum/maximum per hole	1 / 9 / 2
Search orientation (see Table 14.9):	variable
Search distances: major/semimajor/minor (m): first pass; second pass	200 / 200 / 100 (non-oxide only) 100 / 100 / 33 (oxide and non-oxide)
Inverse distance power	3
High-grade restrictions (grade in ppm W and distance in m)	None
Outside Modeled Tungsten Domains	
Composites: minimum/maximum/maximum per hole	2 / 9 / 2
Search orientation (see Table 14.9):	variable
Search distances: major/semimajor/minor (m)	60 / 60 / 20
Inverse distance power	3
High-grade restrictions (grade in ppm W and distance in m)	6ppm W / 5m

14.7 MINERAL RESOURCES

Mr. Ristorcelli reports resources at cutoffs that are reasonable for deposits of this nature given anticipated mining and processing methods and approximate operating costs, while also considering economic conditions, because of the regulatory requirements that a resource exists "*in such form and quantity and of such a grade or quality that it has reasonable prospects for eventual economic extraction*." Mr. Ristorcelli classified the Tiger resources giving consideration to the confidence in the underlying database, sample integrity, analytical precision/reliability, QA/QC results, and confidence in geologic interpretations. Material is predominantly classified as Measured and Indicated because the underlying database, sample integrity, analytical reliability as shown by QA/QC results are sufficient to classify much of the material at a higher level of confidence than Inferred. Essentially the exploration work and procedures, dense systematic drilling, and deposit predictability give high levels of confidence. The detractions, which, while minor, are sufficient to disallow Measured in the oxide and at depth, are the lack of systematic drying of oxide samples, poor core recovery in oxide in early drilling, and the limited down-hole surveys early in the exploration campaigns.

A sense of model reliability was obtained because the sectional modeling demonstrated reasonable continuity from section to section. All material is classified by the gold estimate; the tungsten is carried along with the gold. Blocks outside the domains are reported as part of the resource for those volumes that lie above the surface defining potential open-pit minable material. Blocks outside the domains are not reported as part of the resource for those volumes that lie below the surface defining potential open-pit minable material. Resource classification is presented in Table 14.12.

Table 14.12 Mineral Resource Classification

Measured	
Description	**Parameter**
Number of composites	7
Range to closest composite (m)	10
Other constraints	Not in oxide Above open-pit surface In underground grade shells
Indicated	
Number of composites used to estimate a block for the range to closest composite (m) (or average to all composites – "avg")	7 for 25m 3 for 20m 5 for 40m (avg)
Other constraints	Above open-pit surface In underground grade shells
Inferred in Domains	
Number of composites	1
Other constraints	Above open-pit surface In underground grade shells

Inferred outside Domains	
Number of composites	2
Range to closest composite (m)	15
Other constraints (note- there extreme search range restrictions on well mineralized composites)	Above open-pit surface only

All Resources tabulated in this Technical Report are based on the presumption that the most likely method of exploitation would be from open pit with some potential underground mining below the pit. Technical and economic factors likely to influence the "*reasonable prospects for eventual economic extraction*" were evaluated using the best judgement of Mr. Ristorcelli. Only gold grades were used for cutoff calculations but tungsten is reported and carried at the gold-only cutoff grades. For open pit mining, the reporting cutoff 0.75g Au/t was determined by using mining costs (US$5/ton), processing costs (US$29/ton), anticipated metallurgical recoveries (95% for gold), and appropriate G&A (US$20/ton) costs for similar size operations in northern Canada.

Potential for underground mining was assessed by determining underground mining costs for similar operations. A grade shell with grades above 1.4g Au/t was made. Isolated and discontinuous zones were eliminated, and then that solid was used to constrain the reported resources. For underground mining, the reporting cutoff 1.5g Au/t was determined by using mining costs (US$60/ton) for similar size operations in northern Canada. The cutoff grades are based on US$1,625/oz Au.

The Tiger reported mineral resources are the fully block-diluted estimates. The blocks are 5m by 5m by 5m.

Table 14.13, Table 14.14, Table 14.15 and Table 14.16 present tabulations of Measured, Indicated, Measured and Indicated and Inferred gold and tungsten Mineral Resources at Tiger, respectively. Table 14.17 presents a breakdown of resources by mining method. The bolded lines in the tables report the current Tiger resources. Those resources are all material that have "*reasonable prospects for eventual economic extraction*". Representative cross sections for the gold and tungsten block models are shown in Figure 14.7 and Figure 14.8, respectively.

Table 14.13 Tiger Measured Resources

Block-Diluted Gold & Tungsten, "Oxide", Measured

There are no oxide Measured Resources

Block-Diluted Gold & Tungsten, "Sulfide", Measured

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
0.50	968,000	2.57	80,000	169	164
0.75	833,000	2.88	77,000	172	143
1.00	725,000	3.18	74,000	166	120
1.25	650,000	3.40	71,000	165	107
1.50	586,000	3.66	69,000	164	96
Variable*	**828,000**	**2.89**	**77,000**	**171**	**142**
1.75	508,000	3.92	64,000	161	82
2.00	439,000	4.32	61,000	157	69
2.50	340,000	4.85	53,000	147	50
3.00	269,000	5.43	47,000	145	39
4.00	175,000	6.58	37,000	143	25
5.00	119,000	7.58	29,000	151	18

Block-Diluted Gold & Tungsten, Total Measured

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
0.50	968,000	2.57	80,000	169.0	164
0.75	833,000	2.88	77,000	172.0	143
1.00	725,000	3.18	74,000	166.0	120
1.25	650,000	3.40	71,000	165.0	107
1.50	586,000	3.66	69,000	164.0	96
Variable*	**828,000**	**2.89**	**77,000**	**171.0**	**142**
1.75	508,000	3.92	64,000	161.0	82
2.00	439,000	4.32	61,000	157.0	69
2.50	340,000	4.85	53,000	147.0	50
3.00	269,000	5.43	47,000	145.0	39
4.00	175,000	6.58	37,000	143.0	25
5.00	119,000	7.58	29,000	151.0	18

*Open pit cutoff 0.75g Au/t; UG cutoff 1.5g Au/t

Table 14.14 Tiger Indicated Resources

Block-Diluted Gold & Tungsten, "Oxide", Indicated

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
0.50	2,549,000	3.21	263,000	262	667
0.75	2,161,000	3.67	255,000	280	604
1.00	1,891,000	4.08	248,000	290	548
1.25	1,674,000	4.46	240,000	303	508
1.50	1,479,000	4.86	231,000	313	463
Variable*	**2,145,000**	**3.68**	**254,000**	**280**	**601**
1.75	1,312,000	5.29	223,000	322	423
2.00	1,184,000	5.65	215,000	332	393
2.50	954,000	6.46	198,000	344	328
3.00	786,000	7.28	184,000	358	281
4.00	572,000	8.65	159,000	371	212
5.00	434,000	10.03	140,000	385	167

Block-Diluted Gold & Tungsten, "Sulfide", Indicated

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
0.50	1,832,000	2.38	140,000	164	300
0.75	1,623,000	2.61	136,000	165	268
1.00	1,494,000	2.75	132,000	164	245
1.25	1,398,000	2.87	129,000	163	228
1.50	1,245,000	3.05	122,000	162	202
Variable*	**1,553,000**	**2.66**	**133,000**	**165**	**257**
1.75	1,038,000	3.33	111,000	159	165
2.00	874,000	3.59	101,000	152	133
2.50	598,000	4.21	81,000	151	90
3.00	418,000	4.91	66,000	148	62
4.00	230,000	6.09	45,000	152	35
5.00	146,000	7.03	33,000	151	22

Block-Diluted Gold & Tungsten, Total Indicated

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
0.50	4,381,000	2.86	403,000	221	967
0.75	3,784,000	3.21	391,000	230	872
1.00	3,385,000	3.49	380,000	234	793
1.25	3,072,000	3.74	369,000	240	736
1.50	2,724,000	4.03	353,000	244	665
Variable*	**3,698,000**	**3.26**	**387,000**	**232**	**858**
1.75	2,350,000	4.42	334,000	250	588
2.00	2,058,000	4.78	316,000	256	526

table continues...

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
2.50	1,552,000	5.59	279,000	269	418
3.00	1,204,000	6.46	250,000	285	343
4.00	802,000	7.91	204,000	308	247
5.00	580,000	9.28	173,000	326	189

*open pit cutoff 0.75g Au/t; UG cutoff 1.5g Au/t

Table 14.15 Tiger Measured and Indicated Resources

Block-Diluted Gold & Tungsten, "Oxide", Measured and Indicated

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
0.50	2,549,000	3.21	263,000	262	667
0.75	2,161,000	3.67	255,000	280	604
1.00	1,891,000	4.08	248,000	290	548
1.25	1,674,000	4.46	240,000	303	508
1.50	1,479,000	4.86	231,000	313	463
Variable*	**2,145,000**	**3.68**	**254,000**	**280**	**601**
1.75	1,312,000	5.29	223,000	322	423
2.00	1,184,000	5.65	215,000	332	393
2.50	954,000	6.46	198,000	344	328
3.00	786,000	7.28	184,000	358	281
4.00	572,000	8.65	159,000	371	212
5.00	434,000	10.03	140,000	385	167

Block-Diluted Gold & Tungsten, "Sulfide", Measured and Indicated

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
0.50	2,800,000	2.44	220,000	166	464
0.75	2,456,000	2.70	213,000	167	411
1.00	2,219,000	2.89	206,000	164	365
1.25	2,048,000	3.04	200,000	164	335
1.50	1,831,000	3.25	191,000	163	298
Variable*	**2,381,000**	**2.74**	**210,000**	**168**	**399**
1.75	1,546,000	3.52	175,000	160	247
2.00	1,313,000	3.84	162,000	154	202
2.50	938,000	4.44	134,000	149	140
3.00	687,000	5.12	113,000	147	101
4.00	405,000	6.30	82,000	148	60
5.00	265,000	7.28	62,000	151	40

Block-Diluted Gold & Tungsten, Total Measured and Indicated

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
0.50	5,349,000	2.81	483,000	211	1,131
0.75	4,617,000	3.15	468,000	220	1,015
1.00	4,110,000	3.44	454,000	222	913
1.25	3,722,000	3.68	440,000	226	843
1.50	3,310,000	3.97	422,000	230	761
Variable*	**4,526,000**	**3.19**	**464,000**	**221**	**1,000**
1.75	2,858,000	4.33	398,000	234	670
2.00	2,497,000	4.70	377,000	238	595
2.50	1,892,000	5.46	332,000	247	468
3.00	1,473,000	6.27	297,000	259	382
4.00	977,000	7.67	241,000	278	272
5.00	699,000	8.99	202,000	296	207

*open pit cutoff 0.75g Au/t; UG cutoff 1.5g Au/t

Table 14.16 Tiger Inferred Resources

Block-Diluted Gold & Tungsten, "Oxide", Inferred

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
0.50	76,000	2.13	5,200	132	10
0.75	64,000	2.43	5,000	125	8
1.00	59,000	2.58	4,900	119	7
1.25	53,000	2.70	4,600	113	6
1.50	48,000	2.33	3,600	125	6
Variable*	**61,000**	**2.04**	**4,000**	**131**	**8**
1.75	40,000	2.80	3,600	125	5
2.00	31,000	3.41	3,400	129	4
2.50	19,000	3.60	2,200	158	3
3.00	13,000	5.26	2,200	154	2
4.00	6,000	5.70	1,100	167	1
5.00	2,000	1.56	100	500	1

Block-Diluted Gold & Tungsten, "Sulfide", Inferred

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
0.50	146,000	1.92	9,000	75	11
0.75	124,000	2.13	8,500	89	11
1.00	123,000	2.15	8,500	89	11
1.25	122,000	2.17	8,500	90	11
1.50	102,000	2.29	7,500	98	10

table continues…

Variable*	**104,000**	**2.24**	**7,500**	**96**	**10**
1.75	71,000	2.41	5,500	127	9
2.00	48,000	2.85	4,400	146	7
2.50	20,000	3.58	2,300	150	3
3.00	14,000	5.11	2,300	214	3
4.00	8,000	4.28	1,100	125	1
5.00	4,000	7.78	1,000	-	-

Block-Diluted Gold & Tungsten, Total Inferred

Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
0.50	222,000	1.99	14,200	95	21
0.75	188,000	2.23	13,500	101	19
1.00	182,000	2.29	13,400	99	18
1.25	175,000	2.33	13,100	97	17
1.50	150,000	2.30	11,100	107	16
Variable*	**165,000**	**2.17**	**11,500**	**109**	**18**
1.75	111,000	2.55	9,100	126	14
2.00	79,000	3.07	7,800	139	11
2.50	39,000	3.59	4,500	154	6
3.00	27,000	5.18	4,500	185	5
4.00	14,000	4.89	2,200	143	2
5.00	6,000	5.70	1,100	167	1

*Open pit cutoff 0.75g Au/t; UG cutoff 1.5g Au/t

Table 14.17 Breakdown of Resources by Mining Method

Type	Classification	Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
			Open Pit				
Oxide	Indicated	0.75	1,980,000	3.74	238,000	282	559
Sulfide	Measured	0.75	799,000	2.92	75,000	171	137
Sulfide	Indicated	0.75	847,000	2.68	73,000	164	139
Ox + S	M+I	0.75	3,626,000	3.31	386,000	230	835
			Underground				
Oxide	Indicated	1.50	165,000	3.09	16,000	253	42
Sulfide	Measured	1.50	29,000	2.06	2,000	188	5
Sulfide	Indicated	1.50	706,000	2.64	60,000	167	118
Ox + S	M+I	1.50	900,000	2.70	78,000	183	165
			Open Pit + Underground				
Ox + S	M+I	Variable	4,526,000	3.19	464,000	221	1,000
			Open Pit				
Oxide	Inferred	0.75	20,000	1.54	1,000	139	3
Sulfide	Inferred	0.75	7,000	2.41	500	123	1
Ox + S	Inferred	0.75	27,000	1.73	1,500	*135	4
			Underground				
Oxide	Inferred	1.50	41,000	2.62	3,000	112	5
Sulfide	Inferred	1.50	97,000	2.26	7,000	94	9
Ox + S	Inferred	1.50	138,000	*2.37	10,000	101	14
			Open Pit + Underground				
Ox + S	Inferred	Variable	165,000	2.17	11,500	109	18

* These two grades are calculated slightly differently than all other grades in the resource tables. They are calculated based on full-precision tonnages, whereas all other grades are based on rounded tonnages. This was done to remove what looked like an inconsistency resulting from rounding small tonnages.

Figure 14.7 Tiger Gold Domains, Geology and Block Model– Section 10200

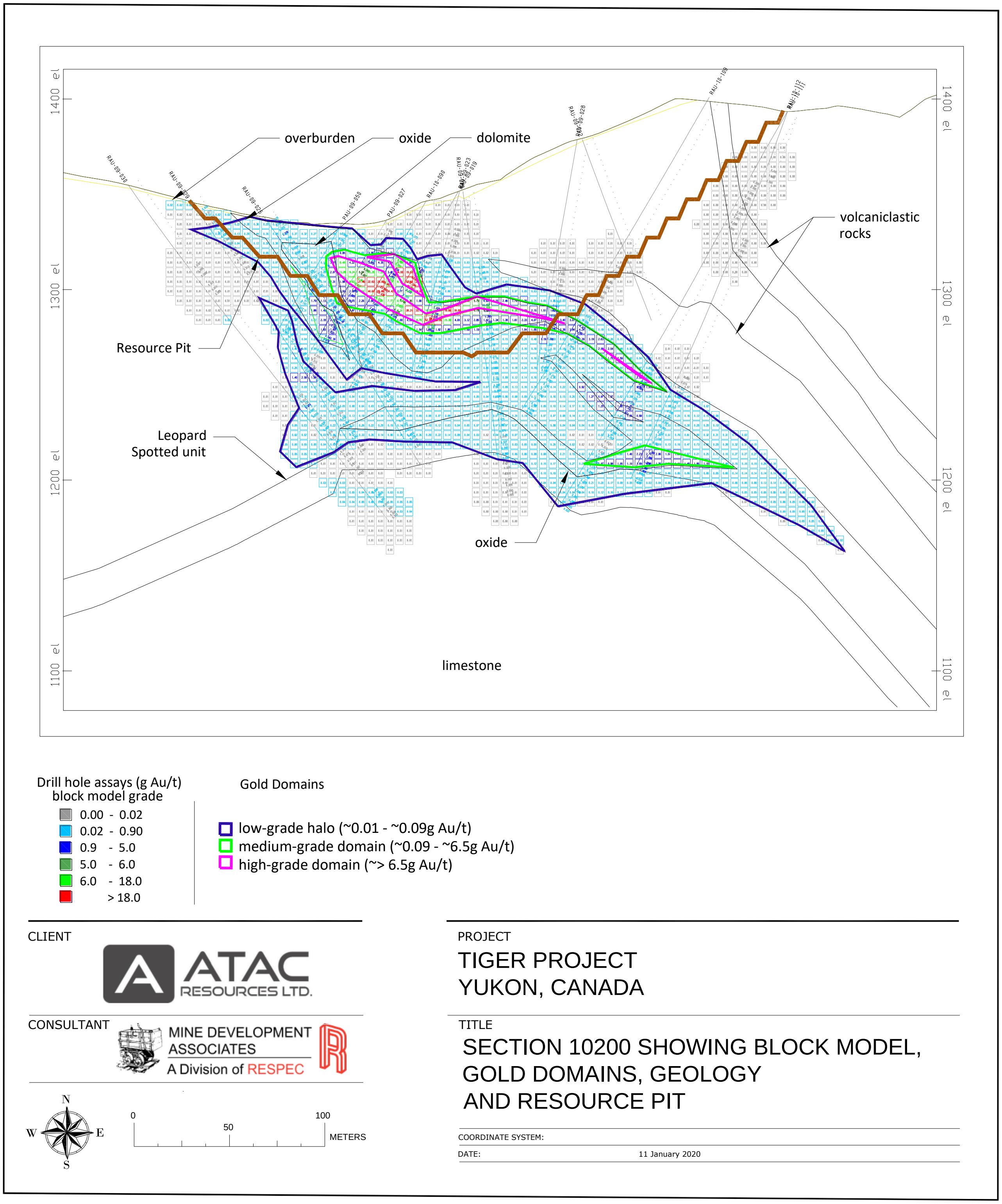
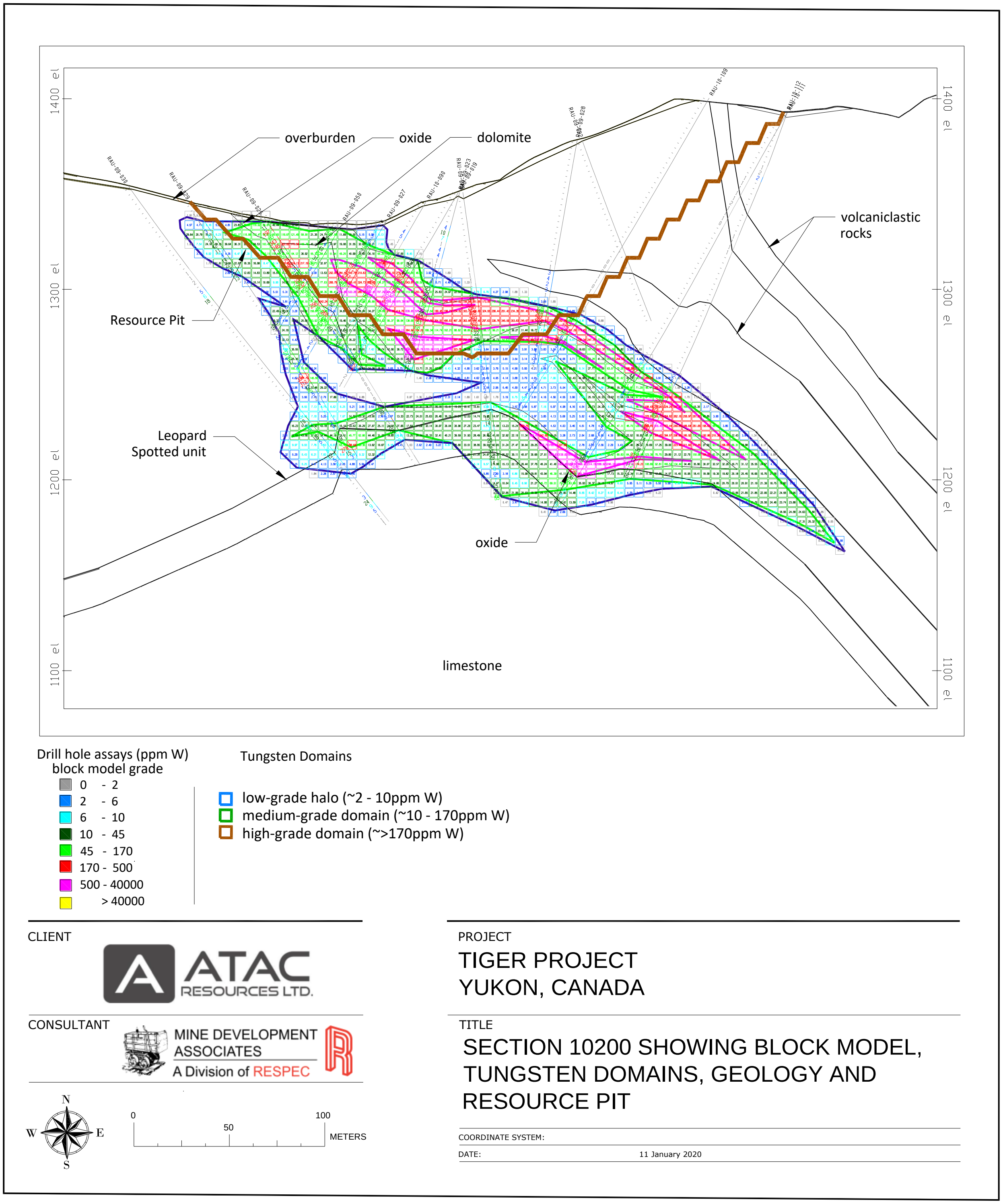


Figure 14.8 Tiger Tungsten Domains, Geology and Block Model – Section 10200



14.8 DISCUSSION OF RESOURCES

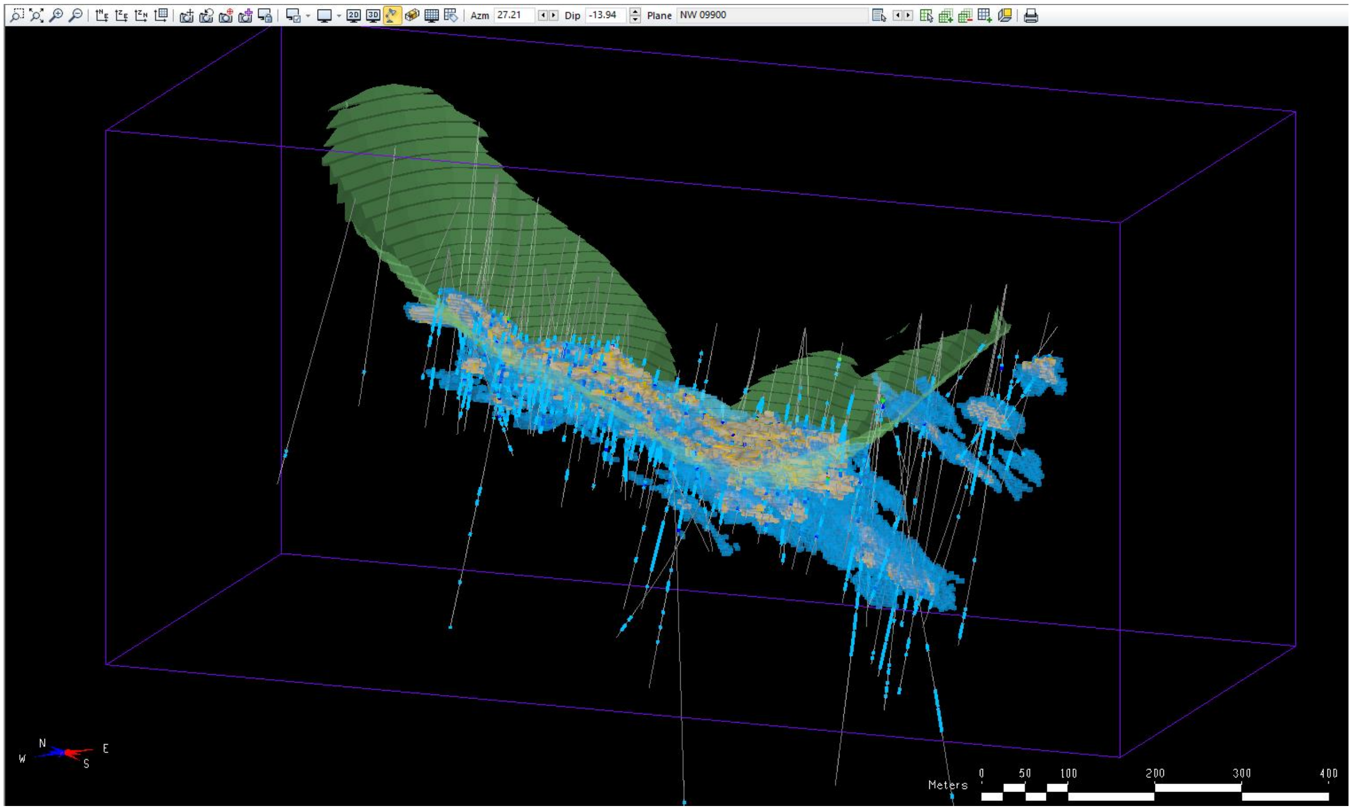
ATAC has shown that important gold resources, with some accompanying potentially economically viable tungsten, exist at Tiger. MDA finds that the risks at Tiger are minimal for several reasons. First, the drilling density is tight. Second, while modeling domains section to section, the predictability of the mineralization was found to be good and in some cases very good.

The risks include core recovery from drilling in the oxide early in the exploration of Tiger, density measurements in the oxide, and having only one standard with a certified tungsten value. The low core recovery does not seem to have an impact on sample integrity as there seems to be no bias, nevertheless, because of this and issues with density measurements MDA did not classify any oxide material as Measured. The main issue with the density measurements is that the samples measured were allowed to only air dry for unspecified time intervals before density measurements were done, and with clays, drying may not have been complete. To account for some potential remnant moisture, MDA dropped the assigned density by 4% from the average measurement of 2.4g/cm3 to 2.3g/cm^3. While there was only one standard with a certified tungsten grade, the tungsten is not a major component of the resource so this lack of certified standards for tungsten is not considered material.

Regulatory guidelines require that resources meet "*reasonable prospects for eventual economic extraction*", so it is often the case that not all the mineralization is reported as part of the resource. Figure 14.9 shows the mineralization estimated throughout the deposit and the relationship it has with the reporting pit surface and underground solids. It is clear that the reported resource is a small part of the total mineralization.

The most likely scenario for exploitation will be by open pit but underground methods may also be applicable for material under the open-pit surface, so this Technical Report is reporting both resources that would meet the requirement of "*reasonable prospects for eventual economic extraction*" by open pit, and resources below the reporting pit surface that could potentially be exploited by underground mining. A large volume of mineralization exists that would, if deemed more reasonable, be potentially exploitable by underground mining methods if metal prices were to increase (Figure 14.9).

Figure 14.9 Confining Volumes and Additional Mineralization



Note: Looking northeast at -15o; scale bar in meters in bottom right; blue is the 0.5g Au/t grade shell; orange is the 1.4 Au/t grade shell)

15.0 MINERAL RESERVE ESTIMATES

A Mineral Reserve has not been estimated for the Project as part of this PEA.

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource.

16.0 MINING METHODS

A PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that PEA results will be realized.

16.1 INTRODUCTION

This section outlines the input data, procedures, and results of a PEA-level pit optimization, design, mine schedule, mine equipment, and labor requirements for a nominal process capacity of approximately 1,500 t/d.

16.2 PIT OPTIMIZATION

Tetra Tech completed open pit optimizations and mine production scheduling using GEOVIA Whittle™ software, which is based on the Lerchs-Grossmann algorithm. Tetra Tech prepared pit optimization parameters using inputs from other engineering consultants retained by ATAC and experience from other projects.

16.2.1 BLOCK MODEL

MDA provided Tetra Tech with a 5 m by 5 m by 5 m block model in a .csv format. This block model forms the basis of this mining study.

16.2.2 PIT SLOPE ANGLE

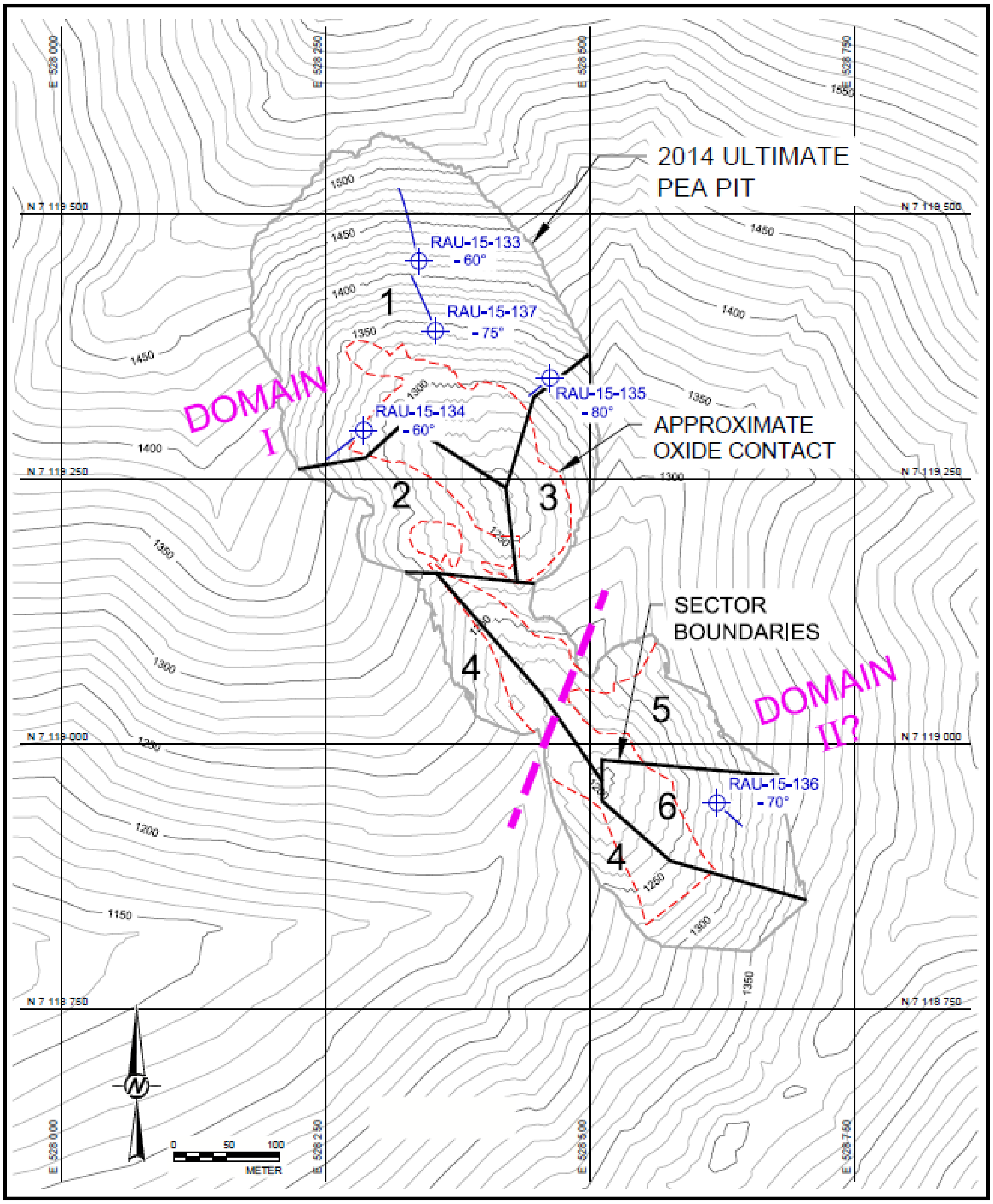
Golder Associates Inc. (Golder) completed a scoping-level pit slope evaluation report (Golder 2016) in March of 2016. Table 16.1 lists the scoping-level pit slope design recommendations as proposed by Golder and Figure 16.1 shows the sector numbers described in Table 16.1.

Table 16.1 Recommended Design Inter-ramp Slope Angles and Bench Configurations by Sector

Sector No.	Slope Dip Direction Range (°)	Major Rock Types	Vertical Separation Between Stacked 5-m High Benches (m)	Minimum Catch Bench Width (m)	Blasting Method	Bench Face Angle (°)	Design Inter-ramp Slope Angle (°)
All	All	Oxide <30 m High	10 (double benching)	6.5	Buffer if required*	63	41
		Oxide >30 m High	10 (double benching)	8.0	Buffer if required*	63	37
1, 3, 4, 5	All except 25-90 clockwise and 265-295 clockwise	Carbonates, Volcaniclastic Rocks	15 (triple benching)	7.5	Pre-splitting	71	50
2	25-90 clockwise	Carbonates	15 (triple benching)	7.5	Trim Blasting	50	37
6	265-295 clockwise	Carbonates, Volcaniclastic Rocks	15 (triple benching)	7.5	Trim Blasting	57	41

Note: *Otherwise trim with dozer or excavator
Source: Golder (2016)

Figure 16.1 2014 PEA Pit Design with 2015 Oriented Geotechnical Core Holes and Sector boundaries



Source: Golder (2016)

16.2.3 SURFACE TOPOGRAPHY

Digital topographical drawings of the Property were provided by ATAC.

16.2.4 PIT OPTIMIZATION PARAMETERS

Table 16.2 lists the pit optimization parameters.

Table 16.2 Pit Optimization Parameters

Items	Units	Value
Exchange Rate	USD:CAD	0.77:1.00
Discount Rate	%	5
Production Rate		
Daily Processing Capacity	t/d	1,500
Working Days	d/a	365
Yearly Processing Capacity	t/a	547,500
Metal Price (Market)		
Gold	USD/troy oz	1,400
Process		
Method	-	Tank Leaching
Recovery		
Oxide	%	90.5
Limestone	%	63
Dolomite	%	1.074 x CAURI*
Volcanics	%	63
Leopard Spotted Unit	%	63
Off-site Costs		
Refining Cost – Au Dore	USD/troy oz	1.00
Percent Payment	%	99.50
Transportation (insurance and security included) – Au Doré	USD/troy oz	5.00
Private Royalty	%	0.00
Operating Cost		
Mining		
Oxide	CAD/t mined	4.50
Non-oxide	CAD/t mined	5.00
Processing, G&A, Surface Services and Tailing		
G&A and Surface Services	CAD/t processed	18.00
Processing	CAD/t processed	29.00
Tailing	CAD/t processed	1.70
Total Processing, G&A, Surface Services and Tailing	CAD/t processed	48.70
Block Model		
Block Model	m	5 x 5 x 5
Gold Grade	g/t	Variable

table continues…

Items	Units	Value
Density		
Oxide Mineralization	t/m^3	Based on Block Model
Sulphide Mineralization	t/m^3	Based on Block Model
Volcanics	t/m^3	Based on Block Model
Waste Outside Mineralized Solids	t/m^3	Based on Block Model
Overburden	t/m^3	Based on Block Model
Mining Technical Assumptions		
Mining Recovery	%	95
Mining Dilution	%	5

*CAURI is an attribute in the block model and is Cyanide soluble estimated gold recovery

16.2.5 Pit Optimization Results

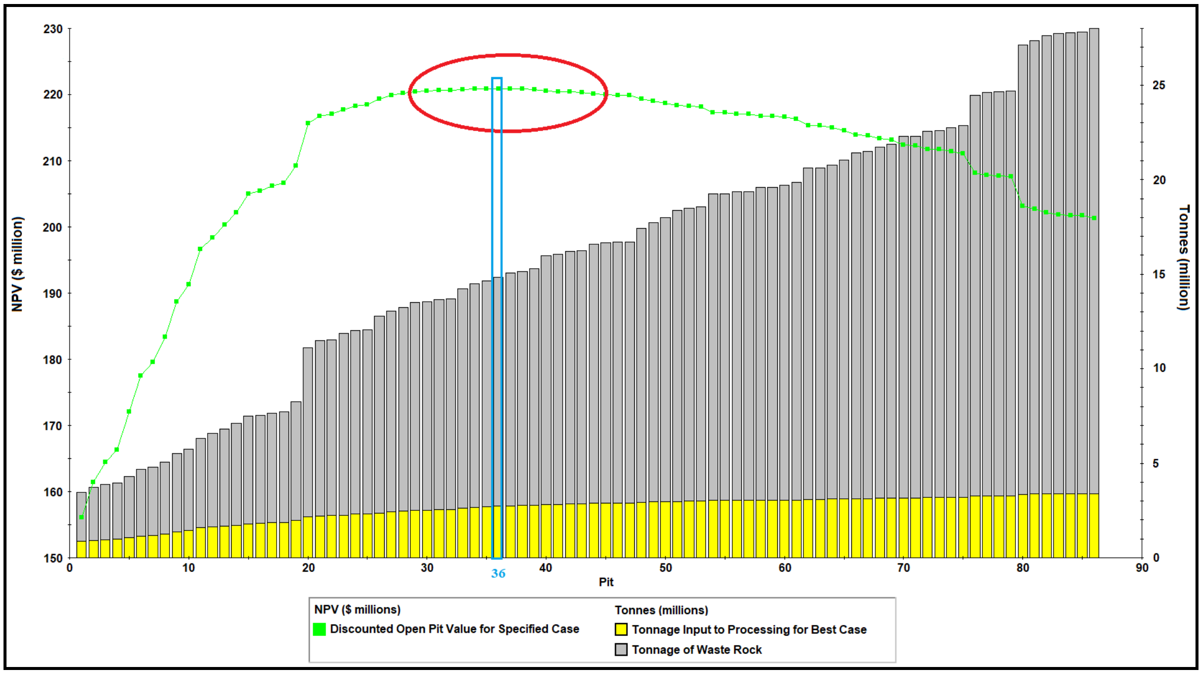
The open pit limits were determined by considering the physical and economic constraints to mining using Whittle 4X pit limit optimization software. The Whittle 4X software uses the industry-standard Lerchs-Grossman algorithm to define a three-dimensional ("3D") shape for the open pit which is considered the "optimal" economic shell for mining.

The terminology "Pit limit optimization" refers to a process which aims to identify the highest value mining pit shape for a given series of inputs and constraints. It does not imply that mining has been "optimized" in other ways, such as equipment optimization, or labor optimization, or grade optimization for example.

A key outcome of the economic pit limit optimization generation is a series of 3D surfaces or "nested pit shells" based on a range of metal selling prices. The metal price sensitivity analysis is conducted by applying a "Revenue Factor" ("RF") to the base case metal price. In this study, price factors ranged between 0.3 to 2.0.

Pit optimizations were completed using the Measured, Indicated, and Inferred oxide and sulphide Mineral Resources. No capital costs were considered when generating these discounted values. The pit optimization results are provided in Figure 16.2. Since pits with revenue factor in the range of 0.96 to 1.0 are very close in NPV value (as highlighted in the figure), in order to maximize the contained gold, pit 36 with revenue factor of 1 was selected as the final pit for further detailed design and production scheduling.

Figure 16.2 Discounted Cash Flow Analysis by Revenue Factor Pit Shell



16.3 Mine Design

16.3.1 Bench Height and Pit Wall Slope

The final pit was designed based on the geotechnical parameters provided in Table 16.1.

16.3.2 Minimum Working Area

Benches were designed to accommodate a 3.8-m^3 excavator and a 39.5-t articulated truck.

16.3.3 Haul Road

Main haul roads for the pit area were designed to accommodate a 39.5-t articulated truck, with two-way traffic on most of the haulage roads, and one-way traffic for the last two to three benches at the pit bottom. In-pit ramps were designed with a maximum grade of 10%. The widths of the one-way and two-way traffic were set at 10 m and 16 m, respectively.

16.3.4 Pit Hydrology/Dewatering

No detailed pit hydrology/dewatering is included in this PEA; however, an allowance is included in the mining operating cost to account for pit dewatering costs.

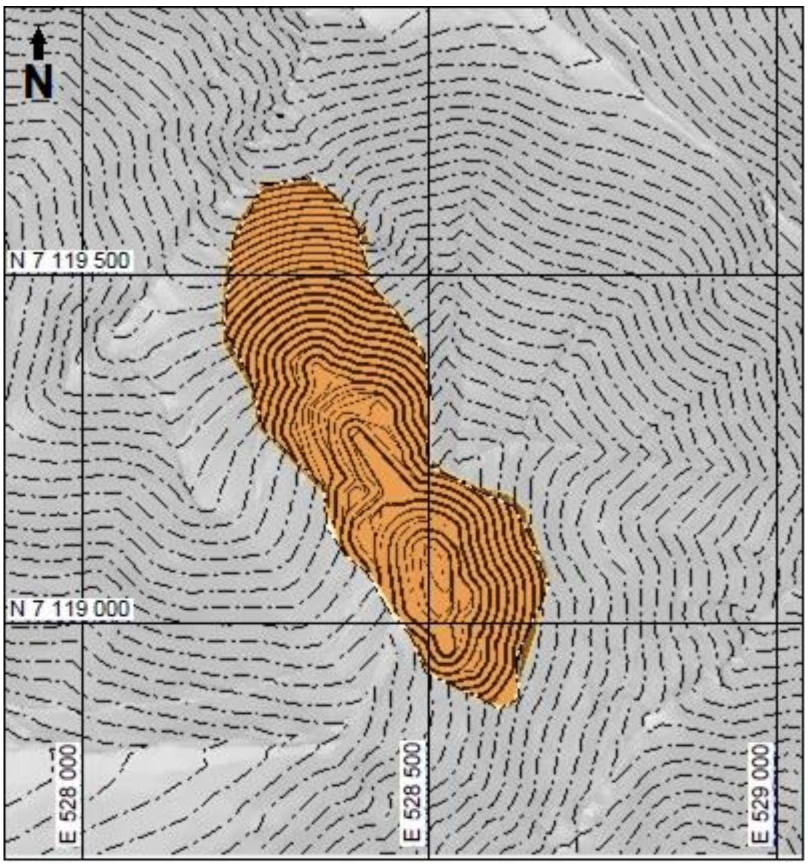
16.3.5 PIT DESIGN RESULTS

The final designed pit includes 1.66 Mt of oxide Mineral Resource and 1.05 Mt of sulphide Mineral Resource, with a LOM strip ratio of 5.3. A material summary for the final pit is provided in Table 16.3. Figure 16.3 shows a general plan view of the final pit.

Table 16.3 Pit Design Results

Material	Tonnage (kt)	Au (g/t)
Mineralized Oxide to Mill	1,658	4.0
Mineralized Sulphide to Mill	1,052	3.5
Waste (Rock and Low Grade Oxide/Sulphide)	14,378	-
Total Material Mined	**17,088**	-

Figure 16.3 General Plan View of the Final Pit



16.4 PRODUCTION SCHEDULE

The mining schedule was developed based on a nominal processing capacity of 1,500 t/d for 365 d/a. Oxide and sulfide material above the economic cut-off will be scheduled for processing. Oxide and sulfide material below the economic cut-off will be handled as waste.

Relatively high-grade material will be sent directly to the primary crusher, located southwest of the pit close to the primary crusher. Waste material will be stored in two waste dumps located at the northwest and southwest side of the pit.

The Project's total LOM is approximately seven years, including one year of pre-stripping followed by six years of mill production. The production schedule is shown in Table 16.4 and Figure 16.4. Over the seven-year mine life, the pit will produce 2.7 Mt of mineralized material and 14.4 Mt of waste rock. The LOM average gold grade of oxide and sulfide material is 4.0 g/t and 3.5 g/t, respectively. The LOM stripping ratio (defined as waste material mined divided by mineralized material mined) is 5.3. Figure 16.3 shows the final pit at the end of mine life.

Figure 16.4 Production Schedule

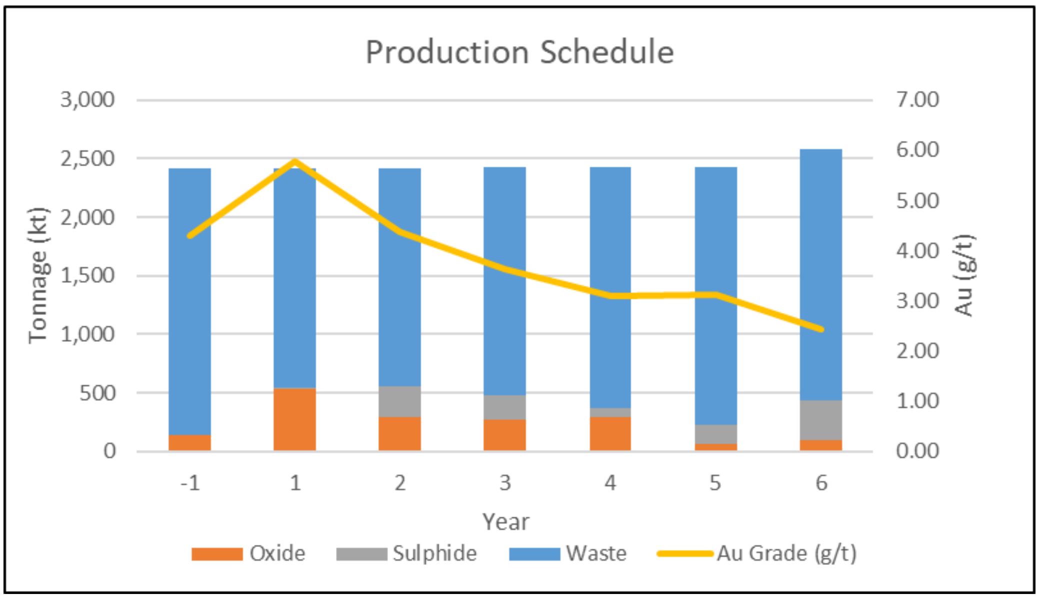


Table 16.4 Production Schedule

Production and Processing Schedule									
Production Year			**-1**	**1**	**2**	**3**	**4**	**5**	**6**
Project Year			**1**	**2**	**3**	**4**	**5**	**6**	**7**
Production Schedule	**Unit**								
Oxide Mined	000 t	1,658	133	528	289	270	289	57	91
Sulphide Mined	000 t	1,052	0	20	258	204	71	158	341
Total Ore Mined	000 t	**2,711**	**133**	**548**	**548**	**475**	**360**	**216**	**432**
Waste Mined	000 t	14,378	2,286	1,872	1,872	1,944	2,059	2,203	2,142
Total Material Mined	**000 t**	**17,088**	**2,419**	**2,419**	**2,419**	**2,419**	**2,419**	**2,419**	**2,574**
Strip Ratio	**:**	**5.3**	**0.0**	**3.4**	**3.4**	**4.1**	**5.7**	**10.2**	**5.0**
Ore to Stockpile	000 t	110	133	110	0	0	0	0	0
Stockpile to Mill	000 t	243	0	0	0	73	170	0	0
Stockpile Balance	**000 t**	**109**	**133**	**243**	**243**	**170**	**0**	**0**	**0**
Processing Schedule	**Unit**								
Oxide Ore Milled	000 t	1,658	0	418	289	343	459	57	91
Sulphide Ore Milled	000 t	1,052	0	20	258	204	71	158	341
Total Ore Milled	**000 t**	**2,711**	**0**	**438**	**548**	**548**	**530**	**216**	**432**
Gold Grade	g/t Au	3.8	4.3	5.8	4.4	3.6	3.1	3.1	2.4
Gold Milled	**000 oz**	**331**	**0**	**82**	**77**	**64**	**53**	**22**	**34**
Tungsten Grade	ppm	231.53	0.00	258.60	294.33	269.20	151.54	210.93	204.55
Recovery	%	77.4%	0.0%	89.3%	80.9%	79.4%	86.9%	65.0%	62.9%
Gold Produced	**000 oz**	**267.1**	**0.0**	**72.9**	**62.3**	**50.6**	**46.0**	**14.1**	**21.2**

16.5 Mine Waste Rock and Stockpile Management

Please refer to Section 18.4 of this report.

16.6 Mining Equipment

16.6.1 Mine Equipment Fleet

Mining operations will be performed using leased mining equipment. Small mining equipment with operating flexibility was selected to match the pit production schedule and the nature of site. The equipment selection, sizing, and fleet requirements were based on anticipated site operating conditions, haulage profiles, cycle times, and overall equipment utilization. In determining the number of units for major equipment such as drills, excavators, and trucks, annual operating hours were calculated and compared to the available hours for the equipment. Mine support equipment, such as track dozers, motor graders, water/sanding trucks, etc., was matched with the major mining equipment.

16.6.2 Operating Hours

Mine operations are assumed to operate 365 d/a, with 2 shifts/d and 12 h/shift. As shown in Table 16.5, the expected delay per shift is 3 hours.

Table 16.5 Operational Delay per Shift

Delay	Time (min)
Weather	50
Breaks	60
Shift Change	15
Blasting	30
Fuel, Equipment Moves, Others	25
Total	**180**

16.6.3 Primary Equipment

Loading will be performed using the 3.82-m^3 hydraulic excavator, and hauling will be performed using 39.5-t articulated trucks. Haul truck cycle times were estimated using the Caterpillar® Fleet Production and Cost software. Estimated travel times are provided in Table 16.6.

Table 16.6 Haulage Cycle Times

Production Year	Crusher Stockpile (min)	Northwest Waste Dump (min)	Southwest Waste Dump Tailing Facility (min)
-1	11	-	18
1	16	14	15
2	15	17	12
3	15	17	12
4	15	-	10
5	10	-	8
6	12	-	10

Blasthole drilling will be performed using 4.5-inch percussion crawler drills. Blasting will be performed using ammonium nitrate/fuel oil (ANFO) and emulsion with mix proportions of 0.7 and 0.3, respectively. Based on Golder (2016), no blasting will be required for the oxide material; excavation will be performed directly by the hydraulic excavator.

The primary equipment requirements for the LOM are summarized in Table 16.7.

Table 16.7 Primary Equipment Requirements

Production Year	Diesel Drill 4.5"	Hydraulic Excavator 3.82 m^3	Articulated Trucks 39.5 t
-1	1	1	4
1	1	1	5
2	1	1	5
3	1	1	5
4	1	1	3
5	1	1	2
6	1	1	2

16.6.4 Support and Ancillary Equipment

Selection of the support and ancillary equipment considers the size and type of the main fleet for loading and hauling, the geometry and size of the pit, and the number of roads and waste dumps that will operate at the same time. It reflects experience at operations of similar size and considers the specific characteristics of the Project. The LOM support and ancillary equipment requirements are listed in Table 16.8

Table 16.8 Support and Ancillary Equipment Requirements

Equipment	Maximum Fleet Size
Track Dozer 9.8 feet (2.9 m)	2
Wheel Dozer 12 feet (3.6 m)	1
Grader 12 feet (3.6 m)	1
Water/Sanding Tanker	1
Service Loader	1
Secondary Drill	1
Vibratory Compactor	1
Excavator	1
Flatbed Truck	1
Mechanics Service Truck	1
Pickup Truck	3
Mobile Crane	1
Rough Terrain Forklift	1
Shop Forklift	1
Light Plant	8
Mobile Radios	100
Safety Equipment	1
Engineering/Geology Equipment	1
Maintenance Management System	1
Surveying	1

16.7 MINING LABOUR

Mining labour requirements were estimated based on 12-hour shifts, 2 shifts/d, and a 2-week-on/2-week-off rotation schedule. Mine operator and maintenance staff requirements are estimated based on the scheduled hours. Salaried mine staff numbers were estimated from experience, historic data, and anticipated operating conditions for the Project.

The average ratio of maintenance labour complement to operator labour complement was estimated at 0.6:1. The maintenance labour estimate is based on historical ratios between equipment operators and maintenance mechanics and electricians.

A benefit package of 35% was applied to salaried staff, and 45% to the hourly labour base rates. The labour burden consisted of vacation, statutory holidays, medical and health insurance, employment insurance, long-term disability insurance, overtime, shift differential, and other factors.

Table 16.9 shows the maximum salaried staff requirements during the LOM. The hourly mining operator and maintenance labour on payroll is shown in Table 16.10.

Table 16.9 LOM Maximum Salaried Staff Requirement

Position	Maximum Number of Employees
Technical Services Staff	8
Operations Staff	5
Total	**13**

Table 16.10 Operator and Maintenance Staff on Payroll

Production Year	Operators	Maintenance
-1	37	22
1	41	23
2	42	23
3	41	23
4	33	20
5	25	17
6	24	17

17.0 RECOVERY METHODS

17.1 MINERAL PROCESSING

17.1.1 INTRODUCTION

The proposed processing plant will treat the gold mineralization from the Tiger Deposit at an average process rate of 1,500 t/d. The processing plant will co-process two distinct types of mineralization: oxide mineralization and sulphide mineralization.

17.1.2 SUMMARY

A conventional CIP cyanidation process is proposed for the Project. The processing plant will comprise crushing, grinding, a CIP cyanidation process, and gold recovery from the loaded carbon to produce gold doré. The processing plant will consist of four separate facilities:

- a crushing facility, including mill feed receiving pad, primary crushing by a mineral sizer, and related material handling facility
- a crushed material surge bin and related feeding and reclaim systems
- a main processing facility, including grinding, leaching feed thickening, cyanide leaching, loaded carbon acid wash, elution and carbon reactivation, gold electrowinning, and smelting
- residual cyanide destruction of the leach residue.

A mineral sizer in the crushing circuit will reduce the ROM material to a particle size of approximately 80% passing 120 mm.

The crushed material will be transported by a conveyor to a 1,500-t surge bin and then reclaimed and fed to a primary grinding circuit consisting of a SAG mill and a ball mill in closed circuit with hydrocyclones. The grinding circuit will further reduce the crushed mill feed to a particle size of 80% passing 75 µm.

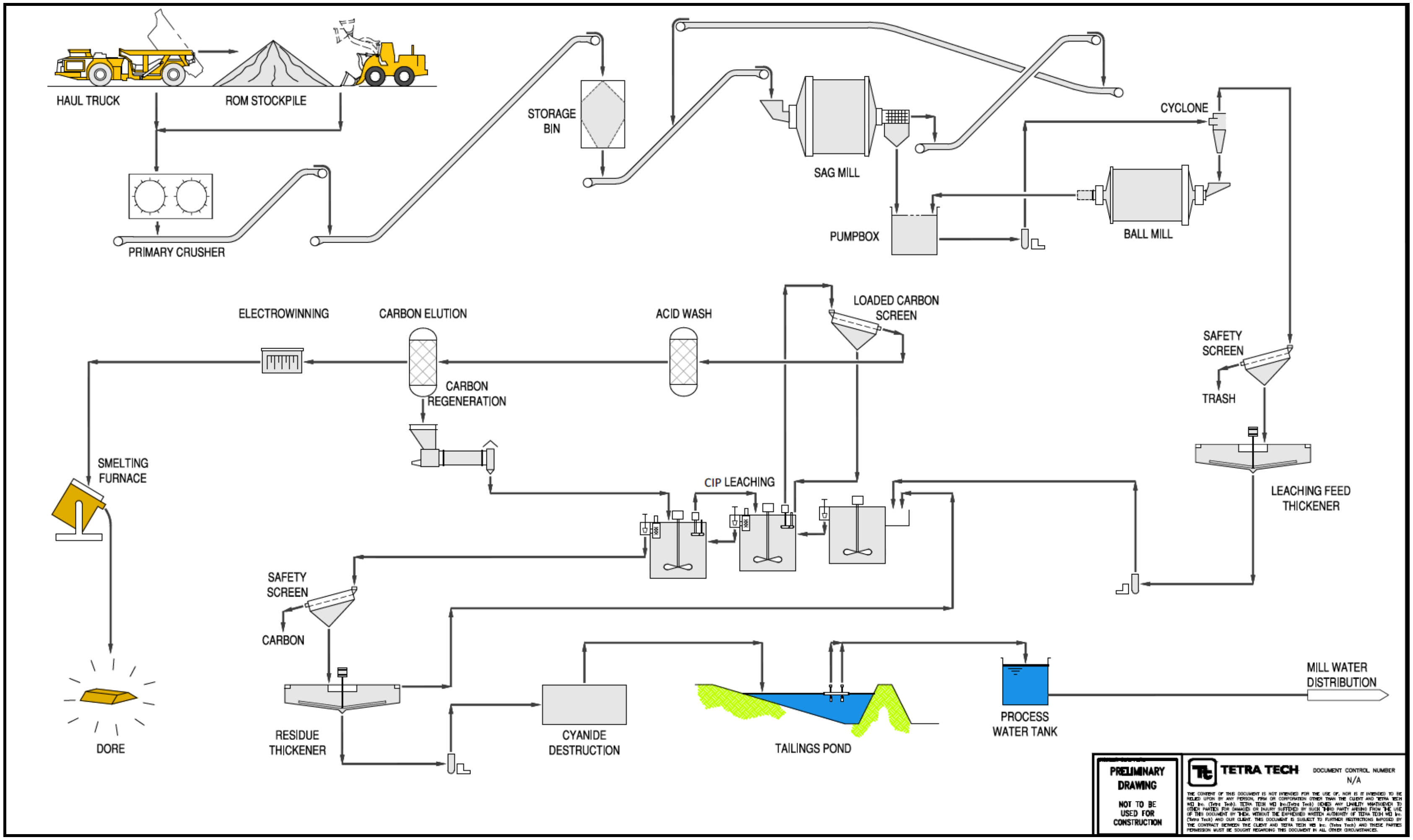
The hydrocyclone overflow from the primary grinding circuit will flow by gravity to a thickener where the slurry will be thickened for downstream cyanidation. The underflow of the thickener will be diluted with process water to the optimum solid density and be cyanide leached in a CIP circuit to recover the gold from the mineralization.

The loaded carbon from the CIP circuit will be washed by diluted acid solution and eluted by a conventional Zadra pressure stripping process. The gold in the pregnant solution will be recovered by electrowinning. The barren solution from the elution circuit will circulate back to the leach/elution circuit. The gold sludge produced from the electrowinning circuit will be smelted to produce gold doré bullion.

The residue from the leach circuit will be thickened to recover the leach solution for reuse as process water in the cyanidation circuit. The thickener underflow will be sent to a cyanide destruction circuit employing a sulphur dioxide/air process to destroy the residual WAD cyanide. The treated residue slurry will flow by gravity to the lined TMF for storage.

These processes are shown in the simplified flowsheet in Figure 17.1 and are detailed in the following sections.

Figure 17.1 Simplified Process Flowsheet



17.2 PLANT DESIGN

A conventional agitated cyanide leaching treatment after the mill feed is ground to 80% passing 75 µm, is proposed for this study.

17.2.1 MAJOR PROCESS DESIGN CRITERIA

The nominal throughput of the processing plant is designed to be 547,500 t/a or 1,500 t/d. The major criteria used in the design are shown in Table 17.1.

Table 17.1 Major Design Criteria

Criteria	Unit	Value
General		
Daily Process Rate	t/d	1,500
Operating Days	d/a	365
Overall Gold Recovery, Oxides	%	90.5
Overall Gold Recovery, Sulphides	%	60.8
Gold Production, Average	oz/a	~44,500
Ore Characteristics		
Head Gold Grade , Average	g/t Au	3.82
Specific Gravity - Oxide	-	2.4
- Sulphide	-	3.4
Primary Crushing		
Availability – Primary Crushing	%	75
Crushing Process Rate	t/h	200
Primary Crushing Product Particle Size, 80% passing	mm	120
Grind/Leach		
Availability	%	92
Nominal Milling Process Rate	t/h	68
Mill Feed Size, 80% passing	µm	120,000
Primary Grind Size, 80% passing	µm	75
Bond Ball Mill Work Index – Design	kWh/t	10.0
Leach Method	-	CIP
Feed Mass to CIP Circuit	t/h	68

17.3 PROCESS PLANT DESCRIPTION

17.3.1 PRIMARY CRUSHING

The crushing facility will have an average process rate of 200 t/h. Since the mill feed is anticipated to be soft with a significant amount of fines, a mineral sizer is proposed for the primary crushing.

The ROM materials will be trucked from the proposed open pit to the plant site and stockpiled in the ROM receiving pad. The stockpiled materials will be reclaimed by a loader to a loading hopper and conveyed to the sizer. The material will be reduced to 80% passing approximately 120 mm by the sizer.

The primary crusher product will be conveyed to the SAG feed surge bin. The crushing circuit will be operated during the day shift only.

The primary crushing area will be equipped with a dust control system to mitigate fugitive dust generation during unloading, crushing, and loading.

17.3.2 Mill Feed Surge Bin

The crushed materials will be fed to a SAG mill feed surge bin having a live capacity of 1,500 t. The crushed material will be reclaimed from the bin by a belt feeder at a nominal rate of 68 t/h onto a belt conveyor to feed the primary grinding circuit. A dust control system will be installed in the area to mitigate fugitive dust generation.

17.3.3 Primary Grinding, Classification

A SAG mill/ball mill circuit is proposed for primary grinding. The primary grinding circuit will consist of a SAG mill and a ball mill in a closed circuit with classifying hydrocyclones. Grinding will be conducted as a wet process at a nominal rate of 68 t/h.

The grinding circuit will include:

- one SAG mill, 4.27 m diameter by 2.59 m long (14 ft by 8.5 ft) (effective grinding length), driven by a 470 kW variable frequency drive
- one ball mill, 3.35 m diameter by 4.42 m long (11 ft by 14.5 ft) (effective grinding length), powered by a 665 kW fixed speed drive
- two hydrocyclone feed slurry pumps
- two 200 mm hydrocyclones
- one particle size analyzer.

The crushed material from the surge bin will be reclaimed onto a belt conveyor that feeds the crushed material to the SAG mill. The SAG mill will be equipped with 40 mm pebble ports to discharge the fine fraction from the SAG mill. The SAG mill discharge will be classified by a trommel screen that is integrated with the SAG mill. The trommel screen will have an opening of 9.5 mm (slot wide). The oversize from the trommel screen will be transported by belt conveyors back to the SAG mill feed conveyor. The screen undersize will discharge by gravity to the hydrocyclone feed pump box in the grinding circuit. Provisions have been made to provide sufficient space in the grinding area to accommodate a pebble crushing circuit if required at a later date.

The ball mill will be operated in closed circuit with hydrocyclones. The hydrocyclone underflow will flow by gravity to the ball mill feed chute and the ball mill discharge will be combined with the SAG mill trommel screen undersize slurry and pumped to the hydrocyclones for classification. The circulating load to the ball mill will be approximately 200 to 300%. The particle size of the hydrocyclone overflow, or the final product of the primary grind circuit, will be 80% passing 75 µm. The pulp density of the hydrocyclone overflow slurry will be approximately 30% w/w solids.

Grinding media will be manually added into the mills on a batch basis. Dilution water will be added to the grinding circuit as required and lime slurry will be added to the mill to adjust the slurry pH. A particle size analyzer will be installed to monitor and optimize the operating efficiency of the grinding circuit.

17.3.4 Cyanide Leaching and Carbon Adsorption

The hydrocyclone overflow will be screened to remove any oversize material and the trash screen undersize will flow by gravity to the leach feed thickener for the optimum solid density control for the downstream cyanidation. The thickener overflow will be pumped to the grinding circuit for reuse.

The thickener underflow with a solids density of 55 to 60% w/w will be pumped to the head of a bank of cyanide leach tanks. The overflow of the leach residue thickener will be used in the grinding circuit. The leach residue thickener overflow and process make-up water will be added to dilute the cyanide leach feed to a solid density of approximately 45% w/w. Cyanidation will be performed in a CIP circuit consisting of six leach tanks and five CIP tanks. Each of the leach tanks, with a dimension of 10 m diameter by 10 m high, will be equipped with an agitator. These leaching tanks will be insulated and located outdoors. The five CIP tanks will be located inside the mill building. Both the leach tanks and CIP tanks will provide a total leaching retention time of 38 hours. The tanks will be aerated with compressed air from two oil-free compressors (one operation and one standby). The CIP tanks will be equipped with in-tank carbon transferring pumps and inter-stage screens to advance the loaded carbon to the preceding CIP leach tank. The activated carbon will be added into the last CIP leach tank and the loaded carbon will leave the CIP circuit from the first CIP tank.

Sodium cyanide will be added to the leach tanks to extract gold. Lime will be added to maintain the slurry pH at approximately 10 to 11.

The loaded carbon leaving the first CIP tank will be transferred to the carbon stripping circuit, while the leach residue will be sent to a carbon safety screen to recover any coarse carbon grains. The screen undersize will report to the residue thickener prior to being pumped to the cyanide destruction circuit.

The key equipment in the leach circuit includes:

- one 10 m diameter high rate thickener
- six 10 m diameter by 10 m high leach tanks
- five 7 m diameter by 7.5 m high CIP leach tanks equipped with in-tank carbon transferring pumps and screens

- one leach thickener feed trash screen
- one loaded carbon screen
- one carbon safety screen
- two dedicated oil-free type air compressors.

Cyanide detection and alarm systems, safety showers and emergency medical stations will be provided in the area for operators' health and safety.

17.3.5 Carbon Stripping

The loaded carbon will be treated by acid washing and a modified Zadra pressure stripping process for gold desorption in one stream, which is capable of processing approximately 2.0 t of loaded carbon in each batch.

The loaded carbon will be acid washed by diluted hydrochloric acid solution to remove inorganic contaminants, such as calcium scale, prior to being transferred to the elution vessel. The acid washed carbon bed will be rinsed with fresh water.

The stripping process will include the circulation of the heated barren solution through the carbon bed. The barren solution will be heated by passing through two heat exchangers, one heated by the pregnant solution and the other by steam from a boiler. The barren solution will then flow up through the bed of the loaded carbon in the elution vessel and overflow near the top of the stripping vessel. The pregnant solution will be cooled by exchanging heat with the barren solution and flow to the pregnant solution holding tank for subsequent gold recovery by electrowinning. The eluded carbon will be discharged from the bottom of the vessel through a regulating valve to the stripped carbon tank.

17.3.6 Gold Electrowinning and Refining

The pregnant solution from the elution system will be pumped from the pregnant solution stock tank through electrowinning cells where the gold will be deposited on stainless steel wool cathodes. The depleted solution will be sent to the barren solution tank prior to being reheated and returned to the stripping vessel or sent directly to the CIP circuit without heating.

Periodically, the stainless steel cathodes will be cleaned to remove precious metals in the form of sludge. The mud will be pumped to a plate and frame filter press for dewatering on a batch basis. The filter cake will be dried in an oven. Dried slimes will be mixed with flux and melted at approximately 1,150°C in an induction furnace to produce gold bullion containing mostly gold and some silver and impurities.

The area will be provided with sufficient ventilation. The gold room will be in a secured facility with a secured room entrance, monitored 24 h/d by closed-circuit television (CCTV) surveillance. Access to the gold room will be restricted to authorized personnel only.

17.3.7 Carbon Reactivation

The eluted carbon from the elution circuit will be transferred by a recessed impeller pump to a stationary dewatering screen for dewatering and then to a kiln feed bin which provides supplemental dewatering for the carbon. The reactivation will be carried out in an electrically heated rotary kiln at a temperature of 650 to 700°C in an inert atmosphere. The reactivated carbon will be discharged into a tank flooded with water, where the carbon is quenched. The regenerated carbon will be circulated back into the CIP circuit after attrition treatment and screen washing. Recessed impeller pumps will deliver the regenerated carbon to the CIP circuit. Make-up fresh carbon will be added, as required. The fresh carbon will be treated by attrition prior to being used in the CIP circuit. Sufficient ventilation will be provided in this area.

17.3.8 Treatment of Leach Residue

Leach Residue Dewatering

The residues from the CIP circuit will be pumped to a 10-m diameter high-rate thickener to recover residual cyanide and water. The thickener overflow will be pumped back to the leach feed box as dilution water. The underflow of the thickener will be sent to the cyanide destruction circuit prior to flowing by gravity to the TMF.

Cyanide Destruction

The underflow of the residue thickener will be pumped to a cyanide destruction circuit.

The WAD residual cyanide in the underflow of the thickener will be decomposed by a sulphur dioxide/air oxidation process. Sodium metabisulfite (SMBS) will be used as the sulphur dioxide source and copper sulphate as a catalyst as required. Lime will also be added to control slurry pH. The equipment used will include two, 6.0 m diameter by 7.0 m high sulphur dioxide oxidation tanks. Air will be provided for the oxidation process.

17.3.9 Tailings Management

The treated leach residue will gravity flow to the TMF located northwest of the process plant. The residue storage pond will be lined with geomembrane liners. The residue will be covered with the water to prevent suphide minerals from oxidation. The supernatant from the residue pond will be reclaimed by pumping to the grinding and cyanidation circuits for reuse. Tailings management is detailed in Section 18.3.

17.3.10 Reagents Handling

The reagents used in the process will include:

- CIP and gold recovery: hydrated lime ($Ca(OH)_2$), sodium cyanide (NaCN), activated carbon, sodium hydroxide (NaOH), hydrochloric acid (HCl)

- Cyanide destruction: sodium metabisulphite (SMBS, $Na_2S_2O_5$), copper sulphate ($CuSO_4$), hydrated lime ($Ca(OH)_2$)
- Others: flocculant and antiscalant.

All the reagents will be prepared in a separate reagent preparation area. Reagent storage tanks will be equipped with level indicators and instrumentation to prevent overflow. Appropriate ventilation, fire protection and safety protection will be provided at the facility.

Undiluted liquid reagents (including hydrochloride acid and antiscalant) will be added to the required process circuits via individual metering pumps.

Solid reagents (including hydrated lime, sodium hydroxide, sodium cyanide, copper sulphate, and sodium metabisulfite) will be diluted to 10% to 25% solution strength with fresh water in respective mixing tanks and stored in separate holding tanks before being added to various addition points by metering pumps.

Flocculant will be received in solid form and prepared in a packaged preparation system, including a screw feeder, a flocculant eductor and mixing devices. The flocculant mixing system will auotmatically replenish the holding tank. Mixed solution will be transferred and stored in an agitated flocculant holding tank. Flocculant will be diluted to 0.2% solution strength with fresh water and distributed to the leach feed thickener and the leach residue thickener via metering pumps.

Cyanide monitoring and alarm systems will be installed in the cyanide preparation and leaching areas. Emergency medical stations and emergency cyanide detoxification chemicals will be provided in the area.

17.3.11 WATER SUPPLY

Two separate water supply systems will be provided to support the process operations: a freshwater system and a process water system serving various process circuits. The freshwater storage tank and the process water storage tank and will be located inside the process plant.

FRESH WATER SUPPLY SYSTEM

Fresh water will be supplied to one 8.0 m diameter by 8.0 m high storage tank from a fresh water reservoir or boreholes. Fresh water will be used primarily for the following:

- fire water for emergency use
- cooling water for mill motors and mill lubrication systems
- gland seal water for slurry pumps
- reagent preparation.

By design, the freshwater tank will be kept full at all times and will provide at least 2 h of firewater supply in case of emergency.

The potable water from boreholes will be treated by filtration and chlorination and stored in a covered tank prior to delivery to various service points.

Process Water Supply System

The process water system will supply the process water for the grinding, CIP leach, gold recovery and cyanide destruction circuits.

The overflow from the leach feed thickener and the water from the TMF will be pumped the process circuits directly or to a 5.0 m diameter by 6.0 m high process water surge tank and used as process makeup water. The water will be pumped to the various service points.

Process water overflow from the residue thickener will be used for diluting the leach feed thickener underflow.

17.3.12 Air Supply

Plant air service systems will supply air to the following areas:

- leach circuits – high pressure air from two dedicated oil-free type air compressors
- cyanide destruction circuits – low pressure air from air blowers
- crushing circuit – high pressure air from an air compressor
- plant services – high pressure air for various services from one dedicated air compressor
- instrumentation – instrument air will come from the plant air compressors and will be dried and stored in a dedicated air receiver.

17.3.13 Assay and Metallurgical Laboratory

The assay laboratory will provide routine standard assays for both the mine and process plant. The laboratory will consist of a set of assay instruments for gold and silver assays, total sulphur and base metal analyses, including:

- fire assay equipment
- a microwave plasma-atomic emission spectrometer
- a Leco furnace
- other instruments such as pH and redox potential meters and experimental balances.

The metallurgical laboratory will perform tests to optimize the process flowsheet and improve metallurgical performance. The facility will be equipped with laboratory crushers, ball mills, particle size analysis devices, leach cells, balances, and pH metres.

17.3.14 PROCESS CONTROL AND INSTRUMENTATION

The plant control system will consist of a distributed control system (DCS) with PC-based operator interface stations (OISs) located in the plant control room. The plant control room will be staffed by trained personnel 24 h/d.

The DCS, in conjunction with the OISs, will perform all equipment and process interlocking, control, alarming, trending, event logging, and report generation.

Programmable logic controllers or other third party control systems supplied as part of mechanical packages will be interfaced to the plant control system via Ethernet network interfaces where possible.

Operator workstations will be capable of monitoring the process operations within the entire plant site, and will be capable of viewing alarms and controlling equipment within the plant. Supervisory workstations will be provided in the offices of the plant superintendent and the electrical and instrumentation superintendent.

In addition to the plant control system, a closed-circuit television (CCTV) system will be installed at various locations throughout the plant site, including the crushing facility, the surge bin conveyor discharge point, the tailings facility, and the gold recovery facilities. The cameras will be monitored from the central control room 24 h/d.

For health and safety, cyanide monitoring and alarm systems will be installed in the cyanide preparation, leaching and destruction areas.

17.4 YEARLY METALLURGICAL PERFORMANCE PROJECTION

According to the test work results and metallurgical performance projections described in Section 13.0 and the proposed mine production schedule, preliminary gold recoveries for the project are projected on a yearly basis and shown in Table 17.2. Further test work is recommended for more accurate metallurgical performance projections.

Table 17.2 Yearly Gold Metallurgical Performance Projections

	Mill Feed		Recovery
Year	Tonnage (t)	Grade (g/t Au)	(%)
1	438,000	5.79	89.3%
2	547,500	4.37	80.9%
3	547,500	3.62	79.4%
4	530,202	3.11	86.9%
5	215,674	3.13	65.0%
6	431,800	2.43	62.9%
Total	**2,710,676**	**3.82**	**80.8%**

18.0 PROJECT INFRASTRUCTURE

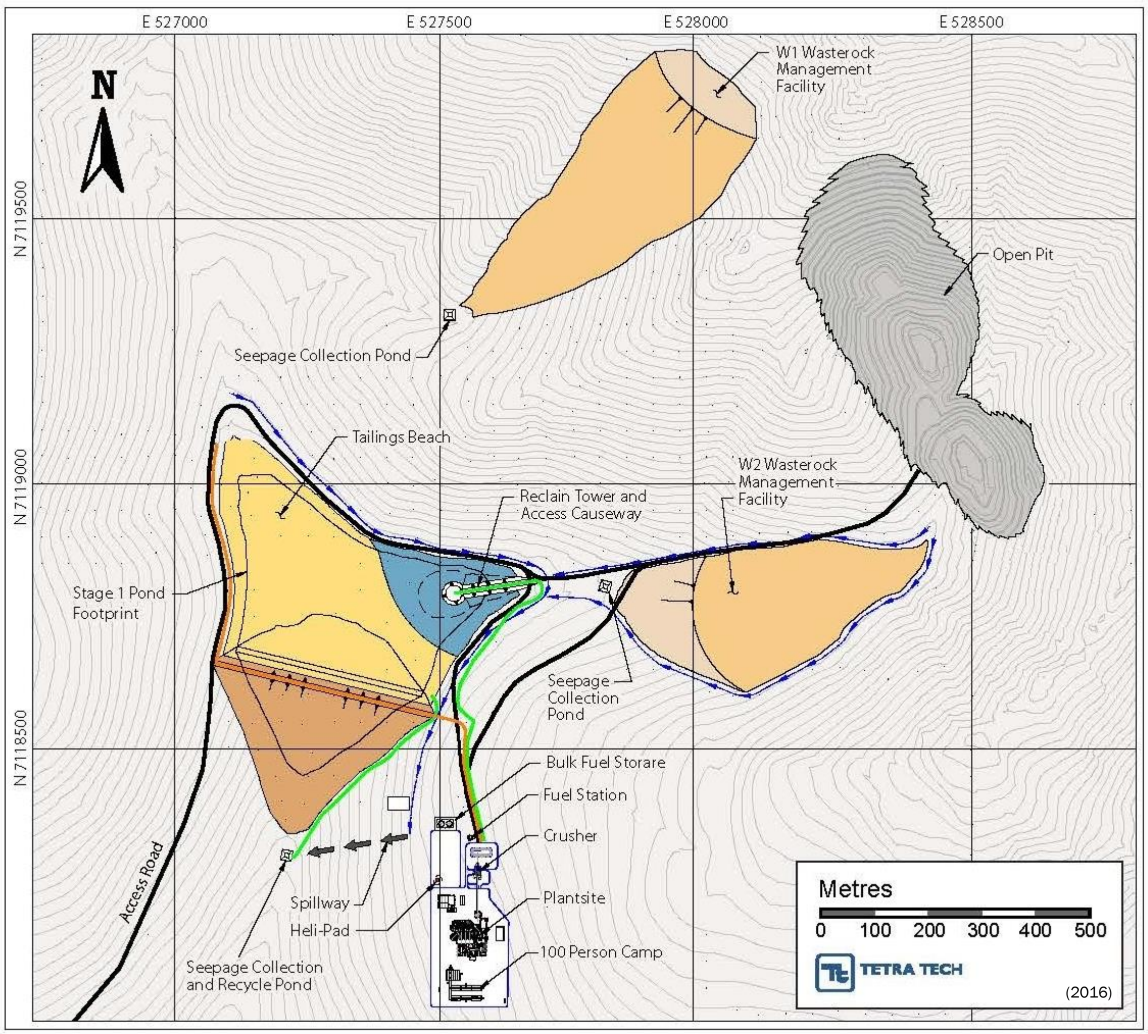
18.1 SITE LAYOUT

The proposed on-site infrastructure for the Project will include:

- a processing plant
- a permanent camp
- an emergency vehicle building with vehicle maintenance shop and warehouse
- administration offices
- power generation units
- a main electrical substation and power distribution system
- potable and fire water storage and distribution system
- plant and camp sewage treatment facilities
- a laydown and a container storage yard
- a fuel storage and fueling station
- a TMF
- two WRMFs
- access and site roads.

The general site layout of the Project is provided in Figure 18.1.

Figure 18.1 General Site Layout



18.1.1 SITE ACCESS

The Project site is currently accessible by air. There is an existing 3,000 foot airstrip approximately 8 km from the site. The airstrip is connected to the site by a service road.

A single-lane, radio-controlled tote road is proposed to link the project site to Mayo, in order to facilitate land transport. The road will be 69 km long with 17 km of construction along the existing trail/winter road and 52 km along new terrain. The design road width is 5.0 m, with a design vehicle speed of 20 to 50 km/h (with design vehicle WB-17 standard large semi-trailer), and a maximum gradient of 11%. Three-meter wide pullouts will be constructed where necessary. It is expected that the road will be temporarily closed for approximately one month during the spring thaw and one month during the fall freeze. As the private road connects to the public road network, gates will be placed that will restrict access to the tote road.

The tote road will be built with the mining equipment fleet prior to mine development. During operation, the tote road will be maintained by the mining equipment fleet during its stand-by time.

18.1.2 PROCESS PLANT

The process plant will comprise the following areas:

- crushing
- grinding
- leaching
- acid wash
- carbon elution
- electrowinning
- smelting
- cyanide destruction
- assay and metallurgical laboratory.

The preliminary plant site arrangement is shown in Figure 18.2 and Figure 18.3.

Figure 18.2 Plate Site Layout

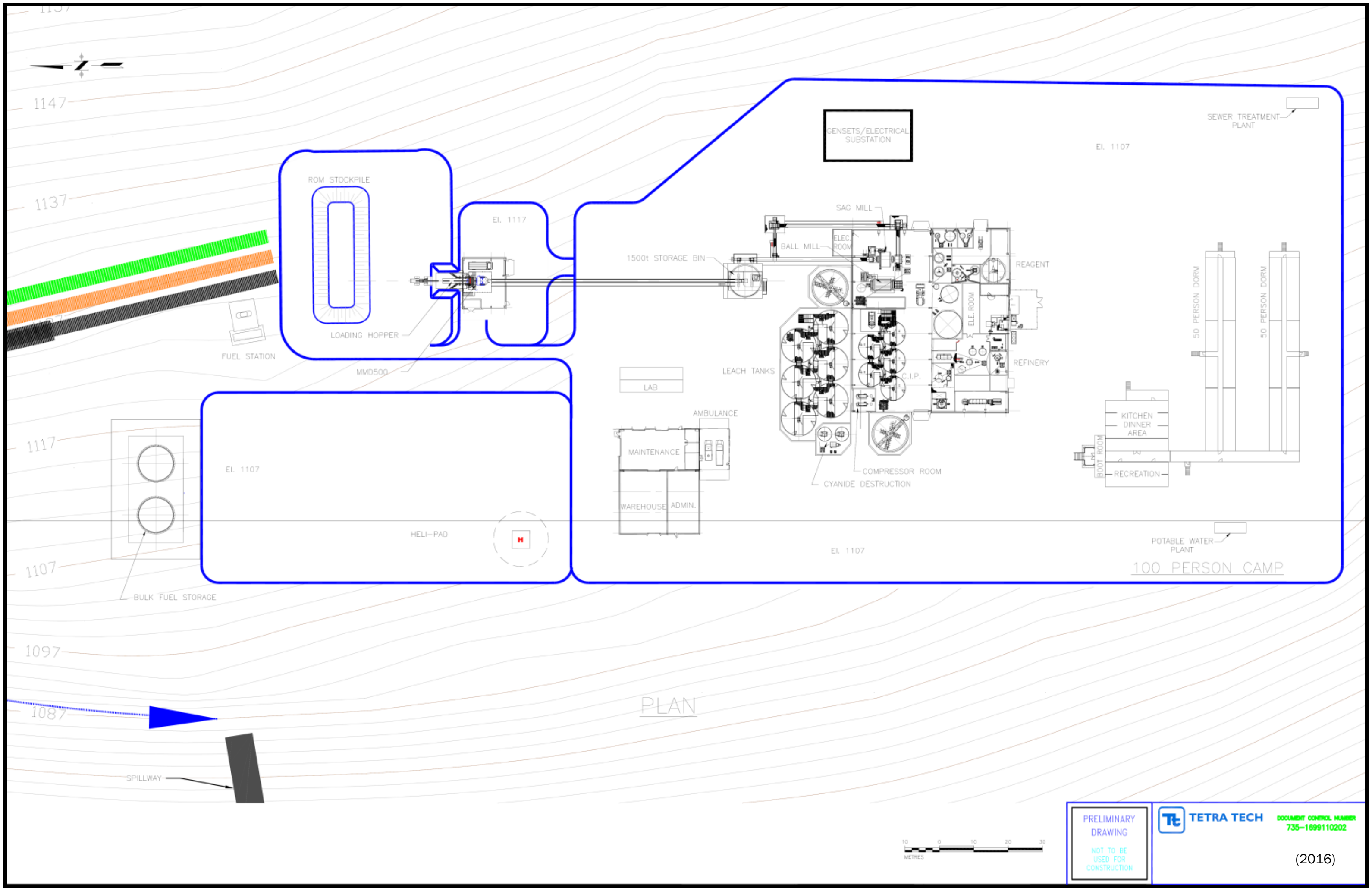
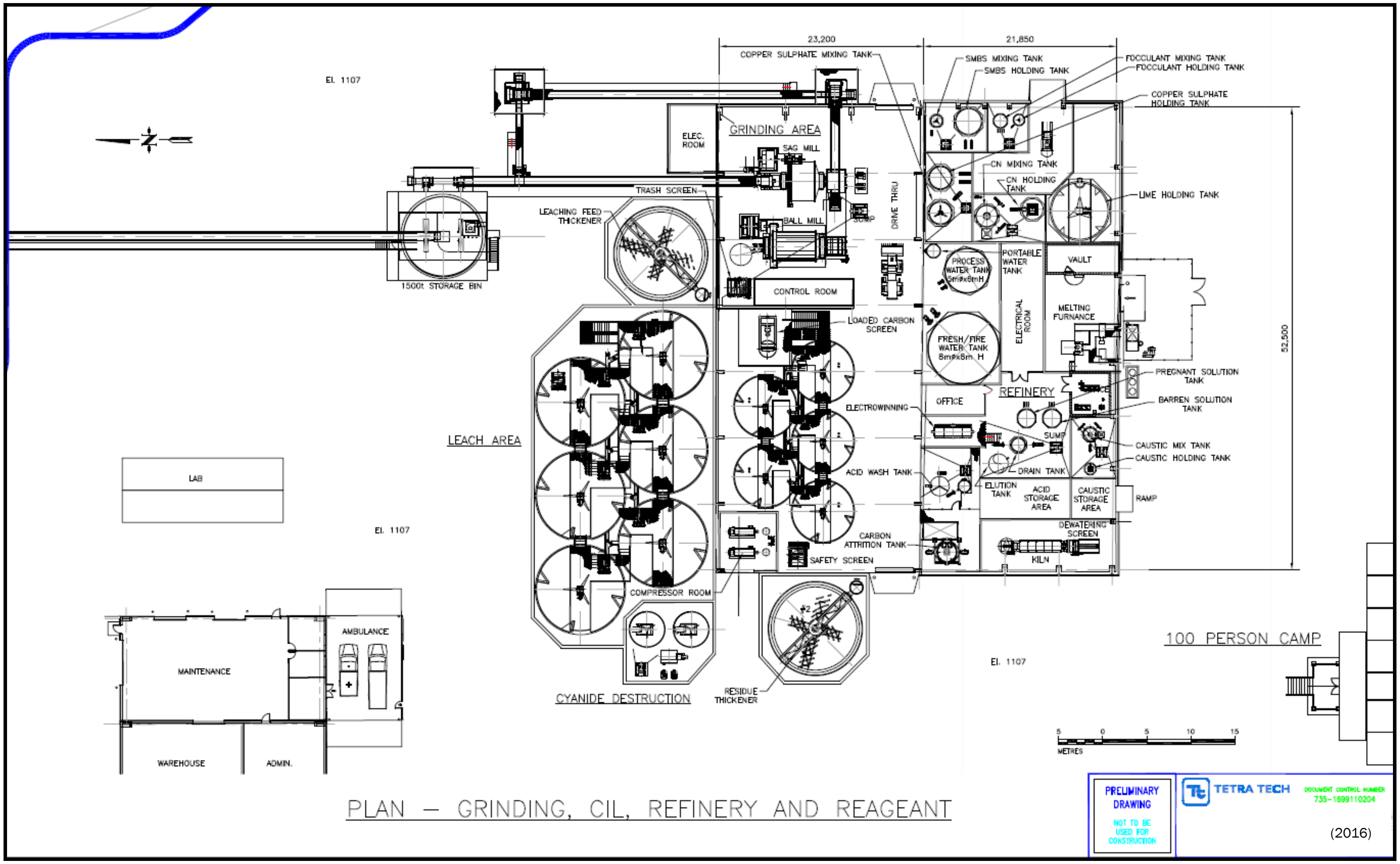


Figure 18.3 Grinding, Leaching and Recovery Area Layout



18.2 POWER

The average electrical demand is estimated to be 2.0 MW for the Project.

The power will be supplied by three, 1.5-MW diesel generators. Two of the three units are expected to operate full time; the third generator will be available as a stand-by unit.

The site electrical distribution system will run on 4,160 V, which is the same voltage as the power generation system. The transmission poles will carry both power and communication lines. Motor control centers (MCCs) and power distribution centers at each facility will manage and control power requirements.

Emergency back-up generators with automatic transfer switches will supply the Project with back-up power.

A waste heat recovery system will collect waste heat generated by the power plant and transfer the heat to the process plant and other buildings via the glycol circulation system. Only non-toxic glycol, such as propylene glycol, will be used.

18.3 TAILINGS MANAGEMENT

18.3.1 DESIGN CRITERIA

The basic design criteria for the Tailings Management Facility (TMF) and Waste Rock Storage Facilities were developed and summarized in Table 18.1.

Table 18.1 Design Criteria Summary

Parameter	Units	Value
Average Mill Throughput	tpd	1,500
Design Life	yrs	7
Total Tonnes of Tailings	Mt	2.7
Tailings Final Settled Dry Density (Average)	t/m^3	1.3
Embankment Crest Width	m	10
Embankment Upstream Slope	-	2.5H:1V
Embankment Downstream Slope	-	2.5H:1V
Freeboard (Storm Storage, Wave Run-Up & Freeboard)	m	5
2 Year Starter Tailings Tonnage	t	681,000
Waste Rock Density	t/m^3	2
Waste Rock Tonnage	M tonnes	14.4
Waste Rock Volume	$M\ m^3$	7.2
Overall Waste Rock Storage Facility Slopes	-	2H:1V

The following assumptions have been taken into consideration for this study:

- All embankments will be constructed using waste rock and waste rock processed materials.
- Cyanide will be used in the process and the TMF will be a fully lined with an engineered liner system in the basin area to protect the natural groundwater.

18.3.2 Tailings Management Design

General

The principal design objectives for the TMF are protection of the regional groundwater and surface waters both during operations and in the long-term (after closure), and to achieve effective reclamation at mine closure. The design of the TMF has taken into account the following requirements:

- Permanent, secure and total confinement of all solid waste materials within an engineered disposal facility
- Control, collection and removal of free draining liquids from the tailings during operations for recycling to the maximum practical extent
- The inclusion of monitoring features within the facility to verify performance goals are achieved and design criteria and assumptions are met
- Staged development of the facility over the life of the Project

TMF Design Features

The overall project general arrangement is shown on Figure 18.1. The TMF has the following specific features for tailings and water management:

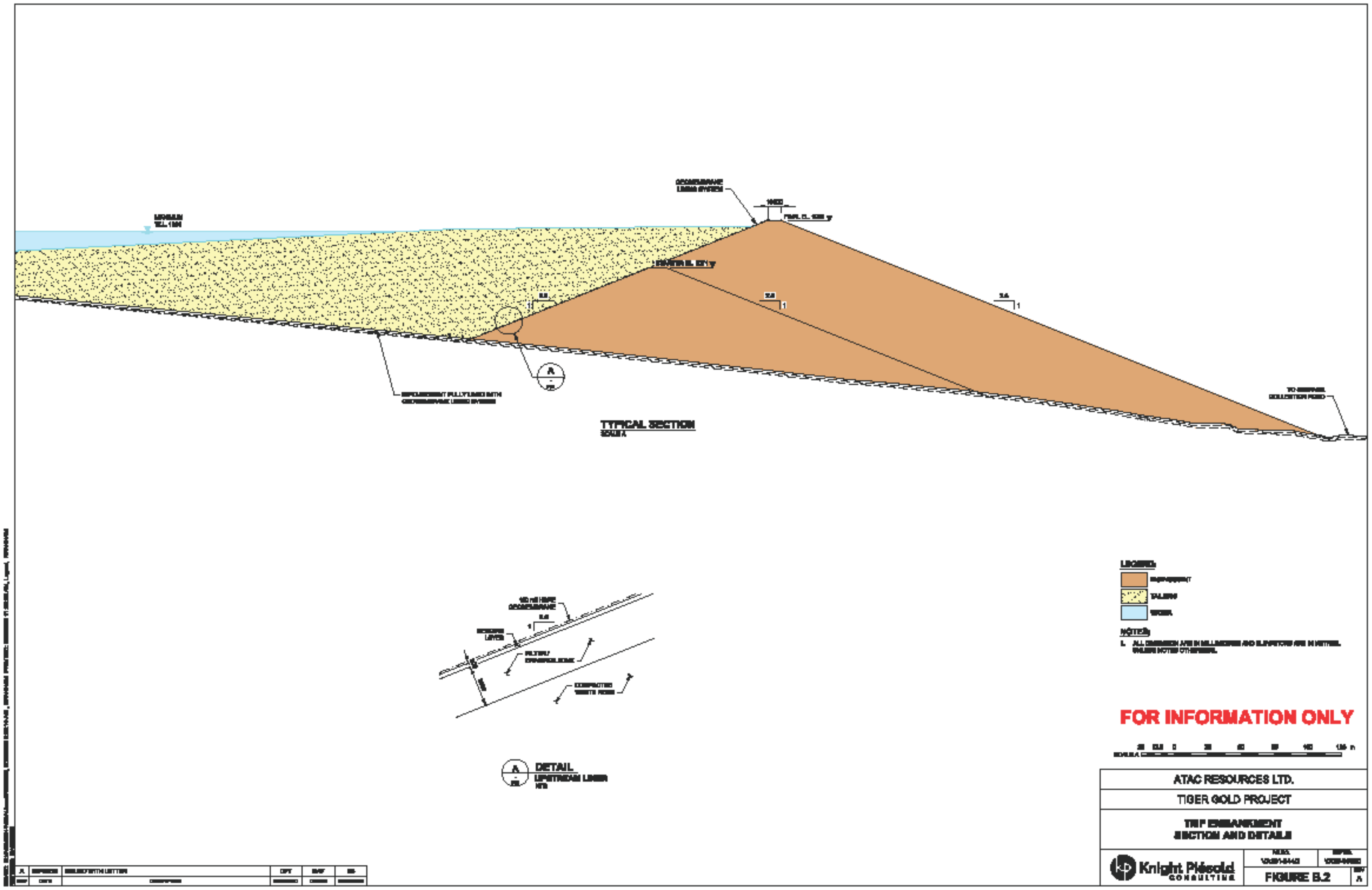
- Rockfill embankment constructed from mine waste rock and processed filter/drainage zones
- Fully lined impoundment to minimize seepage losses
- Basin drainage system
- Foundation drainage system
- Tailings beach
- Tailings distribution system
- Reclaim water system
- Diversion ditches

Embankment Design

The tailings dam is designed as a rock-filled structure with granular filter zones on the upstream face primarily constructed from mine waste rock. The impoundment and upstream face of the dam will be covered with a geomembrane to minimize tailings seepage losses. The filter zones provide a bedding surface for the geomembrane and prevent the migration of fines into the rockfill dam from geomembrane defects.

The TMF will be expanded using downstream construction. An initial starter dam will be constructed to contain the first two years of tailings production to minimize upfront capital expenditure. The dam will be raised twice over the mine life to increase the storage capacity and maintain a minimum of 5 m freeboard at all times. A cross section of the embankment is shown on Figure 18.4.

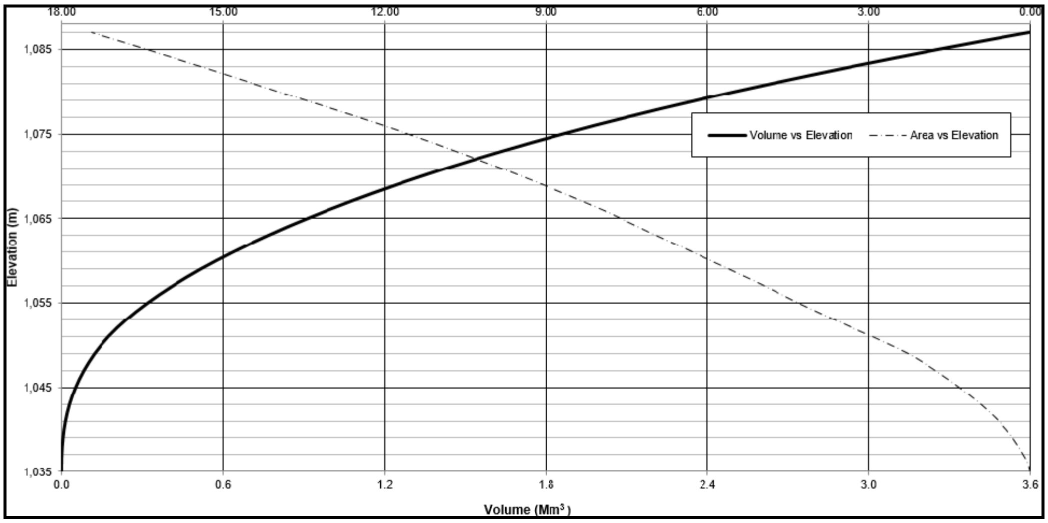
Figure 18.4 TMF Embankment Section



TMF CAPACITY AND FILING SCHEDULE

The TMF layout provides storage capacity for approximately 2.5 million cubic meters for tailings, process water, storm storage and freeboard to an elevation of 1,085 m. This will provide storage for 7 years of mine operations. The depth/area/capacity (DAC) relationship for the TMF to an elevation of 1,085 m is shown on Figure 18.5.

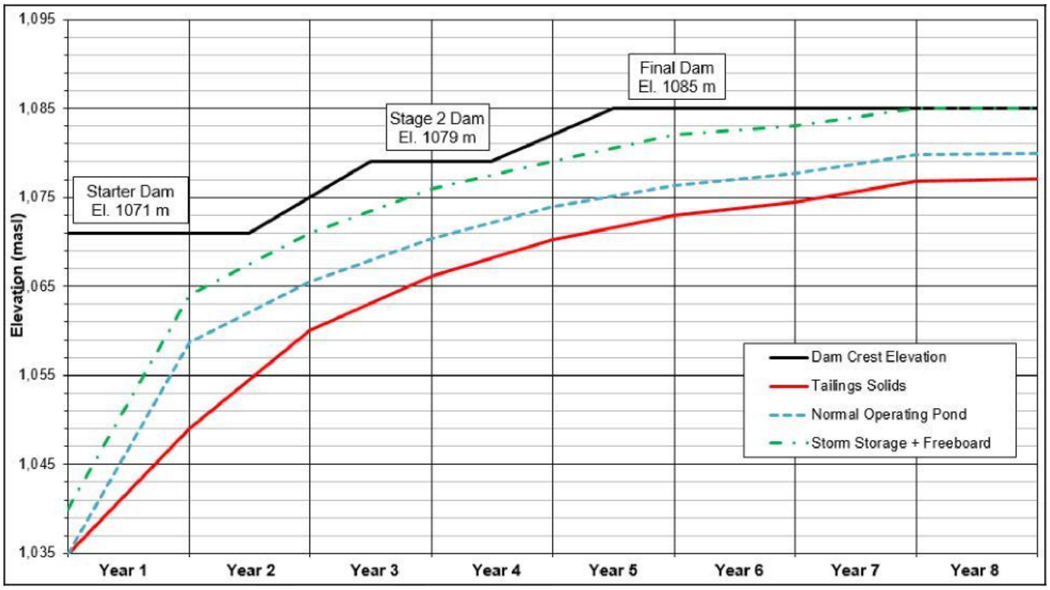
Figure 18.5 TMF Depth-Area-Capacity Relationship



Freeboard allowances to manage run-off, storm storage and process water have been incorporated in dam sequencing. Reclaim water will be recirculated from the supernatant pond back to the mill and used as process water. Reclaim water will be pumped to the mill using a submersible pump installed in a wet well surrounded by coarse rockfill at the end of an access causeway. This configuration allows for water to be reclaimed beneath the ice and snow cover.

The staged filling schedule for the TMF is shown on Figure 18.6.

Figure 18.6 TMF Filling Schedule



TAILINGS DISTRIBUTION SYSTEM

Tailings will be delivered to the TMF from the mill via gravity in a tailings pipeline. The discharge of tailings from the delivery pipelines into the TMF will be from a series of valved off takes located along the TMF dam crest and perimeter road. The sandy coarse fraction of the tailings will settle rapidly after discharge and will accumulate close to the discharge points, forming a gentle beach with a slope of approximately 0.5 to 1 percent. Finer tailings particles will travel farther and settle at a steeper slope adjacent to and beneath the supernatant pond.

The tailings beaches will be developed with the intent to maximize storage volume and to control the location of the supernatant pond. Selective tailings deposition will be used to maintain the supernatant pond away from the embankment.

WATER RECLAIM SYSTEM

The water reclaim system performance objectives are as follows:

- To allow the collection and removal of process water
- To allow the collection and removal of precipitation and runoff
- To remove water beneath ice cover during winter operations

A wet well reclaim tower will be constructed at the northeast of the facility on top of the geomembrane to allow process water to be reclaimed. The reclaim tower will be raised over the life of the facility and will be accessed via an access causeway constructed from mine waste rock. A submersible pump will be installed in the vertical wet well to allow water to be reclaimed beneath the ice cover in the winter months. Appropriate filter zones will be constructed around the wet well to prevent fines migration into the sump area.

Basin Underdrainage System

A basin underdrainage system will be placed above the geomembrane liner system to enhance consolidation of the tailings and collect seepage. Tailings seepage collected in the underdrain system will be returned to the impoundment.

Seepage Collection and Recycle Ponds

Seepage to the embankment drainage zones will be largely controlled by the geomembrane liner system and tailings beach. Seepage intercepted in the embankment drainage zones will be routed to the seepage collection and recycle pond located at the topographic low point at the downstream toe of the embankment. Surface water runoff from the embankment faces or other impacted areas in the vicinity of the TMF embankment will be collected in the pond. Water collected in the pond will be continuously monitored and pumped back into the TMF via a recycle pipeline.

Emergency Spillway Design

The TMF will be designed to safely manage and store the Inflow Design Flood for the project. An emergency spillway will be incorporated into the embankment abutment as a contingency to convey excess water safely from the TMF should the water in the facility be managed incorrectly. The primary objective of the spillway is to protect the integrity of the TMF embankment during an emergency and is not intended to be used at any stage during operations.

18.4 Waste Management

A waste rock development strategy has been identified to assist with future mine planning to take advantage of topographic conditions adjacent to the deposit. The locations of the sites are based on topography and proximity to the proposed pit location and identified as sites W1 and W2 shown on Figure 18.1.

A gully northwest of the deposit (Site W1) was identified for efficient waste rock storage from mining the upper portions of the deposit with haulage roads constructed parallel to the hillslope contours. Waste rock will be hauled along contour and lobes pushed out across the gully in 50 meter increments. Waste rock Management Facility, W1, provides capacity for 3.2 Mt of waste rock storage between elevations 1,230 to 1,500 meter stacked at an overall slope of 2H:1V in the gully. Waste rock would be developed in stages depending on the mining sequence of the hillslope.

Waste Rock Management Facility, W2, is utilized later in the mining operation for storage of up to 7.2 Mt of waste rock between elevations 1,103 to 1,200 meters within the upper reaches of the drainage. Waste rock delivered to this site will be from the lower elevation pit excavation. Diversion ditches will be constructed to divert clean runoff around the facilities and contact water will be collected at the base of the storage areas and directed to the TMF.

18.5 WATER MANAGEMENT

The key facilities for the water management plan are:

- Open pit
- Waste Rock Management Facilities W1 and W2
- Mill (including fresh and process water tanks)
- Tailings Management Facility (TMF)
- Diversion and water management structures
- Fresh water supply
- Sediment and erosion control measures for the facilities

The water management strategy utilizes water within the project area to the maximum practical extent. The plan involves collecting and managing site runoff from disturbed areas and maximizing the recycle of process water. Site runoff water will be stored on site within the TMF. The water supply sources for the project are as follows:

- Precipitation runoff from the mine site facilities
- Water recycle from the tailings supernatant ponds
- Groundwater wells for fresh water supply and potable water
- Treated black and grey water, in small quantities, from the camp

An overall high-level average site water balance assessment was carried out to determine the preliminary water management strategy and process makeup water requirements for the project.

The preliminary results from the water balance assessment are summarized below:

- There will be no surface water discharge from the TMF to the environment during operations.
- Under average precipitation conditions the site will operate in a water-deficit condition with the need for additional makeup water.
- The average annual runoff from the TMF and Waste Rock Management Facilities will increase up to 90,000 m^3 per year.
- Pit runoff and pit dewatering contributes up to 20,000 to 25,000 m^3 of water per year at full footprint under average annual precipitation conditions.
- The average volume of water retained in the tailings voids, and hence not available for water recycle to the processing plant is approximately 260,000 m^3 annually.
- Water will be collected for the process and stored in the TMF by diverting some of the May snowmelt runoff from the upper catchment diversion ditches into the tailings

impoundment and from collection of runoff from the Waste Rock Management Facilities W1 and W2 and the open pit runoff and dewatering system.

- The potable water requirement is approximately 7,000 m^3 annually.
- The freshwater water requirement for plant operation is approximately 20,000 m^3 annually.

18.6 Fresh, Fire, and Potable Water Supply, and Sewage Disposal

Fresh and fire water will be required primarily for start-up and emergency purposes, gland seal water, reagent, flotation cleaning stages, and process water makeup. The gland and seal water will be pumped and distributed to the slurry pumps from the fire-fresh tank.

Fresh water will be supplied from the diversion system that will divert water from areas around the TMF/WRMFs.

Potable water will be supplied from water wells. A potable water tank and hydro-chlorination system will be provided.

The sewage treatment plant will be a pre-packaged rotating biological contactor. The plant will be manufactured off-site and containerized for simple connection to the collection system on site. Once treated, the sewage treatment plant effluent will be discharged into the outfall in accordance with the federal and territorial regulations.

18.7 Communication

On-site communication systems will include a voice over internet protocol telephone system, a local area network with wired and wireless access points, hand-held very-high frequency radios and a satellite television system for the accommodations.

Off-site communications will be achieved with a satellite base system, which will be utilized during the construction phase and the operating phase of the Project.

19.0 MARKET STUDIES AND CONTRACTS

There were no market studies conducted or contracts negotiated for this PEA.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The following description comprises an updated summary of the more detailed text provided in Section 20.0 of the NI 43-101 technical report entitled "Technical Report and Preliminary Economic Assessment of the Tiger Deposit, Rackla Gold Project, Yukon, Canada" (2016 PEA) and dated May 31st, 2016, combined with relevant new information as appropriate.

20.2 ENVIRONMENTAL SETTING

The Tiger Property is located in the Yukon Plateau-North ecoregion of the Yukon, approximately 143 km northeast of Stewart Crossing, 98 km northeast of the community of Mayo, and 55 km northeast of Keno City. The Property's ecoregions are characterized by a series of plateaus and valleys located northeast of the Tintina Trench. The Tiger Deposit, centrally located within the Property, is situated in the Nadaleen Range of the Selwyn Mountains and is drained by creeks that flow into the Rackla and Beaver rivers which are both part of the Yukon River watershed.

Local topography is alpine to sub-alpine, featuring north and south trending rocky spurs and valleys that flank a main east-west trending ridge. Elevations range from 725 masl alongside the Beaver River in the center of the claim block, to 1,800 masl atop a local peak referred to as Monument Hill.

Most hillsides are talus covered at higher elevations and are blanketed by glacial till at lower elevations. Soil development is moderate to poor in most areas. Forest cover is comprised mainly of white pine and black and white spruce up to elevations of 1,500 masl. At higher elevations shrub birch, scattered pine, white spruce, and subalpine fir form the forest cover, with lichen comprising the under-storey. With increasing elevation, vegetation thins to shrub and lichen tundra, with the tree line at approximately 1,500 masl. The density and size of vegetation gradually increases on lower slopes towards the valley floors, where it is well-treed with mature black spruce. The valley floor's under-storey typically consists of low shrubs and moss.

Moderately steep, south facing slopes are well-drained and are often lightly forested with poplar. Steep, north-facing slopes are usually rocky with exposed outcrop and/or talus. Gentler, spruce and moss covered terrain exhibits widespread permafrost.

Much of the overburden in the region is associated with the most recent Cordilleran ice sheet, the McConnell glaciation, that is believed to have covered south and central Yukon between 26,500 and 10,000 years ago (Yukon Geological Survey 2010).

The climate at the Property is typical of northern continental regions, with long cold winters, truncated falls, shortened springs, and mild to warm summers. Although summers are relatively mild, snowfall can occur in any month at higher elevations. Temperatures within the valley area's ecoregions are some of the most extreme in Yukon, ranging from lows of −62°C to highs of +36°C. Higher terrain experiences slightly less extreme temperature ranges. Annual average precipitation for this ecoregion is approximately 300 mm/year, with areas in the east receiving upwards of 600 mm/year of precipitation (Matrix 2010).

20.3 BASELINE ENVIRONMENTAL STUDIES OVERVIEW

The Tiger Deposit is located within the Rau Property, which is a part of the much larger RGP. Due to the widespread and sequential exploration advancement throughout the district, ATAC has begun to develop robust baseline environmental characterization and monitoring programs to support environmental and socio-economic assessment and permitting under Yukon and federal legislation for advanced development.

Table 20.1 of the 2016 PEA outlined the status of all relevant environmental studies on the Property, along with key areas where further studies will be necessary, as the Project progresses. The following is a brief summary of the key environmental components that will require further study for future environmental assessment purposes.

Surface Water Quality: Good quality assessment conducted on a quarterly basis since 2007, and monthly between July 2012 and 2014; surface water monitoring stopped in 2014. Good for stage of project, with the possibility that additional water quality license monitoring and compliance points may need to be added to the network during Type A Water Use Licensing.

Surface Water Hydrology: Good for stage of project, however current hydrology data from project site drainages limited to seasonal open water (May to Oct); year-round operation mine planning and detailed site and process water balance will require continuous year-round monitoring at key sites (Rau 11, Rau 12, Rau 9). Additional water quantity license monitoring and compliance points may need to be added to the network during Type A Water Use Licensing. In the future it will be important to also collect flow measurements at water quality stations as licensing is increasingly setting maximum total metal loadings in receiving environment monitoring rather than just point-source concentrations.

Hydrogeology/Subsurface Water Quality: Gap Identified. Detailed understanding of nature and characteristics of groundwater will be required to support advanced mine development licensing. A subsurface hydrological investigation will need to be undertaken prior to future stages of study to provide an accurate characterization of groundwater depth, flow and quality, where potentially affected by pit, waste rock disposal areas, plant site, and TMF.

Geochemical Characterization: Gap identified. Currently, available data limited to static testing (ABA) conducted on exploration drill core (several samples per lithology). A more thorough, statistically and spatially representative program, will be required to provide predictive ARD/ML information on all site rock disturbance and deposits including waste rock and tailings. ARD is not expected, due to carbonate host rocks and low sulfides. However, more investigation is required to determine propensity for metal leaching, and confirm low ARD potential. Geochemical characterization should represent all proposed excavations including all lithologies and waste products (e.g. tailings streams). Static testing may be adequate to commence assessment under the YESAA; kinetic testing will likely be required for water licensing if identified as necessary during preliminary static assessment.

Permafrost: Good for stage of project; detailed design phase will require more in-depth study. The project will require an accurate and complete understanding of the Project area permafrost regime to support the YESAB Executive Committee submission, major licensing, and final design. A geotechnical drilling program will be required to determine sub-surface conditions to support the detailed design phase of site development.

Climate and Weather: Good for stage or project with continuous monitoring data available from 2013 and 2014 at the Rau airstrip, and 2017 to present at the Rau camp; ongoing continuous weather station data collection is recommended to support future development permitting. Localized temperature, precipitation, and wind data collection (including snowpack surveys) should be continued. Detailed localized climate data analysis will be required for development permitting.

Heritage Values: Adequate HRIAs completed for the access road, airstrip and other high potential areas (included selected physical testing). The expectation is that heritage sites may be found in valleys around lakes. Caution is urged during land disturbance activities. Heritage impact terms and conditions will be part of the Mining Land Use licensing conditions. HRIA avoidance areas can be used for future development planning and design.

20.4 WATER MANAGEMENT

A conceptual overall site water management was developed by Knight Piésold and is described in Section 18.5. Key concepts included the collection and management of site runoff from disturbed areas and maximizing the recycle of process water. Because average precipitation data predict the site will operate in a water-deficit condition with the need for additional makeup water, site runoff water will be directed to the TMF for storage and recycle. Should excess site water conditions prevail, it will need to be conveyed away from project contact areas via appropriate diversions to prevent surface water runoff from entering the process system. Groundwater wells will need to be installed for fresh water supply and potable water.

20.5 PROJECT PERMITTING REQUIREMENTS

20.5.1 CURRENT PERMITS

Prior to August 2019, ATAC was conducting mineral exploration and associated activities at the Rau Property, including the Tiger Deposit, under the terms and conditions of Class 3 Exploration Permit LQ00260C (Mining Land Use Regulations). That permit was renewed on August 14, 2020 under authorization LQ00531 and is valid until August 13, 2024.

In addition, In March 2018, ATAC received conditional approval for a 65 km, all-season, gated and restricted access tote road to the Rackla Gold Project, pending completion of an Access Management Plan and an Adaptive Management Plan. This represented completion of the YESAB Mayo District Office review process, enabling ATAC to obtain key permits and proceed with a submission to the Yukon Water Board. The assessment and permitting for the proposed access road is independent of the Project but is anticipated to facilitate development of the Tiger Deposit.

Discussions regarding access and development of the Tiger Deposit with the Na-Cho Nyak Dun First Nation (NNDFN), local communities and other interested parties has been ongoing for more than seven years.

20.5.2 MINE DEVELOPMENT PERMITS

Mining development in Yukon is governed by a multi-staged process that can be roughly divided into two groups:

- **Project Development Permits and Licenses** (e.g. Quartz Mining License, Water License, etc.) – The acquisition of each of these authorizations requires substantial and detailed submission documentation (e.g. project description, socio-economic and environmental baseline characterization, potential environmental effects, proposed mitigative measures to address potential effects, monitoring plan, component-specific adaptive management plans, and closure plan). Each of these typically takes several months to complete, and will drive the timelines for project development.

- **Ancillary Permits and Licenses** (e.g. camp septic, propane, electrical, solid waste, building permits for site infrastructure, etc.) – These are fairly straightforward to acquire, require relatively minor documentation in application, and can be secured as a project develops, typically without impact to project timeline.

This discussion is focussed on an assessment of the Project Development permits and licenses, as it is assumed that the numerous ancillary permits will be secured as necessary to support the mine development time schedule. Table 20.1 summarizes the Project development authorizations required prior to production.

Table 20.1 Project Development Authorizations Required

Mine criteria trigger	Authorization Required	Issuing Agency	Legislation
>100 t/d gold mine	YESAB Decision Document	Issued by Decision Body (Government of Yukon, Energy, Mines & Resources), after evaluation at the YESAB Executive Committee level	YESAA, Assessable Activities, Exceptions and Executive Committee Projects Regulations
Commencement of commercial production	Quartz Mining License	Yukon Government, Energy Mines & Resources	*Quartz Mining Act*, Mining Land Use Regulations
Use of water for milling, use of >300 m^3/d, deposit of a waste	Type A Water Use License	Yukon Water Board	*Waters Act*, Waters Regulations

The authorizations in Table 20.1, listed in the order in which they will be acquired, will be issued for the full LOM period as described in this updated PEA. The Water Use License may require modification of the security held under the Quartz Mining License. The Project will also be subject to the federal Metal and Diamond Mine Effluent Regulations pursuant to subsections 34(2), 36(5) and 38(9) of the *Fisheries Act*, which will set monitoring requirements and criteria for all effluent discharges emanating from the mine and its infrastructure (e.g. pit, heap, tailings pond, etc.).

As noted by ATAC in its Beaver River Land Use Plan Submission (ATAC 2019), Yukon mine development projects are subject to substantial review and regulatory requirements. These mechanisms provide means for detailed evaluation of project-specific impacts, benefits and mitigations. Existing processes serve as appropriate means of management, versus blanket restrictions on development in any given area.

ATAC has been through the public YESAB review process nine (9) times since 2007 for projects within the planning area, including the previously noted positive decision on the 65-km Rau tote road.

The first stage of review for all projects is a Project Proposal submission to YESAB, an independent agency tasked with conducting public reviews of proposed projects to evaluate potential impacts and make recommendations as to whether the project should proceed. This formalized process provides an opportunity for regulators, members of the public, and the proponent to work together in a transparent manner to evaluate a project.

For this Project, it is anticipated that the YESAB assessment will take the form of an Executive Committee level review. This will ensure that the Project is evaluated at an appropriate level based on the scope of proposed activities, and more fundamentally ensures the environmental and socio-economic impacts of projects are adequately considered.

For projects assessed by the YESAB Executive Committee, timelines would generally be up to a year (or less) and typically not greater than 2.3 years for release of the Screening Report and Recommendations, plus up to 60 days for a government decision. The length of time required for an assessment depends on the type of project and the amount of information provided in the Project Proposal. Proposals with inadequate information take longer as they may require more information requests to be issued. (YESAB 2020 - https://www.yesab.ca/faq/).

Once a Project Proposal completes the YESAB review process, it proceeds to the regulators for final decisions regarding permits and associated terms and conditions. In particular, the Project will require review by the Yukon Water Board – an additional independent public registry process and regulatory decision leading to the issuance of a Type A Water License.

A requirement of the Quartz Mining License will be the approval of a Detailed Decommissioning and Reclamation Plan. This document will be used to set security requirements, which must be met prior to receiving authorization for the commencement of commercial production.

Due to the nature of the Project and geology of the deposit, environmental assessment and permitting is anticipated to proceed with minimal technical challenges. Important environmental considerations include the fact that the Project will process primarily oxide material and the deposit is hosted in strongly neutralizing carbonate rocks. Although a small amount of sulphide material will be excavated for processing and waste, it is anticipated that mixing this material with the dominant neutralizing lithologies will result in net neutralizing waste products.

The abundance of carbonate host rocks with negligible sulphide content, the minimization of the Project footprint and positioning as far as possible from sensitive aquatic values, recycling of process water, and detoxification of cyanide will underscore the environmental assessment and subsequent licensing. The direct agitated leach in a controlled environment concept is considered to be a project design strength.

20.6 SOCIOECONOMIC, COMMUNITY ENGAGEMENTS

The Project is located within the Traditional Territory of the Na-Cho Nyak Dun (NNDFN), whose people have lived a subsistence lifestyle off the land for centuries. The NNDFN represents the most northerly community of the Northern Tutchone language and culture group. The NNDFN resides in the community of Mayo, a town that had its beginnings during the boom years of the silver mines in the area. Since concluding a Land Claim Agreement with the Government of Canada in 1993, the NNDFN have developed the capacity to provide skilled personnel and a broad range of services to mining projects.

ATAC has developed a good working relationship with the NNDFN and in January of 2014 the parties renewed the ECA that was originally signed in 2010. The ECA provides a framework within which exploration activities and environmental regulatory processes for the Project have been and will continue to be carried out.

ATAC began community consultations regarding proposed ground access to the Project area in 2009, and continued engagement on this project in subsequent years (ATAC 2019). In the lead-up to the YESAB submission for the tote road, a series of consultation events were held in March, April and June 2016. These events included open houses in Mayo, Keno City and Whitehorse, as well as presentations to NNDFN at a general meeting and the annual General Assembly.

In response to feedback obtained during the tote road consultation process, additional alternatives were evaluated, and changes were made to the proposal. In particular, ATAC investigated the feasibility of airships instead of ground access, shifted the alignment to minimize impacts on moose, and developed an alternate route that was relocated to cross NNDFN Category A Lands.

The main concerns that arose during consultations were regarding access – specifically, ensuring that new access was not made available to the general public. Participants recognized ATAC's commitment to ensuring a private access-controlled road; however, there were questions about what could happen if the Project or company ownership changed. NNDFN proposed that the route could cross segments of their Category A lands as a way of ensuring NNDFN had an irrevocable say in any future changes to use of the road. To accommodate this request, the road design was revised to potentially cross two limited segments of NNDFN Category A Lands (2.4 km in total).

Similar consultations will need to be undertaken for the anticipated mine development YESAB submission at the Executive Committee level. ATAC will also need to negotiate an enhanced Impacts Benefit Agreement with NNDFN, which will encompass the mine development and production stream.

21.0 CAPITAL AND OPERATING COST ESTIMATES

21.1 SUMMARY

The capital and operating costs for the Project have been estimated and are summarized in Table 21.1.

Table 21.1 Summary of Capital and Operating Costs

Cost Type	Total Capital ($ million)	Average Unit Operating Cost ($/t milled)
Initial Capital Costs	110.1	-
Sustaining Capital for LOM	9.3	-
LOM On-site Operating Costs	214.0	78.95

All costs are reflected in 2019 Q3/Q4 Canadian Dollars unless otherwise specified. The expected accuracy range of the cost estimates is -25%/+40%. Where applicable, costs in this report have been converted from US Dollars to Canadian Dollars using a currency exchange rate of CAD1.00:USD0.77.

21.2 CAPITAL COST ESTIMATE

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $110.1 million. A summary breakdown of the initial capital cost is provided in Table 21.2. This total includes all direct costs, indirect costs, Owner’s costs, and contingency.

Table 21.2 Initial Capital Cost Summary

Description	Initial Capital Cost ($ Million)
Overall Site	3.2
Open Pit Mining	10.4
Materials Crushing and Handling	2.0
Process	30.4
TMF	8.0
On-Site Infrastructure	5.2
External Access Roads	11.6
Project Direct Costs - Subtotal	**70.9**
Project Indirect Costs	20.8
Owner's Costs	1.3
Contingencies	17.2
Total Initial Capital Cost	**110.1**

Note: Totals may not add up due to rounding

21.2.1 Class of Estimate

This Class 4 cost estimate has been prepared in accordance with the standards of AACE International. There is no deviation from AACE International's recommended practices in the preparation of this estimate. The expected accuracy of this estimate is -25%/+40%.

21.2.2 Estimate Base Date and Validity Period

This estimate was prepared with a base date of Q3/Q4 2019 and does not include any escalation beyond this date. The quotations used for this PEA estimate were obtained in Q3/Q4 2019 and have a validity period of 90 calendar days or less.

21.2.3 Estimate Approach

Currency and Foreign Exchange

The capital cost estimate uses Canadian Dollars as the base currency. Where applicable, quotations received from vendors were converted to Canadian Dollars using a currency exchange rate of CAD1.00:USD0.77. There are no provisions for foreign exchange fluctuations.

Duties and Taxes

Duties and taxes are not included in the estimate.

Measurement System

The International System of Units (SI) is used in this estimate.

Work Breakdown Structure

The estimate is organized according to the following hierarchical work breakdown structure (WBS):

- Level 1 = Major Area
- Level 2 = Area
- Level 3 = Sub-Area.

21.2.4 Elements of Cost

This capital cost estimate consists of the four main parts: direct costs, indirect costs, Owner's costs, and contingency.

Direct Costs

AACE International defines direct costs as:

> *...costs of completing work that are directly attributable to its performance and are necessary for its completion. In construction, (it is considered to be) the cost of installed equipment, material, labor and supervision directly or immediately involved in the physical construction of the permanent facility.*

Examples of direct costs include mining equipment, process equipment and permanent buildings.

The total direct cost for the Project is estimated to be $70.9 million.

Indirect Costs

AACE International defines indirect costs as:

> *...costs not directly attributable to the completion of an activity, which are typically allocated or spread across all activities on a predetermined basis. In construction, (field) indirects are costs which do not become a final part of the installation, but which are required for the orderly completion of the installation and may include, but are not limited to, field administration, direct supervision, capital tools, start-up costs, contractor's fees, insurance, taxes, etc.*

The total indirect cost for the Project is estimated to be $20.8 million.

Owner's Costs

Owner's costs are costs provided by the Owner to support and execute the Project.

The Project execution strategy, in particular for construction management, involves the Owner working with an engineering, procurement, and construction management (EPCM) organization and supervising the general contractor(s). The Owner's costs include home office staffing, home office travel, home office general expenses, field staffing, field travel, general field expenses, community relations, and Owner's contingency.

The total Owner's cost allowance for the Project is estimated to be $1.3 million.

CONTINGENCY

Tetra Tech estimated a contingency for each activity or discipline based on the level of engineering effort as well as experience on past projects.

The total contingency allowance for the Project is $17.2 million.

21.2.5 CAPITAL COST EXCLUSIONS

The following items have been excluded from this capital cost estimate:

- working or deferred capital (included in the financial model)
- financing costs
- refundable taxes and duties
- land acquisition (marshalling yard and a satellite office in Mayo)
- currency fluctuations
- lost time due to severe weather conditions
- lost time due to force majeure
- additional costs for accelerated or decelerated deliveries of equipment, materials, or services resultant from a change in project schedule
- warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares
- any project sunk costs (studies, exploration programs, etc.)
- mine reclamation costs (included in the financial model)
- mine closure costs (included in the financial model)
- escalation costs.

21.2.6 SALVAGE

Since the mine life is shorter than other comparable projects and typical equipment life, it is expected that most of the process equipment will be salvageable by the end of the seven years mine life. The estimated salvage values are summarized in Table 21.3.

Table 21.3 Salvage Value

Area	Salvage Value ($ Million)
Primary Crushing	0.3
Grinding	1.5
Leaching	1.9
ADR	1.0
Reagent Systems	0.3
Metallurgical Laboratory	0.5
Control System	0.1
Gensets (Lease to Own)	1.4
Total	**7.0**

21.3 OPERATING COST ESTIMATE

21.3.1 SUMMARY

On average, the LOM on-site operating costs for the Project were estimated to be $78.95/t of material processed. The operating costs are defined as the direct operating costs including mining, processing, site servicing, and G&A costs, including related freight costs. Table 21.4 and Figure 21.1 show the cost breakdown for various areas.

The cost estimates in this section are based upon the consumable prices and labour salaries/wages in Q3/Q4 2019 or based on the information from the database of the consulting firms involved in the cost estimates. Where applicable, costs in this estimate have been converted from US Dollars to Canadian Dollars using the currency exchange rate of CAD1.00:USD0.77. The expected accuracy range of the operating cost estimate is -25%/+40%.

Table 21.4 LOM Average Operating Cost Summary

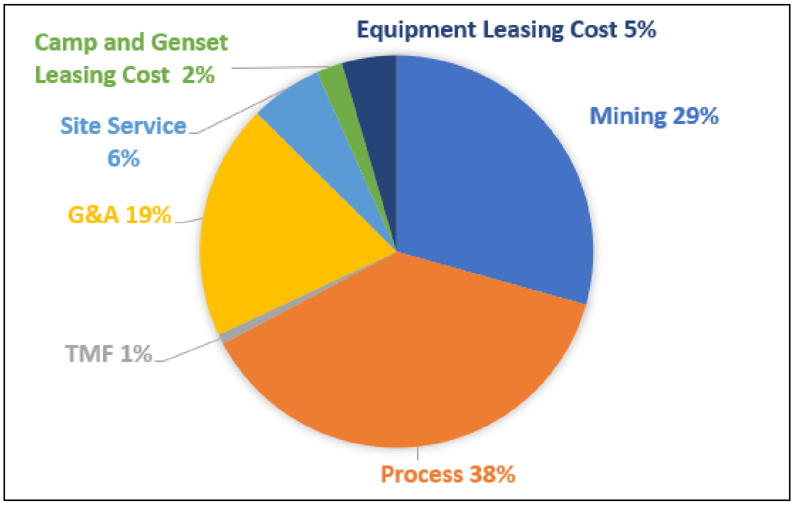
Area	LOM Average Operating Cost ($/t milled)
Mining*	23.18
Process	29.88
TMF	0.64
G&A	15.33
Site Service	4.68
Camp and Genset Leasing Cost	1.68
Equipment Leasing Cost**	3.55
Total Operating Cost	**78.94**

* mining unit operating cost is estimated to $4.28/t mined or $23.18/t milled
** Includes mining equipment leasing cost.

The G&A cost estimate includes off-site operating expenditures for a satellite office in Mayo, Yukon.

The operating costs exclude shipping and refining charges for gold doré, which are included in financial analysis.

Figure 21.1 Operating Cost Distribution



21.3.2 Mining

The operating costs were estimated from equipment productivity calculations, and more generally from "Mine and Mill Equipment Costs – An Estimator's Guide 2019". The annual equipment utilization hours were derived from calculated available hours less estimated operating delays, and then applied to the hourly equipment costs to calculate direct mining operating costs.

Labour

Annual labour operating costs were calculated using the yearly cost per labour category equal to an average of salaries from similar mining studies. The yearly cost of each labour category includes a base salary and a 35% and 45% benefit package for salaried staff and hourly operators, respectively.

Blasting Services

The mine will contract out blasting services, including the supply of a mix truck and trained personnel to carry out the delivery of the explosive mix to the drillholes and blasting operation. Based on Golder (2016), blasting will be performed only on non-oxide rock, while oxide material will be excavated directly by the hydraulic excavator.

Mining Operating Cost Summary

Table 21.5 summarizes the LOM mining operating costs per activity.

Table 21.5 Mining Operating Cost Summary

Description	LOM Average Operating Cost ($/t mined)
Drilling & Blasting	1.50
Loading	1.07
Hauling	1.07
Mine Maintenance	0.30
Technical Services	0.13
Mine General	0.21
Total Mining Operating Cost	**4.28**

21.3.3 PROCESSING

PROCESS OPERATING COSTS

The average LOM unit process operating cost was estimated to be $29.88/t milled, or $27.70/t at a nominal processing rate of 1,500 t/d, or 547,500 t/a, including the power cost for the processing plant. The estimate is based on 12-hour shifts, 24 h/d, and 365 d/a.

The estimated process operating cost at the nominal processing rate of 1,500 t/d is summarized in Table 21.6.

Table 21.6 Process Operating Cost Summary at a 1,500 t/d Processing Rate

Description	Unit Cost* ($/t milled)
Manpower (56 persons)	10.28
Metal Consumables	1.17
Reagent Consumables	5.27
Maintenance Supplies	2.11
Operating Supplies	0.48
Power Supply	8.39
Total Process Operating Cost*	**27.70**

* Average LOM unit process operating cost: $29.88/t milled

The process operating cost estimate includes:

- personnel requirements, including supervision, operation and maintenance; salary/wage levels, including burdens, based on the estimated 2019 Q3/Q4 labour rates in Yukon
- mill liner and grinding media consumption, estimated from the Bond Ball Mill Work Index and Abrasion Index equations and Tetra Tech's experience; steel ball and mill liner prices quoted from potential suppliers.
- maintenance supplies, based on approximately 8% of major equipment capital costs or estimated based on the information from the Tetra Tech's database/experience
- reagent consumptions based on test results and reagent prices quoted from potential suppliers in Q3/Q4 2019 or Tetra Tech's database
- other operation consumables, including laboratory and service vehicles consumables
- power consumption for the processing plant based on the preliminary plant equipment load estimates and a power unit cost of $0.278/kWh; electricity is planned to be generated from gensets on site.

All operating cost estimates exclude taxes unless otherwise specified.

Personnel

The estimated average personnel cost at a nominal processing rate of 1,500 t/d is $10.28/t milled. The projected process personnel requirement is 56 persons, including:

- 8 staff for management and technical support
- 22 operators servicing for overall operations from crushing to doré melting
- 26 personnel for equipment maintenance and personnel at laboratories for quality control, process optimization, and assaying.

The salaries and wages, including burdens, are based on the estimated 2019 Q3/Q4 labour rates in Yukon. The benefit burdens for the workers includes Registered Retirement Savings Plans (RRSPs), various life and accident insurances, extended medical benefits, Canada Pension Plan (CPP), Employment Insurance (EI), Workers' Compensation Board (WCB) insurance, tool allowance, and other benefits.

Consumables and Maintenance/Operation Supplies

The operating costs for major consumables and maintenance/operation supplies were estimated at $9.03/t milled, excluding the costs associated with doré off-site shipment and refining. The costs for major consumables, which include metal and reagent consumables, were estimated to be $6.44/t milled. Most of the consumable prices were quoted from potential suppliers in 2019 Q3/Q4.

The cost for maintenance/operation supplies was estimated at $2.59/t milled. Maintenance supplies were estimated based on approximately 8% of major equipment capital costs and/or based on the information from the Tetra Tech's database/experience.

Power

The total process power cost was estimated to be $8.39/t milled. The electricity will be supplied by gensets on site. The power unit cost used in the estimate was $0.278/kWh.

The power consumption was estimated from the preliminary power loads estimated from major process equipment load list. The average annual power consumption was estimated to be approximately 16.5 GWh.

21.3.4 Tailings Management Facility

The TMF operating cost was estimated by Knight Piésold. The average LOM operating costs for tailings management were estimated to be approximately $0.64/t milled. The operating costs include expenditures for tailings distribution management, reclaim water and basin drain pumping, and pipeline maintenance. TMF embankment raising was excluded from the operating cost estimates, as the related cost was treated as a sustaining capital cost.

21.3.5 General and Administrative and Site Services

G&A and site service costs include the expenditures that do not relate directly to the mining or process operating costs. These LOM average costs were estimated to be $15.33/t milled for G&A and $4.68/t for site services based on a nominal mill feed processing rate of 1,500 t/d. The costs include expenditures for a satellite office at Mayo, Yukon. The G&A and site service costs include:

- personnel – general manager and staffing in accounting, purchasing, environmental, security, site maintenances and other G&A departments. The estimated total employee numbers are 18 for G&A and 12 for site services, including the personnel at a satellite office at Mayo, Yukon.

The salaries and wages are based on the 2019 Q3/Q4 labour rates in Yukon, including base salary or wage and related burdens, including RRSPs, various life and accident insurances, extended medical benefits, CPP, EI, WCB insurance, tool allowance, and other benefits.

- general expenses – general administration, contractor services, insurance, security, medical services, legal services, human resources, travel, camp services, workers' transportation, communication services/supports, external assay/testing, overall site maintenance, electricity and fuel supplies, engineering consulting, and sustainability, including an environment and community liaison.

A summary of the LOM G&A and site service cost estimates is shown in Table 21.7. The average LOM costs for management and service personnel were estimated to be $4.02/t milled for G&A and $2.48/t milled for the site services. On average, the estimated other LOM costs for G&A and site services are $11.31/t milled and $2.20/t milled, respectively. Outside of the salary/wage costs, camp services and worker transportation costs are the major cost components and were estimated to be approximately $3.5 million per year.

Table 21.7 LOM G&A and Site Service Cost Estimates

Description	Manpower	Annual Cost ($ Million/a)	Unit Cost ($/t milled)
G&A			
Labour	18	1.82	4.02
Other Costs	-	5.11	11.31
Subtotal	**18**	**6.93**	**15.33**
Site Services			
Labour	12	1.12	2.48
Other Costs	-	1.00	2.20
Subtotal	**12**	**2.12**	**4.68**

21.3.6 Leasing Cost

A preliminary trade-off analysis was conducted to compare leasing versus purchasing the camp and gensets for the Project. The comparison appears to show potential economic benefits for the leasing option. A similar analysis was conducted for mining equipment leasing. The leasing option also appears a more attractive option than purchasing, attributable by the relatively short life of mine. The estimated leasing costs are shown in the LOM operating costs summary in Table 21.4.

22.0 ECONOMIC ANALYSIS

A PEA should not be considered to be a prefeasibility or feasibility study, as the economics and technical viability of the Project have not been demonstrated at this time. A PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results as reported in the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

An economic evaluation was prepared for the Project based on a pre-tax financial model. The following pre-tax financial parameters were calculated using the base case gold price of USD1,400/oz and an exchange rate of USD0.77:CAD1.00 (all currency units are Canadian dollars unless otherwise specified):

- 54.5% IRR
- 1.24-year payback on $110.1 million initial capital
- $118.2 million NPV at a 5% discount rate.

ATAC commissioned PwC in Vancouver, British Columbia to prepare a tax model for the post-tax economic evaluation of the Project with the inclusion of applicable income and mining taxes (see Section 22.4 for further details).

The following post-tax financial results were calculated:

- 42.6% IRR
- 1.40 year payback on $110.1 million initial capital
- $85.4 million NPV at a 5% discount rate.

Sensitivity analyses were conducted to analyze the sensitivity of the Project merit measures (NPV, IRR, and payback periods) to the main inputs, and can be found in Section 22.3.

22.1 PRE-TAX FINANCIAL ANALYSIS

22.1.1 MINE/METAL PRODUCTION IN FINANCIAL MODEL

The life-of-project average material tonnages, grade, and gold production are shown in Table 22.1.

Table 22.1 Mine/Metal Production from the Tiger Gold Project

Item	Unit	Value
Mill Feed		
Total LOM Oxide Tonnes to Mill	kt	1,658
Total LOM Sulphide Tonnes to Mill	kt	1,052
Average Annual Oxide Tonnes to Mill	kt	276
Average Annual Sulphide Tonnes to Mill	kt	175
Average Gold Head Grade		
Oxide Head Grade	g/t	4.0
Sulphide Head Grade	g/t	3.5
Gold Production		
Total LOM	k oz	267
Average Annual Production	k oz	44.5

22.1.2 Basis of Financial Evaluation

The production schedule was incorporated into the 100% equity pre-tax financial model to develop the annual recovered gold production from the relationships between tonnage processed, head grades, and recoveries.

Payable gold values were calculated using the base case gold price and exchange rate. The net invoice value was calculated for each year by subtracting the applicable refining charges from the payable metal value. The at-mine revenues were then estimated by subtracting the transportation and insurance costs. Operating costs for mining, processing, surface services and G&A were deducted from the revenues to derive the operating cash flows.

Initial and sustaining capital costs, as well as working capital, were incorporated over the LOM. Capital expenditures were then deducted from the operating cash flow to determine the net cash flow before taxes.

Initial capital expenditures include costs accumulated prior to the first production of gold; sustaining capital includes expenditures for mining and processing additions, replacement of equipment, and tailings embankment construction.

The pre-production construction period is assumed to be one year. The NPV of the Project was calculated at the beginning of this one-year period.

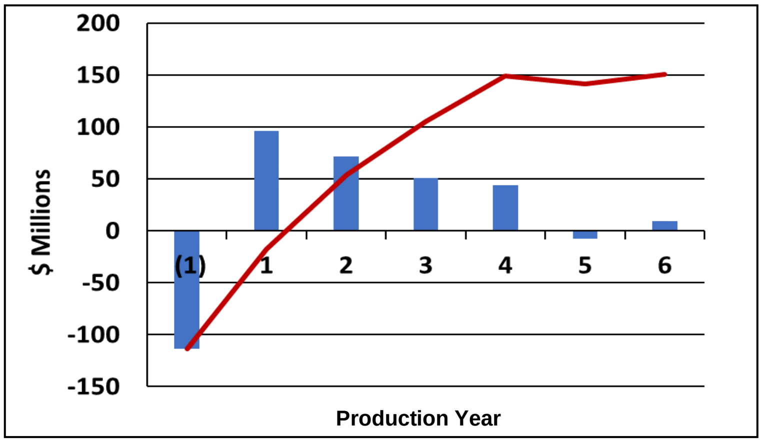
Initial and sustaining capital costs over the LOM were estimated at $110.1 million and $9.3 million, respectively.

Mine closure and reclamation costs were estimated at $4.4 million with costs bonded at 50% in Year -1, 25% in Year 2, and 25% in Year 3.

As indicated in Section 21.0, salvage value of the processing plant was estimated at $7.0 million.

The pre-tax undiscounted annual net cash flow (NCF) and cumulative net cash flow (CNCF) are illustrated in Figure 22.1.

Figure 22.1 Pre-tax Undiscounted Annual and Cumulative Net Cash Flows



Note: Blue bars indicate pre-tax undiscounted annual cash flows. Red line indicates cumulative net cash flows.

22.2 SUMMARY OF FINANCIAL RESULTS

The pre-tax financial model was established on a 100% equity basis, excluding debt financing and loan interest charges. The pre-tax financial results for the base case are presented in Table 22.2.

Table 22.2 Summary of Pre-tax Financial Results

Item	Units	Value
Metal Prices and Exchange Rate		
Gold Price	USD/oz	1,400
	CAD/oz	1,818
Exchange Rate	USD/CAD	0.77
Mine/Mill Production		
Waste	Mt	14.4
Mineralized Material	Mt	2.7
Stripping Ratio	-	5.3
Milling Period	years	6
Gold Grade, Oxide	g/t	4.0
Gold Grade, Sulphide	g/t	3.5
Contained Gold	k oz	331
Recovered Gold	k oz	267
Recovered Gold Value	$ Million	486
Operating Costs		
On-site		
Mining Cost	$ Million	62.8
Process Cost	$ Million	81.0
G&A Cost	$ Million	41.6
Surface Services Cost	$ Million	12.7
Tailings Management Cost	$ Million	1.7
Equipment & Camp/Genset Leasing	$ Million	14.2
Total On-site Operating Costs	$ Million	214.0
Off-site		
Deductions	$ Million	1.6
Refining/Smelting	$ Million	0.3
Doré Transportation and Insurance	$ Million	1.3
Royalty	$ Million	–
Total Off-site Operating Costs	$ Million	3.2
Operating Cash Flows	$ Million	267.6
Capital Costs		
Initial Capital	$ Million	110.1
Sustaining Capital	$ Million	9.3
Salvage value	$ Million	-7.0
Closure/Environmental Bonding	$ Million	4.4
Total Capital Cost	$ Million	116.8
Cash/Total Cost Net of Gold Credit		
Cash Cost (LOM)	$/oz Au recovered	801
Capital Cost (LOM)	$/oz Au recovered	437
Total Cost	$/oz Au recovered	1,239
All-In Sustaining Cost (AISC)	USD/oz Au recovered	661

table continues...

Item	Units	Value
Pre-tax Financial results		
NCF	$ Million	150.8
Discounted Cash Flow NPV @ 3%	$ Million	130.3
Discounted Cash Flow NPV @ 5%	$ Million	118.2
Discounted Cash Flow NPV @ 8%	$ Million	102.1
Payback	years	1.40
IRR	%	54.4%

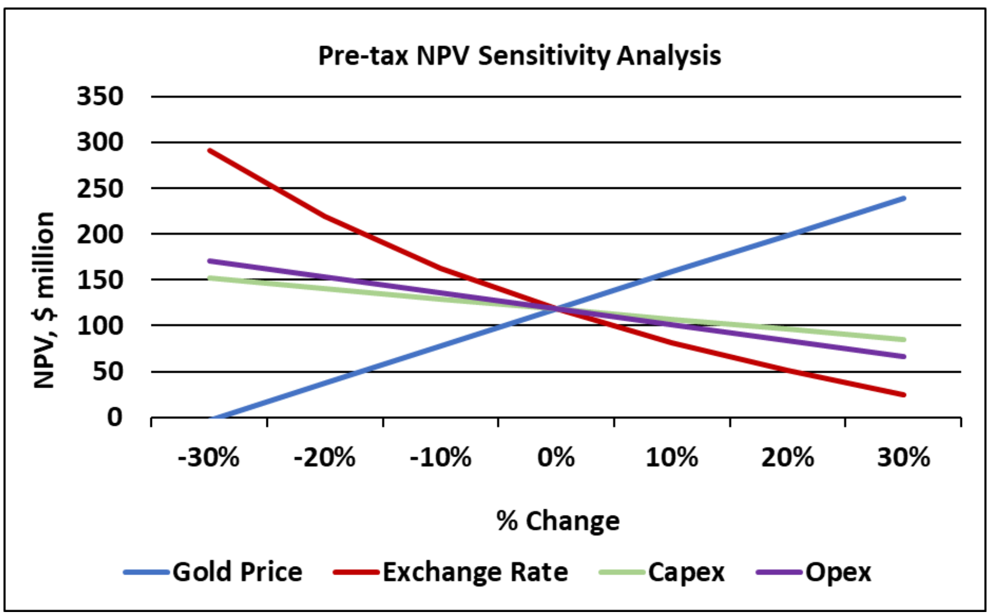
22.3 SENSITIVITY ANALYSES

Tetra Tech investigated the sensitivity of NPV, IRR, and payback period to the key project variables. Using the base case as a reference, each of the key variables was changed between −30% and +30% at 10% intervals, while maintaining the other variables constant. The following key variables were investigated:

- gold price
- exchange rate
- capital costs
- on-site operating costs.

The Project's pre-tax NPV, calculated at a 5% discount rate, is most sensitive to exchange rate and gold price followed by on-site operating costs and capital costs, as shown in Figure 22.2.

Figure 22.2 Pre-tax NPV Sensitivity Analysis



As shown in Figure 22.3, the Project's pre-tax IRR is most sensitive to exchange rate and gold price followed by capital costs and on-site operating costs.

Figure 22.3 Pre-tax IRR Sensitivity Analysis

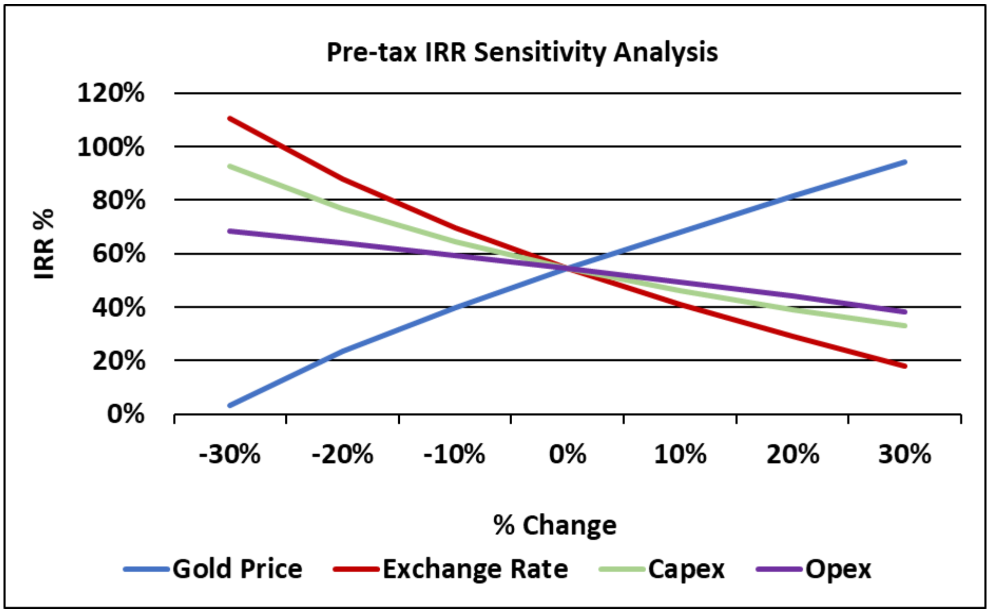
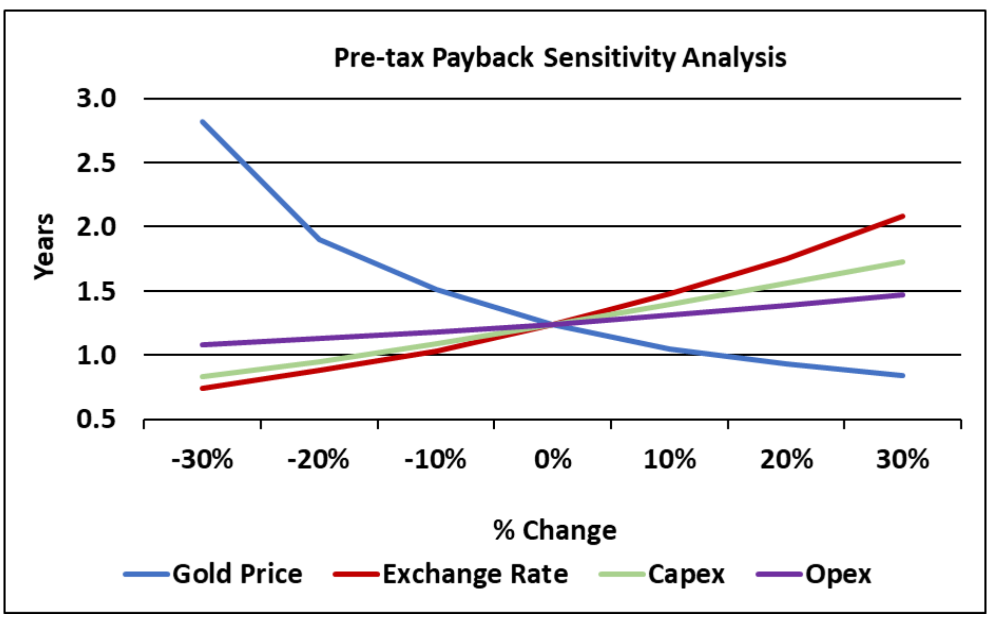


Figure 22.4 shows that the payback period is most sensitive to gold price followed by exchange rate, capital costs and on-site operating costs.

Figure 22.4 Pre-tax Payback Period Sensitivity Analysis



Since the project economics are most sensitive to gold price and exchange rate, the following tables demonstrate the sensitivity of the Tiger Deposit pre-tax economics to changes to the price of gold and exchange rates.

Table 22.3 Summary of Gold Price Sensitivity (USD0.77:CAD1.00)

	Gold Price (USD/oz)						
	1,250	1,300	1,350	1,400	1,450	1,500	1,550
Pre-tax NCF ($ million)	99.0	116.3	133.5	150.8	168.0	185.3	202.6
Pre-tax NPV @ 5% discount rate ($ million)	74.9	89.4	103.8	118.2	132.6	147.0	161.4
Pre-tax IRR (%)	38.7%	44.1%	49.4%	54.5%	59.4%	64.3%	69.2%

Table 22.4 Summary of Exchange Rate Sensitivity ($1,400/oz Gold)

	Exchange Rate (USD:CAD1.00)				
	0.75	0.76	0.77	0.78	0.79
Pre-tax NCF ($ million)	163.7	157.2	150.8	144.6	138.6
Pre-tax NPV @ 5% discount rate ($ million)	129.0	123.5	118.2	113.0	108.0
Pre-tax IRR (%)	58.2%	56.3%	54.5%	52.6%	50.9%

22.4 Post-tax Financial Analysis

ATAC Resources Ltd. ("the Company) engaged PwC in Vancouver, BC to review the tax model for the post-tax economic evaluation of the project with the inclusion of applicable income and mining taxes. The tax model covers years from 2020 to 2026 (7 year period), using tax pool balances from filed 2018 T2 Corporation Income Tax Return and the Company's 2019 financial statement tax provision workpapers. Production year is expected to begin in 2021.

The following general tax regime was recognized as applicable at the time of report writing:

22.4.1 Canadian Federal and Yukon Territorial Income Tax Regime

Federal and Yukon territorial income taxes are calculated using the currently enacted corporate rates of 15% for federal and 12% for Yukon.

For both federal and territorial income tax purposes, capital expenditures are accumulated in pools that can be deducted against mine income at different rates, depending on the type and timing of the capital expenditures.

Costs that are incurred to determine the existence, location, extent, or quality of a mineral resource in Canada are accumulated in the Canadian Exploration Expense (CEE) pool. CEE is fully deductible in the year in which it is incurred and unused balances may be carried forward indefinitely.

Resource property acquisition costs and the costs of mine shafts, main haulage ways and other underground workings as well as pre-production mine development expenses are accumulated in the Canadian Development Expense (CDE) pool. CDE is amortized against income at 30% on a declining balance basis.

Fixed assets purchased before the commencement of production are accumulated in Class 41(a), while purchases made after the commencement of production are accumulated in Class 41(b). Both Class 41(a) and (b) are amortized at 25% on a declining balance basis

22.4.2 RECENT CHANGES TO CANADIAN TAX RULES

ACCELERATED TAX DEPRECIATION

Mining assets (Class 41) purchased after November 20, 2018 that are available for use before 2028 will generally be eligible for first year CCA of 37.5%, which consists of the standard CCA rate of 25% multiplied by 1.5 (with no half-year rule). A phase out period will begin for property that becomes available for use after 2023.

ACCELERATED AMORTIZATION OF RESOURCE PROPERTY

CDE expenditures are also eligible for an enhanced first year deduction. The maximum CDE deduction under the old rules is generally 30% of the cumulative CDE pool. The new rules allow taxpayers to deduct an additional amount in connection with net accelerated CDE (“ACDE”) additions in the year. The additional deduction is 15% for expenses that are incurred after November 20, 2018 and before 2024 and reduced to 7.5% for expenses incurred after 2023 and before 2028. Note that the additional deduction is only available in connection with certain eligible CDE additions in the year, not the cumulative pool.

The amount eligible for the additional deduction is “net” ACDE additions, which is calculated based on the net amount of current CDE addition less regular 30% of regular CDE deduction.

22.4.3 YUKON MINERAL TAX REGIME

Under the Yukon Quartz Mining Act, a mining royalty applies to all ore, minerals, or mineral-bearing substances mined in the Yukon in a calendar year, based on the following schedule (refer to Table 22.5).

Table 22.5 Yukon Mineral Tax Schedule

	From ($)	To ($)	Rate (%)
Mine Profit	0	10,000	0
	10,000	1,000,000	3
	1,000,000	5,000,000	5
	5,000,000	10,000,000	6
	10,000,000	15,000,000	7
	15,000,000	20,000,000	8
	20,000,000	25,000,000	9
	25,000,000	30,000,000	10
	30,000,000	35,000,000	11
	35,000,000	>35,000,000	12

Mine profit for the purpose of computing the royalty is generally based on the value of the mine's output (i.e.: proceeds from the sale of minerals) less eligible deductions, such as mine operating expenses, a development allowance, and a depreciation allowance, as well as a community and economic development allowance (CEDEA).

Pre-production exploration and mine development costs are pooled and, once production has commenced, are deducted through the development allowance on an estimated unit of production basis over the life of the mine.

Tangible assets (other than land) acquired before and after the commencement of production are pooled and are deducted through the depreciation allowance on a straight-line basis at 15% annually.

Eligible mining expenditures in community and economic developments are pooled and deducted at the lesser of 15% of the deductions and development and depreciation allowance, or 20% of the value of the mine's output after all deductions. In order to qualify for CEDEA, an expense must be approved by Natural Resources Canada.

22.4.4 Taxes and Post-tax Results

At the base case gold price and exchange rate used for this study, the total estimated taxes payable on the Project profits are $38.74 million over the six-year production life. The components of the various taxes that will be payable for the base case are shown in Table 22.6.

Table 22.6 Components of the Various Taxes

Item	Value
Gold Price (USD/oz)	1,400
Exchange Rate (USD:CAD)	0.77:1.00
Income Tax Payable ($ million)	25.08
Yukon Mining Tax Payable ($ million)	11.67
Total Taxes ($ million)	**36.75**

The base case post-tax financial results are summarized in Table 22.7.

Table 22.7 Summary of Post-tax Financial Results

Description	Base Case
Gold Price (USD/t)	1,400
Exchange Rate (USD:CAD)	0.77:1.00
NCF ($ million)	112.05
Discounted Cash Flow NPV ($ million) at 3%	72.25
Discounted Cash Flow NPV ($ million) at 5%	85.36
Discounted Cash Flow NPV ($ million) at 8%	95.23
Payback (years)	2.4
IRR (%)	42.6

22.5 Royalties

As advised by ATAC, no private royalties are applicable to the Project.

22.6 Smelter Terms

The following refining terms were applied in the financial analysis:

- percentage payment: 99.5% of delivered gold
- refining charge: $1.00/oz of payable gold

22.7 Transportation and Insurance

The following transportation and insurance cost has been applied in the financial model:

- $5.00/oz of delivered gold from mine site to refinery.

23.0 ADJACENT PROPERTIES

The Property forms the western part of ATAC's 1,758 km^2 Rackla Gold Project, while the Orion and Osiris properties, also known as the Nadaleen Trend, form the eastern part. Exploration within the Nadaleen Trend has focused on the 12 km^2 Osiris and 18 km^2 Anubis clusters. Mineralization within the Nadaleen Trend has been categorized as Carlin-type gold. Although the Property and Nadaleen Trend form a contiguous claim group, known mineralization in the two trends are separated by approximately 100 km. Given this large distance and the different styles of mineralization identified, it is unlikely that common infrastructure would be utilized to develop mineral deposits in both trends.

A number of other advanced projects are located within 10 km of the Property, or the proposed access route to the Tiger Deposit. Although these deposit types are not representative of mineralization found on the Property, their proximity to the Property and proposed infrastructure is important to recognize.

The Blende Deposit, owned by Blind Creek Resources Ltd., is located 28 km northwest of the Tiger Deposit, 10 km north of the Property. Lead-zinc-silver mineralization is hosted by upper Gillespie Lake Group dolostone and is spatially associated with a Middle Proterozoic fault zone (Price 2011). Access to the Blende Deposit is by winter trail, via the Wind River Trail or by helicopter.

The Marg Deposit, a copper-lead-zinc-silver bearing volcanogenic hosted massive sulphide deposit located 33 km southwest of the Tiger Deposit, is owned by Golden Predator Corp. It is located 6 km from the proposed Tiger tote-road route. Access to the Marg deposit is by winter trail, from Keno City, or by light aircraft to an airstrip on the property.

Alexco Resources owns the historic Keno Hill Silver District, 50 km southwest of the Tiger Deposit. The Bellekeno silver mine commenced commercial production in 2011 and was placed in interim suspension in 2013 due to low metal prices. Access to the Keno Hill Silver District is via the all-season Silver Trail, a part of the Yukon highway system.

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information to complete this report.

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 MINERAL RESOURCE ESTIMATE

The Mineral Resource estimate was completed using 6,861 assays taken from 166 diamond drillholes, totalling 28,610 m. The modeling and estimation of the mineral resources were completed on January 3, 2020 under the supervision of Mr. Steven J. Ristorcelli, a qualified person with respect to mineral resource estimations under NI 43-101. The effective date of this Mineral Resource estimate is January 3, 2020. The Effective Date of the Tiger database on which this Resource estimate is based is November 23, 2019.

Rock units and gold and tungsten domains were explicitly modelled on cross-sections and long-sections. This produced pseudo-solids which were used to code the model. Gold and tungsten domains were defined based on population breaks on cumulative probability plots and grade changes in the drill-hole samples. Capping of gold and tungsten was conducted within each domain separately based on a review of detail statistics. Samples were composited to 3m intervals down-hole honoring domain boundaries.

Inverse distance estimation with a power of three was used to estimate grade. The search parameters were based on variography.

The block model was classified into potential open pit and potential underground material based on technical and economic factors chosen to meet the “reasonable prospects for eventual economic extraction” test.

Mineral Resources are reported at a 0.75 g Au/t cut-off for open pit material and a 1.5 g Au/t cut-off in underground material, as shown in Table 25.1. These cut-off grades were selected based on a review of economic parameters for similar deposits.

Table 25.1 Combined Oxide and Sulphide Resources by Mining Method

Type	Classification	Cutoff g Au/t	Tonnes	g Au/t	oz Au	ppm W	Tonnes W
			Open Pit				
Oxide	Indicated	0.75	1,980,000	3.74	238,000	282	559
Sulfide	Measured	0.75	799,000	2.92	75,000	171	137
Sulfide	Indicated	0.75	847,000	2.68	73,000	164	139
Ox + S	M+I	0.75	3,626,000	3.31	386,000	230	835
			Underground				
Oxide	Indicated	1.50	165,000	3.09	16,000	253	42
Sulfide	Measured	1.50	29,000	2.06	2,000	188	5
Sulfide	Indicated	1.50	706,000	2.64	60,000	167	118
Ox + S	M+I	1.50	900,000	2.70	78,000	183	165
			Open Pit + Underground				
Ox + S	**M+I**	**Variable**	**4,526,000**	**3.19**	**464,000**	**221**	**1,000**
			Open Pit				
Oxide	Inferred	0.75	20,000	1.54	1,000	139	3
Sulfide	Inferred	0.75	7,000	2.41	500	123	1
Ox + S	Inferred	0.75	27,000	1.73	1,500	*135	4
			Underground				
Oxide	Inferred	1.50	41,000	2.62	3,000	112	5
Sulfide	Inferred	1.50	97,000	2.26	7,000	94	9
Ox + S	Inferred	1.50	138,000	*2.37	10,000	101	14
			Open Pit + Underground				
Ox + S	**Inferred**	**Variable**	**165,000**	**2.17**	**11,500**	**109**	**18**

* These two grades are calculated slightly differently than all other grades in the resource tables. They are calculated based on full-precision tonnages, whereas all other grades are based on rounded tonnages. This was done to remove what looked like an inconsistency resulting from rounding small tonnages.

ATAC has shown that important gold resources, with some accompanying potentially economically viable tungsten, exist at Tiger. MDA finds that the risks at Tiger are minimal for several reasons. First, the drilling density is tight. Second, while modeling domains section to section, the predictability of the mineralization was found to be good and in some cases very good.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

25.2 MINERAL PROCESSING AND METALLURGICAL TESTING

Since 2010, metallurgical test programs have been conducted on various oxide and sulphide samples from the Project. Test work results show that oxide mineralization usually responds well to gold cyanidation at a wide range of grind and even fine crushing sizes. Variability testing using the bottle roll test procedure on ground samples yielded average gold extractions of 88%, ranging from 77 to 97%. A single Bond Ball Mill Work Index test suggested the oxide material is very soft (BWi = 8.5 kWh/t). For the sulphide mineralization, it appears that gold occurs both as solid solution in sulphides and as discrete mineralization. In the samples studied, mineralogical analysis has shown that 10 to 20% of the gold was hosted in pyrite, 23 to 59% in arsenopyrite, and 28 to 67% as discrete gold. Variability tests showed that the ground samples yielded widely variable extractions, ranging from 13 to 95%. As with the oxides, primary grind size had a limited effect on extraction rates. The sulphide mineralization responded reasonably well to flotation concentration. The flotation concentrates showed a good amenability to cyanidation after being treated by pressure oxidation or bacterial oxidation although a better gold extraction of 97 to 99% was achieved with pressure oxidation, compared to an extraction of 92 to 93% after bacterial leach pretreatment. The economics of sulphide oxidation pre-treatment have not been investigated in detail but are likely to be marginal and perhaps driven by gold price.

Gravity concentration tests suggest that both types of mineralization contain little coarse discrete, gravity recoverable gold.

With the promising metallurgical test results from the oxide mineralization, extensive engineering studies were conducted using a combined heap leach and tank processing treatment to extract the gold from the mineralization. However, as identified by the previous work, there are numerous potential technical risks and challenges associated with the heap leach option. Therefore, a conventional circuit consisting of grinding and agitated cyanide leaching for both the sulphide and oxide resources is proposed for this study.

25.3 MINING

The open pit mine will utilize a conventional truck-and-excavator fleet. The Project's LOM is approximately seven years, including one year of pre-stripping followed by six years of mill production. Over the seven-year LOM, the pit will produce 2.7 Mt of mineralized material and 14.4 Mt of waste rock. The overall LOM average gold grade of oxide and sulphide material is 3.82 g/t. The LOM stripping ratio (defined as waste material mined divided by mineralized material mined) is 5.3.

Factors which may affect the mine plan include changes to the geotechnical parameters, gold price, exchange rate, operating costs, marketing assumptions, and metallurgical recoveries.

25.4 Recovery Methods

The 1,500 t/d processing plant will utilize conventional crushing, grinding, cyanidation by CIP, and gold recovery from loaded carbon to produce gold doré. The residue from the leach circuit will be treated by a sulphur dioxide/air process to destroy the residual WAD cyanide prior to being sent to the lined tailings storage facility.

The flowsheet and equipment that have been selected for the project have been widely used in mining industry and can be operated and maintained effectively in a cold environment.

25.5 Project Infrastructure

The proposed on-site infrastructure for the Project will include:

- a process plant
- a permanent camp
- an emergency vehicle building with vehicle maintenance shop and warehouse
- administration offices
- power generation units
- a main electrical substation and power distribution system
- potable and fire water storage and distribution system
- plant and camp sewage treatment facilities
- a laydown and container storage yard
- fuel storage and fueling station
- a TMF
- two WRMFs
- access and site roads.

25.6 Capital and Operating Costs

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $110.1 million. A summary breakdown of the initial capital cost is provided in Table 25.2. This total includes all direct costs, indirect costs, Owner's costs, and contingencies. All costs are shown in Canadian Dollars unless otherwise specified.

Table 25.2 Capital Cost Summary

Description	Initial Capital Cost ($ Million)
Overall Site	3.2
Open Pit Mining	10.4
Materials Crushing and Handling	2.0
Process	30.4
TMF	8.0
On-Site Infrastructure	5.2
External Access Roads	11.6
Project Direct Costs - Subtotal	**70.9**
Project Indirect Costs	20.8
Owner's Costs	1.3
Contingencies	17.2
Total Initial Capital Cost	**110.1**

On average, the LOM on-site operating costs for the Project were estimated to be $78.94/t of material processed. The operating costs are defined as the direct operating costs including mining, processing, surface services, G&A, equipment leasing and freight costs (Table 25.3).

Table 25.3 LOM Average Operating Cost Summary

Area	LOM Average Operating Cost ($/t milled)
Mining	23.18
Process	29.88
TMF	0.64
G&A	15.33
Site Service	4.68
Camp and Genset Leasing Cost	1.68
Equipment Leasing Cost	3.55
Total Operating Cost	**78.94**

25.7 ECONOMIC ANALYSIS

A PEA should not be considered to be a prefeasibility or feasibility study, as the economics and technical viability of the Project have not been demonstrated at this time. A PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results as reported in the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

An economic evaluation was prepared for the Project based on a pre-tax financial model. The following pre-tax financial parameters were calculated using the base case gold price of USD1,400/oz and an exchange rate of USD0.77:CAD1.00 (all currency units are Canadian dollars unless otherwise specified):

- 54.5% IRR
- 1.24-year payback on $110.1 million initial capital
- $118.2 million NPV at a 5% discount rate.

ATAC commissioned PwC in Vancouver, British Columbia to prepare a tax model for the post-tax economic evaluation of the Project with the inclusion of applicable income and mining taxes (see Section 22.4 for further details).

The following post-tax financial results were calculated:

- 42.6% IRR
- 1.40 year payback on $110.1 million initial capital
- $85.4 million NPV at a 5% discount rate.

Sensitivity analyses were conducted to analyze the sensitivity of the Project merit measures (NPV, IRR, and payback periods) to changes in gold price, exchange rate, operating costs, and capital costs. The Project's pre-tax NPV, calculated at a 5% discount rate, was found to be most sensitive to exchange rate and gold price followed by on-site operating costs and capital costs. The Project's pre-tax IRR was found to be most sensitive to exchange rate and gold price followed by capital costs and on-site operating costs. The payback period was found to be most sensitive to gold price followed by exchange rate, capital costs and on-site operating costs.

26.0 RECOMMENDATIONS

It is recommended that the Project proceed to the feasibility level of study. The total cost for future recommended work is $3.6 million; Table 26.1 shows the cost breakdown by discipline.

Table 26.1 Recommended Costs for Future Work

Area	Budget Amount ($)
Geology and Mineral Resources	1,000,000
Geotechnical and Hydrogeological	1,100,000
Mineral Processing and Metallurgical Testing	200,000
Mining	500,000
Process	80,000
Infrastructure	120,000
Environmental	600,000
Total	**3,600,000**

26.1 GEOLOGY AND MINERAL RESOURCES

Exploration at the Tiger Deposit has defined significant gold resources. In order to reduce uncertainty and improve quality of the existing resources, the following should be undertaken:

- Review and refine the geological model to better separate underlying lithology from alteration and mineralization and ensure uniform nomenclature of rock units across all drilling.
- Confirmatory drilling of PQ core holes in areas of low oxide recovery to improve confidence in grades. Approximately 500 m of drilling is recommended but may increase depending on how results match existing drill data.
- Trenching, mapping and surface sampling of exposed and near-surface oxide to better define the extent of overburden cover, and model grade to surface where reasonable.
- Collect additional samples for cyanide-soluble gold assay data to improve the quality of the cyanide solubility model in sulphide material and confirm low recovery variability in oxide material. Run cyanide solubility assays on all samples grading over 0.5 g Au/t in new drilling. Review samples in storage (both coarse

rejects and core storage) for any areas that could be re-assayed (older sulphide samples in storage may have oxidized and no longer be suitable).

- Review previous drilling for areas that were not initially sampled. Sample areas adjacent to grade and where the new geological model indicates low-grade halos may exist.

- Collect additional density data on all future drilling of all lithologies, with particular emphasis on oxide. Density samples should include more oxide samples that are dried before weight measurements, however care should be taken to dry at low temperature so as not to drive water out of hydrated mineral species. Consider utilizing commercially available downhole gamma survey tools to measure in-situ rock density.

- Obtain before and after drying weights from the assay laboratory in all future drilling to assess change in moisture content from as-drilled to as-assayed weights.

- Considering that tungsten may, in the future, contribute to the economics of the Tiger deposit, the QA/QC protocol should be modified to provide full QA/QC for tungsten. Standards should be certified for tungsten and material used for coarse blanks should be checked to ensure that it contains negligible tungsten.

- In future drilling programs a subset of sample pulps should be re-assayed at a second laboratory, as was done in 2010.

The Tiger Deposit also demonstrates potential for expansion of existing resources in areas with limited drilling. To evaluate expansion potential, the following should be undertaken:

- Step-out drilling to the northwest to test extension of high-grade oxide trends outside the existing resource. Three holes totalling approximately 400 m are recommended for initial assessment, with additional step-outs if results are promising.

- Deep step-out drilling on the east side of the deposit to test extension of sulphide horizons at depth and improve definition of the upper horizons. Five holes totalling approximately 1,000 m are recommended in this area, with refinement of targeting based on initial results.

All drilling should be conducted with oriented core tooling to capture structural information for modeling and geotechnical purposes.

Following completion of the above work, the Mineral Resource should be updated.

The cost for this proposed work is estimated to be $1,000,000.

26.2 GEOTECHNICAL

The following recommendations are made to improve the geotechnical and hydrogeological understanding of the Tiger Deposit:

- Existing VWPs should be read regularly during future field seasons. Replace any malfunctioning VWPs as needed.
- Failure type should be recorded during future point load testing.
- Additional structural orientation data is recommended on the southeast slope (Sector 6) to evaluate whether the bench face angle, and therefore the inter-ramp angle can be increased above 40° in this sector. This can be achieved by either by drilling an additional hole or by trenching and mapping.
- Geotechnical conditions and stability should be documented for any test pits excavated in the oxide zone. Of particular importance are the observed relative density of the oxide and the stability of any steep-sided test pits that might simulate bench face angles in the oxide.
- A detailed hydrogeological data collection program should be established, including installation of 8 monitoring wells with regular sampling and data collection.

The cost for the proposed work is estimated to be $1,100,000.

26.3 MINERAL PROCESSING AND METALLURGICAL TESTING

Future work on the project must take into account the size of the resource, which is best suited to a straightforward process flowsheet as has been designed for this PEA.

The greatest metallurgical risk to this project lies in the quality of the metallurgical forecasts, so the focus of future work needs to be on improving the understanding of forecasted recoveries. Oxide materials generally leach quite well but recoveries from the nine variability composites tested so far have varied from 77% to 98%. Effort needs to be expended into establishing a means of predicting gold recoveries better than it is currently done, so a more accurate metallurgical forecast can be created for the project.

Care must also be taken in future sample selection to ensure head grades are reasonably reflective of expected mill feed grades.

The same applies to the sulphide materials. Both the mineralogy data and the leach data have shown that the sulphides do not host refractory gold in a great enough abundance to warrant economic recovery using current technologies, so further work to explore ways of extracting this refractory gold component is not warranted. Instead, work needs to be done on the leach process including exploring pre-aeration and testing further the use of lead nitrate to (a) reduce cyanide consumption, and, (b) enhance gold recoveries. The greatest effort, however, needs to be devoted to developing a good system for predicting gold recoveries. Recoveries range from 11% to 98% in the current dataset and while there is a link between AuCN/AuFA ratios and leach recovery that is usable for the project at PEA level, there is a great deal of random error in this relationship. The reason for this needs to be ascertained and methods (potentially alternative AuCN assay methods) should be identified that allow for much better prediction of gold extraction. In addition, the AuCN database should be expanded to include all rock types in the sulphide zones.

Tungsten recovery to a marketable grade will also need to be demonstrated if tungsten processing is to be incorporated in the project in a future study.

The following test work is recommended:

- grindability:
  - SAG mill and ball mill grindability work index
  - abrasion index
  - grinding circuit simulations
- leach variability:
  - AuCN/AuFA parameter determination tests, and establishment of a more precise link between geochem-based gold recovery estimates and actual bottle roll test data
  - leach condition verification/optimization tests
  - lithological and spatial location variability tests.
- determining process design related parameters, such as settling tests
- gold extraction improvement:
  - further exploration of economical pretreatment methods to improve gold extraction from sulphide mineralization and project economics, possibly including selective oxidation of gold bearing arsenopyrite.

The cost for the proposed test work is estimated to be $200,000.

26.4 MINING

Tetra Tech makes the following recommendations for future mining work:

- The Project should proceed to the feasibility level. A detailed mining production schedule and design should be developed with detailed mining activities to understand the potential constraints and cost reduction opportunities.
- As the pit optimization and scheduling results are highly dependent on the geotechnical parameters, more detailed geotechnical studies and/or fieldwork should be conducted to better define the appropriate pit slope angles and design parameters for the pit, stockpile, and waste dump
- To estimate pit dewatering requirements, a hydrogeological study should be completed.

- A detailed characterization of mine waste material should be completed to enhance the waste management.
- A trade-off study between Owner-operated and contract mining is recommended. Given the short LOM, leasing the mining fleet could enhance the Project economics.

The estimated cost for the proposed mining work is approximately $500,000.

26.5 PROCESSING

According to the results of next phase test work, further optimizations on plant design and layout are recommended. The costs associated with the optimizations will be part of the costs for the next phase of study.

In addition to gold recovery, there is a potential economic value of recovering tungsten as a by-product from the scheelite skarn mineralization. A preliminary assessment shows that the recovery of the tungsten may be economically feasible. Further scheelite recovery and grade improvement testwork is recommended in Section 26.3. If tungsten recovery is demonstrated to be economical for the Project, a cost-effective recovery method should be investigated. Likely the tungsten recovery circuit can be incorporated after gold extraction. Marketing studies for the tungsten concentrate should be conducted. The cost for the marketing study and the engineering work is estimated to be $80,000.

26.6 INFRASTRUCTURE

Opportunities to reduce the construction timeline, such as early mobilization and modularization of the process plant, should be investigated during next phase of the study. The opportunity of optimizing the cash flow by expediting the construction schedule and building process plant modules offsite should be evaluated further as part of the next phase of study, with an estimated budget of $120,000.

26.7 ENVIRONMENTAL

The following works are recommended: hydrology monitoring program. hydrological investigation, geochemical characterization, site geotechnical and permafrost condition studies, continuous weather monitoring and a manage plan for mapping the path forward in conducting assessments and permitting in the future. The estimated cost for the recommended environmental work will be approximately $600,000.

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28.0 CERTIFICATES OF QUALIFIED PERSONS

28.1 HASSAN GHAFFARI, P.ENG.

I, Hassan Ghaffari, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Director of Metallurgy with Tetra Tech Inc. with a business address at Suite 1000, 10th Fl., 885 Dunsmuir St., Vancouver, BC, V6C 1N5.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Tiger Deposit, Rackla Gold Project, Yukon, Canada" with an effective date of February 27, 2020 (the "Technical Report").
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#30408). My relevant experience includes 27 years of experience in mining and plant operation, project studies, management, and engineering. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I have not conducted a personal inspection of the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.1, 1.10, 1.11, 1.12, 1.14, 2.0, 3.0, 18.1, 18.2, 18.6, 18.7, 19.0, 20.0, 21.0, 24.0, 25.5, 25.6, 26.6, 26.7, 27.4, and 28.1 of the Technical Report.
- I am independent of ATAC and all its subsidiaries as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
- My prior involvement with the Property includes acting as a Qualified Person for the 2016 preliminary economic assessment titled "Preliminary Economic Assessment (PEA) NI 43-101 Technical Report on the Tiger Gold Project, Yukon Territory, Canada" with an effective date of May 31, 2016.
- I have read the Instrument and sections of the Technical Report I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 09th day of April, 2020 at Vancouver, British Columbia.

"Original document signed and sealed by
Hassan Ghaffari, P.Eng."

Hassan Ghaffari, P.Eng.
Director of Metallurgy
Tetra Tech Inc.

28.2 SURAJ (RAJ) PRIYADARSHI, P.ENG.

I, Suraj (Raj) Priyadarshi, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech Inc. with a business address at Suite 1000, 10th Fl., 885 Dunsmuir St., Vancouver, BC, V6C 1N5.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Tiger Deposit, Rackla Gold Project, Yukon, Canada" with an effective date of February 27, 2020 (the "Technical Report").
- I am a graduate of Nagpur University (B.E. Mine Engineering, 1996). I am a member in good standing of the Association of Professional Engineers and Geoscientists of Alberta. My relevant experience is mine evaluation, with more than 10 years of experience in the evaluation of mining projects, financial analysis, and mine planning and optimization. I have been involved in technical studies of several base metals, gold, coal, and aggregate mining projects in Canada and abroad. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent inspection of the Property was on October 1st, 2019.
- I am responsible for Sections 1.8, 1.13, 15.0, 16.0, 22.0, 25.3, 25.7, 26.4, 27.3, and 28.2 of the Technical Report.
- I am independent of ATAC and all its subsidiaries as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and sections of the Technical Report I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 09th day of April, 2020 at Vancouver, British Columbia.

"Original document signed and sealed by Suraj (Raj) Priyadarshi, P.Eng."

Suraj (Raj) Priyadarshi, P.Eng.
Senior Mining Engineer
Tetra Tech Inc.

28.3 MATTHEW DUMALA, P.ENG.

I, Matthew Richard Dumala, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Engineer and Partner with Archer, Cathro & Associates (1981) Limited with a business address at 1016-510 West Hastings Street., Vancouver, BC, V6B 1L8.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Tiger Deposit, Rackla Gold Project, Yukon, Canada" with an effective date of February 27, 2020 (the "Technical Report").
- I am a graduate of the University of British Columbia (BASc Geological Engineering, 2002). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#32783). I have been involved in the exploration and deposit modeling of a variety of deposit types, including epithermal veins, carbonate replacement, porphyry, skarn and volcanogenic massive sulphide, continuously since 2004. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was from April 30 to September 1, 2011.
- I am responsible for Sections 1.2, 1.3, 1.4, 1.5, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 23.0, 26.1, 26.2, 27.1, and 28.3 of the Technical Report.
- I am not independent of ATAC Resources Ltd. as defined by Section 1.5 of the Instrument.
- I have supervised and managed the various exploration projects conducted on the Property between 2008 and 2011.
- I have read the Instrument and sections of the Technical Report I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 09h day of April, 2020 at Vancouver, British Columbia.

"Original document signed and sealed by Matthew Richard Dumala, P.Eng."

Matthew Richard Dumala, P.Eng.
Senior Engineer and Partner
Archer, Cathro & Associates (1981) Limited

28.4 PETER ARTHUR RONNING, P.ENG.

I, Peter Arthur Ronning, P.Eng. of Gibsons, British Columbia, do hereby certify:

- I am a consulting geological engineer, doing business under the registered name New Caledonian Geological Consulting, with a business address at 1450 Davidson Road, Gibsons, B.C., VON 1V6.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Tiger Deposit, Rackla Gold Project, Yukon, Canada" with an effective date of February 27, 2020 (the "Technical Report").
- I am a graduate of the University of British Columbia in geological engineering, with the degree of B.A.Sc. granted in 1973. I also hold the degree of M.Sc. (applied) in geology, granted by Queen's University in Kingston, Ontario, in 1983. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#16,883).
- I have worked as a geologist and latterly as a Professional Engineer in the field of mineral exploration since 1973, in many parts of the world. Since 2006 I have participated in or conducted numerous audits of exploration data, reviews and evaluations of mining and mineral exploration project quality control and quality assurance ("QA/QC") data, including data derived from many gold deposits. I have studied QA/QC topics relating to the sampling and analysis of mineralized material independently and in formal continuing education sessions. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I have not conducted a personal inspection of the Property that is the subject of this Technical Report.
- I am responsible for Sections 12.0 and 28.4 of the Technical Report, and share joint responsibility for Section 11.0.
- I am independent of ATAC and all its subsidiaries as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
- The work described in section 12 of this report and similar work in 2017-18 on one other project owned by ATAC constitute the only involvement I have had with ATAC.
- I have read the Instrument and sections of the Technical Report I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 09th day of April, 2020 at Gibsons, British Columbia.

"Original document signed and sealed by Peter Arthur Ronning, P.Eng."

Peter Arthur Ronning, P.Eng.

New Caledonian Geological Consulting

28.5 BRUNO BORNTRAEGER, P.ENG.

I, Bruno Borntraeger, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Specialist Geotechnical Engineer, Associate with Knight Piésold Ltd. with a business address at 1400-750 West Pender Street, Vancouver, BC, V6C 1G8.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Tiger Deposit, Rackla Gold Project, Yukon, Canada" with an effective date of February 27, 2020 (the "Technical Report").
- I am a graduate of the University of British Columbia, May 1990, with a Bachelor of Applied Science in Geological Engineering. I am a member in good standing with the Association of Professional Engineers and Geoscientist of British Columbia (#20926) and Yukon. My relevant experience includes 29 years. I have been directly involved in geotechnical engineering, mine water and water management, mine development with practical experience from feasibility studies, detailed engineering construction, operations and closure. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I have not conducted a personal inspection of the Property that is the subject of this Technical Report.
- I am responsible for Sections 18.3, 18.4, 18.5, and 28.5 of the Technical Report.
- I am independent of ATAC and all its subsidiaries as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
- My prior involvement with the Property includes acting as a Qualified Person for the 2016 preliminary economic assessment titled "Preliminary Economic Assessment (PEA) NI 43-101 Technical Report on the Tiger Gold Project, Yukon Territory, Canada" with an effective date of May 31, 2016.
- I have read the Instrument and sections of the Technical Report I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 09th day of April, 2020 at Vancouver, British Columbia.

"Original document signed and sealed by Bruno Borntraeger, P.Eng."

Bruno Borntraeger, P.Eng.
Specialist Geotechnical Engineer, Associate
Knight Piésold Ltd.

28.6 JIANHUI (JOHN) HUANG, PH.D., P.ENG.

I, Jianhui (John) Huang, Ph.D., P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Metallurgist with Tetra Tech Inc. with a business address at Suite 1000, 10th Fl., 885 Dunsmuir St., Vancouver, BC, V6C 1N5.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Tiger Deposit, Rackla Gold Project, Yukon, Canada" with an effective date of February 27, 2020 (the "Technical Report").
- I am a graduate of North-East University, China (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals, China (M.Eng., 1988), and Birmingham University, United Kingdom (Ph.D., 2000). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#30898). My relevant experience includes over 30 years involvement in mineral processing for base metal ores, gold and silver ores, rare metal ores, and industrial minerals. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not conducted a personal inspection of the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.9, 17.0, 25.4, 26.5, 27.6, and 28.6 of the Technical Report.
- I am independent of ATAC and all its subsidiaries as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
- My prior involvement with the Property includes acting as a Qualified Person for the 2016 preliminary economic assessment titled "Preliminary Economic Assessment (PEA) NI 43-101 Technical Report on the Tiger Gold Project, Yukon Territory, Canada" with an effective date of May 31, 2016.
- I have read the Instrument and sections of the Technical Report I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 09th day of April, 2020 at Vancouver, British Columbia.

"Original document signed and sealed by
Jianhui (John) Huang, Ph.D., P.Eng."

Jianhui (John) Huang, Ph.D., P.Eng.
Senior Metallurgist
Tetra Tech Inc.

28.7 STEVEN J. RISTORCELLI, C.P.G.

I, Steven J. Ristorcelli, C.P.G., of Reno, Nevada, do hereby certify:

- I am a Principal Geologist of Mine Development Associates, Inc. (a Division of RESPEC), with a business address at 210 South Rock Blvd., Reno, Nevada, 89502.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Tiger Deposit, Rackla Gold Project, Yukon, Canada" with an effective date of February 27, 2020 (the "Technical Report").
- I graduated with a Bachelor of Science degree in Geology from Colorado State University in 1977 and a Master of Science degree in Geology from the University of New Mexico in 1980. I am a Certified Professional Geologist (#10257) in good standing with the American Institute of Professional Geologists. I am also registered as Professional Geologist in the state of California (#3964). I have worked as geologist for over 40 years. I have conducted exploration, definition, modeling, and estimation of sediment-hosted epithermal gold-silver deposits in the Western US and Canada. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not conducted a personal inspection of the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.6, 14.0, 25.1, 27.2, and 28.7 of the Technical Report, and share joint responsibility for Section 11.0.
- I am independent of ATAC Resources Ltd. as defined by 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 and sections of the Technical Report I am responsible for and they have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 09th day of April, 2020 at Reno, Nevada, USA

"Original document signed and sealed by
Steven J. Ristorcelli, C.P.G."

Steven J. Ristorcelli, C.P.G.
Principal Geologist
Mine Development Associates, Inc.

28.8 CHRISTOPHER JOHN MARTIN, C.ENG., MIMMM

I, Christopher John Martin, C.Eng., MIMMM, of Parksville, British Columbia, do hereby certify:

- I am a Principal Metallurgist with Blue Coast Metallurgy Ltd. with a business address at 2-1020 Herring Gull Way, Parksville, BC, V9P 1P2.
- This certificate applies to the technical report entitled “Technical Report and Preliminary Economic Assessment for the Tiger Deposit, Rackla Gold Project, Yukon, Canada” with an effective date of February 27, 2020 (the “Technical Report”).
- I am a graduate of the Camborne School of Mines, Cornwall, UK (B.Sc., 1984) and McGill University, Montreal, Quebec (M.Eng. 1988). I am a Chartered Engineer registered in good standing with the Engineering Council (#423115) and a Member of the Institute of Materials, Minerals and Mining (#46116). My relevant experience includes 10 years in the start-up and management of mineral processing operations in the gold, platinum, base metal, and silver industries, plus 23 years as a consultant both in the development and engineering of greenfield projects and support of existing operations. I am a “Qualified Person” for the purposes of National Instrument 43-101 (the “Instrument”).
- I have not conducted a personal inspection of the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.7, 13.0, 25.2, 26.3, 27.5, and 28.8 of the Technical Report.
- I am independent of ATAC and all its subsidiaries as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
- My prior involvement with the Property includes acting as a Qualified Person for the 2016 preliminary economic assessment titled “Preliminary Economic Assessment (PEA) NI 43-101 Technical Report on the Tiger Gold Project, Yukon Territory, Canada” with an effective date of May 31, 2016.
- I have read the Instrument and sections of the Technical Report I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 09th day of April, 2020 at Parksville, British Columbia.

“Original document signed by
Christopher John Martin, C.Eng., MIMMM”

Christopher John Martin, C.Eng., MIMMM
Principal Metallurgist
Blue Coast Metallurgy Ltd.