ARCHER, CATHRO & ASSOCIATES (1981) LIMITED
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TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT

for the

TIGER DEPOSIT, RACKLA GOLD PROJECT

ACX 1-234-310

Gam 1-21

Q 27-52, 59-84, 89-89, 91-109

R 11-12, 25-54, 71-103, 125-158, 183-214, 239-270, 329-416, 436, 438, 440, 442, 444, 446, 448, 450, 452, 454, 435, 437, 439, 441, 443, 445, 447, 449, 451, 453, 455-456, 490-495, 536-539, 578-581, 617-620, 652-655, 686-689, 720-723, 754-757, 784-787, 811-814, 837-840, 841-908, 934-955, 982-1003, 1030-1051, 1078-1107, 1132-1161, 1182-1211, 1220-1249, 1252-1295, 1296-1337

Rau 1-10, 12, 17-31, 33-44, 49-53, 65-69

Rau f 100

S 1-9, 11-42, 93-700, 701 -842, 843, 844-1154

NTS 105M/14 and 15; and 106D/01, 02, 03, 06, 07, 08
Latitude 63°57´N to 64°24´N; Longitude 134°10´W to 135°18´W

in the

Mayo Mining District,
Yukon Territory

prepared by

Archer, Cathro & Associates (1981) Limited

for

ATAC RESOURCES LTD.

by

M. Dumala, P.Eng.

October 2016

CONTENTS

Statement of Expenditures	1-2
Claim List	3

APPENDICES

I Physical copy of Technical Report and Preliminary Economic Assessment for the Tiger Deposit

Statement of Expenditures

April 22, 2016

Group I - 386 mineral claims (Q 27-52, 59-84, 89, 91-109, R 11-12, 25-54, 71-103, 125-158, 183-214, 239-270, 329-416, 841-860, Rau 1-10, 12, 17-31, 33-44, and 49-53),

Group K - 748 mineral claims (47 R, Rau 65, 100 and 699 S) and

Group L - 749 mineral claims (341 R, Rau 66-68 and 405 S)

Labour

R. Carne (P. Geo) 12 hours June to January at $145/hr	1,827.00
G. Downs (manager) 160 hours June to January at $113/hr	18,984.00
M. Dumala (P. Eng.) 266 1/2 hours June to January at $106/hr	29,661.45
J. Lane (geologist) 95 hours June to January at $106/hr	10,573.50
W. Schneider (office) 76 hours June to January at $92/hr	7,341.60
A. Carne (EIT) 443 1/2 hours June to January at $90/hr + bonus	42,803.25
J. Itkin (office) 73 1/4 hours June to January at $90/hr	6,922.13
J. Mariacher (office) 63 1/2 hours June to January at $90/hr	6,000.75
D. Arnold-Wallinger (office) 6 hours June to January at $85/hr	535.50
D. Jones (office) 109 1/2 hours June to January at $85/hr	9,772.88
R. Phillips (geologist) 110 hours June to January at $85/hr	9,817.50
J. Stevens (EIT) 86 hours June to January at $85/hr	7,675.50
S. Newman (office) 10 hours June to January at $66/hr	693.00
	$152,607.76

Expenses (including management fee)

Blue Coast Metallurgy	1,921.00
Blue Coast Research	11,254.24
Corebox	5,695.20
J. Gibson Environmental	3,971.95
Giroux Consultants Ltd.	10,113.50
Golder Associates	37,595.83
Innovat Mineral Process Solutions	14,464.00
Knight Piesold Consulting	11,370.12
Stantec Consulting Ltd.	2.697.86
Tetra Tech Wei Inc.	12,392.71
	$111,476.41
	$264.084.17

1883 claims = $140.25/claim

Statement of Expenditures

February 24, 2016

Group M - 331 mineral claims (ACX 1-310 and Gam 1-21)

Expenses (including management fee)

Matrix Research Ltd.	19,070.16
NELPCo Limited Partnership	**49,406.55**
	$68,476.71

Claim List

ACX 1-234	YD08251-YD08484
235-310	YD33163-YD33238
Gam 1-21	YC98437-YC98457
Q 27-52	YC92387-YC92412
59-84	YC92419-YC92444
89-89	YC92449-YC92449
91-109	YC92451-YC92469
R 11-12	YC68344-YC68345
25-54	YC68358-YC68387
71-103	YC68404-YC68436
125-158	YC68458-YC68491
183-214	YC68516-YC68547
239-270	YC68572-YC68603
329-416	YC68662-YC68749
R 436	YC68769
438	YC68771
440	YC68773
442	YC68775
444	YC68777
446	YC68779
448	YC68781
450	YC68783
452	YC68785
454	YC68787
435	YC68768
437	YC68770
439	YC68772
441	YC68774
443	YC68776
445	YC68778
447	YC68780
449	YC68782
451	YC68784
453	YC68786

R 455-456	YC68788- YC68789
490-495	YC68823-YC68828
536-539	YC68869-YC68872
578-581	YC68911-YC68914
617-620	YC68950-YC68953
652-655	YC68985-YC68988
686-689	YC69019-YC69022
720-723	YC69053-YC69056
754-757	YC69087-YC69090
784-787	YC69117-YC69120
811-814	YC69144-YC69147
837-840	YC69170-YC69173
841-908	YC69174-YC69241
934-955	YC69267-YC69288
982-1003	YC69315-YC69336
1030-1051	YC69363-YC69384
1078-1107	YC69411-YC69440
1132-1161	YC69495-YC69494
1182-1211	YC69515-YC69544
1220-1249	YC69553-YC69582
1252-1295	YC69585-YC69628
1296-1337	YC70595-YC70636
Rau 1-10	YC50268-YC50277
12-12	YC50279-YC50279
17-31	YC50284-YC50298
33-44	YC50300-YC50311
49-53	YC50316-YC50320
65-69	YC57529-YC57533
Rau f 100	YC69962
S 1-9	YC90801-YC90809
11-42	YC90811-YC90842
93-700	YC90893-YC91500
701 -842	YC91901-YC92042
843	YC92355
844-1154	YC92044-YC92354

APPENDIX I

PHYSICAL COPY OF TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT FOR THE TIGER DEPOSIT

Report to:

ATAC Resources Ltd.

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

Document No. 735-1699110200-REP-R0001-01

Report to:

ATAC RESOURCES LTD.

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

EFFECTIVE DATE: MAY 31, 2016

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TABLE OF CONTENTS

1.0 SUMMARY 1-1
1.1 INTRODUCTION 1-1
1.2 PROPERTY DESCRIPTION 1-1
1.3 HISTORY 1-1
1.4 GEOLOGICAL SETTING AND MINERALIZATION 1-3
1.5 EXPLORATION AND DRILLING 1-3
1.6 MINERAL RESOURCE ESTIMATE 1-4
1.7 MINERAL PROCESSING AND METALLURGICAL TESTING 1-5
1.7.1 OXIDE STUDIES 1-5
1.7.2 SULPHIDE STUDIES 1-5
1.7.3 PROCESS SELECTION 1-5
1.8 MINING METHODS 1-6
1.9 RECOVERY METHODS 1-7
1.10 PROJECT INFRASTRUCTURE 1-9
1.11 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT 1-11
1.12 CAPITAL AND OPERATING COST ESTIMATES 1-11
1.12.1 CAPITAL COST ESTIMATE 1-11
1.12.2 OPERATING COST ESTIMATE 1-12
1.13 ECONOMIC ANALYSIS 1-12
1.14 RECOMMENDATIONS 1-13
2.0 INTRODUCTION 2-1
2.1 QUALIFIED PERSONS 2-1
2.2 SOURCES OF INFORMATION 2-3
2.3 UNITS OF MEASUREMENT AND CURRENCY 2-3
3.0 RELIANCE ON OTHER EXPERTS 3-1
4.0 PROPERTY DESCRIPTION AND LOCATION 4-1
5.0 ACCESSIBILITY, CLIMATE, INFRASTRUCTURE, LOCAL RESOURCES AND PHYSIOGRAPHY 5-1
6.0 HISTORY 6-1
7.0 GEOLOGICAL SETTING AND MINERALIZATION 7-1
7.1 REGIONAL GEOLOGY 7-1
7.2 PROPERTY GEOLOGY 7-5
7.3 DEPOSIT SCALE LITHOLOGY 7-8

7.4 Mineralization 7-12
7.4.1 Replacement Style Gold Mineralization – Tiger Deposit 7-12
7.4.2 Scheelite Tremolite-Actinolite Skarns 7-19
7.4.3 Gold-Bearing Quartz-Boulangerite Vein Style Mineralization 7-20
7.4.4 Transported Gossans and Carbonate Replacement Lead-Zinc-Silver Mineralization 7-20

8.0 DEPOSIT TYPE 8-1

9.0 EXPLORATION 9-1
9.1 Soil Sampling 9-1
9.2 Surface Rock Sampling 9-5
9.3 Geophysics 9-6

10.0 DRILLING 10-1
10.1 Diamond Drilling Summary 10-1
10.2 Diamond Drilling Specifications 10-6
10.3 Core Logging Procedures 10-8
10.4 Density Measurements 10-9
10.5 Drill Collar and Down Hole Surveys 10-10
10.6 Oriented Core Surveys 10-10

11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY 11-1
11.1 Sample Shipment and Security 11-1
11.2 Sample Preparation and Analysis 11-2
11.3 Quality Assurance and Quality Control 11-2
11.3.1 Results of Certified Reference Materials 11-3
11.3.2 Results of Blank Samples 11-4
11.3.3 Results of Duplicates 11-5
11.3.4 Results of External Check Analysis 11-5
11.4 Data Validation 11-6
11.4.1 Database verification 11-6
11.4.2 Collar location verification 11-7
11.4.3 Assay verification 11-7
11.5 Conclusions 11-8

12.0 DATA VERIFICATION 12-1

13.0 MINERAL PROCESING AND METALLURGICAL TESTING 13-1
13.1 Sequence of Mineral Processing and Metallurgical Testing 13-1
13.2 Oxides 13-2
13.2.1 Samples Tested 13-2
13.2.1 Oxide Mineralogical Analysis 13-2
13.2.1 Comminution Test Work 13-2
13.2.2 Heap Leach Studies 13-4
13.2.3 Agitation Leach Studies 13-5
13.3 Sulphides 13-6

13.3.1 SAMPLES TESTED 13-6
13.3.2 SULPHIDE MINEROLOGY 13-8
13.3.3 GRINDABILITY STUDIES 13-9
13.3.4 CYANIDATION 13-10
13.3.5 FLOTATION 13-12
13.3.6 CONCENTRATE PRESSURE OXIDATION AND BIOLOGICAL LEACHING WITH SUBSEQUENT CYANIDATION 13-12
13.4 METALLURGICAL INTERPRETATION 13-13

14.0 MINERAL RESOURCE ESTIMATES 14-1

14.1 INTRODUCTION 14-1
14.2 DATA ANALYSIS 14-1
14.3 COMPOSITES 14-5
14.4 VARIOGRAPHY 14-6
14.5 BLOCK MODEL 14-7
14.6 BULK DENSITY 14-8
14.6.1 WEIGHT IN AIR FROM WHOLE COMPETENT CORE 14-8
14.6.2 WEIGHT IN AIR OF ENTIRE BOX OF CORE 14-10
14.6.3 WEIGHT OF CRUSHED DRY OXIDE MATERIAL 14-10
14.6.4 SINTREX GRAVILOG BHG SYSTEM 14-11
14.6.5 CONCLUSIONS 14-12
14.7 GRADE INTERPOLATION 14-12
14.8 CLASSIFICATION 14-13
14.9 BLOCK MODEL VERIFICATION 14-21
14.10 COMPARISON WITH PREVIOUS MINERAL RESOURCE ESTIMATE 14-27

15.0 MINERAL RESERVE ESTIMATES 15-1

16.0 MINING METHODS 16-1

16.1 INTRODUCTION 16-1
16.2 PIT OPTIMIZATION 16-1
16.2.1 BLOCK MODEL 16-1
16.2.2 PIT SLOPE ANGLE 16-1
16.2.3 SURFACE TOPOGRAPHY 16-4
16.2.4 PIT OPTIMIZATION PARAMETERS 16-4
16.2.5 PIT OPTIMIZATION RESULTS 16-5
16.3 MINE DESIGN 16-7
16.3.1 BENCH HEIGHT AND PIT WALL SLOPE 16-7
16.3.2 MINIMUM WORKING AREA 16-7
16.3.3 HAUL ROAD 16-7
16.3.4 PIT HYDROLOGY/DEWATERING 16-7
16.3.5 PIT DESIGN RESULTS 16-7
16.4 PRODUCTION SCHEDULE 16-8
16.5 MINE WASTE ROCK AND STOCKPILE MANAGEMENT 16-11
16.6 MINING EQUIPMENT 16-11
16.6.1 MINE EQUIPMENT FLEET 16-11

16.6.2 OPERATING HOURS 16-11
16.6.3 PRIMARY EQUIPMENT 16-12
16.6.4 SUPPORT AND ANCILLARY EQUIPMENT 16-13
16.7 MINING LABOUR 16-13

17.0 RECOVERY METHODS 17-1

17.1 MINERAL PROCESSING 17-1
17.1.1 INTRODUCTION 17-1
17.1.2 SUMMARY 17-1
17.2 PLANT DESIGN 17-4
17.2.1 MAJOR PROCESS DESIGN CRITERIA 17-4
17.3 PROCESS PLANT DESCRIPTION 17-5
17.3.1 PRIMARY CRUSHING 17-5
17.3.2 MILL FEED SURGE BIN 17-5
17.3.3 PRIMARY GRINDING, CLASSIFICATION 17-5
17.3.4 CYANIDE LEACHING AND CARBON ADSORPTION 17-6
17.3.5 CARBON STRIPPING 17-7
17.3.6 GOLD ELECTROWINNING AND REFINING 17-8
17.3.7 CARBON REACTIVATION 17-8
17.3.8 TREATMENT OF LEACH RESIDUE 17-8
17.3.9 TAILINGS MANAGEMENT 17-9
17.3.10 REAGENTS HANDLING 17-9
17.3.11 WATER SUPPLY 17-10
17.3.12 AIR SUPPLY 17-10
17.3.13 ASSAY AND METALLURGICAL LABORATORY 17-11
17.3.14 PROCESS CONTROL AND INSTRUMENTATION 17-11
17.4 YEARLY METALLURGICAL PERFORMANCE PROJECTION 17-11

18.0 PROJECT INFRASTRUCTURE 18-1

18.1 SITE LAYOUT 18-1
18.1.1 SITE ACCESS 18-3
18.1.2 PROCESS PLANT 18-3
18.2 POWER 18-6
18.3 TAILINGS MANAGEMENT 18-6
18.3.1 DESIGN CRITERIA 18-6
18.3.2 TAILINGS MANAGEMENT DESIGN 18-7
18.4 WASTE MANAGEMENT 18-12
18.5 WATER MANAGEMENT 18-13
18.6 FRESH, FIRE, AND POTABLE WATER SUPPLY, AND SEWAGE DISPOSAL 18-13
18.7 COMMUNICATION 18-14

19.0 MARKET STUDIES AND CONTRACTS 19-1

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT 20-1

20.1 ENVIRONMENTAL SETTING 20-1
20.2 BASELINE ENVIRONMENTAL STUDIES OVERVIEW 20-2

20.3 Tailings Management 20-7
20.4 Waste Rock Management 20-7
20.5 Water Management 20-8
20.6 Project Permitting Requirements 20-8
20.6.1 Current Permits 20-11
20.7 Socioeconomic, Community Engagements 20-11
20.8 Mine Closure/Reclamation Requirements and Costs 20-12
20.8.1 Conceptual Closure/Reclamation Measures and Cost Estimates 20-13
20.8.2 Closure Cost Estimate and Security Bonding Requirements 20-15

21.0 CAPITAL AND OPERATING COST ESTIMATES 21-1
21.1 Summary 21-1
21.2 Capital Cost Estimate 21-1
21.2.1 Class of Estimate 21-2
21.2.2 Estimate Base Date and Validity Period 21-2
21.2.3 Estimate Approach 21-2
21.2.4 Elements of Cost 21-3
21.2.5 Capital Cost Exclusions 21-4
21.2.6 Salvage 21-4
21.3 Operating Cost Estimate 21-5
21.3.1 Summary 21-5
21.3.2 Mining 21-6
21.3.3 Processing 21-7
21.3.4 Tailings Management Facility 21-9
21.3.5 General and Administrative and Site Services 21-9
21.3.6 Camp and Genset Leasing Cost 21-10

22.0 ECONOMIC ANALYSIS 22-1
22.1 Pre-tax Financial Analysis 22-1
22.1.1 Mine/Metal Production in Financial Model 22-1
22.1.2 Basis of Financial Evaluation 22-2
22.2 Summary of Financial Results 22-3
22.3 Sensitivity Analyses 22-5
22.4 Post-tax Financial Analysis 22-7
22.4.1 Canadian Federal and Yukon Territorial Income Tax Regime 22-7
22.4.2 Yukon Mineral Tax Regime 22-8
22.4.3 Taxes and Post-tax Results 22-9
22.5 Royalties 22-10
22.6 Smelter Terms 22-10
22.7 Transportation and Insurance 22-10

23.0 ADJACENT PROPERTIES 23-1

24.0 OTHER RELEVANT DATA AND INFORMATION 24-1

25.0 INTERPRETATIONS AND CONCLUSIONS 25-1

25.1 Mineral Resource Estimate 25-1
25.2 Mineral Processing and Metallurgical Testing 25-2
25.3 Mining 25-2
25.4 Recovery Methods 25-3
25.5 Project Infrastructure 25-3
25.6 Capital and Operating Costs 25-3
25.7 Economic Analysis 25-4

26.0 RECOMMENDATIONS 26-1

26.1 Geology and Mineral Resources 26-1
26.2 Geotechnical 26-2
26.3 Mineral Processing and Metallurgical Testing 26-2
26.4 Mining 26-3
26.5 Processing 26-3
26.6 Infrastructure 26-3
26.7 Environmental 26-3

27.0 REFERENCES 27-1

27.1 Geology 27-1
27.2 Mining 27-4
27.3 Environmental 27-4
27.4 Mineral Processing 27-5

28.0 CERTIFICATES OF QUALIFIED PERSONS 28-1

28.1 Hassan Ghaffari, P.Eng. 28-1
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28.8 Christopher John Martin, C.Eng., MIMMM 28-8

LIST OF TABLES

Table 1.1	Combined Oxides and Sulphide Resource	1-4
Table 1.2	Capital Cost Summary	1-12
Table 1.3	LOM Average Operating Cost Summary	1-12
Table 1.4	Recommended Costs for Future Work	1-14
Table 2.1	Summary of QPs	2-2
Table 4.1	Claim Data	4-3
Table 7.1	Regional Lithological Units (after Roots in Cathro, 2006)	7-5
Table 9.1	Geochemical Characteristics for Soil Samples, Rau Property	9-2
Table 10.1	Tiger Deposit Mineral Resource Database Summary	10-3
Table 11.1	QA/QC Samples by Year	11-3
Table 11.2	Recommended Values of Certified Reference Materials	11-4
Table 11.3	2015 Results of Certified Reference Material	11-4
Table 13.1	Test Work Completed for the Tiger Deposit	13-1
Table 13.2	Coarse Crush Bottle Roll Test Results	13-4
Table 13.3	Column Leach Test Work Results	13-4
Table 13.4	Oxide Variability Test Data	13-6
Table 13.5	Mineralogical Occurrence of Gold in Three Sulphide Composites vs. Leach Extraction	13-9
Table 13.6	24-hour Leach Variability Data from Testing 12 Different Sulphide Composites	13-11
Table 13.7	Results from Flotation of Four Sulphide Composites	13-12
Table 14.1	Assay Statistics from Solids	14-3
Table 14.2	Gold populations in Oxide Domain	14-4
Table 14.3	Capped Assay Statistics from Solids	14-5
Table 14.4	3 m Composite Statistics from Solid	14-6
Table 14.5	Semivariogram Parameters for Rau	14-7
Table 14.6	Average Unit Density	14-11
Table 14.7	Average Unit Density Scintrex	14-11
Table 14.8	Kriging Parameters for Tiger Zone	14-13
Table 14.9	Tiger Zone Oxide Blocks - Classified Measured	14-18
Table 14.10	Tiger Zone Oxide Blocks - Classified Indicated	14-18
Table 14.11	Tiger Zone Oxide Blocks - Classified Inferred	14-19
Table 14.12	Tiger Zone Oxide Blocks - Classified Measured plus Indicated	14-19
Table 14.13	Tiger Zone Sulphide Blocks - Classified Indicated	14-20
Table 14.14	Tiger Zone Sulphide Blocks - Classified Inferred	14-20
Table 14.15	Combined Oxides and Sulphide Resource	14-21
Table 14.16	Comparison of Mineral Resource Estimates (2011 Cut-offs)	14-28
Table 14.17	Comparison of Mineral Resource Estimates (2015 Cut-offs)	14-28
Table 16.1	Recommended Design Inter-ramp Slope Angles and Bench Configurations by Sector	16-2
Table 16.2	Pit Optimization Parameters	16-4
Table 16.3	Pit Optimization Results	16-6
Table 16.4	Pit Design Results	16-7
Table 16.5	Production Schedule	16-9
Table 16.6	Operational Delays per Shift	16-11
Table 16.7	Haulage Cycle Times	16-12
Table 16.8	Primary Equipment Requirements	16-12
Table 16.9	Support and Ancillary Equipment Requirements	16-13

Table 16.10	LOM Maximum Salaried Staff Requirement	16-14
Table 16.11	Operator and Maintenance Staff on Payroll	16-14
Table 17.1	Major Design Criteria	17-4
Table 17.2	Yearly Gold Metallurgical Performance Projections*	17-12
Table 18.1	Design Criteria Summary	18-6
Table 20.1	Environmental Studies Overview	20-3
Table 20.2	Senior Authorizations Required	20-9
Table 20.3	TMF Closure Concepts	20-12
Table 20.4	Conceptual Liability Components and Closure/Reclamation Measures	20-13
Table 21.1	Summary of Capital and Operating Costs	21-1
Table 21.2	Capital Cost Summary	21-2
Table 21.3	Salvage Value	21-5
Table 21.4	LOM Average Operating Cost Summary	21-5
Table 21.5	Relevant Consumables Prices	21-6
Table 21.6	Mining Operating Cost Summary	21-7
Table 21.7	Process Operating Cost Summary	21-8
Table 21.8	G&A and Site Service Cost Estimates	21-10
Table 22.1	Mine/Metal Production from the Tiger Gold Project	22-2
Table 22.2	Summary of Pre-tax Financial Results	22-4
Table 22.3	Summary of Gold Price Sensitivity (USD0.78:CAD1.00)	22-7
Table 22.4	Summary of Exchange Rate Sensitivity ($1,250/oz Gold)	22-7
Table 22.5	CEE and CDE (%)	22-8
Table 22.6	Percentage Allowed During the Phase-out Period (%)	22-8
Table 22.7	Yukon Mineral Tax	22-8
Table 22.8	Components of the Various Taxes	22-9
Table 22.9	Summary of Post-tax Financial Results	22-10
Table 25.1	Combined Oxides and Sulphide Resource	25-1
Table 25.2	Capital Cost Summary	25-4
Table 25.3	LOM Average Operating Cost Summary	25-4
Table 26.1	Recommended Costs for Future Work	26-1

LIST OF FIGURES

Figure 1.1	Project Location	1-2
Figure 1.2	Simplified Process Flowsheet	1-8
Figure 1.3	General Site Layout	1-10
Figure 4.1	Project Location Map	4-2
Figure 4.2	Claim Locations	4-4
Figure 4.3	Nearby Mineralization and Occurrences	4-5
Figure 5.1	Project Access	5-2
Figure 7.1	Tectonic Setting	7-2
Figure 7.2	Regional Geology	7-3
Figure 7.3	Regional Geology Legend	7-4
Figure 7.4	Local Geology	7-6
Figure 7.5	Stratigraphic Section – Tiger Zone	7-10
Figure 9.1	Gold-in-soil Geochemistry	9-3
Figure 9.2	Arsenic-in-soil Geochemistry	9-4

Figure 9.3 Modelled VTEM Anomoilies 9-7
Figure 9.4 Modelled ZTEM Anomolies 9-9
Figure 10.1 Tiger Zone Drillhole Locations 10-2
Figure 10.2 Tiger Section 10+000 10-4
Figure 10.3 Tiger Section 10+250 10-5
Figure 10.4 Grade Plan 10-7
Figure 13.1 Source of Oxide Samples Tested for Extraction by Cyanidation 13-3
Figure 13.2 Effect of Grind Size on Tails Assays 13-6
Figure 13.3 Source of Sulphide Samples for Metallurgical Testing 13-7
Figure 13.4 Photomicrograph of Tiger Sulphides 13-8
Figure 13.5 Comparative Grind Calibration Data for Tiger Deposit Sulphide and Oxide Samples 13-9
Figure 13.6 Effect of Primary Grind Size on Leach Residue Assay 13-10
Figure 13.7 Correlation between AuCN/AuFA Ratio and Bottle Roll Leach Extraction 13-11
Figure 14.1 Drillhole Location Map Showing 2015 Drillholes in Red 14-2
Figure 14.2 Isometric View Looking Northeast of the Oxide Solids in Orange, Sulphide Solids in Purple, Volcanic Solids in Green and Drillhole Traces 14-3
Figure 14.3 Lognormal Cumulative Probability Plot for Gold in Oxides 14-4
Figure 14.4 Histogram of Assay Sample Lengths 14-6
Figure 14.5 View Looking Northeast Showing Oxide Solids in Orange, Sulphide Solids in Purple, Volcanic Solids in Green, Topography in Grey and Overburden in
Brown 14-8
Figure 14.6 Scatterplot Comparing Density to Specific Gravity in Sulphide Samples 14-9
Figure 14.7 Scatterplot Showing ATAC Measured Density Compared to Scintrex Density 14-12
Figure 14.8 Isometric Views Looking Northeast Showing Classified Blocks with Measured in Red and Indicated in Green 14-17
Figure 14.9 1300 Level Plan Showing Estimated Gold in Blocks and Composites 14-22
Figure 14.10 1280 Level Plan Showing Estimated Gold in Blocks and Composites 14-23
Figure 14.11 1260 Level Plan Showing Estimated Gold in Blocks and Composites 14-24
Figure 14.12 1240 Level Plan Showing Estimated Gold in Blocks and Composites 14-25
Figure 14.13 1220 Level Plan Showing Estimated Gold in Blocks and Composites 14-26
Figure 14.14 1200 Level Plan Showing Estimated Gold in Blocks and Composites 14-27
Figure 16.1 2014 PEA Pit Design with 2015 Oriented Geotechnical Core Holes and Sector boundaries 16-3
Figure 16.2 Pit Design 16-8
Figure 16.3 Production Schedule 16-10
Figure 16.4 LOM Mine Status Map 16-10
Figure 17.1 Simplified Process Flowsheet 17-3
Figure 18.1 General Site Layout 18-2
Figure 18.2 Plate Site Layout 18-4
Figure 18.3 Grinding, Leaching and Recovery Area Layout 18-5
Figure 18.4 TMF Embankment Section 18-9
Figure 18.5 TMF DAC Relationship 18-10
Figure 18.6 TMF Filling Schedule 18-11
Figure 21.1 Operating Cost Distribution 21-6
Figure 22.1 Pre-tax Undiscounted Annual and Cumulative Net Cash Flows 22-3
Figure 22.2 Pre-tax NPV Sensitivity Analysis 22-5
Figure 22.3 Pre-tax IRR Sensitivity Analysis 22-6
Figure 22.4 Pre-tax Payback Period Sensitivity Analysis 22-6

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 potential	AP
acid rock drainage	ARD
acid-based accounting	ABA
Acme Analytical Laboratories Ltd	Acme
all-terrain vehicle	ATC
ammonium nitrate/fuel oil	ANFO
Archer, Cathro & Associates (1981) Ltd.	Archer Cathro
ATAC Resources Ltd.	ATAC
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
Canadian Development Expense	CDE
Canadian Exploration Expense	CEE
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Canadian Pension Plan	CPP
carbon-in-leach	CIL
carbon-in-pulp	CIP
CDN Resource Laboratories Ltd.	CDN Resource
certified reference material	CRM
closed-circuit television	CCTV
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
interim freshwater sediment quality guidelines	ISQG
internal rate of return	IRR
Kappes Cassidy and Associates	KCA
life-of-mine	LOM
mass spectroscopy	MS
metal leaching	ML
Microsoft-SQL Server® database	the Database
motor control centre	MCC
National Instrument 43-101	NI 43-101
National Topographic System	NTS
NDU Resources Ltd.	NDU
net cash flow	NCF
net present value	NPV
neutralization potential	NP
North American Datum	NAD
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
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 May 31, 2016 and the effective date of the Mineral Resource estimate is October 28, 2015.

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 3,128 mineral claims that are 100% owned by ATAC, totaling 62,460 ha. There are no underlying royalties on the Property.

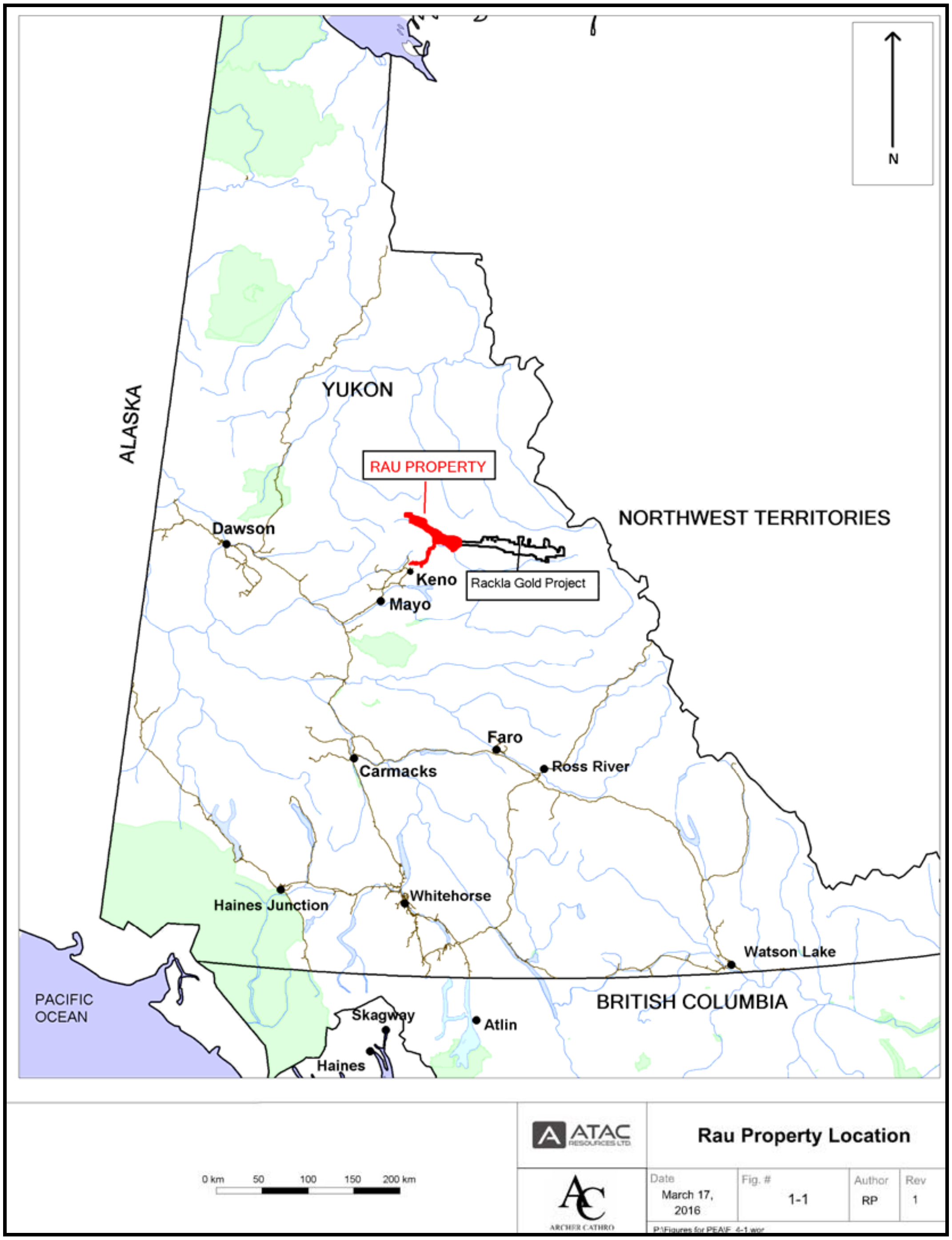
Access to the Property is currently via a 900-m airstrip, located 7.5 km southeast of the the Project. The Wind River Trail, a winter road, starts at McQuesten Lake, near Keno City and crosses the central portion of the Property, 13 km west-northwest of the Project.

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.

Figure 1.1 Project Location



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 the Project. This intrusion post-dates regional thrusting events and is thought to be the source of mineralization.

Carbonate replacement gold-mineralization at the Project is thought to be a distal variety of the reduced intrusion related gold deposit model. The Project 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 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 2015, 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 24,696 soil samples, 1,428 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 26,844 m of drilling in 150 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 6,222 assays taken from 150 diamond drillholes, totalling 26,844 m. The effective date of this Mineral Resource estimate is October 28, 2015. A 3D solid model was constructed to constrain oxide and sulphide mineralization.

Gold distribution, within the mineralized solids, were examined using a lognormal cumulative frequency plot to determine appropriate capping levels. Three-metre composites were formed, honouring solid boundaries, using the capped assay data.

The Project Mineral Resource was estimated by Gary Giroux, P.Eng, MASc. Ordinary Kriging was used to interpolate gold and silver values into a block model. The search parameters were based on variography.

Mineral Resources are reported at a 0.5 g/t cut-off in oxides and 1.0 g/t cut-off in sulphides in Table 1.1. These cut-off grades were selected based on comparison to other analogous deposits.

Table 1.1 Combined Oxides and Sulphide Resource

Type	Classification	Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off Au (g/t)	Grade>Cut-off Ag (g/t)	Contained Metal Au (oz)	Contained Metal Ag (oz)
Oxides	Measured	0.50	2,600,000	3.10	4.77	259,100	398,700
	Indicated	0.50	1,720,000	2.47	4.10	136,300	226,700
Sulphides	Indicated	1.00	1,360,000	2.07	0.56	90,300	24,500
Total	**M+I**	-	**5,680,000**	**2.66**	**3.56**	**485,700**	**649,900**
Oxides	Inferred	0.50	280,000	1.52	5.67	13,700	51,000
Sulphides	Inferred	1.00	2,950,000	1.84	0.47	174,800	44,600
Total	**Inferred**	-	**3,230,000**	**1.81**	**0.92**	**188,500**	**95,600**

Note: Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definition standards (CIM 2014) were used for the Mineral Resource.

The author is not aware of any known environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other similar factors that could materially affect the stated Mineral Resource estimate.

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. During this time nine programs were completed and reported on oxide and sulphide samples.

1.7.1 OXIDE STUDIES

Mineralogical work showed that the oxide samples were dominated by quartz and dolomite, with lesser goethite, talc, calcite, mica, and hematite. Knelson™ concentrator testing recovered 5 to 18% of the gold to concentrate, suggesting there is little coarse discrete gold. A single Bond Ball Mill Work Index (BWi) test showed the material to be very soft (BWi = 8.5 kWh/t).

Bottle roll testing was conducted using a variety of crush/grind sizes and leach conditions. Coarse bottle roll tests yielded gold recoveries of over 86% for crush sizes up to one inch. Fine bottle roll tests yielded similar results, averaging 88%. The effects of grind size and cyanide concentration on recovery were minimal.

1.7.2 SULPHIDE STUDIES

The sulphide zones comprise large but variable amounts of pyrite and arsenopyrite. Non-sulphide mineralization is dominated by carbonates, comprising a mixture of dolomite, ankerite, and calcite. Gold occurs both as solid-solution in sulphides and as discrete mineralization. In the samples studied, 10 to 20% of gold was hosted in pyrite, 23 to 59% in arsenopyrite, and 28 to 67% as discrete gold.

No formal grindability studies have been conducted on the sulphide material, which appears to be somewhat harder than the oxides.

Leach recoveries are highly variable, ranging from 13 to 97% in tests completed to date. As with the oxides, primary grind size had a limited effect on extraction rates.

Flotation testing recovered 82 to 97% of the gold. Pressure pre-oxidation of the concentrates (POX) followed by carbon-in-leach (CIL) extraction extracted 97 to 99% of the gold. Substituting the POX with bacterial oxidation (BOX) yielded CIL gold extractions of 92 to 93% and silver extractions of 71 to 82%. Reagent consumptions were high, especially for POX.

1.7.3 PROCESS SELECTION

Gold in the oxide zone of the Project is mostly free-milling, while in the sulphide zone it is both in free-milling and refractory form. Refractory gold is in solid solution within arsenopyrite, and to a lesser extent pyrite. The grades of solid-solution gold are generally too low for economic pre-oxidation processing at current prices. Accordingly, recoveries are limited by the presence of gold in discrete form, which is higher in the oxide material than in the sulphide.

The discrete gold, though mostly not amenable to gravity recovery, occurs in a way that allows for good cyanide access even at coarser crush sizes.

Column leaching, simulating the heap leach process, yielded excellent recoveries for oxide material. However, the material performed very poorly under compression and agglomeration was needed. Additionally, excessive amounts of cement were needed to make the agglomerates robust enough to resist compression. An engineering assessment of the heap leach option concluded that heap leaching would be risky and problematic for this material, so this option appears not to be viable.

Agitation leaching worked well for the oxides, and recoveries from oxide samples averaged close to 90%. Cyanide consumption rates were modest, but lime consumptions in treating the oxides were sometimes quite high.

The sulphides tended to float quite well with recoveries averaging 85% in the more refractory samples tested. While sulphide pre-oxidation by POX and BOX yielded high to very high gold extraction rates, the low grade of the sulphide concentrates renders such processing uneconomic at current gold prices.

Accordingly, for the sake of the current study, a circuit comprising grinding and agitation cyanide leaching for both the sulphide and oxide resources was envisaged.

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 total life-of-mine (LOM) is approximately eight years, including one year of pre-stripping followed by seven years of mill production. Mill feed during the last production year will come from the low-grade stockpile. Over the LOM, the pit will produce 3.2 Mt of mineralized material and 15.6 Mt of waste rock. The LOM average gold grade of the oxide and sulphide material is 4.06 g/t and 2.99 g/t, respectively. The LOM stripping ratio (defined as waste material mined divided by mineralized material mined) is 4.9.

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. Although previous heap leach test work showed promising results from oxide mineralization, and extensive engineering studies were conducted using a combined heap leach and tank processing treatment to extract the gold from the mineralization, there are numerous potential technical risks and challenges associated with the heap leach option. Therefore, a conventional cyanidation circuit consisting of grinding and agitated cyanide leaching for both the sulphide and oxide resources is proposed for this study.

The process 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.

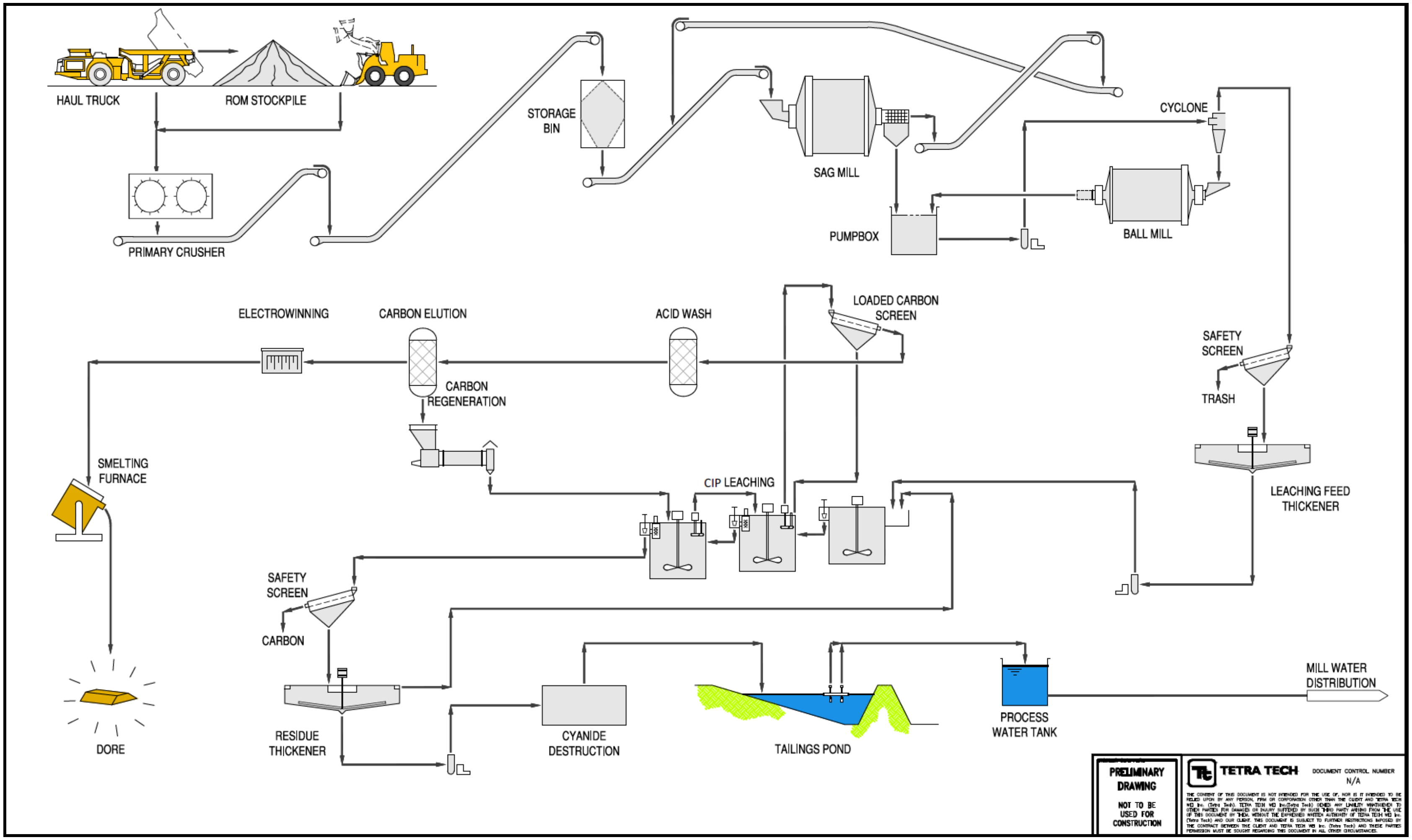
The hydrocyclone overflow from the primary grinding circuit will be thickened, and the underflow of the thickener will be diluted with process water to the optimum solid density, and then cyanide leached in a CIP circuit to recover the gold from the mineralization.

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 1.2. The average gold production to doré is estimated to be approximately 50,000 oz/a.

Figure 1.2 Simplified Process Flowsheet



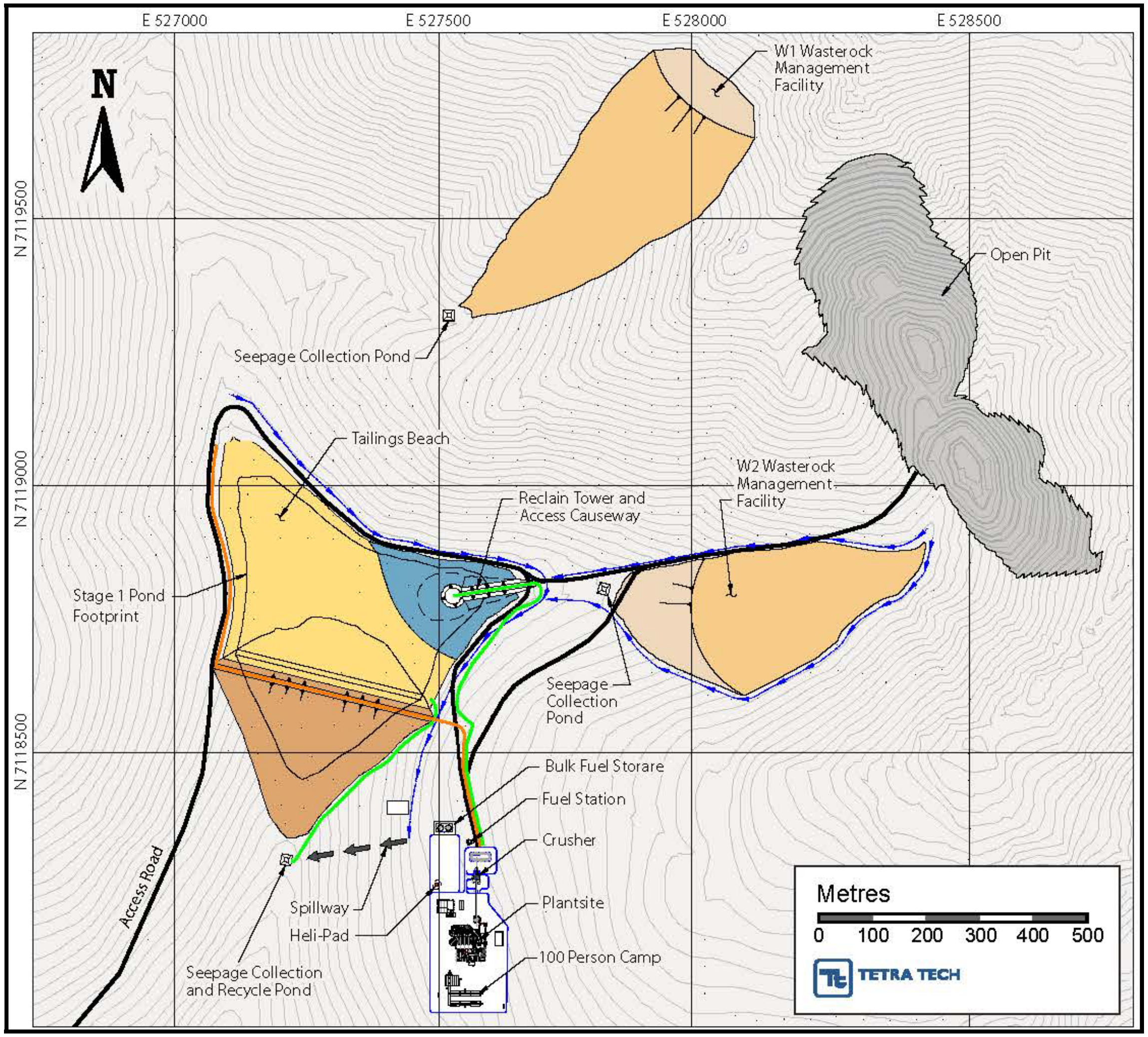
1.10 PROJECT INFRASTRUCTURE

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
- a laydown area
- 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 1.3.

Figure 1.3 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.

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 $109.4 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	Cost ($000)
Overall Site	3,076
Open Pit Mining	13,050
Mine Dewatering	144
Materials Crushing and Handling	1,986
Process	29,683
TMF	7,893
On-Site Infrastructure	5,026
External Access Roads	11,063
Project Indirect Costs	19,818
Owner's Costs	1,197
Contingencies	16,464
Total Initial Capital Cost	**109,400**

1.12.2 OPERATING COST ESTIMATE

On average, the LOM on-site operating costs for the Project were estimated to be $66.59/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	Cost ($/t milled)
Mining	21.75
Process	26.26
TMF	0.72
G&A	12.38
Site Service	3.80
Camp and Genset Leasing Cost	1.68
Total Operating Cost	**66.59**

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 USD1,250/oz and an exchange rate of USD0.78:CAD1.00 (all currency units are Canadian dollars unless otherwise specified):

- 34.8% internal rate of return (IRR)
- 1.85-year payback on $109.4 million initial capital
- $106.6 million net present value (NPV) at a 5% discount rate.

ATAC commissioned PricewaterhouseCoopers (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 22.5 for further details).

The following post-tax financial results were calculated:

- 28.2% IRR
- 1.93 year payback on $109.4 million initial capital
- $75.7 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 $4.65 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	900,000
Geotechnical and Hydrogeological	1,150,000
Mineral Processing and Metallurgical Testing	200,000
Mining	400,000
Infrastructure	1,500,000
Environmental	500,000
Total	4,650,000

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: overall project management, mining methods, mineral processing and recovery methods, project infrastructure, capital and operating cost estimates, and economic analysis.
- Archer Cathro: project description and location, accessibility, history, geological setting, deposit types, exploration, drilling, sample preparation, data verification, and adjacent properties.
- Giroux Consultants Ltd. (Giroux Consultants): Mineral Resource estimate.
- Core Geoscience Services Inc. (Coregeo) and Resource Strategies Ltd. (RSL): environmental studies, permitting, and social or community impact.
- 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 May 31, 2016 and the effective date of the Mineral Resource estimate is October 28, 2015.

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:

- Sabry Abdel Hafez, Ph.D., P.Eng. completed a site visit on November 15, 2013.
- Matthew Dumala, P.Eng., completed a site visit from July to August 2011.
- Gary Giroux, P.Eng., completed a site visit from August 30 to 31, 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	Sabry Abdel Hafez, Ph.D., 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 Mineralisation	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	Archer Cathro	Matthew Dumala, P.Eng.
12.0	Data Verification	Archer Cathro	Matthew Dumala, P.Eng.
13.0	Mineral Processing and Metallurgical Testing	Blue Coast	Christopher Martin, C.Eng, MIMMM
14.0	Mineral Resource Estimates	Giroux Consultants	Gary Giroux, P.Eng., MASc
15.0	Mineral Reserve Estimates	Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.
16.0	Mining Methods	Tetra Tech	Sabry Abdel Hafez, Ph.D., 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	Coregeo/ RSL	Eri Boye, P.Geo.
21.0	Capital and Operating Costs	Tetra Tech	Hassan Ghaffari, P.Eng.
22.0	Economic Analysis	Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.
23.0	Adjacent Properties	Archer Cathro	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 located 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

Tetra Tech relied on Andrew Carne, EIT, 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.

Tetra Tech relied on PwC concerning tax matters relevant to this report. The reliance is based on a letter to ATAC titled "Assistance with preparation of the income and mining tax portions of the economic analysis prepared by Tetra Tech WEI Inc. ("Tetra Tech") in connection with the Preliminary Economic Assessment (the "Report") on ATAC Resources Ltd.'s ("ATAC") Tiger Gold Deposit Project (the "Project") and dated May 27, 2016.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Project, 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 1,741 km^2 RGP and comprises 3,128 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 624.6 km^2 (62,460 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 June 7, 2016, are tabulated in Table 4.1 and the locations shown on Figure 4.2.

Figure 4.1 Project Location Map

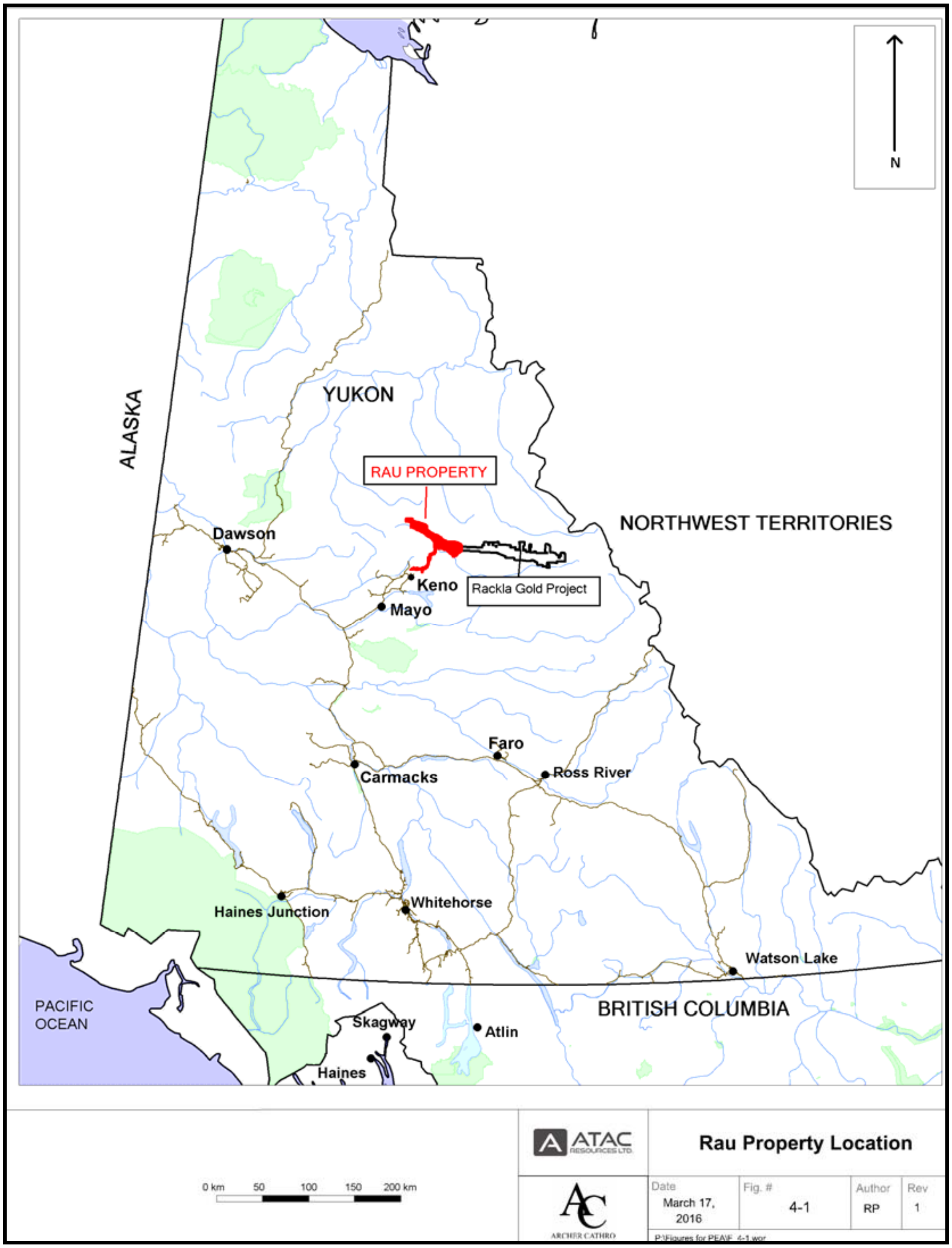


Table 4.1 Claim Data

Claim Name(s)	Grant Number(s)	Expiry Date
GF 3-4	YC32305-YC32306	April 28, 2033
Q 1-13	YC92361-YC92373	April 28, 2030
Q 14	YC92470	April 28, 2030
Q 15-109	YC92375-YC92469	April 28, 2030
R 1-103	YC68334-YC68436	April 28, 2032
R 105-1295	YC68438-YC69628	April 28, 2032
R 1296-1337	YC70595-YC70636	April 28, 2029
Rau 1-64	YC50268-YC50331	April 28, 2039
Rau 65-96	YC57529-YC57560	April 28, 2032
Rau 97F-98F	YC69925-YC69926	April 28, 2033
Rau 99F-100F	YC69961-YC69962	April 28, 2033
S 1-700	YC90801-YC91500	April 28, 2030
S 701-842	YC91901-YC92042	April 28, 2030
S 843	YC92355	April 28, 2030
S 844-1154	YC92044-YC92354	April 28, 2028
S 1155-1244	YD09635-YD09724	March 1, 2019
S 1245	YD09725	March 1, 2023
S 1246-1247	YD09726-YD09727	March 1, 2021
S 1248	YD09728	March 1, 2023
S 1249-1250	YD09729-YD09730	March 1, 2019
ACX 1-234	YD08251-YD08484	April 28, 2027
ACX 235-310	YD33163-YD33238	April 28, 2027
GAM 1-21	YD98437-YD98457	April 28, 2023

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

Figure 4.2 Claim Locations

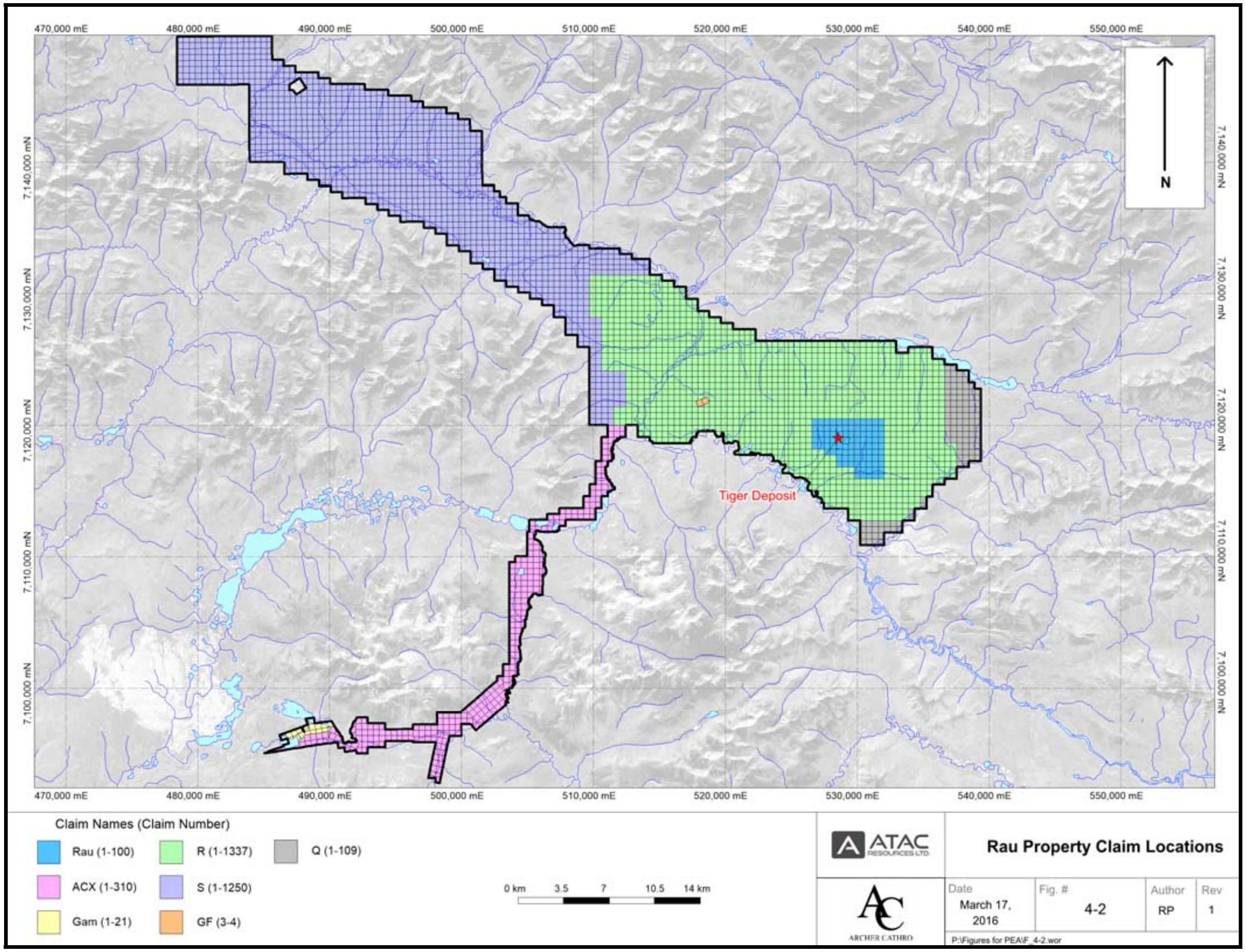
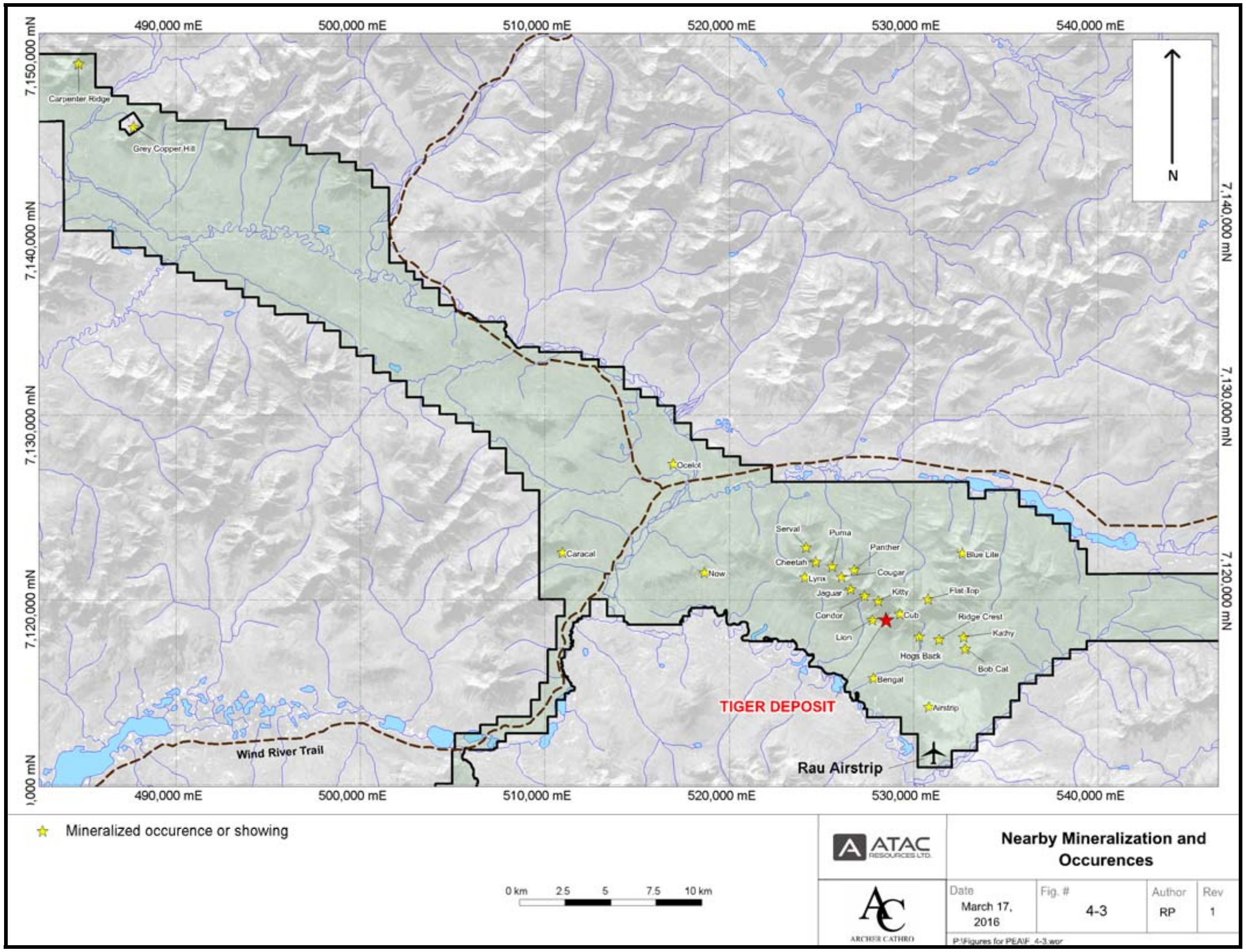


Figure 4.3 Nearby Mineralization and Occurrences



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 YESAA. 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 LQ00260C, which expires August 13, 2019.

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 NNDFN. To the best of the Author's knowledge there are no encumbrances to the Property relating to First Nation Settlement Lands.

In August 17 2010, ATAC and NNDFN signed an exploration co-operation agreement related to ATAC's exploration activities on the Property, which is located within the NNDFN traditional territory. Through this agreement, both parties recognize a mutually beneficial approach to the exploration Project that fosters an understanding and awareness of their respective interests, and to co-operate with each other to establish a mutually respectful relationship.

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 Project 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. ATAC is currently permitted to use the southern part of the Wind River Trail between December 1 and March 31 of each year to access the Property under the terms of Territorial Land Use Permit #2013-F561. However, recent warmer average winter temperatures have greatly reduced the period this route is suitable to use as a winter trail.

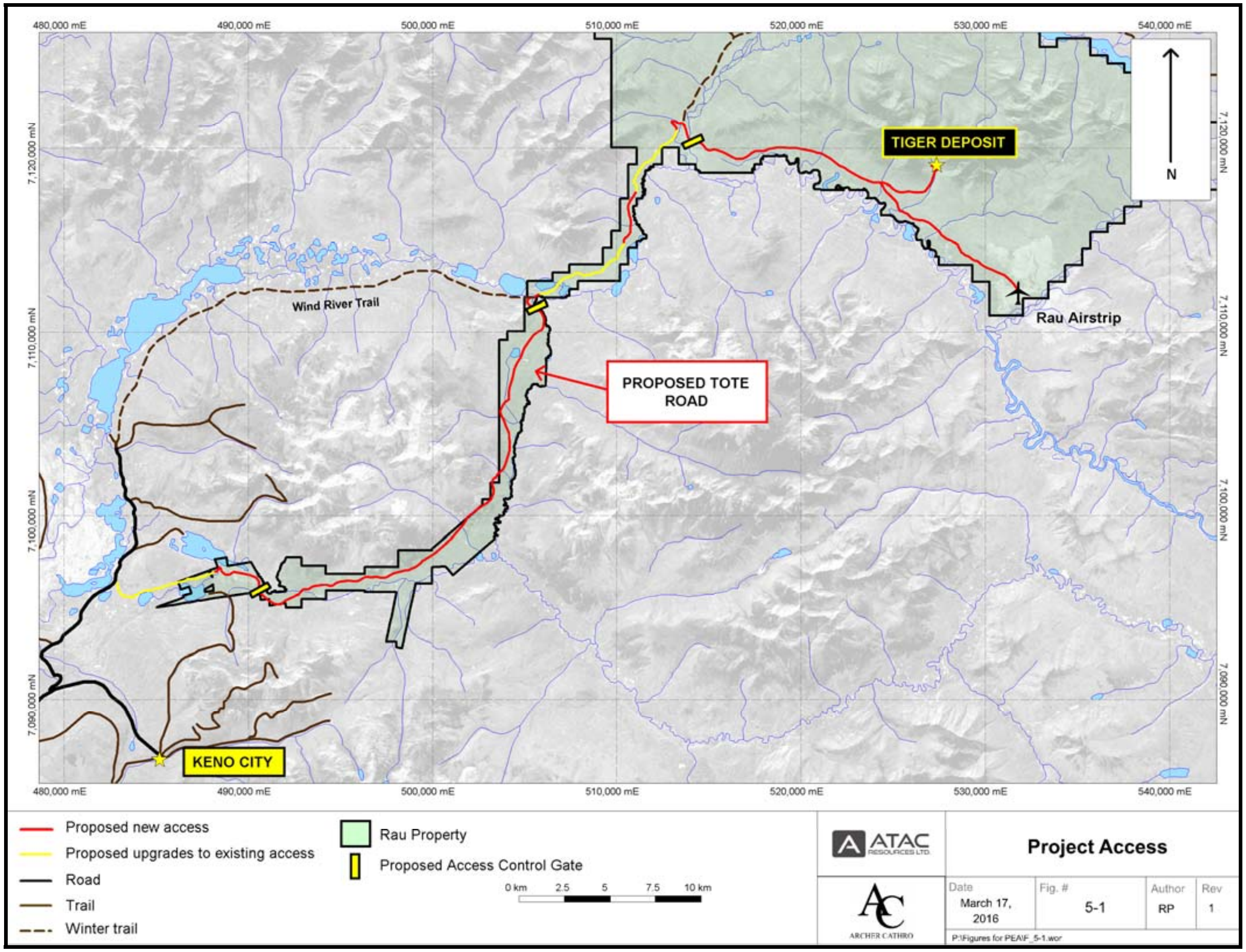
Access to the Property is currently via fixed wing aircraft to ATAC's 900-m-long (2,950-ft) airstrip located 7.5 km southeast of the Project near the southeastern boundary of the Property. Helicopters are used locally to access to exploration targets within the Property. A local trail system provides access within the valley containing the Project. 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 66 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 2016). 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 Project, 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 troy ounces per ton of 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 Project.

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 at between 2008 and 2009 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.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

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

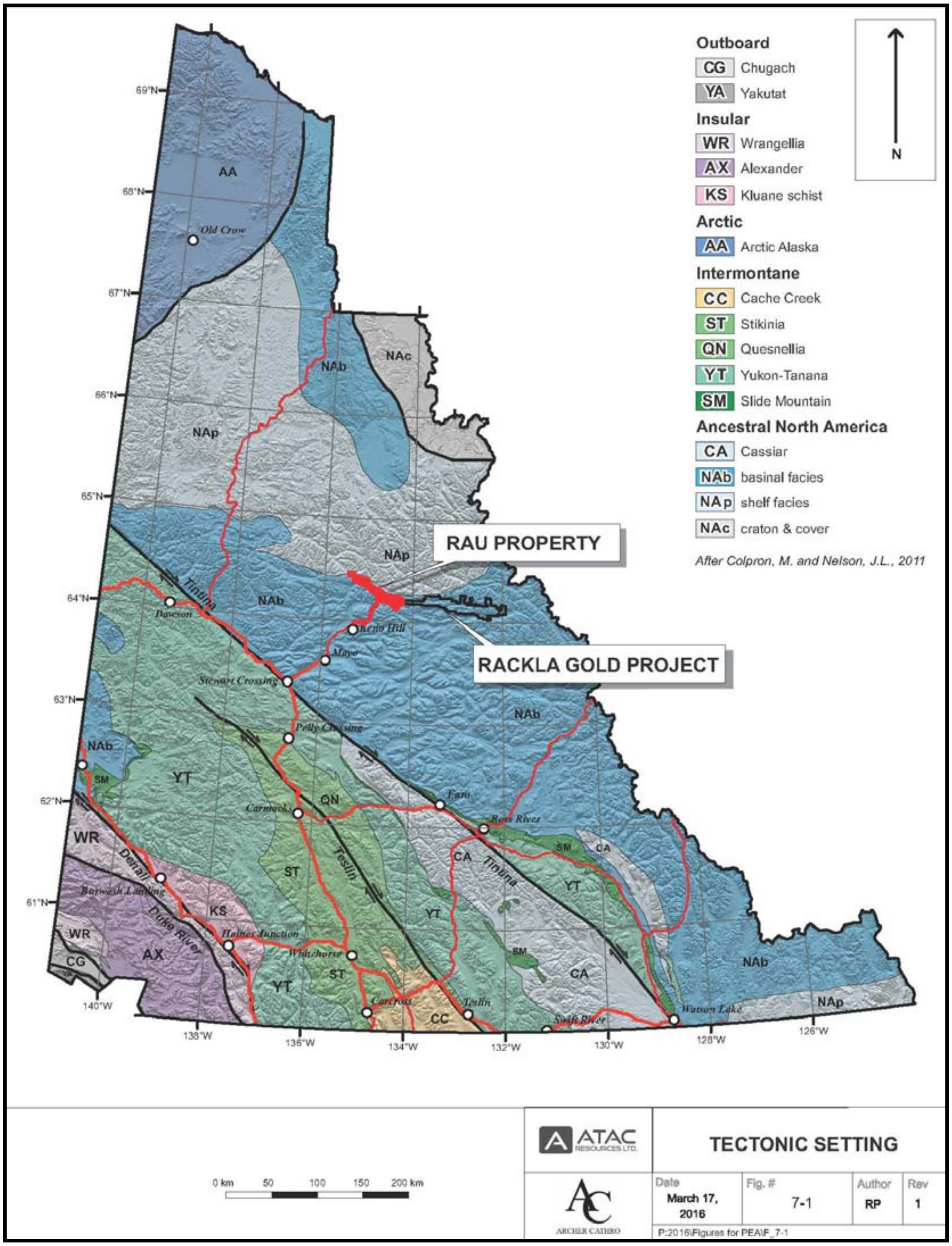


Figure 7.2 Regional Geology

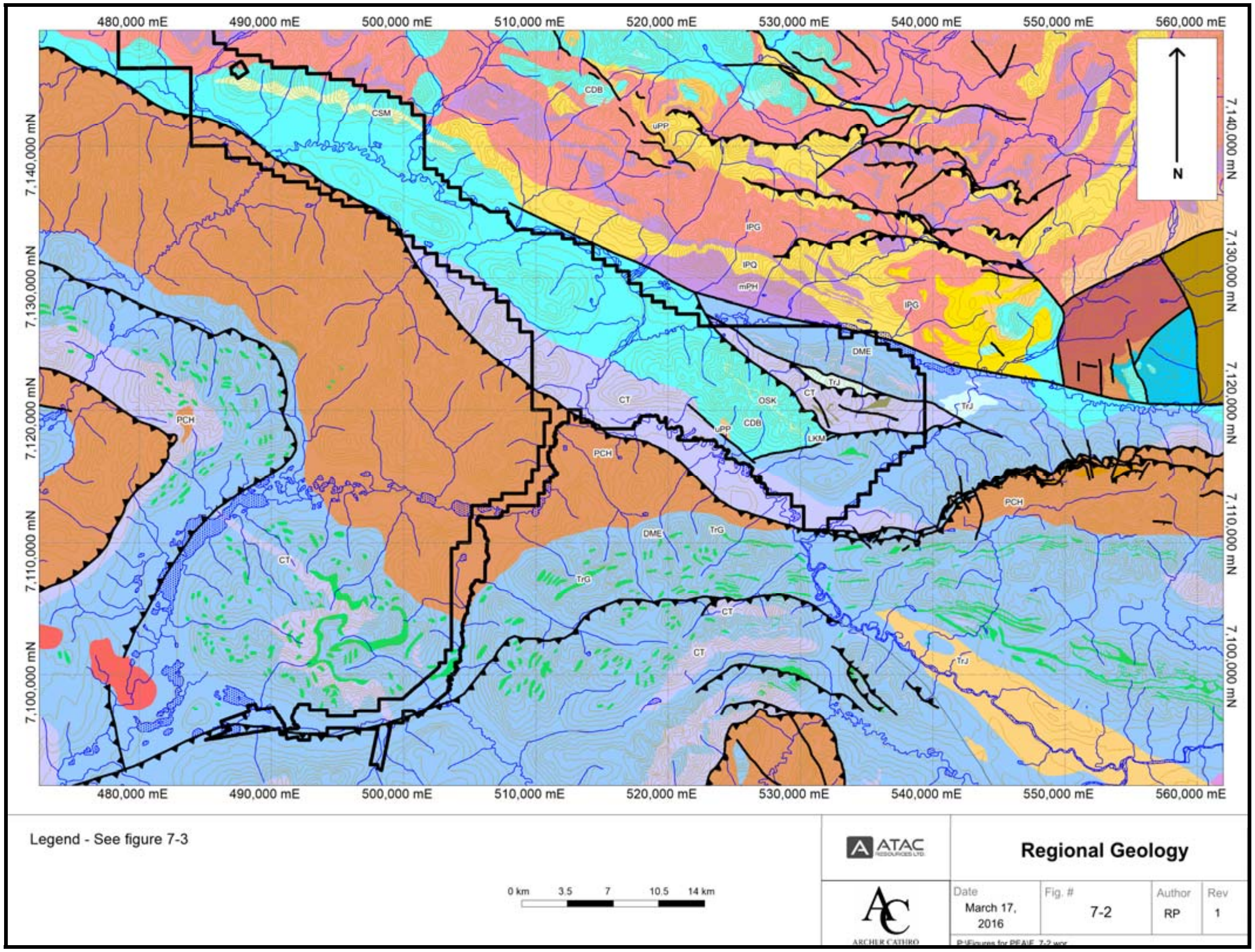


Figure 7.3 Regional Geology Legend

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)

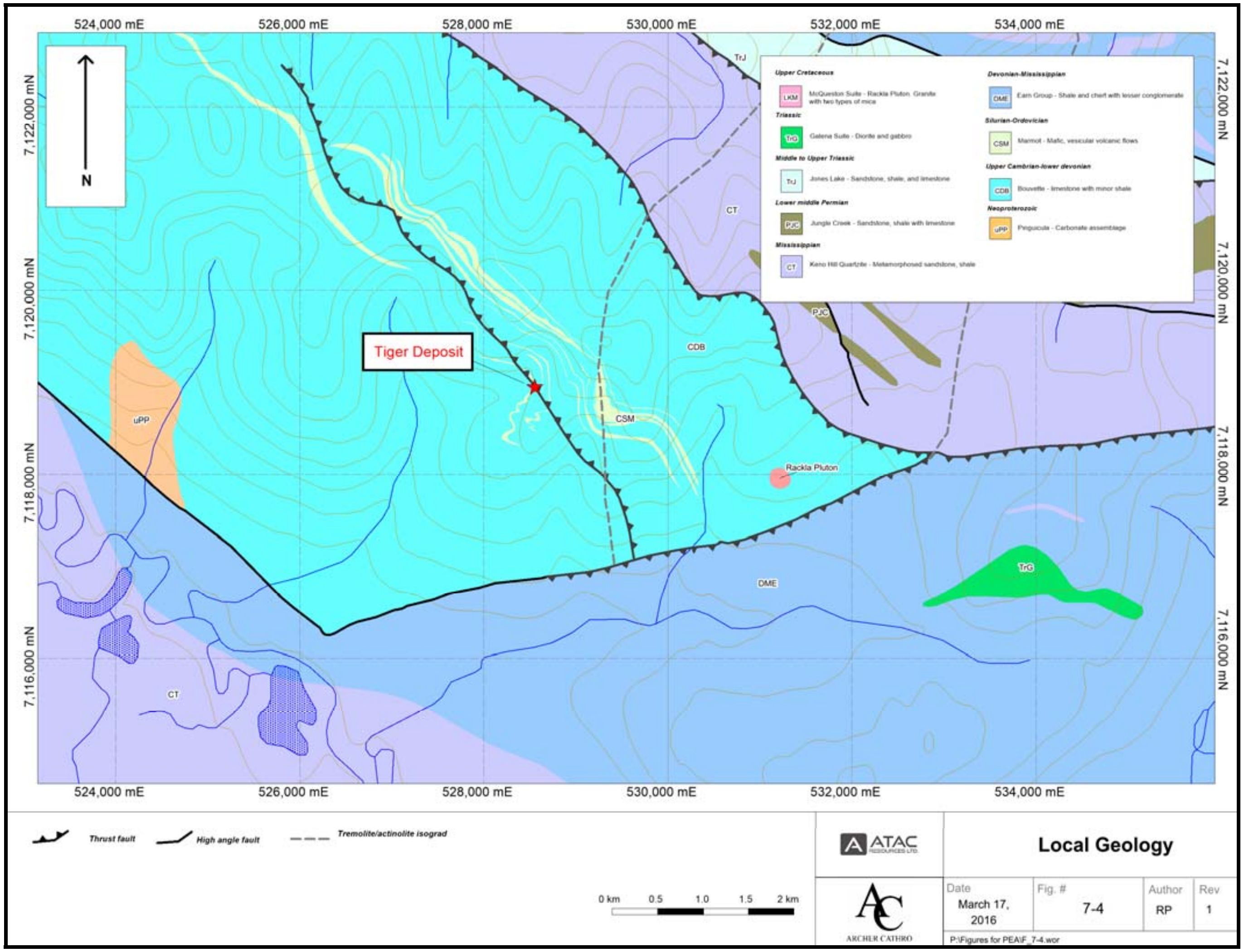
	Unit	Tectonic Element	Age (Ma)	Description
Ancestral North America	Gillespie Lake Group (IPG)	Mackenzie Platform	1700 - 1800	Dolostone and sandstone
	Bouvette Formation (CDB)	Mackenzie Platform	540 - 390	Limestone with rare black shale
	Marmot Formation (CSM)	Mackenzie Platform	540 - 420	Mafic, vesicular 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 detail geological mapping has been conducted within the Property boundary, except within the vicinity of the Tiger Deposit. Most work has focused within the favorable Bouvette Formation 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



The Bouvette Formation is the most abundant inferred rock type shown on government maps and is the principal focus of ATAC's exploration. It can be divided into three main 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 the Bouvette Formation 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 Bouvette Formation has not been mapped in detail and remains undifferentiated.

A narrow sliver of Middle Proterozoic Fifteen Mile Group dolostone lies beneath the Bouvette Formation, to the southwest. This unit is composed of chocolate to orange brown weathering, cryptalgal laminated, medium- to thick-bedded dolostone, overlain by rusty brown weathering, olive green siltstone and shale with lesser maroon black and buff shale.

The Marmot Formation consists of thin volcanic horizons that are inter-bedded with the Ordovician and/or Silurian Bouvette Formation. 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 bounds the Bouvette Formation 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 Bouvette Formation, 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 Bouvette Formation 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 Bouvette Formation 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

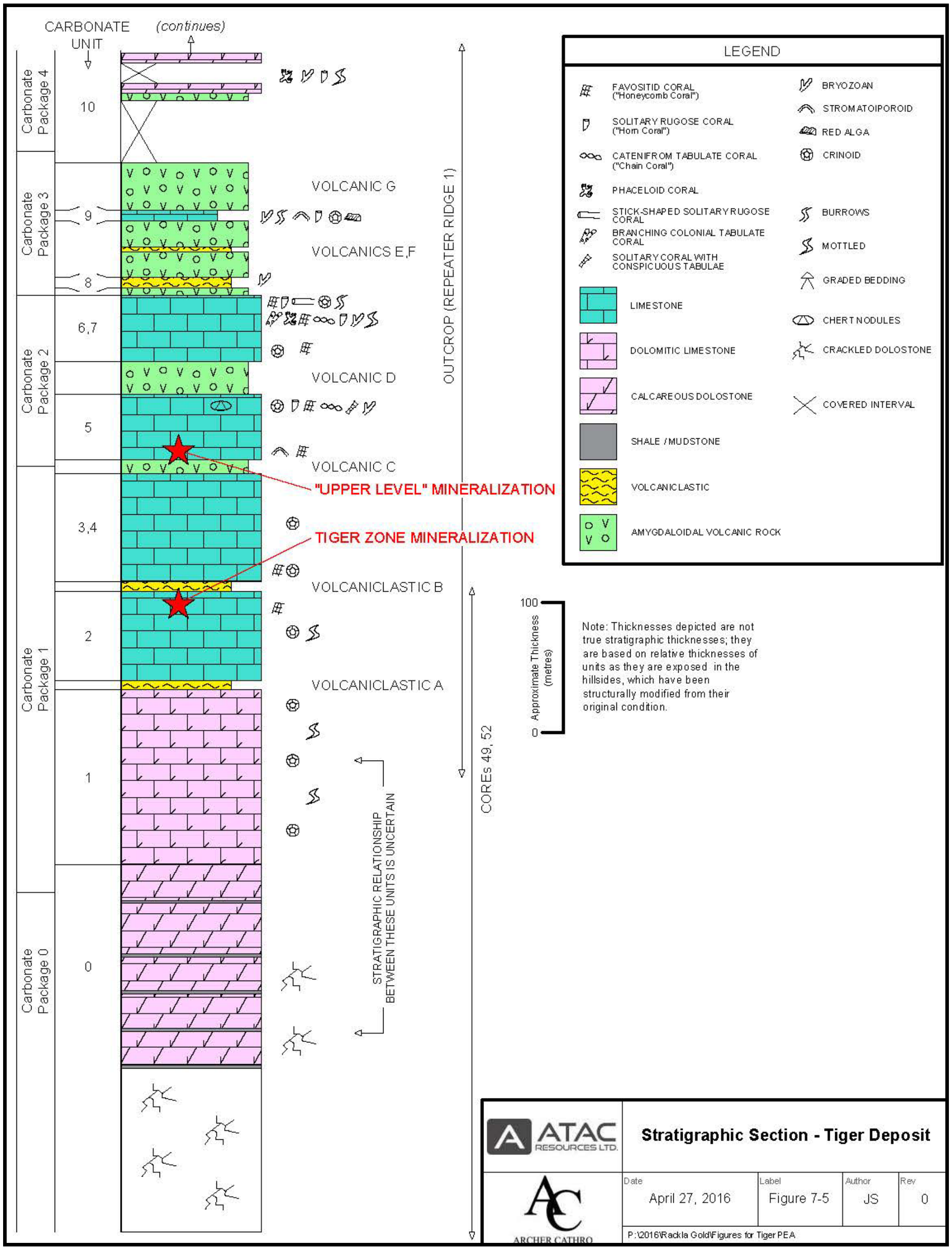
During 2010 detail deposit scale mapping was conducted within the Bouvette Formation carbonate sequence in the vicinity of the Tiger Deposit mineralization. This work was conducted at different periods during the field season by Dr. Elizabeth Turner (Laurentian University), Dr. Harry E. Cook (Nevada 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 by detailed examination of outcrop and drill core 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 carbonates 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 carbonates 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 examined 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, and is presumed to be 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 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 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 moderately northeast dipping horizon. It is currently 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 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 a highly sericite altered volcaniclastic unit. Two samples of sub crop collected in 2008 from the area 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. A small amount of disseminated scheelite is also present. The main sulphide minerals exhibit at least three stages of mineralization.

Oxide mineralization is 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 highest-grade and deepest oxidation occurs where northerly trending extensional faults intersect the northwest trending regional shear structure. Detail 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 'Alizarin Red and Potassium Ferricyanide' carbonate staining technique. 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.

Minerals in this phase are not always observed together but when they are 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 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 has only been observed in diamond drill hole Rau-08-18 (later extended as Rau-09-18). 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-grain which are subsequently annealed around grain margins. These pyrite grain margins exhibit no fracturing and appear to be a recrystallization of the brittley deformed 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 differ from 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. 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.

7.4.2 Scheelite Tremolite-Actinolite Skarns

Four known tungsten skarn showings occur on the Rau Property (Figure 4.3): the Bobcat, Flat Top, Hogs Back and Ridge Crest Showings. Three of these are located less than 1.5 km from the Rackla Pluton, while the fourth is located 4.8 km to the north. They are a combination of scheelite-tremolite and actinolite skarns containing varying concentrations of pyrite, pyrrhotite and rare chalcopyrite. These showings have associated moderate to strong tungsten, gold and copper soil geochemical anomalies (Dumala 2009).

The Hogs Back Showing is located 800 m southwest of the pluton. This showing is exposed on the north and south side of a northwest trending gully. It was first identified in 2006 and followed up in 2007. The showing consists of three actinolite skarn layers occurring conformably within a portion of the Bouvette limestone sequence. Mineralization comprises finely disseminated to patchy pyrrhotite, pyrite and lesser chalcopyrite. Marble in the enclosing limestone is variable but extends up to 5 m in areas. The skarn layers have been traced to the northwest for over 750 m and vary in thickness from 0.3 to 6.0 m, averaging 0.8 m. The thickest and best mineralized exposure occurs at the southeast edge of a crosscutting drainage before disappearing to the southeast beneath cover. In general the exposed skarn horizons appear too thin to the northwest.

The Ridge Crest Showing, located on the southwest side of the Rackla Pluton, was discovered in 2009 while following up a gold-in-soil anomaly (990 ppb) within the 2007 soil sample grid. A 70-cm deep hand pit dug at this location revealed glacial till and grey limestone fragments. Two cobbles of rusty dark green pyroxene skarn and several oxidized skarn fragments were also extracted from this pit. Grab samples of the skarn yielded 0.02 g/t gold and 850 ppm tungsten. A dyke containing equigranular, coarse grained white to smoky quartz and minor muscovite with occasional patches of chlorite and trace fine grained sulphides occurs 15 m to the west. A nearby float sample returned 1,060 ppm tungsten.

The Flat Top Showing occurs along a north trending ridge approximately 1.3 km northwest of the Rackla Pluton. It is localized along the contact between the Bouvette Formation and Earn Group strata. The showing is marked by approximately coincident, moderately to strongly anomalous gold, copper and tungsten soil geochemical values approximately 600 m long and up to 300 m wide. Scheelite in tremolite skarn was first found at this locale in 1979 by Prism. Prospecting in 2009 traced skarn mineralized float around the nose of the ridge for 400 m.

Four types of skarn mineralization occur across a stratigraphic thickness of 40 m. The first, found immediately above the contact, occurs as felted to radiating masses of acicular tremolite/actinolite or wollastonite/actinolite localized in a band that ranges from a few tens of centimetres to a few metres thick. The second, found within unaltered carbonate rock, are masses of tremolite mixed with calcite found in 0.5 mm to 2.0 cm thick veinlets. Thirdly, an iron rich skarn consisting of coalescing aggregates of radiating acicular masses of tremolite/actinolite preferentially replaces the host carbonate. Rare interstitial green tourmaline or vesuvianite, calcite and quartz also occur with this skarn type. Finally extending upward from the contact is the most iron rich species. It contains felted masses of light green actinolite with local black tourmaline, biotite books, light grey to smoky quartz and patches of massive medium brown limonite.

The Blue Lite Showing, first discovered by the Prism Joint Venture in 1979, is a scheelite tremolite skarn located 4.8 km northeast of the Rackla Pluton. The skarn mineralization is well exposed on a cliffy outcrop on the north side of a prominent peak. Mineralization consists of scheelite as disseminations with massive pyrrhotite and minor chalcopyrite. The skarn horizon disappears under talus to the east and grass to the west. The Blue Lite Showing is located along a high angle normal fault that dips to the south. This fault marks the contact between Devono-Mississippian clastics to the north and Devonian to Jurassic clastics to the south.

7.4.3 Gold-Bearing Quartz-Boulangerite Vein Style Mineralization

The Now Showing is situated within a pronounced northwest trending gully located approximately 8 km west of Monument Hill. Little outcrop is exposed here but abundant float occurs along more than 400 m of the gully. Mineralization was first discovered in 1969 by Cominco Limited. Lead, zinc, and copper anomalies were identified but no clear relationship between them was determined (Johnston and Richardson 1969).

Prospecting by ATAC in 2009 revealed that these anomalies appear to follow stratigraphy but are locally overprinted by one or more vein faults. The deep gully located near the Now Showing trends slightly oblique to bedding and corresponds to a large fault.

7.4.4 Transported Gossans and Carbonate Replacement Lead-Zinc-Silver Mineralization

Two transported gossans occur on the Property separated by approximately 18 km. The Kathy Showing comprises a 40 m wide by 30 m long brick red gossanous ferricrete slab, located approximately 750 m southeast of the Rackla Pluton. The gossan is situated downhill of a thrust fault that places Earn Group shale, to the south, over Bouvette Formation carbonates to the north. The interpretation is that the gossan is formed by fluids traveling along this thrust fault.

The Ocelot Showing (historically referred to as EL) is located 11.5 km northwest of Monument Hill and was first identified by the Prism Joint Venture in 1978. The showing is marked by a 110 m long by 25 m wide gossan that formed by the precipitation of iron oxides containing silver, lead and zinc from solutions traveling along a permeable horizon (Montgomery and Cavey 1978). The gossan has a northwest orientation and parallels an adjacent topographic linear feature. The showing is most pronounced in a kill zone to the northwest that originates at a weakly flowing spring.

The gossan is predominately dolomite rubble cemented by iron oxides and is surrounded by buff to orange weathering dolomite and limestone. Diamond drilling conducted by ATAC in 2011 intersected a steeply dipping band of massive pyrite-sphalerite-galena ± tetrahedrite mineralization hosted in dolomite. Mineralization was traced for 200 m along strike, and tested up to 250 m from surface. True width ranges from 6 to 51 m (Dumala 2012).

8.0 DEPOSIT TYPE

The Tiger Deposit represents a distal variety of the reduced intrusion related gold deposit (RIRGD) 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 deposits 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-sulphide volumes, has a reduced sulphide mineral assemblage, and are associated with felsic, moderately reduced plutons. Oxide systems are mostly gold-rich (or relatively copper-poor) variants of the porphyry copper deposit model, associated with more mafic, oxidized, magnetite-series plutons.

RIRGDs 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 impinging upon the thermally driven hydrothermal system.

The dominant structural control on RIRGDs 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. RIRGDs exhibit predictable zonation and differing deposits styles outward from the central, mineralizing pluton, with increasing structural control on the more distal mineralization.

Low-sulphide contents (0.1 to 2.0%) are common in intrusion-hosted systems, and is 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 RIRGDs, both representative of the more characteristic sheeted vein deposit style, whereas Brewery Creek (Yukon) 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 other RIRGDs 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 2014 are referenced in this report as historical activities and are described in Section 6.0.

The diamond drilling programs, conducted between 2008 and 2015, 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 2015 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 24,696 soil samples have been collected on the Property.

Grid samples were collected at 50 m intervals along lines spaced 100 m apart in most areas, while detailed 50 m by 50 m grids were established over the Now, Jaguar, Tiger, Bengal, and Kathy Showings. In areas where there were no known showings, samples were collected along contour lines at 50 m intervals. The relative line positions were established using a handheld GPS, while sample spacing 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 to Figure 9.2, respectively.

Figure 9.1 Gold-in-soil Geochemistry

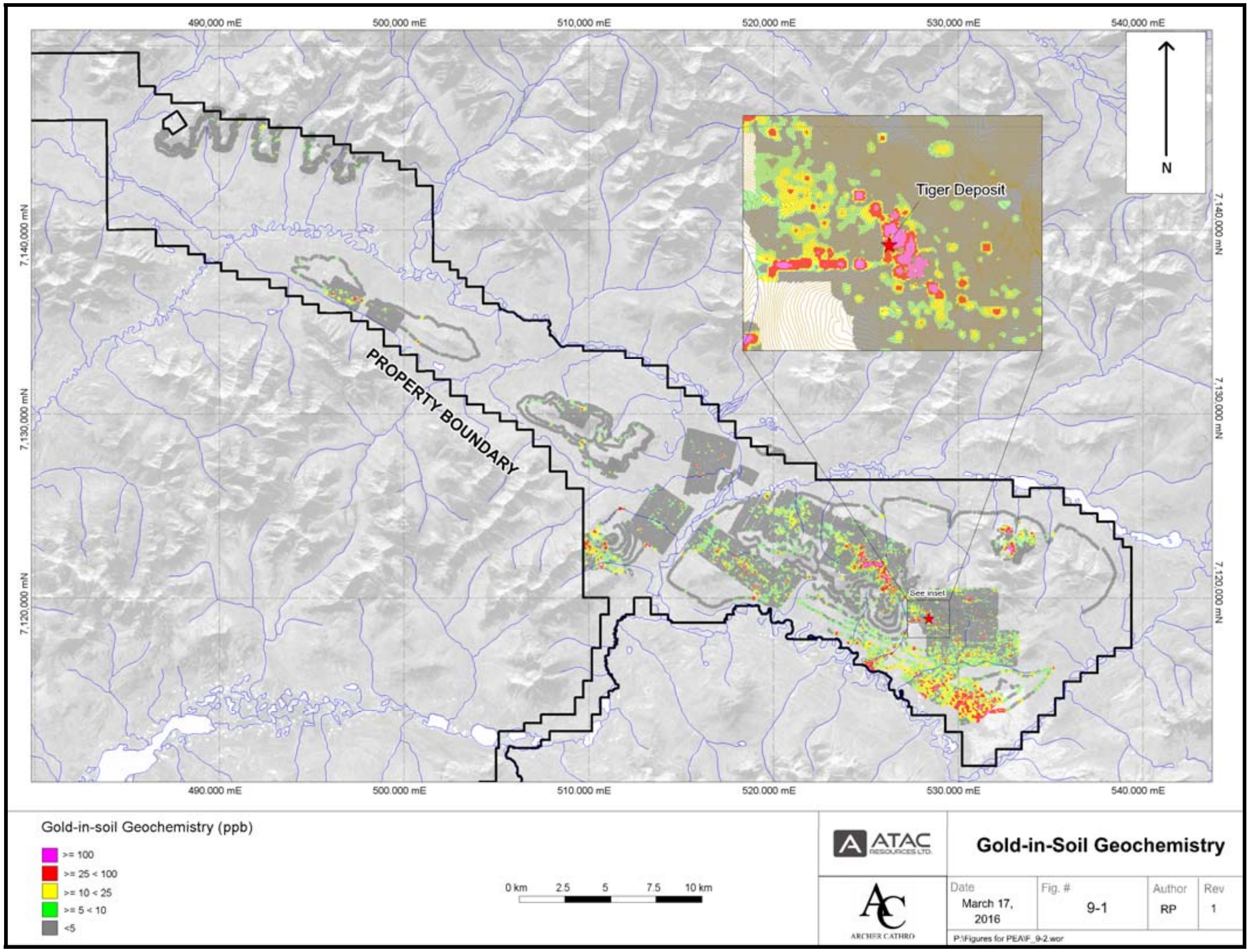
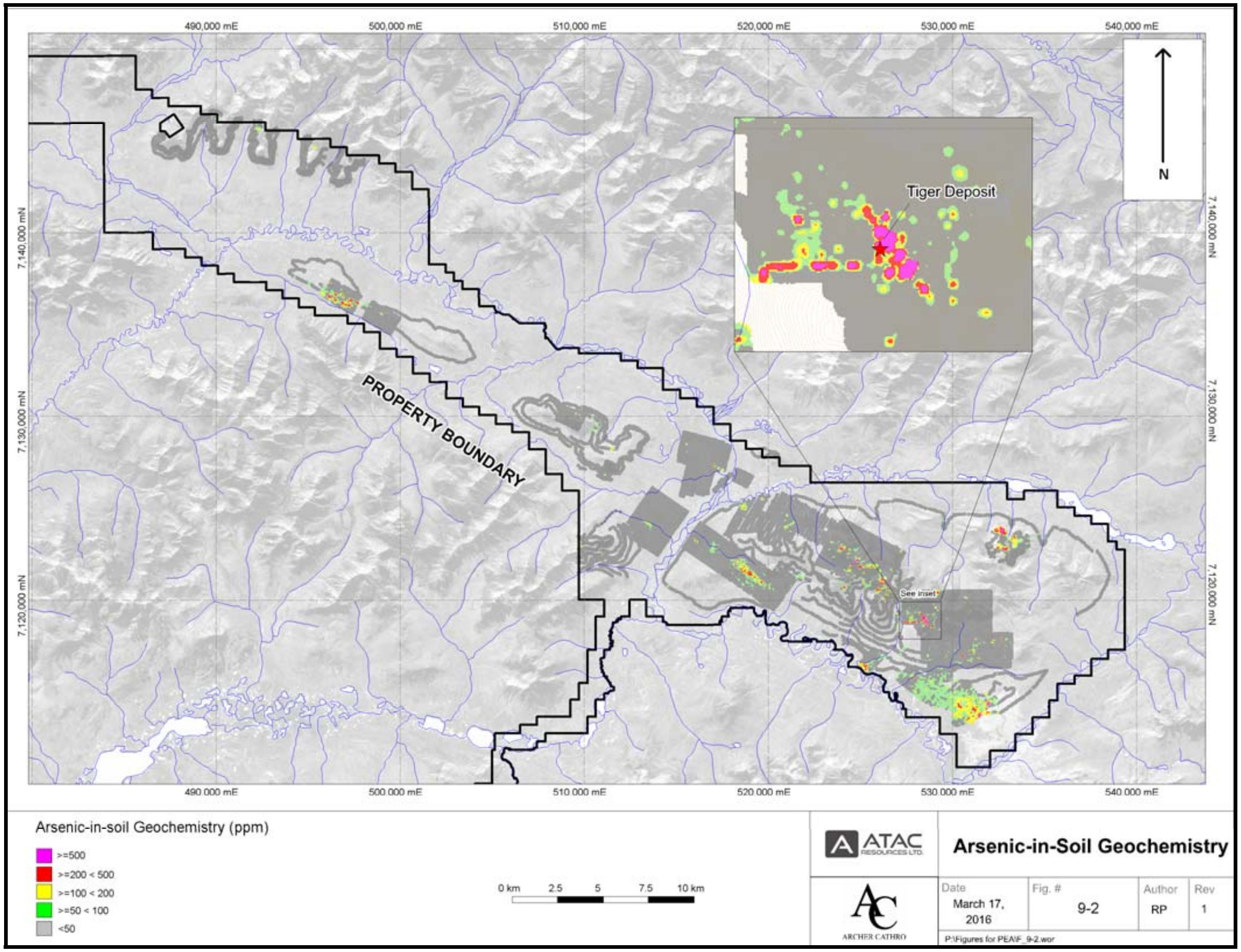


Figure 9.2 Arsenic-in-soil Geochemistry



9.2 SURFACE ROCK SAMPLING

A total of 1,428 rock samples were collected from various targets along the Rau Trend between 2007 and 2015.

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 2015 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, and Serval.

Mineralized float (Cub Showing) was found 575 m to the east 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.

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.

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 2010. Airborne surveys consisted of 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 and isolated VTEM/ZTEM anomalies.

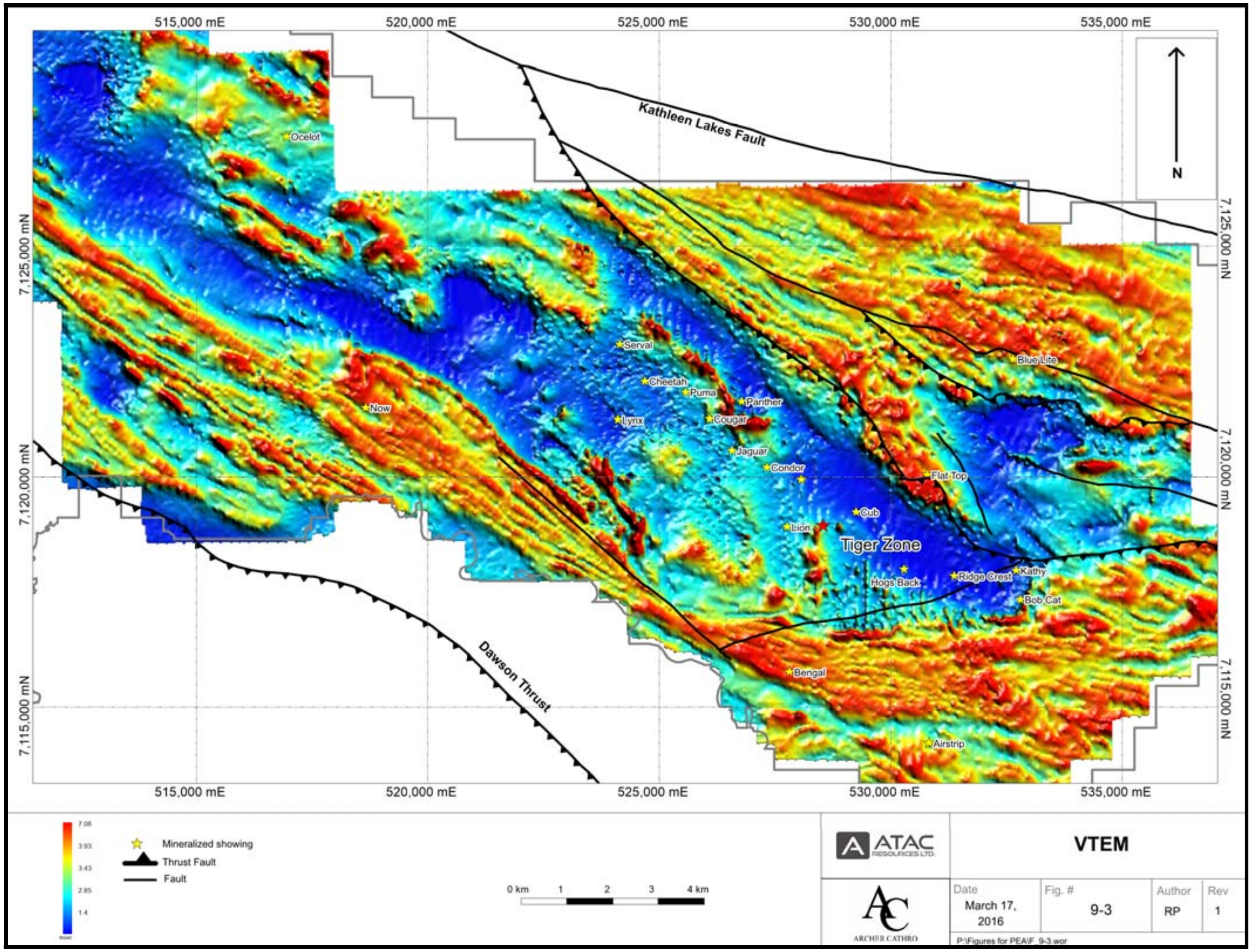
In addition to these airborne and ground based surveys, Scintrex Limited was contracted to conduct a specialized borehole gravity meter survey in a number of drillholes that intersected significant oxide mineralization for purposes of determining in situ specific gravity. This survey is discussed in more detail in Section 14.6.4.

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 Anomoilies



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.

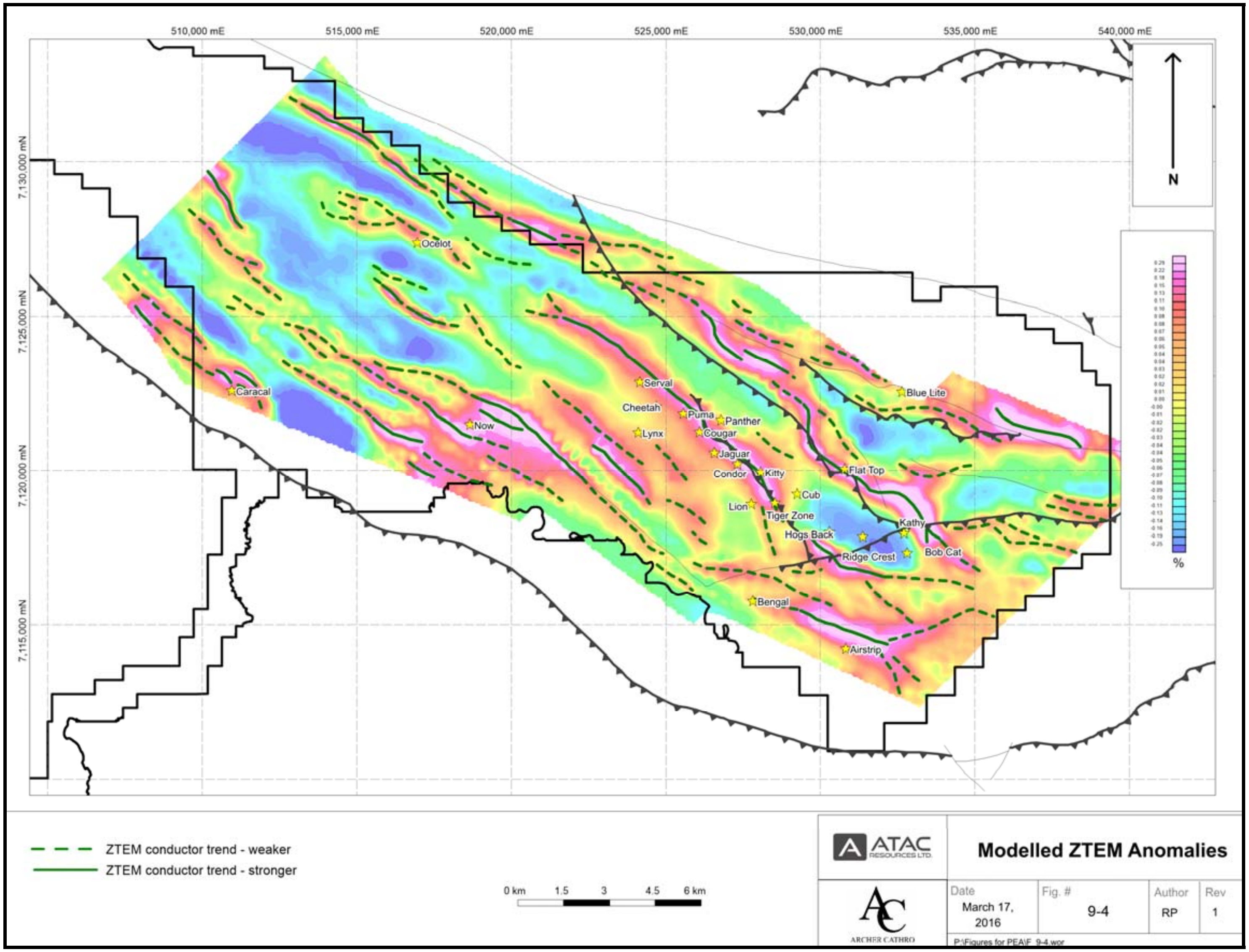
Strings of ZTEM conductors were identified throughout the Property, many of which correspond with the VTEM conductors. A number of new anomalies were also identified a short distance from the Tiger Deposit with similar geophysical signatures which were the focus of some of the 2010 exploration. Figure 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.

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 2015, which was provided by ATAC. No diamond drilling was conducted between 2011 and 2014. 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 26,843.6 m of exploration and definition drillholes were completed through 2015 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 178.8 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

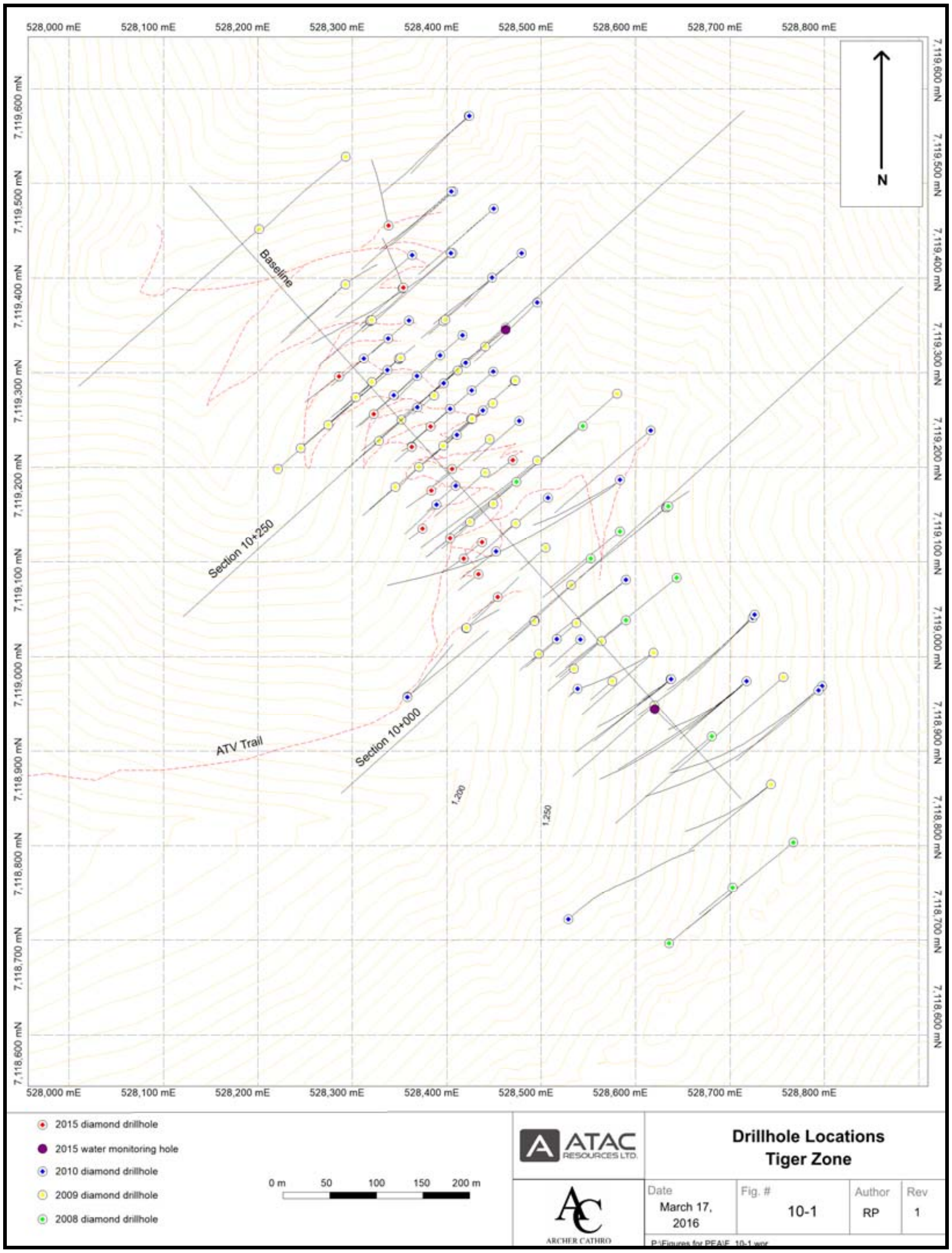


Table 10.1 Tiger Deposit Mineral Resource Database Summary

Year	Holes Drilled	Total Drilled (m)
2008	18	3,375.89
2009	53	8,651.09
2010	61	13,398.02
2015	18	1,418.60
Total	150	26,843.60

Some of the 2015 drilling was done in part for geotechnical and environmental purposes. To monitor water levels, vibrating wireline piezometers (VWPs) were installed in two holes.

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 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. 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

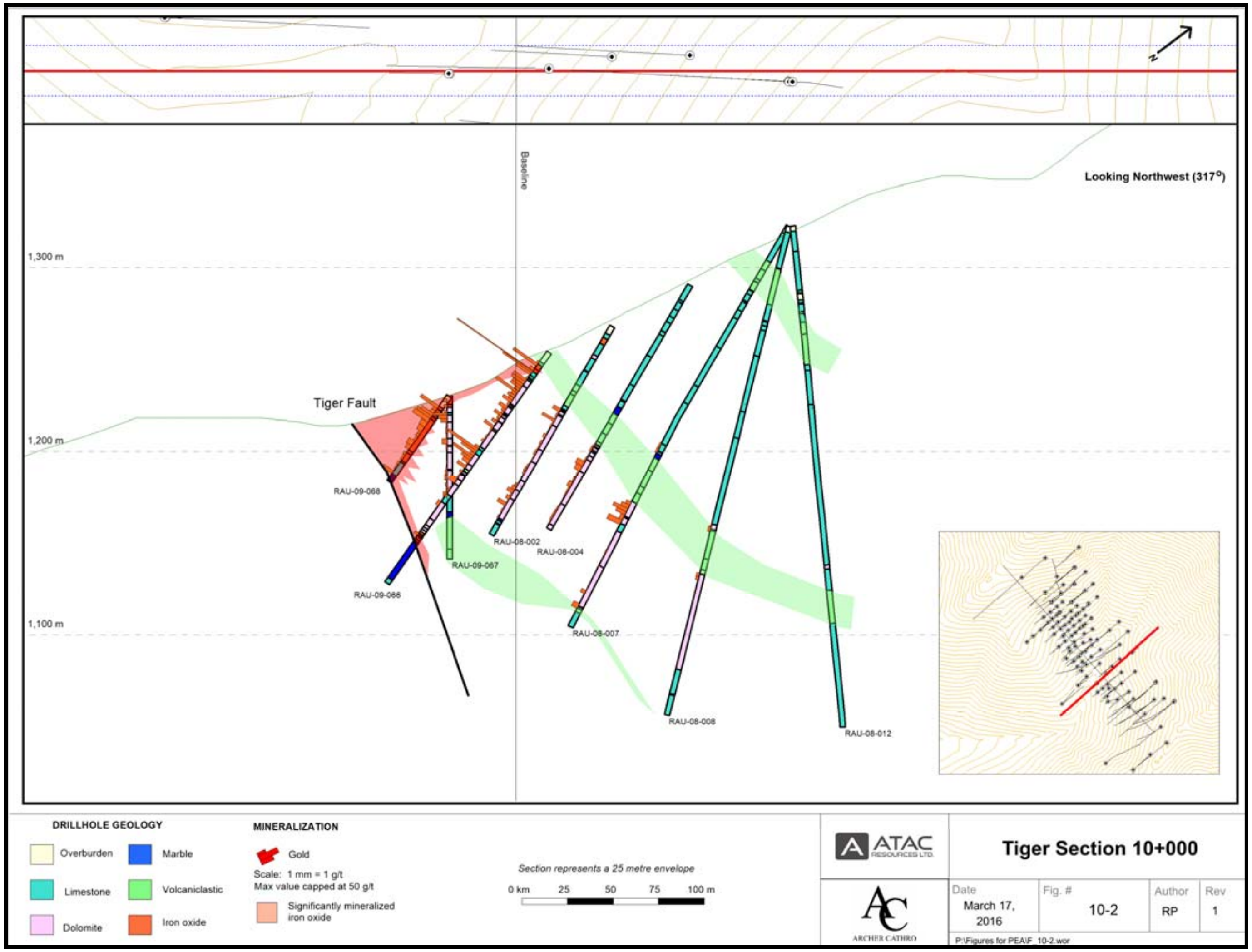
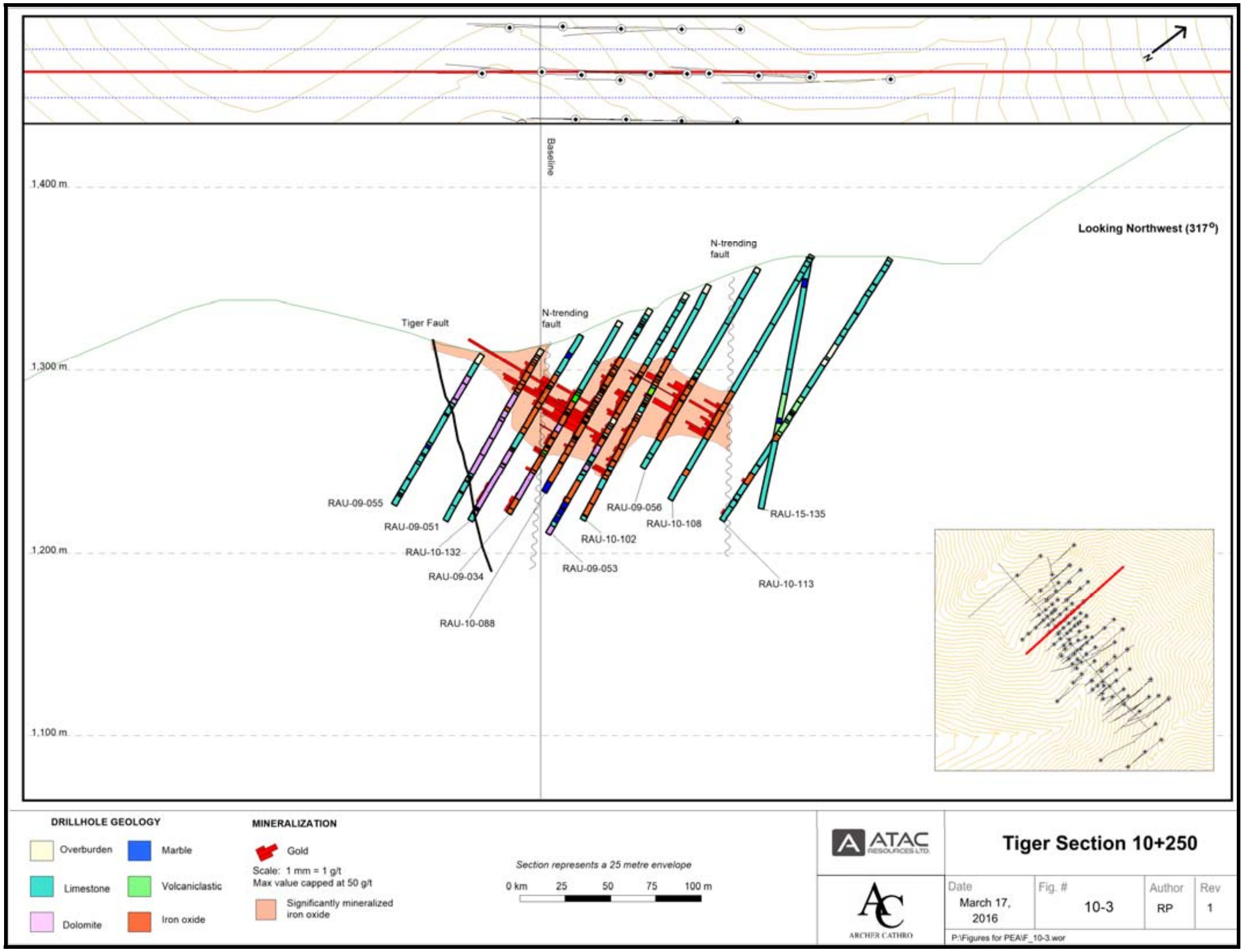


Figure 10.3 Tiger Section 10+250



Oxide mineralization is 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, as is the gold distribution within the mineralized horizon.

Figure 10.4 shows the distribution of gold grades within the block model, which is discussed in more detail in Section 14.0. The figure illustrates the elevated grades associated with the northerly trending extensional faults, as well as the segregation between oxide and sulphide mineralization.

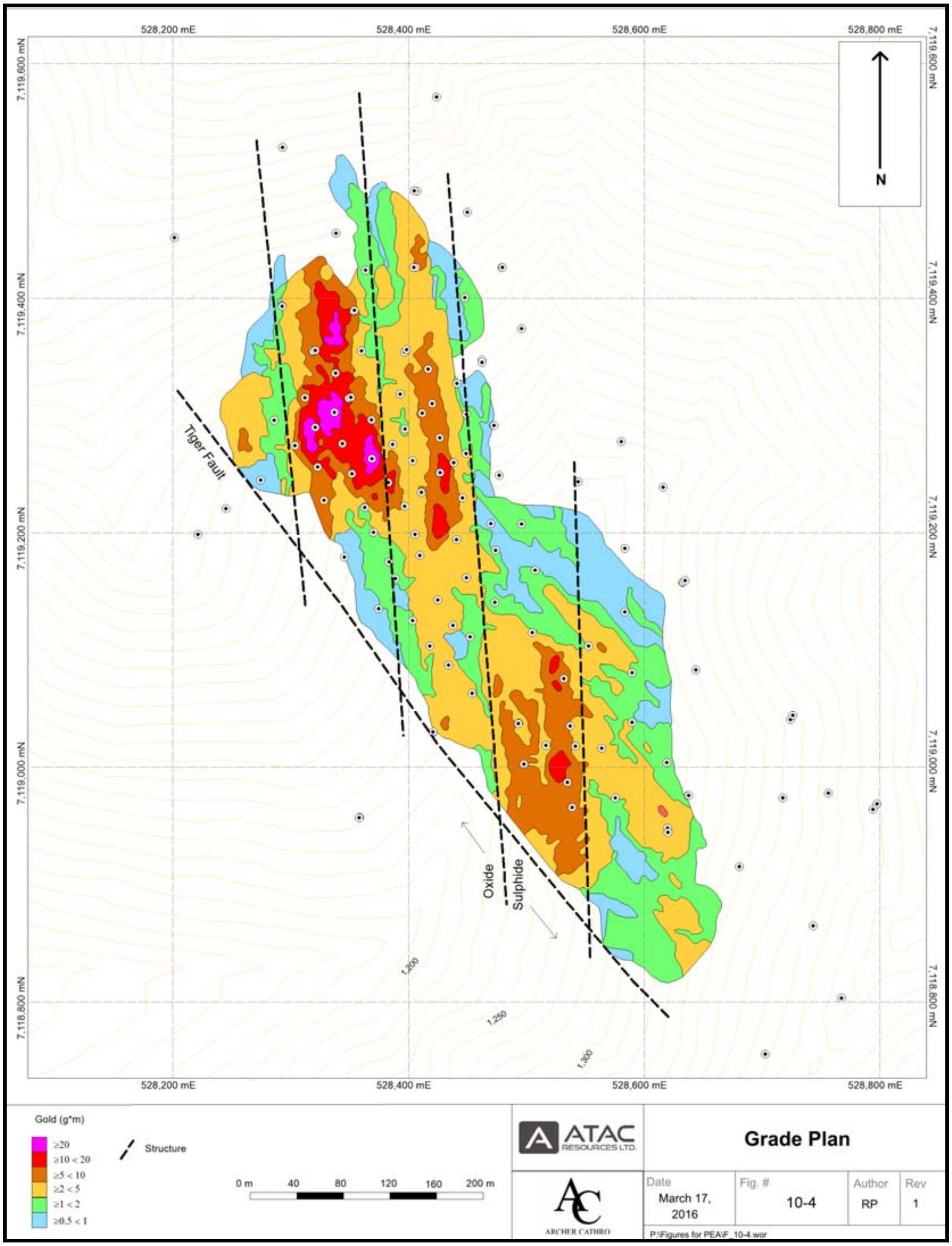
10.2 DIAMOND DRILLING SPECIFICATIONS

All drill campaigns were contracted to Superior Diamond Drilling of Kelowna, British Columbia which conducted all 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.

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.

Figure 10.4 Grade Plan



10.3 Core Logging Procedures

Core logging started very basic in 2008 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, logging was recorded as hardcopy and then entered into a Microsoft-SQL Server® database (the Database). All of the pre-2015 data were transferred to 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. Core recovery was calculated for each sample interval.
7. 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.
8. 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.
9. Two blank and two standard samples were randomly included in every batch of 31 core samples.
10. Duplicate samples were collected by quartering sample intervals after the initial half split. One duplicate was included in every batch of 31 core samples. In 2015, a coarse reject duplicate sample was included in batches comprising 30 core samples.

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. 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.

A total of 133 point load measurements were taken on core in 2015 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 each of the 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:

*Density = weight in air / [Pi * (diameter of core / 2)^2 * 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 Trimble SPS882 and SPS852 base and rover 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, 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. Except in 2008 and 2015, all surveys were completed using a "Ranger Explorer" system provided by Ranger Survey Systems. Holes completed in 2008 and 2015 were surveyed using multi-shot tools provided by Icefield Tools and Reflex, respectively.

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.

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. The QP has supervised the 2008 to 2010 exploration programs and reviewed the data from the 2015 program. 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. In 2015, samples were flown by helicopter to the Rau Airstrip and transferred to a fixed wing aircraft and flown to Mayo. From here they 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.

In 2010 and 2015, 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 any affected samples were not processed until a resolution was reached regarding the security of the samples.

Also in 2010 and 2015, 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. In 2015, samples 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% passing -2 mm, before a 250 g split was taken and pulverized to better than 85% passing -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 routinely 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 sulfide /oxide content in the core. It is the opinion of the QP that the drill core sampling is reliable and is representative of the mineralization with the Tiger Deposit.

11.3 QUALITY ASSURANCE AND QUALITY CONTROL

For all of its exploration programs, ATAC routinely inserted certified reference materials (CRMs), blanks and duplicates into each batch. 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. All assay standards were certified by Smee & Associates Consulting Ltd. of North Vancouver, British Columbia.

In 2015, batches comprised 30 samples. Two standards and blanks were inserted into the sample sequence in each batch. Standards were placed randomly, while blanks were placed following visually mineralized intervals where possible. One quarter-core duplicate was also inserted into each batch at random locations chosen by the geologist while logging. One sample in each batch was selected at random and a duplicate pulp sample was created from the original coarse reject material and analyzed at the same time as the rest of the batch. Pulp duplicates were not included in sample batches prior to 2015.

Table 11.1 summarizes the number of quality assurance (QA)/quality control (QC) samples analyzed each year during drilling within the Tiger Deposit by ATAC. No drilling was conducted at the Tiger Deposit between 2011 and 2014.

Table 11.1 QA/QC Samples by Year

Year	All Samples	Core Samples	Standards	Blanks	Quarter-core Duplicates	Coarse Reject Duplicates
2008	816	737	35	44	0	0
2009	2,590	2,229	148	142	71	0
2010	3,390	2,917	182	188	96	0
2015	453	386	22	23	11	11
Total	7,249	6,269	394	397	178	11

Note: Coarse reject duplicates, as defined in this report, are a second pulp prepared from the same coarse reject material as the main sample.

Results from the QA/QC program are reviewed immediately upon receipt. Over time as data were accumulated, results are reviewed to identify potential biases and other issues.

ATAC's 2015 QA/QC program comprised 453 samples, including 22 CRMs, 23 blanks, 11 quarter-core duplicates, and 11 pulp duplicates.

A total of 268 pulp and coarse reject samples were randomly selected from the 2009 exploration program and submitted to Acme Analytical Laboratories Ltd. for re-analysis. These samples were selected using a random number generator in Microsoft® Excel. No external check analysis was completed during the 2008, 2010 or 2015 programs.

Below are summaries of results for all of the QA/QC programs.

11.3.1 RESULTS OF CERTIFIED REFERENCE MATERIALS

A total of thirteen different CRMs have been used during ATAC's drill programs. Different CRMs were used for batches containing primarily samples of oxide material and sulphide. Oxide CRMs were obtained from Geostats, while sulphide CRMs were made from coarse reject material from previous year's drilling. One sulphide CRM, used in 2008, was obtained from CDN Resource laboratories. The CRMs all have certified gold values which is monitored in ATAC's QA/QC process. Although not certified, all other elements are routinely inspected for indications of sample switches or analytical errors.

Table 11.2 shows the recommended values for the CRMs used during all of the drill programs at the Tiger Deposit.

Table 11.2 Recommended Values of Certified Reference Materials

ID	CRM Name	Type	Au (g/t)	Standard Deviation
CDN15A	CDN-GS-15A	Sulphide	14.830	0.305
RESS-1	2010-D	Sulphide	2.720	0.175
ROS-1	G306-1	Oxide	0.410	0.030
ROS-2	G399-2	Oxide	1.460	0.090
ROS-3	G999-4	Oxide	3.020	0.170
ROS-4	G306-3	Oxide	8.660	0.330
ROS-5	G912-7	Oxide	0.420	0.020
RSS-1	RAU-1	Sulphide	0.514	0.029
RSS-2	RAU-2	Sulphide	1.527	0.067
RSS-3	RAU-3	Sulphide	5.705	0.249
RSS-5	2010-A	Sulphide	0.437	0.024
RSS-6	2010-B	Sulphide	1.830	0.105
RSS-7	2010-C	Sulphide	3.830	0.165

Table 11.3 summarizes results from ATAC's 2015 QA/QC program. For information on QA/QC programs prior to 2015, please see the Technical Report dated September 4, 2014.

Table 11.3 2015 Results of Certified Reference Material

ID	Expected Au Value (g/t)	No. of Assays	Warnings	Fails
ROS-3	3.02	10	0	0
ROS-4	8.66	4	0	1
ROS-5	0.42	6	1	0
RSS-6	1.83	1	0	0
RSS-7	3.83	1	1	0
Total	-	22	2	1

A CRM fails when the assay value is outside three standard deviations of the mean. When a CRM assay fails or there are two warnings within a batch, the assay batch is re-run. A warning is generated when a CRM is outside of two standard deviations of the mean, but still within three.

11.3.2 RESULTS OF BLANK SAMPLES

Coarse blanks test for contamination during both the sample preparation and assay process. Two blanks are inserted into each batch sent to the laboratory.

Blanks are prepared from commercially available marble and are kept in bags in the core shack, away from any possible sources of contamination. Blank samples are prepared in advance and are weighed to ensure the total mass of each blank is close to the average mass of samples submitted. A total of 23 blank samples were inserted into the sample sequence during ATAC's 2015 program.

A warning was generated if a blank returned greater than 10 ppb gold, two times the detection limit (5 ppb gold). All warnings were investigated. In the event of a failure, new pulps are prepared for the whole batch and assayed.

Early in the 2009 program, a higher than normal number of warnings were generated for blanks inserted into batches containing oxide material. This was determined to be related to the sticky, clay rich nature of the oxide material. To mitigate this, the laboratory was requested to perform a second cleaning of preparation equipment using a quartz sand. This lead to a decrease in all warnings generated. This process has been used in all subsequent drill programs.

No warnings were generated in 2015. For details of the blanks in programs prior to 2015, please see the Technical Report dated September 4, 2014.

11.3.3 RESULTS OF DUPLICATES

ATAC resource analyzed a total of 184 quarter-core (field) duplicates to test for repeatability in drill programs completed between 2008 and 2010, and in 2015. These samples were selected at random during core logging. As a result, only 88 field duplicates were greater than 0.10 g/t, the ten times detection limit. Below this, higher variance is expected as the values are nearer the detection limit, where precision is lower.

In general, duplicate samples are consistent with the original values. Sample with values below 1.0 g/t showed an average increase of 15%, while samples above this showed an average decrease of 11%. The greater variability at lower-grades can be expected, as even a minor change in the value results in a significant percentage change. While some nugget effect is likely present, the decreased grades obtained from the duplicates of the higher-grade samples is likely partially attributed to the smaller sample size of the quarter core.

In 2015, ATAC included coarse reject duplicates in each of the sample batches. A total of 11 coarse reject duplicates were analyzed. Of these, eight were greater than 0.10 g/t. The variability seen between duplicates and their corresponding original samples were within the expected range.

11.3.4 RESULTS OF EXTERNAL CHECK ANALYSIS

A total of 136 coarse rejects and 132 pulp rejects from core samples analyzed in 2009 by ALS Minerals were randomly selected for check analysis. Samples were chosen using a random number generator within Microsoft® Excel. These samples represent approximately 12% of the samples analyzed in 2009. These samples were submitted to

Acme Analytical Laboratories Ltd. (Acme) in Vancouver, British Columbia, now Bureau Veritas Minerals Laboratories (BVM). Samples were analyzed for gold by 50 g fire assay followed by ICP with emission spectroscopy (ES).

Results from the Acme assays are consistent with the assays completed by ALS Minerals. The greatest variability occurs in samples that initially yielded 0.1 g/t (ten times the detection limit. Below this, higher variance is expected as the values are nearer the detection limit, where precision is lower. Above this, 72% of the sample pairs showed less than 20% relative difference, with the greatest variation in samples grading greater than 40 g/t.

11.4 DATA VALIDATION

The QP has supervised the exploration programs at the Property from 2008 through 2010 and has reviewed the 2015 program. He has helped establish the data collection and quality control procedures used since 2008. He was directly involved in daily operations between 2008 and 2010.

Over the duration of each field program, sample information, drillhole surveys, drill logs, and other collected data were reviewed on a daily basis and corrections immediately made if necessary. Any changes to the collection procedure were made or additional training was provided as needed.

Drillhole locations, downhole surveys and mineral intersections were plotted as they became available. These were inspected and compared to the existing geological model. Any discrepancies identified were investigated further and addressed as needed. In addition to the QA/QC procedures outlined in Section 11.3, assay data was routinely spot checked against the original ALS Minerals assay certificates.

Prior to commencing the updated Mineral Resource estimate in fall 2015, geotechnical, geological, sample, mineralization and density logs were reviewed. Intervals were checked for missing data, overlaps and data entry errors. Spot checks were performed against original paper logs where available. Any erroneous data were reported and steps were taken to either correct the errors or remove the affected data from further use.

The following sub-sections provide details of the data validation procedures, primarily focusing on data associated with the diamond drilling.

11.4.1 DATABASE VERIFICATION

Prior to 2015, geological and geotechnical logging was initially recorded as a hardcopy and then transcribed into Microsoft Excel®. In 2015, logging was recorded as hardcopy and then entered into the Database. All of the pre-2015 data has been transferred to the Database. Deposit modeling and the Mineral Resource estimate was completed using exports from the Database in comma separated values (.csv) format.

Algorithms within the database automatically check all data as it is entered to ensure accuracy and consistency. These checks include interval checks that alert the user if overlapping or missing intervals are detected. Alerts are also generated if a downhole depth has been entered that is greater than the final hole depth. Drop-down menus and internal libraries ensure consistency between users by requiring the use of pre-approved lithological units, minerals and other logging codes.

Visual comparisons of hardcopy data and digital data were conducted on selected holes to ensure accuracy. Any discrepancies identified by this process were investigated, by examining the core stored on the Property, and corrected.

11.4.2 COLLAR LOCATION VERIFICATION

Drillhole locations were determined using a local survey grid and total stations survey equipment in 2009 and 2010. Once complete, each drill collar was surveyed using a Trimble RTK GPS system. In 2015, drillhole locations were determined using the RTK GPS.

Drillhole collars from previous years were re-surveyed in 2015 and corrected where necessary.

Elevation data obtained during the RTK GPS survey were compared to elevation data calculated from low level orthorectified photographs. Any discrepancies identified were investigated and corrected, if possible. If no resolution to a discrepancy was immediately apparent, an additional RTK GPK survey was conducted.

11.4.3 ASSAY VERIFICATION

During the 2008 to 2010 drill programs, digital assay certificates were obtained from ALS Minerals and manually merged with sample data in a Microsoft Excel® spreadsheet. These spreadsheets were used prior to 2014 to track sample results and for the 2011 Mineral Resource estimation.

In 2014, sample data from the drill programs was entered into the Database. New digital assay certificates were requested from ALS Minerals at this time, obtained in .csv format, and were imported directly into the Database.

Internal algorithms built into the Database ensure that the correct assay data were matched with the correct sampling data. Errors detected by the Database were inspected and corrected. Spot checking of data within the Database against hard copy certificates issued by ALS Minerals was also implemented and did not reveal any issues.

Sample data was exported from the Database in .csv format for use in the 2015 Mineral Resource estimation.

11.5 CONCLUSIONS

It is the QP's opinion that the results obtained during ATAC's exploration programs are representative of the mineralization of the Tiger Deposit and that sample preparation, quality control, security, and analytical procedures for work conducted on the Property meet the standards as set out in NI 43-101.

12.0 DATA VERIFICATION

M. R. Dumala was directly involved in and supervised diamond drill programs at the Tiger Deposit between 2008 and 2010, and supervised a regional exploration program along the Rau Trend in 2011. He has verified the data collected during the 2015 diamond drill program and all pre-2015 data. This verification consisted of:

- review of sampling and logging procedures
- review of validation algorithms within the Database
- spot check assay certificate data with Database .csv exports
- review of QA/QC procedures
- inspection of QA/QC results
- inspection of oriented core data
- re-survey and cross check pre-2015 drillhole collar locations
- review of geological model
- visual inspection of cross sections showing assay and lithological data overlaid onto the geological model

In the QP's opinion, the geological data is suitable for the purposes of Mineral Resource estimation and was collected in line with industry best practice as defined in the CIM Exploration Best Practice Guidelines and the CIM Mineral Resource, Mineral Reserve Best Practice Guidelines.

13.0 MINERAL PROCESING AND METALLURGICAL TESTING

13.1 SEQUENCE OF MINERAL PROCESSING AND METALLURGICAL TESTING

Nine significant mineral processing, metallurgical, and mineralogical programs have been conducted on samples from the Tiger deposit (Table 13.1)

Table 13.1 Test Work Completed for the Tiger Deposit

Year	Test Work Report
2010	Petrographic Report on Six Polished Section from the Rau Project, Yukon
2010	G&T Metallurgical Services, Scoping Level Metallurgical Testing Rau Gold Project
2010	Surface Science Western Reference 05310.ARC.Final report (general mineralogy of three concentrates
2011	SGS Lakefield Project 12510-001, An Investigation into the Extraction of Gold from Rau Gold Project Samples
2012	SGS Lakefield Project 12510-001 and -002, An Investigation into the Extraction of Gold from Rau Gold Project Samples
2012	SGS (Vancouver) Coarse Leach Study Project 50266-001
2014	Kappes Cassidy and Associates – Tiger Project Report of Metallurgical Test Work
2015	Innovat Mineral Process Solutions Ltd., Preliminary Scoping Work – Tiger Gold Project
2016	Blue Coast Research PJ5190 – Tiger Deposit Metallurgical Testwork Report

G&T Metallurgical Services (G&T) conducted a scoping-level metallurgical test program on sulphide samples in 2010. This program included automated mineralogical work (Quantitative Evaluation of Minerals by Scanning [QEMSCAN®]), whole-ore cyanidation test work, flotation testing, and pressure oxidation/cyanide leaching of the flotation concentrates, and leaching of the flotation tails.

Also in 2010, concentrate samples from the G&T program were submitted to Surface Science Western (SSW) for mineralogical characterization of gold. The combination of the G&T metallurgical work and the SSW mineralogical work started to build a picture of the potential metallurgy of the Mineral Resource.

In 2011, SGS Lakefield conducted work on sulphide samples, including flotation and bacterial leaching of the flotation concentrate, followed by cyanidation of the bioleach residue. In addition, leach tests were conducted on the untreated concentrate.

In 2011, SGS Lakefield completed work on oxide samples using samples from a single hole in the heart of the oxide zone. This work included mineralogical characterization and bottle roll and preg-robbing testing. In 2012, SGS Mineral Services in Vancouver, British Columbia (SGS Vancouver) completed initial column leach testing, and this was followed up by more extensive column leach work completed by Kappes Cassidy and Associates (KCA) in 2014. High cement requirements and poor predicted percolation rates led to the decision to evaluate the Innovat continuous vat leaching (CVL) process in 2015.

In 2016, Blue Coast Research (BCR) conducted a variability program focused entirely on agitated leaching of both the sulphide and oxide materials. This program aimed to provide a sufficiently robust database to allow for defendable metallurgical projections to a PEA level of confidence.

13.2 Oxides

13.2.1 Samples Tested

Figure 13.1 shows the drillhole locations of the samples tested in the 2011 SGS Lakefield program, the 2014 KCA column leach program, and the 2016 BCR program. The KCA test program was conducted on near-surface auger samples, and the SGS Lakefield and BCR test work was conducted on crushed drill core. Drillholes RAU-10-079, RAU-10-122, RAU-10-125, RAU-10-121, RAU-10-127, RAU-10-150, RAU-10-068, and RAU-10-126 were all used to create the composites tested in the BCR program.

13.2.1 Oxide Mineralogical Analysis

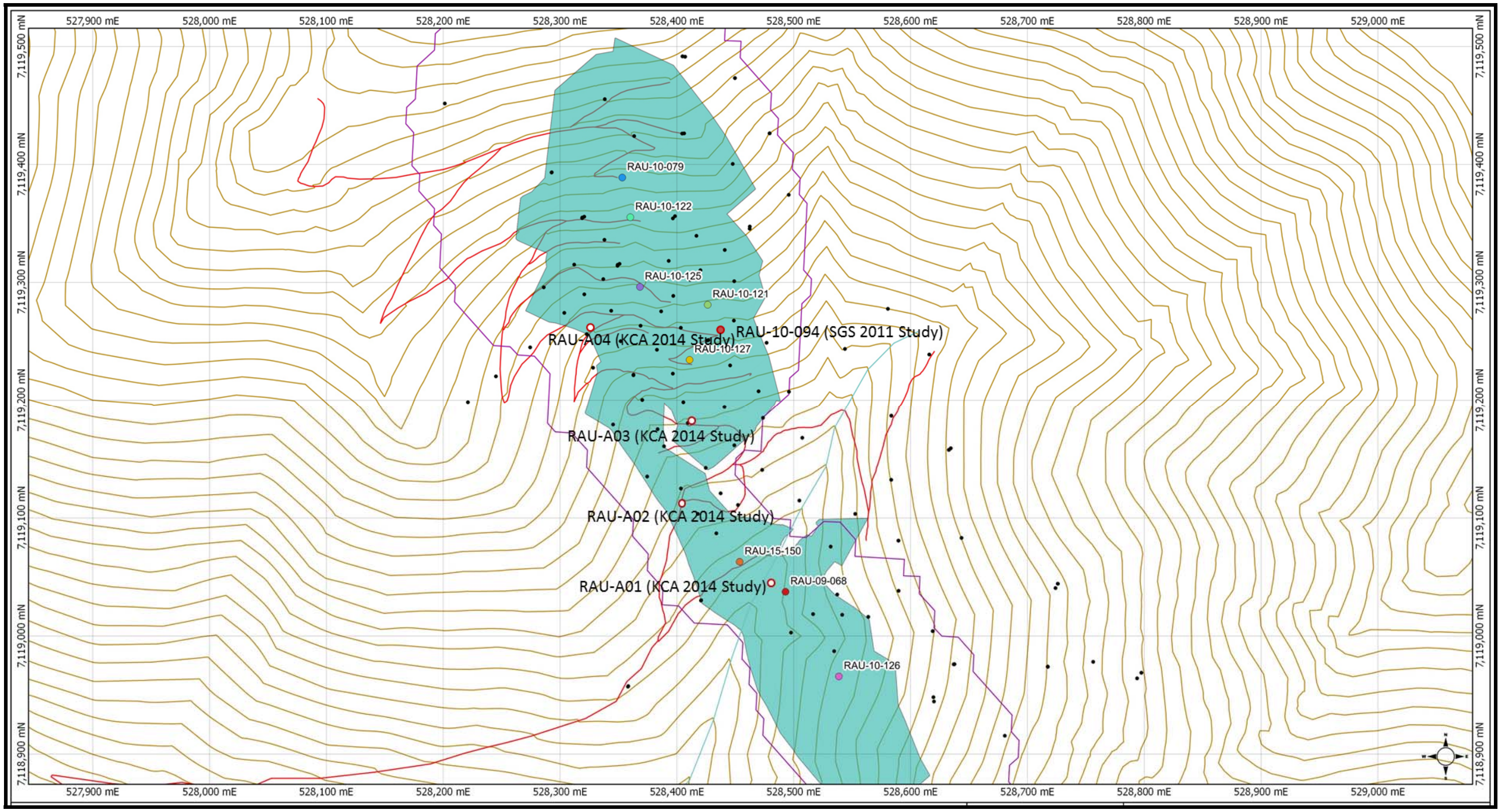
In 2010, SGS Lakefield conducted a rapid mineral scan. This brief study described the oxide sample from drillhole RAU-10-94 as dominated by quartz and dolomite with a moderate abundance of goethite and minor talc, calcite, mica, and hematite. Trace amounts of pyrite were also noted; this is usually as remnants within goethite. Acid-based accounting (ABA) studies pointed to the presence of small amounts of acid leachable sulphates; however, the neutralization potential (NP)/acid potential (AP) ratio was high at 89.9.

No gold mineralogical studies were conducted, but screened metallic work pointed to the presence of very little coarse gold, while gravity test work using Knelson™ concentration recovered 5 to 18% gold. Overall, the data points to the likely presence of relatively fine discrete gold as the dominant form of the metal.

13.2.1 Comminution Test Work

A single grindability Bond Ball Mill Work Index test has been conducted to date, yielding a hardness of 8.5 kWh/t.

Figure 13.1 Source of Oxide Samples Tested for Extraction by Cyanidation



13.2.2 Heap Leach Studies

In 2012, SGS Lakefield conducted coarse crush bottle roll tests, which yielded very good leach recoveries of 86% or higher, even at a crush size of 1 inch (Figure 13.2).

Table 13.2 Coarse Crush Bottle Roll Test Results

Test ID	Sample ID	Feed K_{100} (inch)	Pulp Density (%)	Leach Time (h)	Cyanide Consumed (kg/t)	Lime Consumed (kg/t)	Au in Residue (g/t)	Au in Calculated Head (g/t)	Au Recovery (%)
CN 1	Master Composite	1	50	192	0.41	3.21	0.61	4.27	-
CN 2	Master Composite	3/4	50	192	0.50	3.08	0.51	3.63	85.9
CN 3	Master Composite	1/2	50	192	0.47	3.12	0.48	3.81	87.4
CN 4	Master Composite	1/4	50	192	0.55	3.40	0.53	4.15	87.2

Source: SGS Lakefield (2012)

However, follow-up column studies, initially at SGS Vancouver and later at KCA in Reno, Nevada, encountered problems with percolation, which could only be resolved through agglomeration with high doses of cement. SGS Vancouver used 10 kg/t of cement which allowed for a successful column test, and yielded an 89% gold recovery. Lime and cyanide consumptions were not heavy and would likely be lower in practice.

KCA conducted follow-up studies on near-surface material. Coarse crush tests once again yielded excellent gold extractions of 86 to 89%; however, compacted column tests revealed the same problems with permeability that were seen in the SGS Vancouver work, with extraction levels dropping to 12 to 21%. The use of agglomeration, at cement addition rates of 2 kg/t yielded mixed results—good extractions, but the physical characteristics of one of the tests points to continued potential problems with compaction and permeability. Tests with higher doses of cement worked better, yielding 89% gold extraction. Figure 13.3 shows the results from four tests under a compressive load equivalent to 47 m.

Table 13.3 Column Leach Test Work Results

Test ID	Reagent Dose (kg/t)			Slump (%)	Effluent Colour	Gold Extraction (%)	
	Cement	Lime	Cyanide			7 days	14 days
70207	2	0	2.85	27.0	Brown/Cloudy	85	89
70213	2	0	2.55	9.7	Colourless/Clear	89	89
70275	16	0	1.43	3.8	Colourless/Clear	88	88
70276	20	0	1.28	3.7	Colourless/Clear	86	86

Source KCA (2014)

The promising results from the heap leach test work prompted some extensive engineering studies culminating in the Project's first PEA (Kappes et al. 2014). However, the PEA revealed numerous technical risks and challenges associated with the heap leach option, including:

- the potential effect of the freeze-thaw cycle on agglomerate strength
- the paucity of suitable sites near the mine, prompting the need for 3 km of conveyors
- concerns over the ability to run the heap year round
- possible resistance from the Yukon government to permit a heap leach operation
- concerns over the suitability of the sulphide Mineral Resource to heap leaching.

Accordingly, heap leaching was shelved as an option for the Project.

13.2.3 AGITATION LEACH STUDIES

Two studies conducted to date have evaluated the effect of primary grind on agitation leach recovery. The first, conducted at SGS Lakefield in 2012, evaluated:

- the effect of varying the primary grind from 69 to 131 µm on direct leach recovery
- the effect of regrinding gold gravity tails to 76 to 158 µm on overall gravity/leach recovery.

The second study, completed by BCR in 2016, included a test program conducted on a composite widely representing the deposit as a whole, and evaluated the effect of grinding to 80% passing 172 and 65 µm on ensuing leach recovery.

Both studies showed a very minor beneficial effect of fine grinding. This effect was masked by variations in head grade when comparing recoveries with primary grind, but became clearer when comparing residue grades as a function of primary grind (Figure 13.2).

While most leach tests ran for 24 hours, four samples were leached to 48 hours, including one in the SGS Lakefield 2012 program and three in the BCR 2016 study. Both the SGS Lakefield and BCR studies demonstrated the challenges in achieving repeatable results; however, overall a small (approximately 1%) enhancement in recovery was achieved from the extra leach time.

The study at BCR included a variability program of leach tests, each using the same grind time (targeting a nominal 65 µm grind), 1 g/L cyanide concentrate, and 24 hours of residence time. The average gold extraction from the eight tests was 88%, the average cyanide consumption 0.29 kg/t, and the average lime consumption was 5 kg/t. Weighted for head grade, the average gold recovery rose to 91% (which matches what was achieved using the Master Composite).

Figure 13.2 Effect of Grind Size on Tails Assays

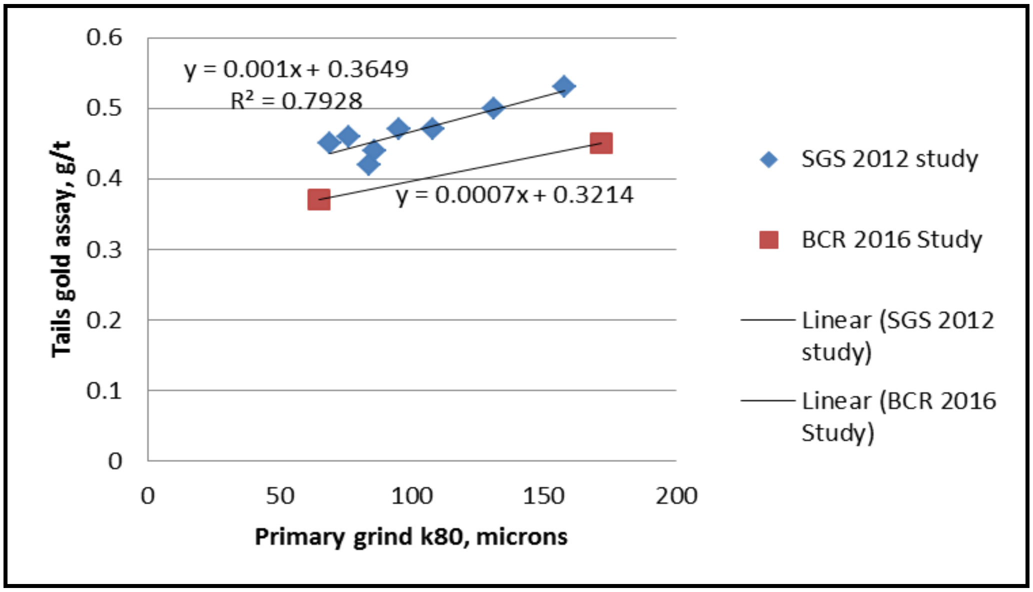


Table 13.4 Oxide Variability Test Data

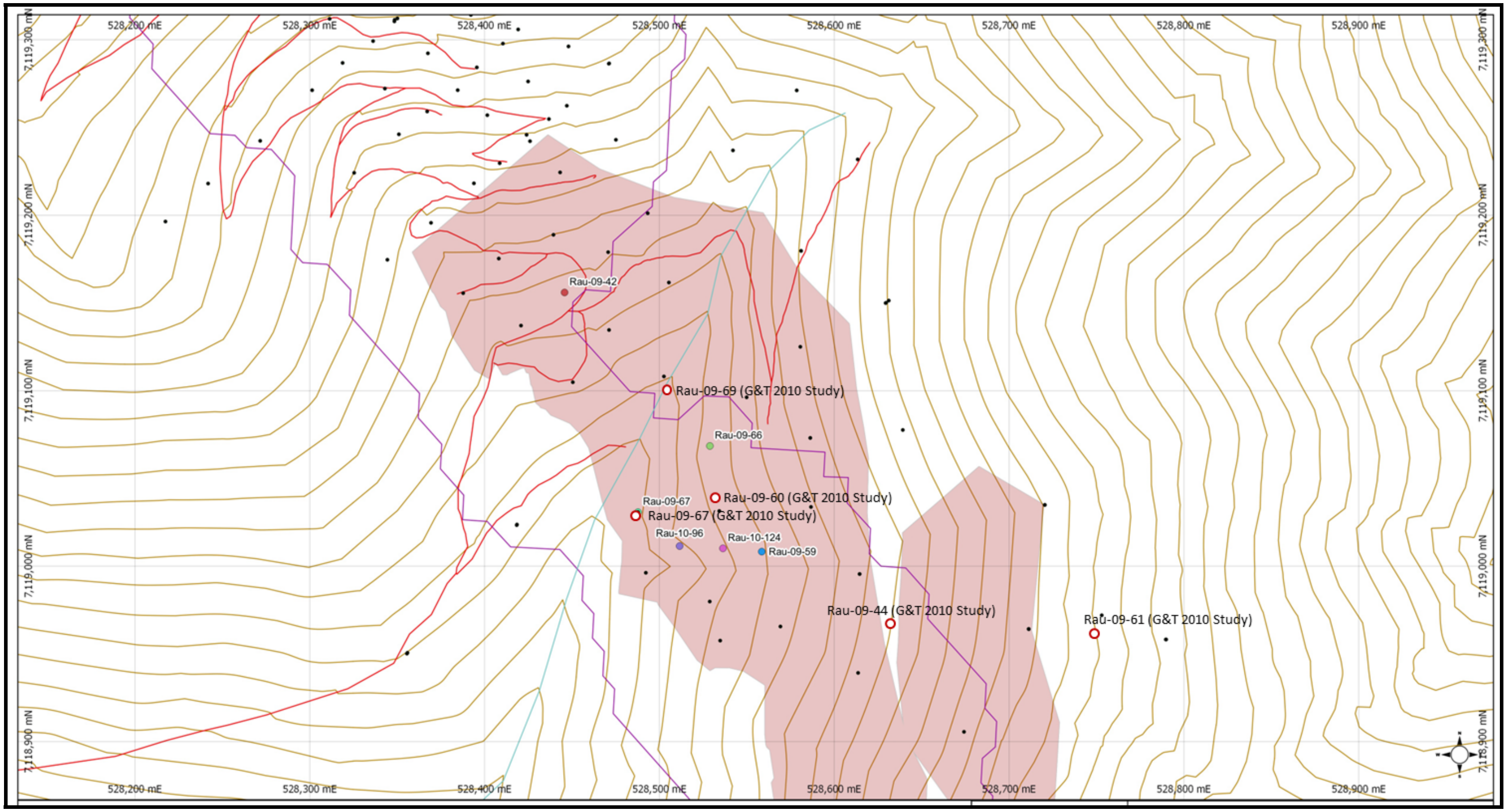
Test ID	Composite	Head Grade (g/t)	Au Recovery (%)	Cyanide Consumption (kg/t)	Lime Consumption (kg/t)
CN-21	Composite 01 (RAU-09-68)	3.26	88.2	0.33	12.00
CN-22	Composite 02 (RAU-09-79)	5.11	97.1	0.30	1.91
CN-23	Composite 03 (RAU-10-121)	1.58	92.7	0.33	3.40
CN-24	Composite 04 (RAU-10-122)	4.24	76.5	0.36	5.51
CN-25	Composite 05 (RAU-10-125)	13.5	97.7	0.32	4.30
CN-26	Composite 06 (RAU-10-126)	1.69	76.9	0.18	2.90
CN-27	Composite 07 (RAU-10-127)	2.88	80.9	0.18	3.44
CN-28	Composite 08 (RAU-10-150)	1.43	91.0	0.33	6.22
n/a	SGS 12501-001 MC1	4.82	91.1	0.27	3.94
Average	-	4.28	88.0	0.29	4.85

13.3 SULPHIDES

13.3.1 SAMPLES TESTED

The source location of the sulphide samples is shown in Figure 13.3. One composite, created from RAU-09-61, is outside of the pit boundary and on a different horizon, and was excluded from interpretation of the variability leach data (described later in this section).

Figure 13.3 Source of Sulphide Samples for Metallurgical Testing

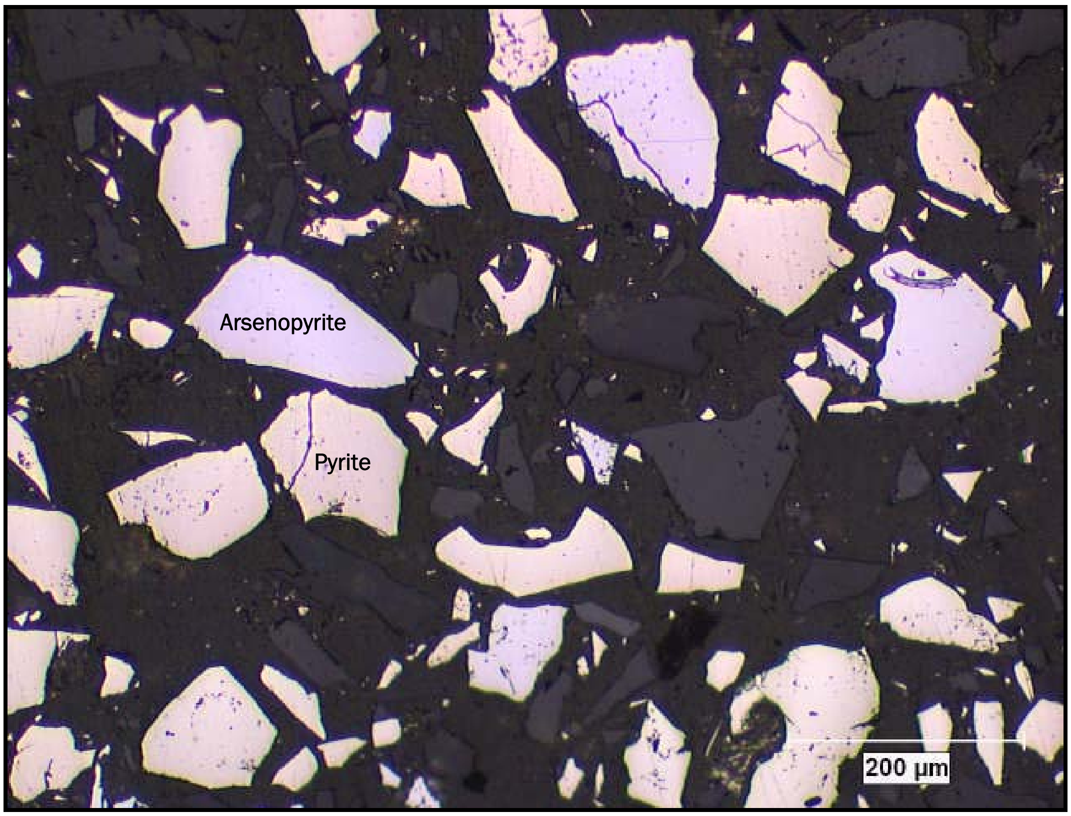


13.3.2 Sulphide Minerology

The sulphide zones comprise variable amounts of pyrite and arsenopyrite. QEMSCAN® analyses of five composites conducted in 2011 described the pyrite and arsenopyrite contents as varying between 31 to 71% for pyrite and 0 to 12% for arsenopyrite.

Non-sulphide mineralization is dominated by carbonates, comprising a mix of dolomite, ankerite, and calcite. Typically, sulphides and carbonates together comprise 85 to 90% of the mineralization, with quartz, at 1 to 7%, as the next largest component. The sulphides are coarse and generally well formed (Figure 13.4).

Figure 13.4 Photomicrograph of Tiger Sulphides



Gold occurs both as solid solution in sulphides and as discrete mineralization. The use of QEMSCAN® to determine the abundance of pyrite and arsenopyrite, coupled with Dynamic Secondary Ionization Mass Spectrometry (D-SIMS) assaying of pyrite and arsenopyrite grains for gold content, has allowed for the mineralogical balancing of free and sulphide-hosted solid solution gold. These mineralogical data pointed to the pyrite hosting 10 to 20% of the gold in the three composites tested, with 23 to 59% of the gold in the arsenopyrite, leaving 28 to 67% of the gold as discrete form within the three samples.

This agreed well with the leach data conducted on the same samples (Table 13.5).

Table 13.5 Mineralogical Occurrence of Gold in Three Sulphide Composites vs. Leach Extraction

Comp	Mineral Content		Gold Grade (g/t)			Mineralogical Balance (%)			Leach Extractions (%)	
	Pyrite	Arseno	Pyrite	Arseno	Total	Pyrite	Arseno	Discrete	70-77 µm	10-20 µm
2	31.4	10.4	2.5	17	7.6	10	23	67	65	69
3	34.4	11.9	1.7	23	4.6	13	59	28	23	30
4	24.9	5.9	2.0	11	2.4	21	27	52	42	46
Average	30.2	9.4	2.1	17	4.9	15	36	49	44	48

Note: arseno = arsenopyrite

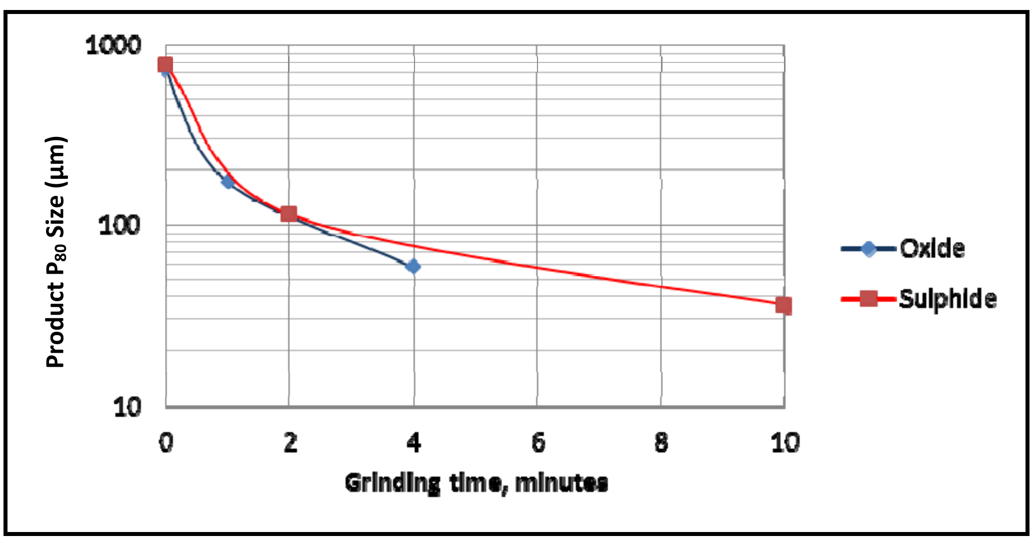
The grades of gold are generally too low for economic sulphide pre-oxidation and leaching. The gold grade in pyrite, at 1.7 to 2.5 g/t is far too low for any conventional sulphide pre-oxidation process; however, the grade of gold in arsenopyrite, at 11 to 23 g/t is more marginal. The selective flotation of arsenopyrite may lend itself to production of a concentrate that is able to be economically processed at a moderately high gold price. This has not been tested so far on the Tiger Deposit.

The discrete gold is moderately fine (as evidenced by the poor gravity recovery data), but more than 90% is leached at a grind of 70 to 77 µm. Overall, it appears likely that variability in gold recovery is driven by the solid-solution component.

13.3.3 Grindability Studies

No formal grindability studies have been conducted on the sulphide material. However, using the same laboratory mill at BCR, it took up to 50% more time to reach the same final grind size as the oxide, especially at grind sizes finer than 100 µm. This suggests a work index in the range of 10 to 12 kWh/t, which in the authors experience would be quite typical of sulphide/carbonate rich material.

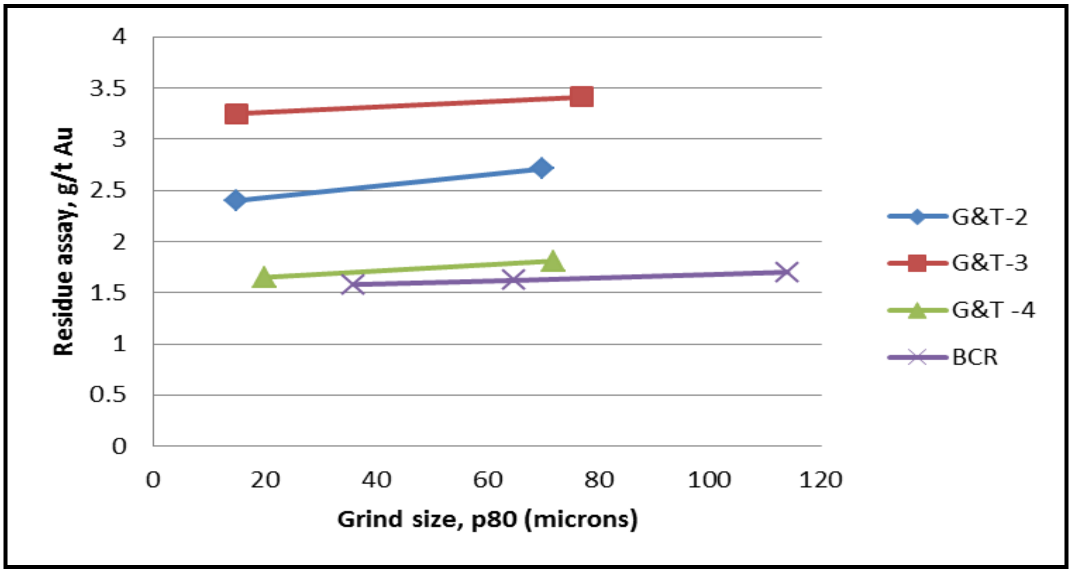
Figure 13.5 Comparative Grind Calibration Data for Tiger Deposit Sulphide and Oxide Samples



13.3.4 CYANIDATION

Figure 13.6 shows the effect of grind size on leach extraction. As with the oxides, the residue assay tends to drop slowly with successively finer grinding ahead of leaching. This has been demonstrated at G&T and BCR on four different sulphide composites. The mean rate of reduction in residue assay is 0.03 g/t gold (or approximatly 0.07% recovery) for every 10 µm finer the primary grind P_{80} is targeted.

Figure 13.6 Effect of Primary Grind Size on Leach Residue Assay



BCR tested the effect of cyanide concentration, with no meaningful benefit obtained from the use of higher cyanide addition rates.

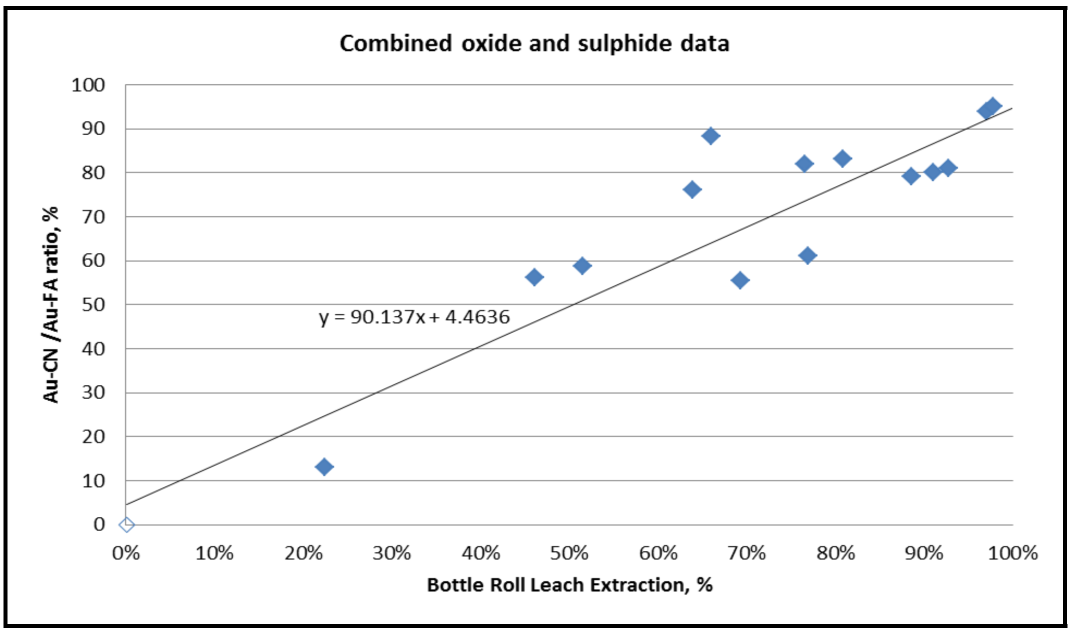
In total, 13 composites have been tested by leaching, including eight by BCR and five by G&T (one composite comprises material sourced from outside of the pit and was removed from the list in Figure 13.6). The results in Figure 13.6 show a wide variability in recovery.

Table 13.6 24-hour Leach Variability Data from Testing 12 Different Sulphide Composites

Test	Composite	Head Grade (Au g/t)	Au Rec (%)	Cyanide Consumption (kg/t)	Lime Consumption (kg/t)
CN-10	Composite S1 (RAU-09-042)	2.20	88.3	0.38	0.91
CN-11	Composite S2 (RAU-09-059)	6.12	76.1	0.74	1.16
CN-12	Composite S3 (RAU-09-066, 19-27)	7.25	87.8	0.60	0.85
CN-13	Composite S4 (RAU-09-066, 31-39)	3.91	58.8	0.58	0.95
CN-14	Composite S5 (RAU-09-067)	2.53	56.3	0.26	0.89
CN-15	Composite S6 (RAU-09-096)	6.10	13.0	0.57	1.75
CN-16	Composite S7 (Rau-09-066)	1.77	55.4	0.49	1.04
CN-17	Composite S8 (RAU-09-124, 28-31)	4.93	79.1	0.47	0.77
KM2537-01	G&T Composite 1 (hole 44)	9.07	95.3	1.20	3.10
KM2537-02	G&T Composite 2 (hole 60)	7.79	65.2	1.90	2.70
KM2537-03	G&T Composite 3 (hole 67)	4.43	23.4	1.70	2.40
KM2537-04	G&T Composite 4 (hole 69)	3.12	42.4	1.20	1.90
Average	-	4.90	61.8	0.84	1.53

Overall, including both the oxide and sulphide leach data, there is a reasonable correlation between the cyanide leachable gold (AuCN)/fire assay gold (AuFA) ratio, with the cyanide leachable gold test tending to somewhat underestimate the actual 24-hour leach recovery (Figure 13.7).

Figure 13.7 Correlation between AuCN/AuFA Ratio and Bottle Roll Leach Extraction



13.3.5 FLOTATION

Four of the five composites tested at G&T were floated, using a flowsheet employing a primary grind of 80% passing 70 to 80 µm, xanthate at doses of 140 to 160 g/t, and at natural pH. Mass pull rates to concentrate were 37 to 51%, and the concentrates assayed 30 to 36% sulphur and 5 to 17 g/t gold. Aside from Composite 1 (which also leached extremely well) the recoveries were in the range of 82 to 87% (Table 13.7).

Table 13.7 Results from Flotation of Four Sulphide Composites

	Grind (P_{80} µm)	Xanthate Dose (g/t)	Mass Pull (%)	Concentrate Grade		Recovery	
				S (%)	Au (g/t)	S (%)	Au (%)
Comp 1	76	160	43.8	30.1	17.4	94.3	97.1
Comp 2	70	140	44.6	35.9	14.9	82.1	84.1
Comp 3	77	140	50.9	35.2	7.5	92.4	82.3
Comp 4	72	140	36.8	34.1	5.4	94.3	87.3

13.3.6 CONCENTRATE PRESSURE OXIDATION AND BIOLOGICAL LEACHING WITH SUBSEQUENT CYANIDATION

Sulphide concentrates from G&T composites 2, 3, and 4 were subjected to sulphide pre-oxidation by pressure leaching and bacterial leaching at SGS Lakefield.

SGS Lakefield ran three pressure oxidation tests, with subsequent gold leaching by CIL. All three tests ran for 90 minutes at 225°C. Sulphide oxidation was complete in all the tests (99.4 to 99.8%). With the feeds running at 40% sulphur, neutralization of the CIL feed required significant lime (160 to 210 kg/t) and cyanide consumption was 12 to 15 kg/t. Gold extractions from the pressure oxidation residue were 97 to 99%. This led to overall recoveries (including flotation/pressurized oxidization /CIL) of 76 to 90%, compared to 23 to 65% by leaching alone.

Five bacterial leaching tests were conducted on a sulphide concentrate assaying 4.5 g/t gold, 8.1% carbonate, and 38% sulphur. This concentrate, when leached directly, yielded 39% gold extraction at a grind of 80% passing 52 µm, rising to 49% at a grind of 80% passing 10 µm.

The tests each ran for different retention times (10, 15, 25 and 30 days [twice]). Limestone use during and after the bacterial leach was 110 to 300 kg/t (as the Mineral Resource is rich in carbonates this may not necessarily reflect as a significant cost to the Project).

While sulphide oxidation rose from 95% after 5 days to 99% after 30 days, gold extractions were largely unchanged in all the tests at 92% to 93%. Silver extractions were 71% to 82%. Cyanide consumptions were lower than for the POX test work at 5 to 7 kg/t for any of the tests using 10+ days, and lime use was also modest for all the tests employing 10+ days of residence time.

13.4 METALLURGICAL INTERPRETATION

Gold within the Tiger Deposit occurs both in discrete and in refractory form, the latter as a solid solution within arsenopyrite, and to a lesser extent pyrite, occurring in the sulphide zone. The discrete gold is generally fine-grained, but appears to occur in a way that allows for good cyanide access even at coarser crush sizes. The grades of solid solution gold vary by sample, but current data point to a mean of 17 ppm in arsenopyrite and 2 ppm in pyrite. At such grades, arsenopyrite is probably not economic for pre-oxidation processing at current prices; however, pyrite hosted in gold is definitely sub-economic for processing.

Accordingly, recoveries are limited by the presence of gold in discrete form, which is higher in the oxide material than in the sulphide material.

Host rock mineralization is dominated by goethite and carbonates, with only very minor primary silicates. While this renders the material very amenable to grinding, it also adversely affects its ability to create strong agglomerates, making heap leaching a challenge.

Significant test work has been conducted evaluating the potential for gold recovery by heap leaching (oxides only), agitation leaching and gravity recovery (sulphides and oxides), and flotation and sulphide concentrate pre-oxidation ahead of leaching (sulphides only).

Column leaching, simulating the heap leach process, yielded excellent recoveries with only a small discount (if any) in recovery versus agitation leaching after grinding. However, the material performed very poorly under compression and agglomeration was needed. Additionally, excessive amounts of cement were needed to make the agglomerates robust enough to resist compression. An engineering assessment of the heap leach option concluded heap leaching would be risky and problematic for this material. A study investigating the use of hybrid heap/agitation leaching, using the coarser, more competent material for the heap only, also failed to yield promising economics.

Agitation leaching worked well for the oxides, and recoveries from oxide samples averaged close to 90%. While kinetics on many of the earlier samples tested were fast, some of the more recent samples leached more slowly, and leach kinetics were quite variable (even between batches of essentially the same sample). Cyanide consumption rates were modest, but lime consumptions in treating the oxides were sometimes quite high.

Gold extraction by agitation leaching from sulphide samples was highly variable, and driven by the proportion of gold present in solid solution form. Extractions ranged from 13% to 95% and averaged 62%. Both cyanide and lime consumptions were quite low in the processing of these samples.

The effect of grind size on gold extraction was minor for both sulphides and oxides, as evidenced by the good recoveries achieved, even with heap leach at a moderately coarse crush size. However, there was a small benefit from finer grinding and this, combined with a high head grade and a low work index (especially for oxide materials) means a trade-off between grinding cost and leach extraction is warranted.

Gravity recoveries were less than 20% in all samples tested.

The sulphides tended to float quite well with recoveries averaging 85% in the more refractory samples tested. Both POX and BOX were tested, both yielding very high sulphide oxidation levels. The POX yielded very high subsequent gold extractions (99%), and the BOX somewhat lower (approximately 92%). Overall, the incremental gain of gold recovery over direct cyanidation alone is approximately 20%, probably not enough to render either process economic at current metal prices.

14.0 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

At the request of Rob Carne from ATAC, Giroux Consultants Ltd. was contracted to complete a Mineral Resource estimate update on the Tiger Deposit. The update is based on additional drillholes completed in 2015. Gary Giroux, P.Eng, MASc, estimated the Mineral Resources and is a QP and independent of the both the issuer and the title holder, based on the criteria outlined in NI 43-101.

This Mineral Resource estimate supersedes the previous estimate outlined in Stroshein et al. (2011) and restated in Kappes et al. (2014). The effective date of this Mineral Resource estimate is October 28, 2015.

14.2 DATA ANALYSIS

The database consisted of 150 diamond drillholes totaling 26,844 m. This included an additional 18 drillholes completed in 2015 (Figure 14.1). A total of 6,222 assays were provided by the effective date for the Mineral Resource estimate (October 28, 2015). Gaps in the “from” and “to” records totaled 218 and represented missing or non-sampled intervals in waste areas, and no recovery in mineralized zones. Values of 0.001 g/t were inserted for missing or non-sampled intersections. Gaps where there was no material recovered within mineralized sections were left blank.

A 3D solid model was constructed by M. R. Dumala of Archer Cathro to constrain the oxide zones and the sulphide zones in dolomite between a series of confining fault surfaces. The solids are shown below in Figure 14.2.

Drillholes were “passed through” these solids with the point each hole entered and left each solid recorded. Assay values were then back tagged with a solid designation and the assay statistics tabulated in Table 14.1. Three oxide solids were modelled and the remaining lithologies were lumped into a main footwall solid and a smaller hanging wall solid sitting above the main mineralization. Thus, the assays were subdivided into oxides and sulphide bearing lithologies.

Figure 14.1 Drillhole Location Map Showing 2015 Drillholes in Red

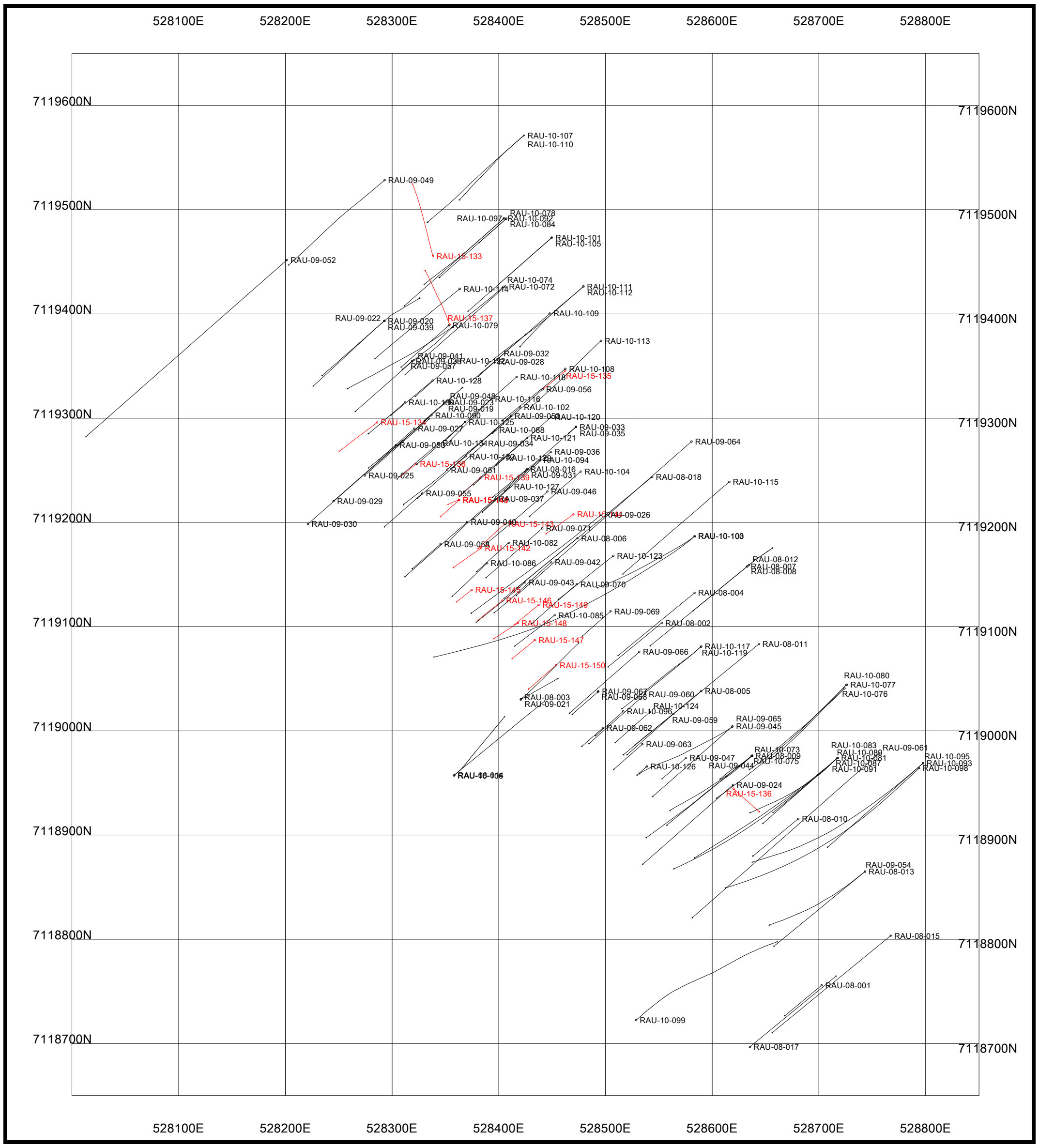


Figure 14.2 Isometric View Looking Northeast of the Oxide Solids in Orange, Sulphide Solids in Purple, Volcanic Solids in Green and Drillhole Traces

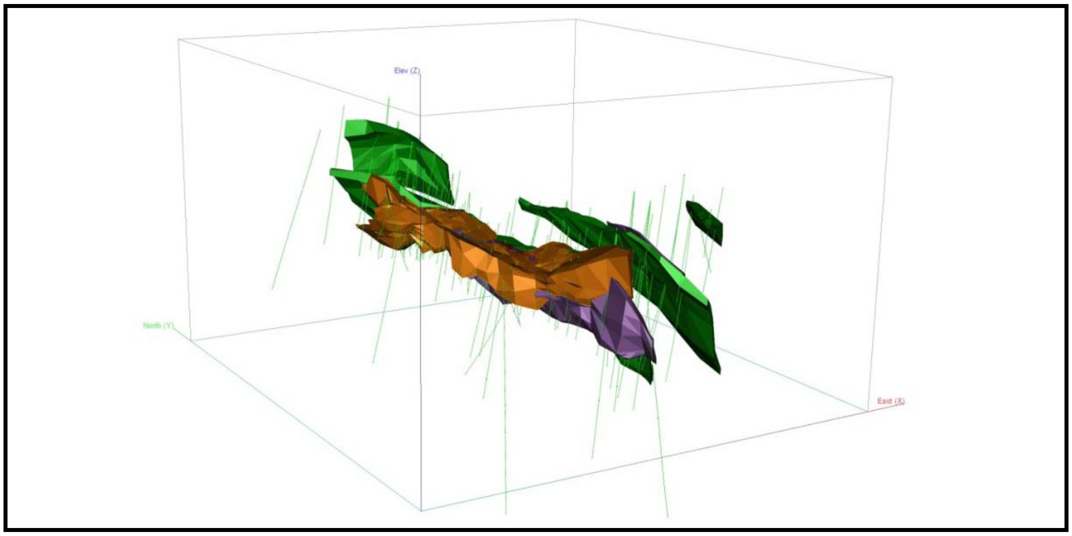


Table 14.1 shows the assay statistics for oxides, sulphides in dolomite, volcanics, and material outside these solids coded as waste.

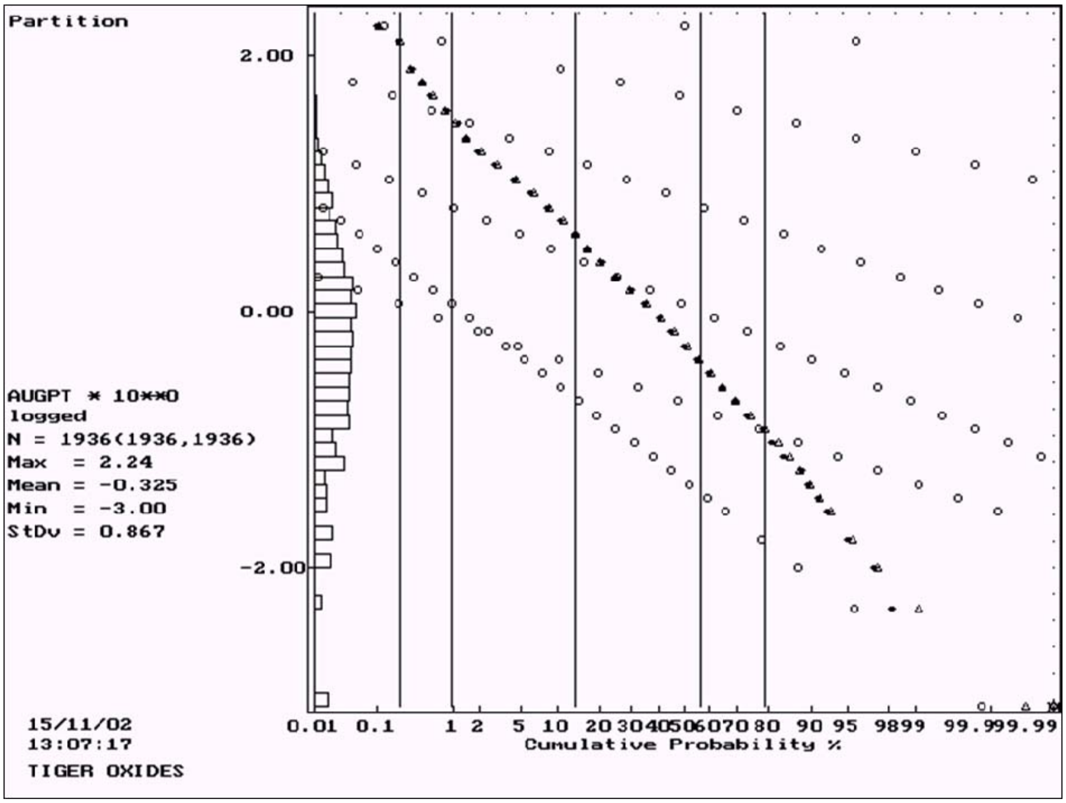
Table 14.1 Assay Statistics from Solids

	Oxides		Sulphides		Volcanics		Waste	
	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)
Number of Assays	1,934	1,934	1,055	1,055	561	561	2,888	2,888
Mean Value	2.53	10.54	1.10	0.73	0.03	0.42	0.09	1.16
Standard Deviation	9.50	151.67	2.08	2.08	0.15	2.61	0.33	6.64
Minimum Value	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001
Maximum Value	175.00	6280.00	19.35	30.10	1.89	54.60	8.13	239.00
Coefficient of Variation	3.75	14.39	1.89	2.85	4.57	6.18	3.65	5.73

Gold distribution within the mineralized solids was examined using a lognormal cumulative frequency plot to determine if capping was required, and, if so, at what level. The procedure used is explained in a paper by Dr. A.J. Sinclair titled *Applications of Probability Graphs in Mineral Exploration* (Sinclair 1974). In short, the cumulative distribution of a single normal distribution will plot as a straight line on probability paper, while a single lognormal distribution will plot as a straight line on lognormal probability paper. Overlapping populations will plot as curves separated by inflection points. Sinclair (1974) proposed a method of separating out these overlapping populations using a technique called partitioning. In 1993, a computer program called P-RES was made available to partition probability plots interactively (Sinclair and Bentzen 1993). A screen dump from this program is shown for oxide gold in Figure 14.3. On this plot the actual gold distribution is shown as black dots. The inflection points that separate the populations are shown as vertical lines, and each population is shown by the straight lines of open circles. The interpretation is tested by recombining the data in the proportions selected and is shown as triangles compared to the original distribution.

In each case the grade distributions for gold and silver were positively skewed with multiple overlapping lognormal populations present.

Figure 14.3 Lognormal Cumulative Probability Plot for Gold in Oxides



In the case of gold in oxides a total of six overlapping populations were identified as tabulated in Table 14.2.

Table 14.2 Gold populations in Oxide Domain

Population	Mean Au (g/t)	Percentage of Total (%)	Number of Assays
1	166.7	0.21 %	4
2	46.75	0.75 %	15
3	7.41	12.70 %	246
4	1.11	42.93 %	831
5	0.19	22.61 %	438
6	0.05	20.80 %	402

Population 1, with a mean grade of 166 g/t gold and representing 0.21% of the data, can be considered erratic outlier mineralization. The samples in this population are scattered through the zone and don't represent a cohesive zone. A cap level of two standard deviations above the mean of Population 2 was chosen to cap Population 1 assays. A total of four gold assays in oxides were capped at 90 g/t.

A similar exercise was completed for silver in oxides. The top population averaged 947 g/t silver and represented 0.32% of the data and was capped at two standard deviations above the mean of Population 2. A total of seven assays were capped at 200 g/t silver.

For gold in sulphides, the top population averaging 14.85 g/t gold and representing 0.38% of the data was considered erratic and capped at two standard deviations above the mean of Population 2. A total of six samples were capped at 12 g/t gold.

For silver in sulphides the top erratic population averaged 30.1 g/t silver and represented 0.31% of the data. Two silver assays were capped at 25.0 g/t.

The results of capping are tabulated in Table 14.3.

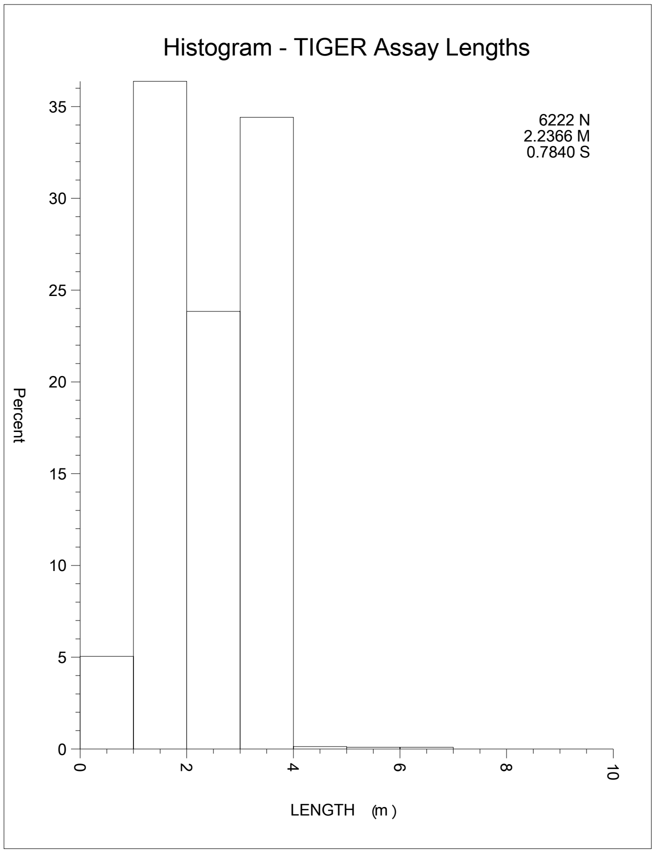
Table 14.3 Capped Assay Statistics from Solids

	Oxides		Sulphides	
	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)
Number of Assays	1,934	1,934	1,055	1,055
Mean Value	2.37	5.71	1.08	0.72
Standard Deviation	7.00	17.64	1.94	1.98
Minimum Value	0.001	0.001	0.001	0.001
Maximum Value	90.00	200.00	12.00	25.00
Coefficient of Variation	2.95	3.09	1.79	2.75

14.3 COMPOSITES

With 89.4% of assays less than or equal to 3.05 m in length (Figure 14.4), a 3 m composite interval was chosen. Composites were formed within each mineralized solid honouring the solid boundaries. Intervals at boundaries less than 1.5 m in length were combined with adjoining samples to produce composites of equal support 3 ±1.5 m in length. The statistics for composites in the oxide and sulphide solids are tabulated in Table 14.4.

Figure 14.4 Histogram of Assay Sample Lengths



In addition to gold and silver in sulphide composites, arsenic and iron were also accumulated for metallurgical purposes.

Table 14.4 3 m Composite Statistics from Solid

	Oxides		Sulphides			
	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	As (ppm)	Fe (%)
Number of Assays	1,263	1,263	843	843	843	843
Mean Value	2.25	5.04	1.01	0.65	8303.6	12.63
Standard Deviation	5.57	12.46	1.57	1.52	15,287.4	6.69
Minimum Value	0.001	0.001	0.001	0.001	0.1	0.05
Maximum Value	86.00	160.31	9.93	20.58	125,235.0	45.10
Coefficient of Variation	2.48	2.47	1.55	2.34	1.84	0.53

14.4 VARIOGRAPHY

Pairwise relative semivariograms were produced for both gold and silver in oxide and sulphide domains. The modelling procedure consisted of first examining the horizontal plane by producing semivariograms at azimuths of 90°, 0°, 45°, and 135° with a dip of 0°. The vertical direction was also examined with this semivariogram setting the nugget effect. Once the direction of maximum continuity was established in the horizontal plane, the vertical plane perpendicular to this direction was examined. For both variables in both domains geometric anisotropy was demonstrated. In all cases, nested spherical models were fit to the data.

The semivariogram parameters are tabulated in Table 14.5.

Table 14.5 Semivariogram Parameters for Rau

Domain	Variable	Azimuth/ Dip (°)	C0	C1	C2	Short Range (m)	Long Range (m)
Oxides	Au	173/0	0.25	0.47	0.25	38.0	200.0
		83/-55				20.0	68.0
		263/-35				10.0	20.0
	Ag	173/0	0.25	0.50	0.30	20.0	100.0
		83/-55				20.0	100.0
		263/-35				12.0	20.0
Sulphides	Au	135/0	0.10	0.16	0.55	15.0	60.0
		45/0				15.0	25.0
		0/-90				10.0	60.0
	Ag	0/0	0.10	0.30	0.40	50.0	120.0
		90/0				20.0	80.0
		0/-90				8.0	20.0

14.5 BLOCK MODEL

A block model with blocks 5 m by 5 m by 5 m in dimension was superimposed over the mineralized solids. The model was rotated to match the drillhole fences. The block model origin and details follow:

Lower Left Corner of Model:

528621.697 East	Size of Column: 5 m	94 columns
7118647.860 North	Size of Row: 5 m	200 rows

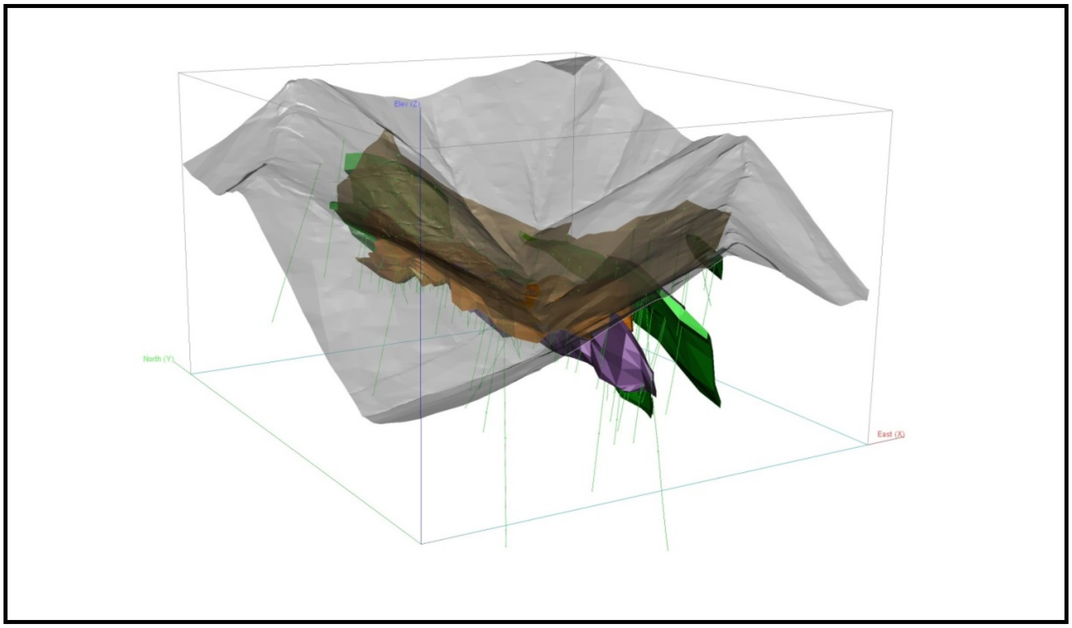
Top of Model

1575 Elevation	Size of Level: 5 m	127 levels

X axis rotated 40° counter clockwise

For each block the percentage below surface topography, below overburden, and within each mineralized solid was recorded. Figure 14.5 shows the various mineralized solids with the overburden and topographic surfaces.

Figure 14.5 View Looking Northeast Showing Oxide Solids in Orange, Sulphide Solids in Purple, Volcanic Solids in Green, Topography in Grey and Overburden in Brown



14.6 BULK DENSITY

Due to the extensive oxide content in this deposit, and the inherent problems with measuring bulk density on soft, broken up oxide material, a lot of care and attention went into determining bulk density for the various units.

Representative densities for each lithology, alteration type, and mineralization style were calculated by ATAC staff in the field using three methods: the first from weights in air from whole competent core, the second from measuring the weight of an entire box of core, and the third measuring the weight of crushed, dried oxide material.

The first method was used most often. Several of these measurements were taken from each lithological unit within each hole. The second and third methods were used exclusively for oxidized material.

14.6.1 WEIGHT IN AIR FROM WHOLE COMPETENT CORE

For the first method, 710 lengths of competent core were cut perpendicular to the core axis with a rock saw to ensure a cylindrical shape for accurate volume calculations. The final length of each cylinder was then measured to the nearest millimeter. Volumes were calculated using the measured sample length and predetermined core diameters

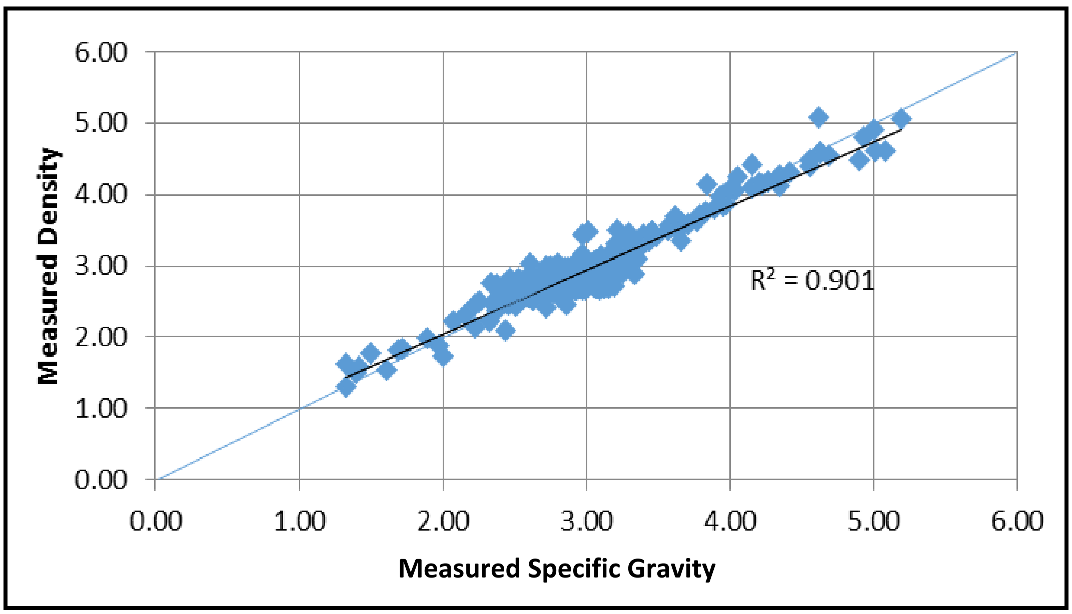
(84.61 mm for PQ, 60.68 mm for HQ, 50.34 mm for NQ, and 41.59 mm for BTW core). Core diameters were determined by averaging numerous measurements taken on representative pieces of core using calipers accurate to 0.01 mm.

All weights were measured with an Ohaus Scout Pro digital scale to an accuracy of 0.1 g. The scale was calibrated, leveled, and zeroed prior to all measurements. Weight in air was measured first and then the sample was placed in a submerged metal basket suspended from a hook on the underside of the scale and weighed. In air and in water weights were recorded along with the depth of the sample, its hole number, lithology, alteration, and the percentage of sulphide mineralization if applicable.

To calculate density, the in air weight of the sample was divided by its volume. Specific gravity was also calculated using the Archimedes method from the sample weights in air and water. These calculations were used as a quality control check for the density calculations. In general, specific gravity measurements are comparable to density measurements. Minor variations can be attributed to porosity and inaccuracy while measuring the volume of each sample. In addition, the Archimedes method also assumes the measurements were taken at sea level and with constant air and water temperatures. Field measurements were taken roughly 1,100 masl and under highly variable temperatures, which could have affected specific gravity calculations.

A scatterplot showing non-oxide samples measured for both density and specific gravity is shown as Figure 14.6. No bias is indicated with a best fit regression line (black) mirroring the equal value line (blue). The coefficient of correlation is 0.901 showing good agreement. This means there is no significant porosity in the sulphide samples.

Figure 14.6 Scatterplot Comparing Density to Specific Gravity in Sulphide Samples



14.6.2 WEIGHT IN AIR OF ENTIRE BOX OF CORE

Much of the oxidized core was too fragile to weigh using the above procedure; instead it was weighed in the core box. After geotechnical analysis, full boxes containing homogeneous intervals of oxide material were weighed using a bathroom scale. The weight of the box was subtracted from the measurement. The weight of each core box size was determined at the start of the program by taking an average weight of several boxes. To do these calculations, the length of core within the weighed box was measured. Some of the recovered core may be rubbly or broken, so estimation for measuring recovery may be required. The volume was then calculated assuming a perfect cylinder using predetermined core diameters and the measured length of core within the weighed box.

A high variability in the weights of core boxes, variations in recovery estimates, error in volume calculations, limited accuracy of a bathroom scale, and occasional excess drill mud in core boxes make box density calculations suspect and are only considered an approximation.

14.6.3 WEIGHT OF CRUSHED DRY OXIDE MATERIAL

A third method to calculate density of oxide material was used later in the field season when drill crews were consistently achieving high recoveries of oxide material. A total of 18 samples were collected from competent oxide material within intervals of 100% recovery that could be extracted from the core box. Lengths of oxide material were cut perpendicular to the core axis with sharp, metal edges. Cut sections were measured to the nearest millimeter prior to being extracted from the box. The volume of the sample was calculated from the same method described above. The sample was placed in a metal pan of known mass and was weighed using an Ohaus Scout Pro digital scale. The weight of the pan was subtracted from this measurement and the weight of the sample was recorded.

The initial sample weight includes the weight of water from natural ground conditions and fluids added during drilling and these samples are therefore considered saturated. To expedite drying, the sample was roasted in an oven at 400°C to remove all water. The sample was taken from the oven hourly and re-weighed until a consistent weight within the 0.1 g error of the scale was achieved. This process would typically take 5 to 6 hours. The density of the sample was calculated at each stage using the previously defined method.

Because the oxide material could not be submerged, density calculations using the roasted method could not be checked against a specific gravity calculation. The most likely source of error in the roasting method would come from inaccurate measurements used in the volume calculations.

Oxidized core often contains significant pebbles and cobles of limonite and goethite, which makes selection and extraction of representative intervals difficult. The amount of solid pieces within a particular interval of core can range from one or two pebbles per metre to nearly solid boxwork limonite. When selecting core for measurements, the uneven distribution of limonite and goethite produces a bias towards the softer, easier to cut, and extract intervals. Although a close approximation, density results obtained from this method for oxidized core should be considered a minimum value and not entirely representative of more competent oxidized core.

The results are summarized in Table 14.6.

Table 14.6 Average Unit Density

Unit	Modifier	Count	Average	Minimum	Maximum	Comments
OX	Wet	65	2.38	1.81	3.10	-
OX	Dry	18	1.80	1.25	2.37	Average 24% density loss after roasting
OX	Box	156	2.53	1.75	3.43	Using box weight method
DOL	MX	67	3.46	2.58	5.19	Mineralized Dolomite (incl. Tiger Zone)
DOL	-	35	2.98	2.71	3.36	Unaltered/Non-mineralized Dolomite
LST	-	387	2.83	2.37	3.46	-
LEP	-	38	2.94	2.75	3.33	-
VOL	-	78	2.93	2.60	3.30	Not including partially oxidized VOL
MBL	-	40	2.79	2.52	3.05	-

14.6.4 Sintrex Gravilog BHG system

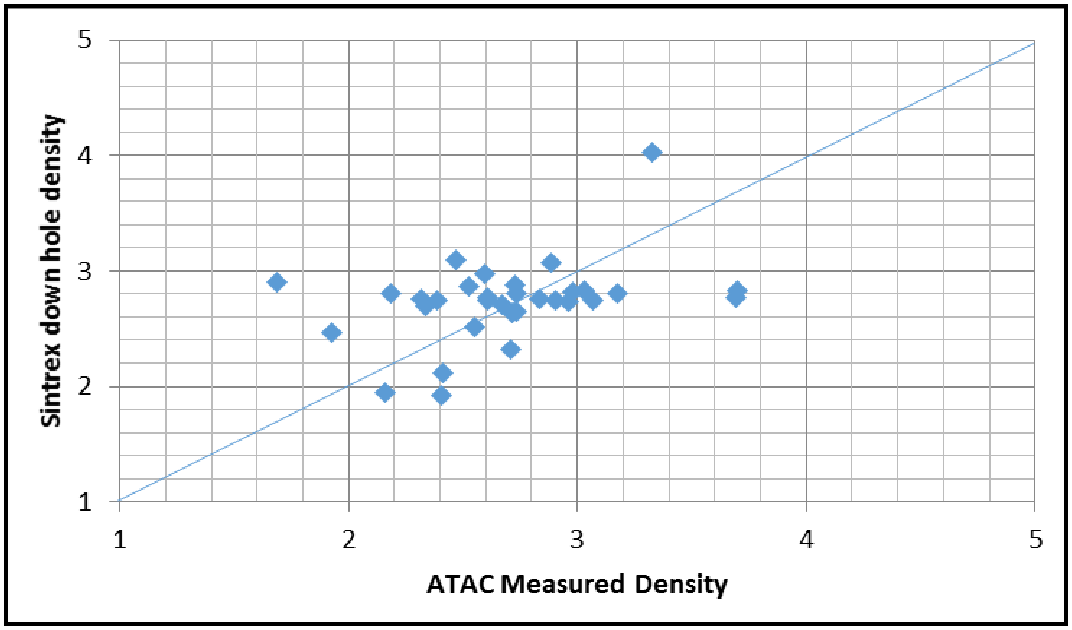
A fourth and relatively new method of determining density was conducted by Scintrex using a borehole gravity meter (Seigel et al. 2009). The Scintrex Gravilog BHG system can be deployed down drillholes to determine the bulk density determination of formations intersected by the borehole. This methodology was tried on eight drillholes through oxide material during the 2010 drill program (MacQueen 2011). Results from this survey are summarized in Table 14.7.

Table 14.7 Average Unit Density Scintrex

Unit	Count	Average	Minimum	Maximum
OX	80	2.46	1.62	3.36
HOST ROCKS	114	2.71	1.96	3.73

As some of the holes tested by Scintrex were outside the mineralized zone, a better comparison is made by comparing intervals from holes sampled by both ATAC and Scintrex. Figure 14.7 shows a scatterplot comparing density in intervals measured by both ATAC and Scintrex. These were not exactly the same intervals, however, as the Scintrex method was over intervals from 2 to 10 m, while the ATAC samples were from small pieces of drill core contained within the Scintrex interval. The plot shows no bias with samples plotting on either side of an equal value line.

Figure 14.7 Scatterplot Showing ATAC Measured Density Compared to Scintrex Density



14.6.5 Conclusions

Deposits containing significant oxide horizons present a challenge to bulk density determinations. The standard methods of measuring a sample in air and again in water do not work on this soft, often poorly consolidated material. A number of different methods were tried on the Tiger Deposit oxide samples. The results from the first method, using a measured volume and a weight in air seem reasonable when compared to other methods.

For the purpose of this Mineral Resource estimate, a density of 2.38 was used for oxide material. For the mineralized dolomite unit (mineralized sulphide domain) a value of 3.38 was used, which represents the average of 63 samples within the limits of 2 standard deviations above and below the mean of all mineralized dolomite samples. For volcanics, an average specific gravity of 2.93 was used based on 78 samples. The material outside the mineralized solids was assigned a density of 2.86; the average of 578 samples outside the solids. Finally a specific gravity of 1.8 was used for overburden.

14.7 Grade Interpolation

Grades for gold and silver were interpolated into the block model using Ordinary Kriging (OK). The OK exercise was completed four times, once each for gold and silver in blocks containing some percentage within the oxide solids, and again for gold and silver in blocks containing some percentage within the sulphide mineralized solids. For OK in oxides only, oxide composites were used, and for estimating grades in sulphides only, sulphide composites were used.

The OK exercise was completed for each variable in each domain in a series of four passes. Pass 1 used a search ellipse with dimensions equal to one quarter of the semivariogram range in each of the three principal directions. A minimum of four composites with a maximum of three coming from any one hole was required to estimate the block. For blocks not estimated in Pass 1, a second pass using search ellipse dimensions equal to half the semivariogram range was completed. Pass 3 using the full range and Pass 4 using twice the range rounded out the exercise. In all cases, if more than 12 composites were found the closest 12 were used.

For blocks containing some percentage of both oxide and sulphide material a weighted average was made. Since the ranges for silver in oxides were less than the ranges for gold, the Pass 4 distances for silver were set to the Pass 4 distances for gold. This ensured all blocks estimated for gold had a silver value. The parameters for the OK runs are tabulated in Table 14.8.

Table 14.8 Kriging Parameters for Tiger Zone

Domain	Variable	Pass	Number Estimated	Azimuth/ Dip (°)	Distance (m)	Azimuth/ Dip (°)	Distance (m)	Azimuth/ Dip (°)	Distance (m)
Oxides 100% of Blocks Estimated	Au	1	13,933	173/0	50.0	83/-55	17.0	263/-35	5.0
		2	14,305	173/0	100.0	83/-55	34.0	263/-35	10.0
		3	3,754	173/0	200.0	83/-55	68.0	263/-35	20.0
		4	406	173/0	400.0	83/-55	136.0	263/-35	40.0
	Ag	1	10,628	173/0	25.0	83/-55	25.0	263/-35	5.00
		2	1,548	173/0	50.0	83/-55	50.0	263/-35	10.0
		3	4,482	173/0	100.0	83/-55	100.0	263/-35	20.0
		4	2,455	173/0	200.0	83/-55	200.0	263/-35	40.0
Sulphides 100% of Blocks Estimated	Au	1	237	135/0	15.0	45/0	6.25	0/-90	15.0
		2	7,863	135/0	30.0	45/0	12.5	0/-90	30.0
		3	21,541	135/0	60.0	45/0	25.0	0/-90	60.0
		4	6,892	135/0	120.0	45/0	50.0	0/-90	120.0
	Ag	1	4,619	135/0	30.0	45/0	20.0	0/-90	5.0
		2	21,931	135/0	60.0	45/0	40.0	0/-90	10.0
		3	8,812	135/0	120.0	45/0	80.0	0/-90	20.0
		4	1,171	135/0	240.0	45/0	160.0	0/-90	40.0

14.8 CLASSIFICATION

Based on the study herein reported, delineated mineralization for the Tiger Deposit, Rau Property is classified as a Mineral Resource according to the following definitions from NI 43-101 and from CIM (2014):

In this Instrument, the terms "Mineral Resource", "Inferred Mineral Resource", "Indicated Mineral Resource" and "Measured Mineral Resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards (May 2014) on Mineral Resources and Mineral Reserves adopted by CIM Council, as those definitions may be amended.

The terms Measured, Indicated and Inferred are defined by CIM (2014) as follows:

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.

The term Mineral Resource covers mineralisation 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 economic extraction' implies a judgement 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 cut-off 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.

Mineralisation 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 mineralisation. The Qualified Person must recognise 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 Preliminary 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.

Mineralisation 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 mineralisation 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.

Modifying Factors

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

Geologic continuity for the deposit has been established through geologic mapping and drill hole logging. The geologic continuity has been used to constrain the oxide and sulphide mineralized domains. The grade continuity, which can be quantified by semivariograms, can be used to classify the estimate.

OXIDE ZONES

Oxide blocks with gold estimated in Pass 1 using a search ellipse of one quarter the semivariogram range and formed contiguous zones were considered Measured. Oxide blocks with gold estimated in Pass 2 using a search ellipse of half the semivariogram range were classified as Indicated. All other oxide blocks were classified as Inferred at this time.

SULPHIDE ZONES

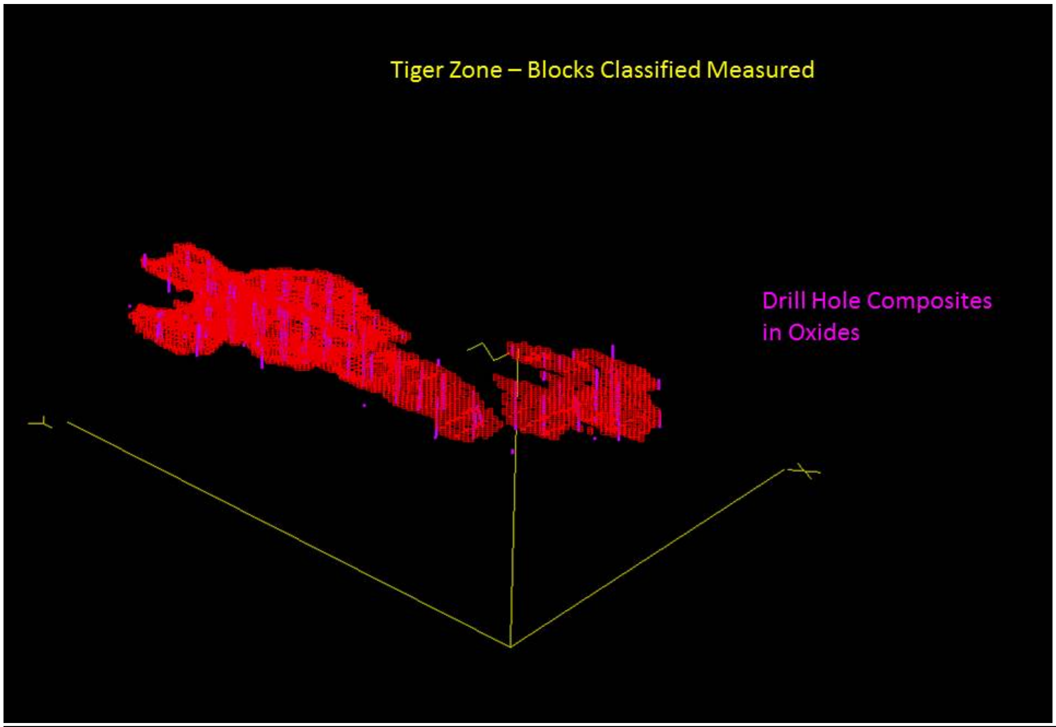
Sulphide blocks with gold estimated in Pass 1 or Pass 2 were classified as Indicated. The remaining sulphide blocks were classified as Inferred.

Figure 14.8 shows blocks classified as Measured and Indicated.

For the Tiger Deposit Mineral Resource, a cut-off of 0.5 g/t gold is highlighted as a possible open pit cut-off for oxides, while a 1.0 g/t gold cut-off is highlighted for sulphides. Cut-off values were chosen based on the somewhat analogous Coffee Deposit (Makarenko et al. 2014), where oxide and transition material is reported at a 0.5 g/t gold cut-off, and sulphide material, which represents greater depths and differing metallurgy, is reported at 1.0 g/t gold. Note, due to rounding off, the totals for all blocks might not equal exactly the sums of oxides plus sulphides.

The Mineral Resource for the Oxide Zone is shown in Table 14.9 to Table 14.12 and for the Sulphide Zone in Table 14.13 and Table 14.14. Table 14.15 summarizes the Mineral Resources for both the oxides and sulphides, above the chosen cut-off grades.

Figure 14.8 Isometric Views Looking Northeast Showing Classified Blocks with Measured in Red and Indicated in Green



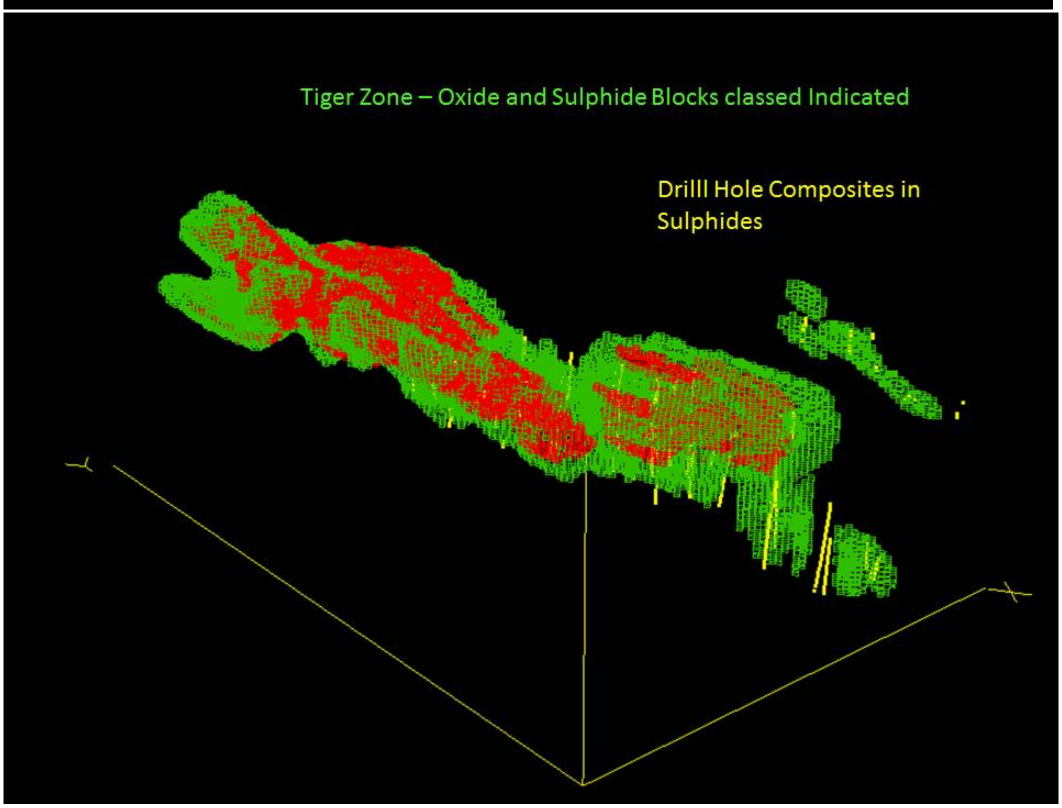


Table 14.9 Tiger Zone Oxide Blocks - Classified Measured

Au Cut-off (g/t)	Tonnes> Cut-off (t)	Grade>Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
0.10	3,320,000	2.49	4.62	265,500	493,100
0.20	3,080,000	2.67	4.64	264,000	459,500
0.30	2,920,000	2.80	4.64	263,100	435,600
0.40	2,750,000	2.95	4.69	261,200	414,700
0.50	**2,600,000**	**3.10**	**4.77**	**259,100**	**398,700**
0.60	2,450,000	3.25	4.88	256,200	384,400
0.70	2,320,000	3.40	5.00	253,300	373,000
0.80	2,180,000	3.57	5.15	249,900	361,000
0.90	2,080,000	3.70	5.26	247,500	351,800
1.00	1,970,000	3.85	5.37	243,900	340,100
1.20	1,800,000	4.12	5.59	238,300	323,500
1.40	1,640,000	4.39	5.75	231,600	303,200
1.60	1,500,000	4.66	5.90	224,600	284,500
1.80	1,370,000	4.93	5.93	217,200	261,200
2.00	1,260,000	5.21	5.96	211,000	241,400

Table 14.10 Tiger Zone Oxide Blocks - Classified Indicated

Au Cut-off (g/t)	Tonnes> Cut-off (t)	Grade>Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
0.10	2,470,000	1.81	5.11	143,700	405,800
0.20	2,290,000	1.94	4.82	142,700	354,900
0.30	2,070,000	2.12	4.47	141,000	297,500
0.40	1,870,000	2.31	4.26	138,600	256,100
0.50	**1,720,000**	**2.47**	**4.10**	**136,300**	**226,700**
0.60	1,620,000	2.59	3.99	134,800	207,800
0.70	1,520,000	2.71	3.81	132,500	186,200
0.80	1,440,000	2.83	3.72	130,800	172,200
0.90	1,370,000	2.92	3.65	128,700	160,800
1.00	1,300,000	3.02	3.56	126,400	148,800
1.20	1,160,000	3.27	3.43	121,800	127,900
1.40	1,040,000	3.48	3.40	116,500	113,700
1.60	950,000	3.67	3.40	112,000	103,800
1.80	870,000	3.86	3.42	107,900	95,700
2.00	790,000	4.06	3.44	103,000	87,400

Table 14.11 Tiger Zone Oxide Blocks - Classified Inferred

Au Cut-off (g/t)	Tonnes> Cut-off (t)	Grade>Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
0.10	620,000	0.86	7.52	17,100	149,900
0.20	580,000	0.91	7.53	17,000	140,400
0.30	460,000	1.07	7.03	15,900	104,000
0.40	340,000	1.33	5.92	14,500	64,700
0.50	**280,000**	**1.52**	**5.67**	**13,700**	**51,000**
0.60	250,000	1.62	5.46	13,000	43,900
0.70	220,000	1.76	5.02	12,400	35,500
0.80	200,000	1.87	5.02	12,000	32,300
0.90	180,000	1.97	4.89	11,400	28,300
1.00	160,000	2.09	4.88	10,800	25,100
1.20	130,000	2.38	4.83	9,900	20,200
1.40	100,000	2.63	4.58	8,400	14,700
1.60	80,000	2.96	4.21	7,600	10,800
1.80	70,000	3.22	3.74	7,300	8,400
2.00	60,000	3.40	3.60	6,600	6,900

Table 14.12 Tiger Zone Oxide Blocks - Classified Measured plus Indicated

Au Cut-off (g/t)	Tonnes> Cut-off (t)	Grade>Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
0.10	5,790,000	2.20	4.83	409,200	899,100
0.20	5,370,000	2.36	4.72	406,800	814,900
0.30	4,990,000	2.52	4.57	404,000	733,200
0.40	4,620,000	2.69	4.51	399,900	669,900
0.50	**4,320,000**	**2.85**	**4.50**	**395,400**	**625,000**
0.60	4,070,000	2.99	4.52	391,000	591,500
0.70	3,840,000	3.13	4.53	385,800	559,300
0.80	3,620,000	3.27	4.58	380,700	533,100
0.90	3,450,000	3.39	4.62	376,100	512,500
1.00	3,280,000	3.52	4.65	371,400	490,400
1.20	2,960,000	3.78	4.74	360,100	451,100
1.40	2,680,000	4.04	4.83	347,900	416,200
1.60	2,450,000	4.27	4.93	336,600	388,300
1.80	2,240,000	4.51	4.95	325,100	356,500
2.00	2,050,000	4.76	4.99	313,900	328,900

Table 14.13 Tiger Zone Sulphide Blocks - Classified Indicated

Au Cut-off (g/t)	Tonnes> Cut-off (t)	Grade>Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
0.10	2,870,000	1.25	0.63	115,200	58,100
0.20	2,690,000	1.32	0.62	114,200	53,600
0.30	2,490,000	1.41	0.63	112,700	50,400
0.40	2,290,000	1.50	0.62	110,500	45,600
0.50	2,120,000	1.59	0.61	108,200	41,600
0.60	1,940,000	1.68	0.59	104,800	36,800
0.70	1,760,000	1.78	0.58	100,900	32,800
0.80	1,610,000	1.88	0.58	97,400	30,000
0.90	1,470,000	1.98	0.57	93,500	26,900
1.00	**1,360,000**	**2.07**	**0.56**	**90,300**	**24,500**
1.20	1,160,000	2.23	0.55	83,200	20,500
1.40	960,000	2.42	0.56	74,800	17,300
1.60	780,000	2.65	0.53	66,300	13,300
1.80	660,000	2.82	0.51	59,800	10,800
2.00	550,000	2.99	0.46	52,900	8,100

Table 14.14 Tiger Zone Sulphide Blocks - Classified Inferred

Au Cut-off (g/t)	Tonnes> Cut-off (t)	Grade>Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
0.10	8,590,000	0.97	0.64	267,300	176,800
0.20	7,970,000	1.03	0.65	263,900	166,600
0.30	7,230,000	1.11	0.64	258,000	148,800
0.40	6,410,000	1.21	0.62	249,000	127,800
0.50	5,680,000	1.31	0.58	238,300	105,900
0.60	5,010,000	1.41	0.54	226,500	87,000
0.70	4,400,000	1.51	0.53	213,900	75,000
0.80	3,850,000	1.62	0.51	200,700	63,100
0.90	3,360,000	1.73	0.51	187,300	55,100
1.00	**2,950,000**	**1.84**	**0.47**	**174,800**	**44,600**
1.20	2,240,000	2.08	0.42	149,700	30,200
1.40	1,740,000	2.31	0.43	129,200	24,100
1.60	1,400,000	2.51	0.45	112,800	20,300
1.80	1,160,000	2.67	0.46	99,600	17,200
2.00	960,000	2.83	0.45	87,300	13,900

Table 14.15 Combined Oxides and Sulphide Resource

Type	Classification	Au Cut-off (g/t)	Tonnes> Cut-off (t)	Grade>Cut-off		Contained Metal	
				Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
Oxides	Measured	0.50	2,600,000	3.10	4.77	259,100	398,700
	Indicated	0.50	1,720,000	2.47	4.10	136,300	226,700
Sulphides	Indicated	1.00	1,360,000	2.07	0.56	90,300	24,500
Total	**M+I**	-	**5,680,000**	**2.66**	**3.56**	**485,700**	**649,900**
Oxides	Inferred	0.50	280,000	1.52	5.67	13,700	51,000
Sulphides	Inferred	1.00	2,950,000	1.84	0.47	174,800	44,600
Total	**Inferred**	-	**3,230,000**	**1.81**	**0.92**	**188,500**	**95,600**

14.9 Block Model Verification

Level plans at 20 m intervals were produced to check estimated block grades with composite grades. Levels 1300 down to 1200 are shown on Figure 14.9 to Figure 14.14. The estimated block grades match the composite grades well with no bias indicated.

Figure 14.9 1300 Level Plan Showing Estimated Gold in Blocks and Composites

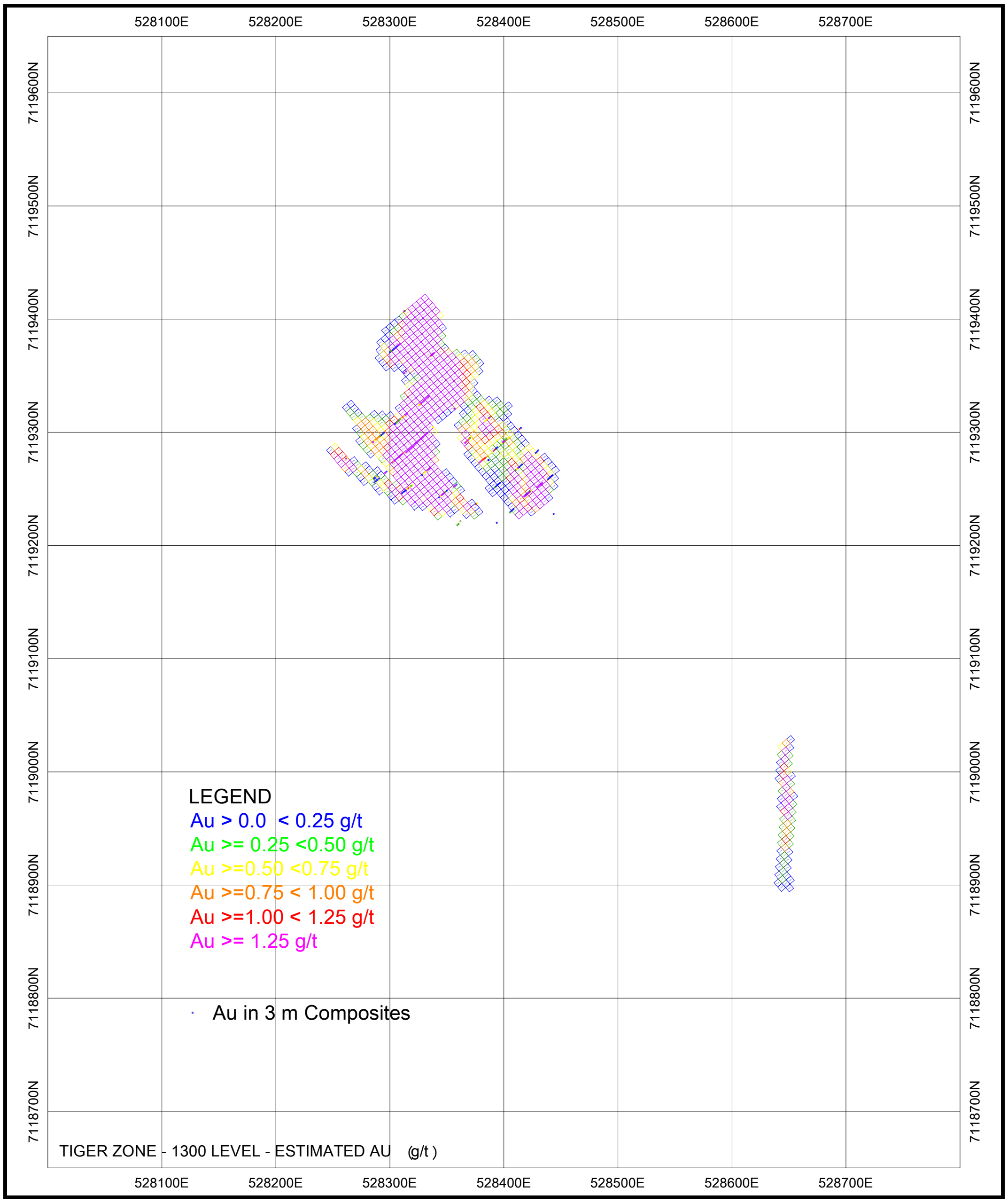


Figure 14.10 1280 Level Plan Showing Estimated Gold in Blocks and Composites

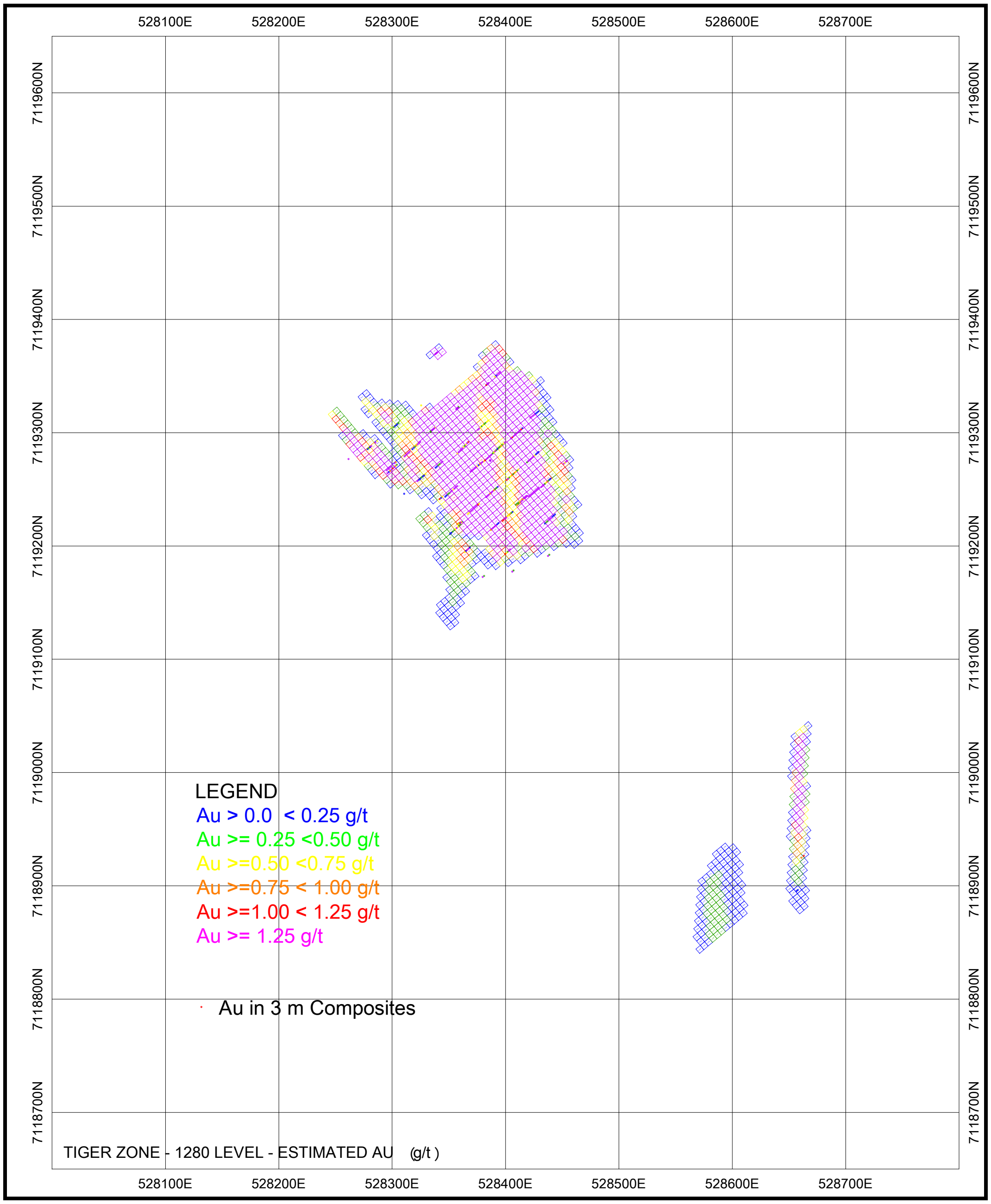


Figure 14.11 1260 Level Plan Showing Estimated Gold in Blocks and Composites

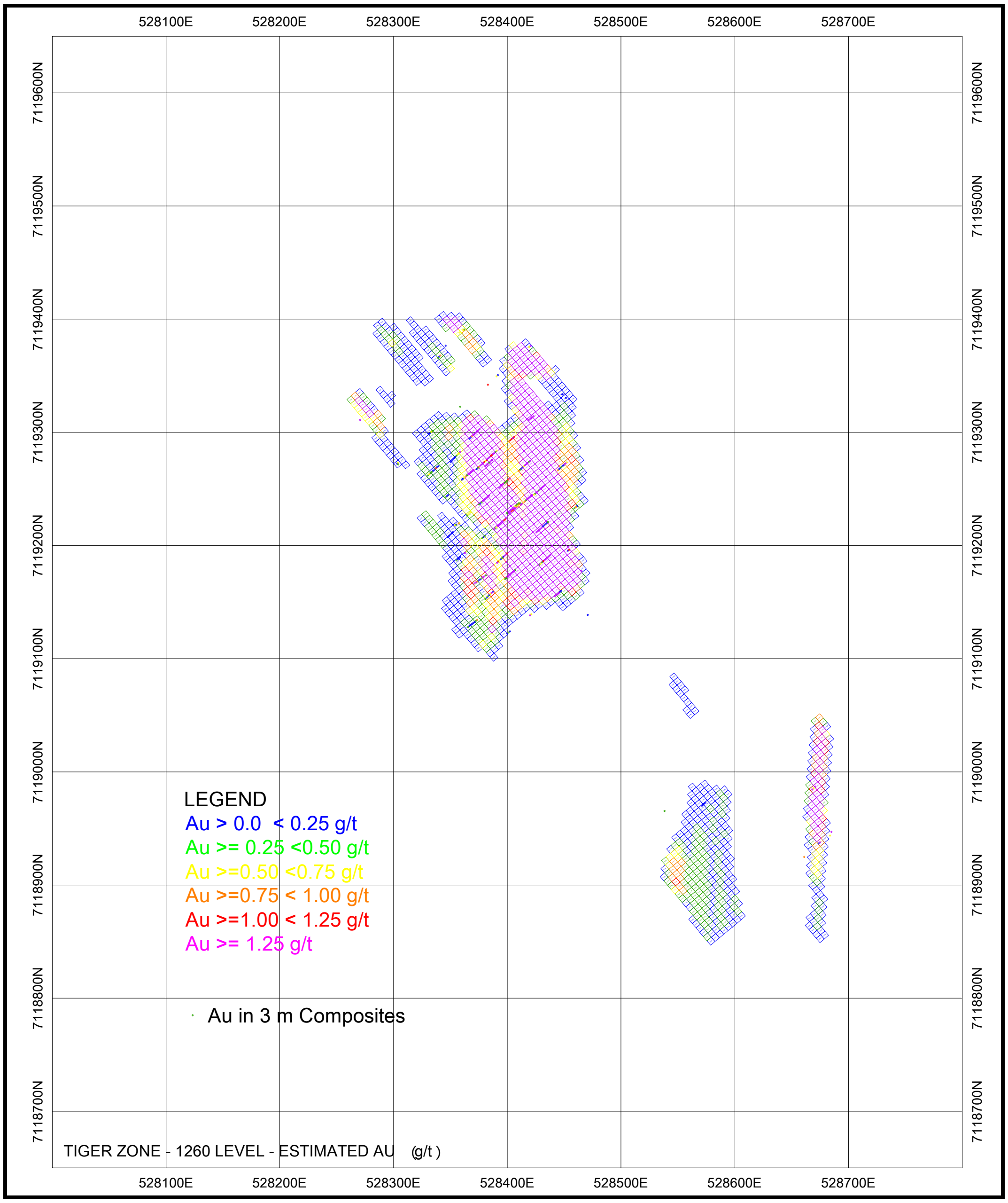


Figure 14.12 1240 Level Plan Showing Estimated Gold in Blocks and Composites

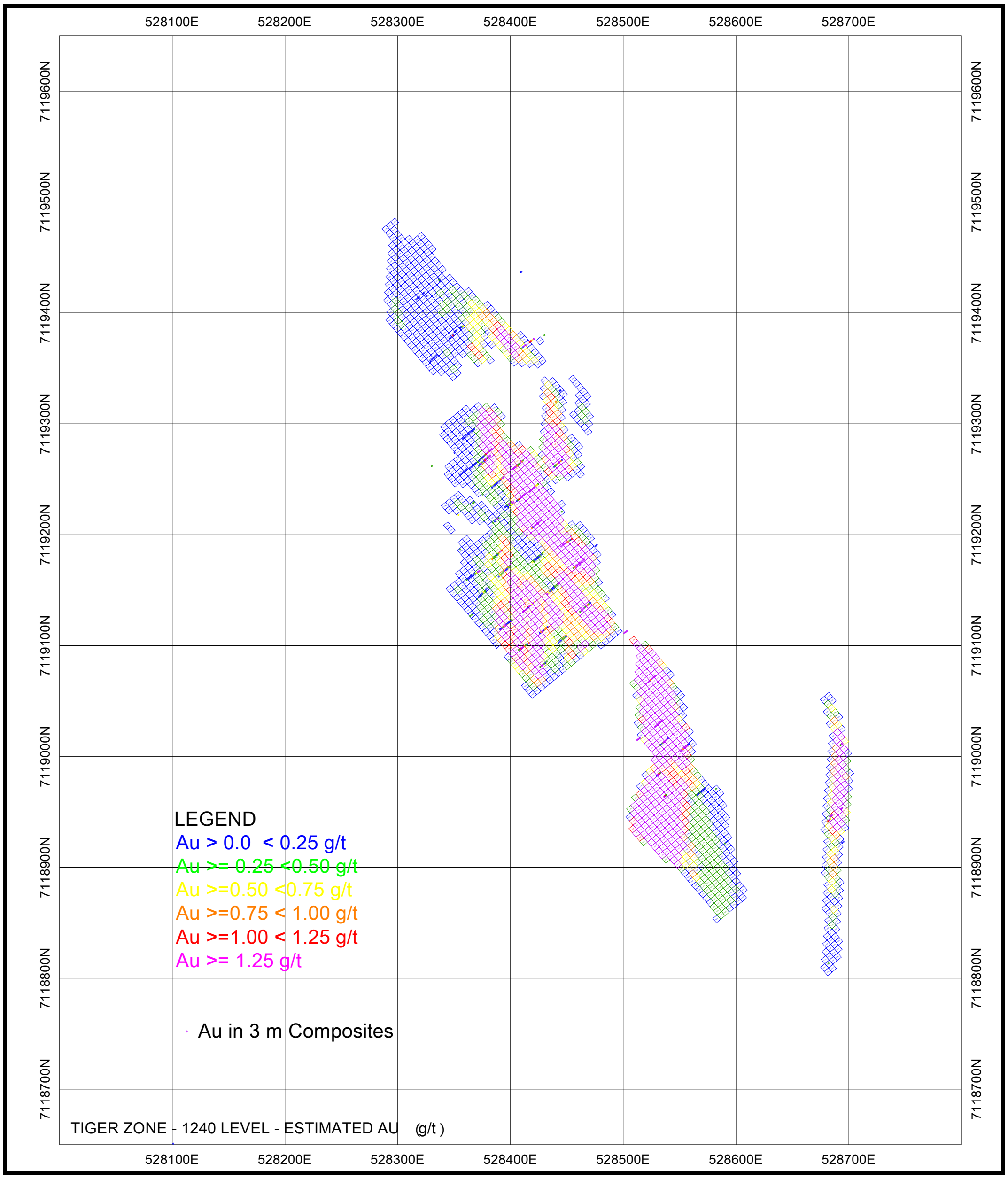


Figure 14.13 1220 Level Plan Showing Estimated Gold in Blocks and Composites

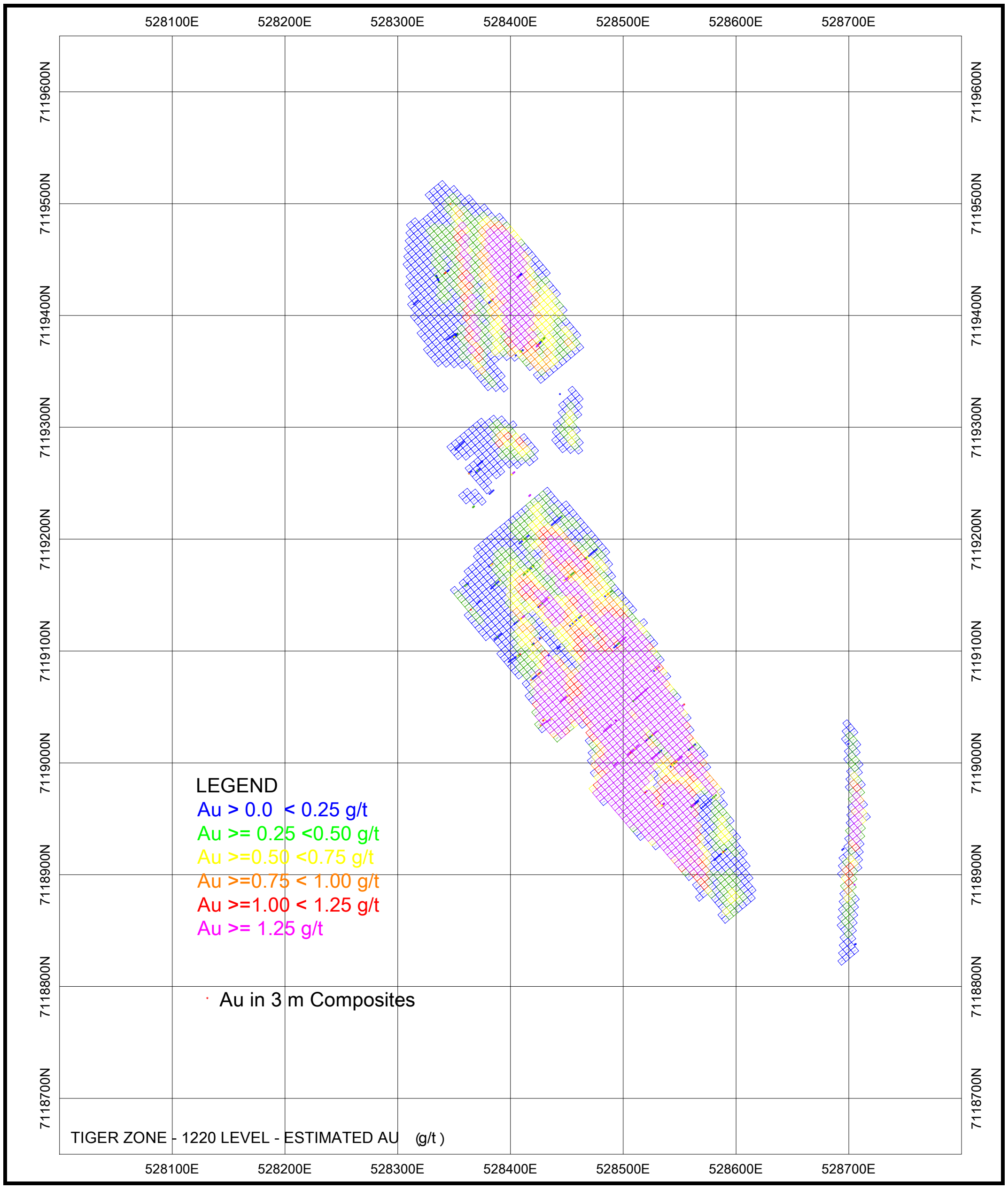
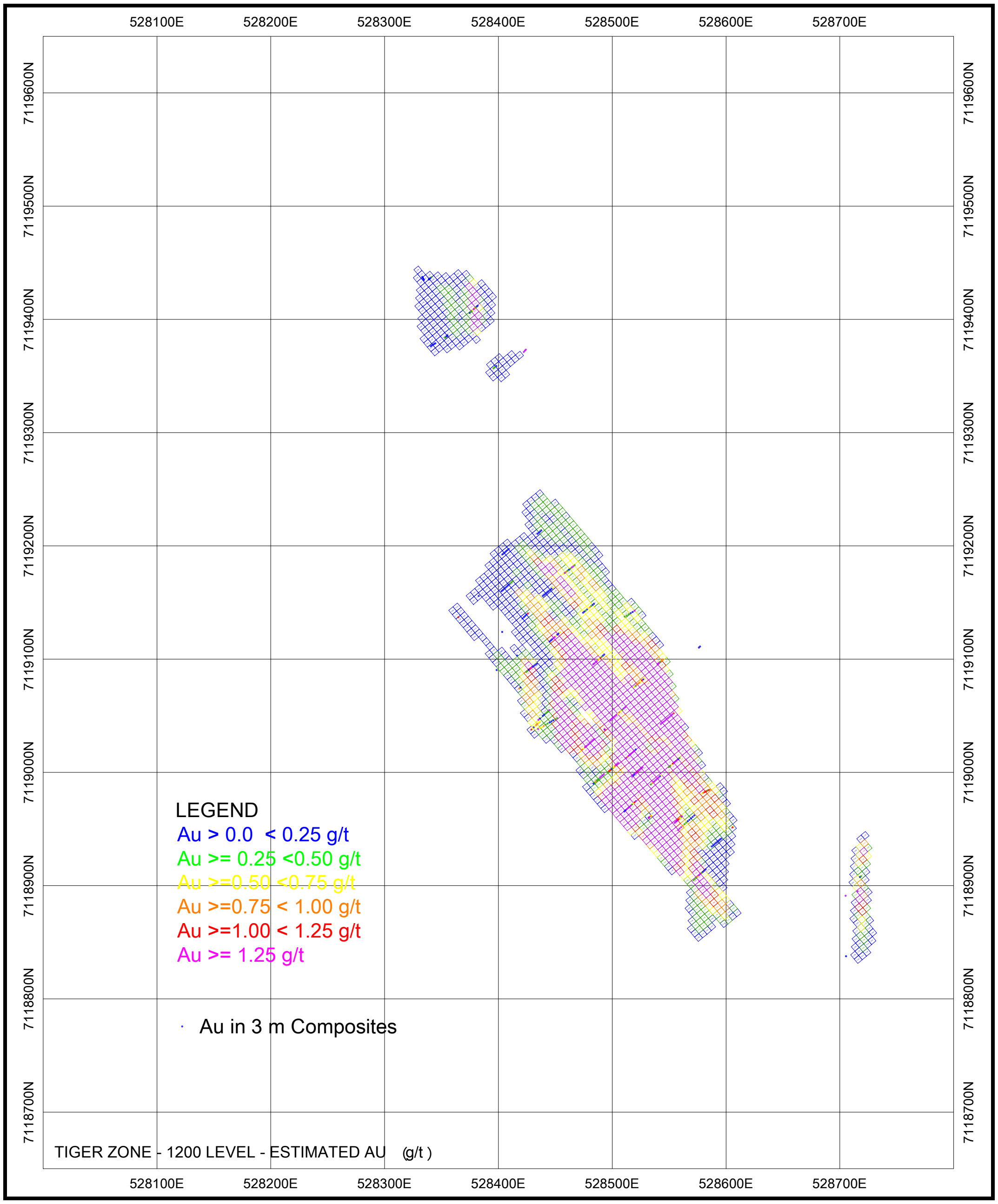


Figure 14.14 1200 Level Plan Showing Estimated Gold in Blocks and Composites



14.10 COMPARISON WITH PREVIOUS MINERAL RESOURCE ESTIMATE

The previous Mineral Resource estimate on the Property was first published in Stroshein et al. (2011) and restated in Kappes et al. (2014).

For comparison purposes, Table 14.16 and Table 14.17 show Mineral Resources for both 2011 and 2015 at the cut-off grades used to report Mineral Resources for each year.

Table 14.16 Comparison of Mineral Resource Estimates (2011 Cut-offs)

		Cut-off Au (g/t)	2015			2011		
			Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)
Oxide	Measured	0.3	2,920,000	2.80	263,100	0	-	0
	Indicated	0.3	2,070,000	2.12	141,000	4,490,000	2.71	391,200
	M+I	0.3	4,990,000	2.52	404,000	4,490,000	2.71	391,200
	Inferred	0.3	460,000	1.07	15,900	620,000	1.42	28,300
Sulphide	Indicated	0.3	2,490,000	1.41	112,700	2,590,000	1.38	114,900
	Inferred	0.3	7,230,000	1.11	258,000	7,640,000	1.06	260,400

Table 14.17 Comparison of Mineral Resource Estimates (2015 Cut-offs)

		Cut-off Au (g/t)	2015			2011		
			Tonnes	Au (g/t)	Au (oz)	Tonnes	Au (g/t)	Au (oz)
Oxide	Measured	0.5	2,600,000	3.10	259,100	0	-	0
	Indicated	0.5	1,720,000	2.47	136,300	3,970,000	3.02	385,000
	M+I	0.5	4,320,000	2.85	395,400	3,970,000	3.02	385,000
	Inferred	0.5	280,000	1.52	13,700	440,000	1.85	26,200
Sulphide	Indicated	1.0	1,360,000	2.07	90,300	1,360,000	2.07	90,500
	Inferred	1.0	2,950,000	1.84	174,800	2,870,000	1.80	166,100

The most notable difference between both Mineral Resources is an increase in the overall confidence of the oxide Mineral Resource, which now includes a Measured component. This is due to the additional drilling in 2015 within the Oxide Zone.

Although there is a small difference within each category, there is no significant change between the total contained metal and total tonnes reported for each of the Mineral Resources.

Previously, due to lack of metallurgical information, low-grade transitional material was included in the sulphide Mineral Resource. In 2015, this material was reclassified as belonging to the Oxide Zone. This was supported by total cyanide solubility assays completed in 2015. The removal of this low-grade material from the Sulphide Zone contributed to an increase in the gold grades within the Sulphide Zone and decrease within the Oxide Zone.

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

ATAC 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

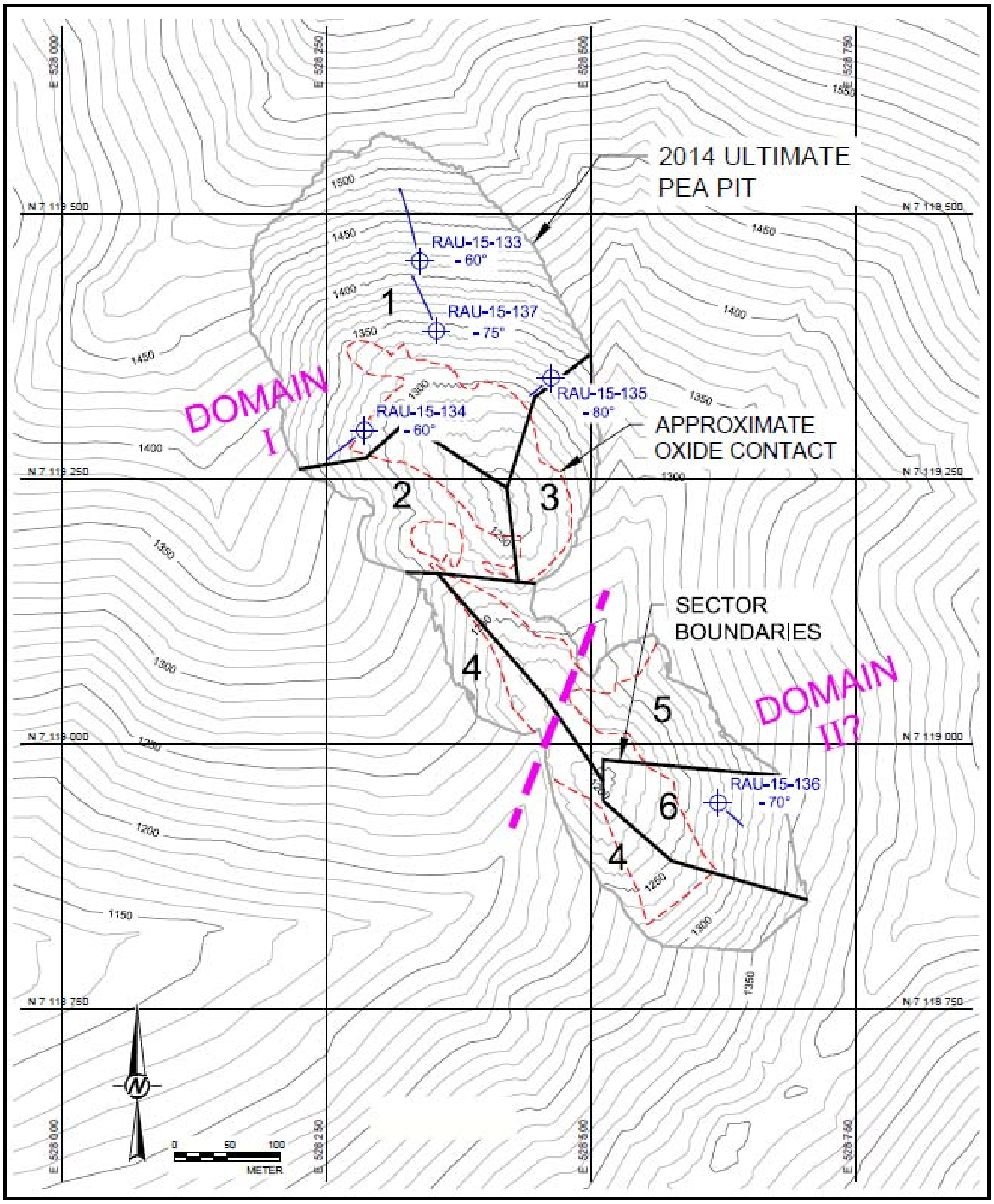
Golder Associates Inc. (Golder) completed a scoping-level pit slope evaluation report (Golder 2016) in March of 2016. Table 16.1 lists the scoping-level pit slope design recommendations as proposed by Golder and Figure 16.1 shows the sector numbers described in Table 16.1.

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

ATAC provided digital topographical drawings to Tetra Tech.

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	CAD = USD	0.78
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	1,200
Process		
Method	-	Tank Leaching
Recovery	%	See formula below
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
Percentage of Oxides/Sulphides in Each Block	%	Variable
Gold Grade	g/t	Variable
Density		
Oxide Mineralization	t/m^3	2.38
Sulphide Mineralization	t/m^3	3.38
Volcanics	t/m^3	2.93
Waste Outside Mineralized Solids	t/m^3	2.86

table continues...

Items	Units	Value
Overburden	t/m^3	1.80
Mining Technical Assumptions		
Mining Recovery	%	95
Mining Dilution	%	5

Process Recovery (Sulphide):

- 55% (if head grade is less than 3 g/t)
- 5.896*head grade (g/t) + 37.83 (if head grade is between 3 and 8.5 g/t)
- 90% (if head grade is greater than 8.5 g/t).

Process Recovery (Oxide):

- 50% (if head grade is less than 0.5 g/t)
- 1.0534*head grade (g/t) + 85.926 (if head grade is between 0.5 and 8.5 g/t)
- 97% (if head grade is greater than 8.5 g/t)

16.2.5 PIT OPTIMIZATION RESULTS

Using the provided block model, pit slope angles, and pit optimization parameters outlined in Table 16.2, 39 pit shells were generated using GEOVIA Whittle™ software, corresponding to price factors ranging between 0.25 and 1.0. Pit optimizations were completed using the Measured, Indicated, and Inferred oxide and sulphide Mineral Resources. The discounted value of each pit was estimated using GEOVIA Whittle™ using a 5% discount rate based on the exchange rate, gold price, process recovery, operating costs and marketing terms listed in Table 16.2. No capital costs were considered when generating these discounted values. The pit optimization results are provided in Table 16.3. Based on the discounted value, Pit 34 (corresponding to a revenue factor of 0.90) has the maximum discounted value. However, since the difference in the discounted value is not significant, in order to maximize the contained gold pit 39 (corresponding to a revenue factor of 1.00) was selected as the ultimate pit for further detailed design and production scheduling.

Table 16.3 Pit Optimization Results

Pit No.	Total Material Mined (t)	Waste Mined (t)	Mineralized Material Mined				Strip Ratio	Discounted Value at 5% (CAD million)
			Tonnage (t)	Oxide Contained (oz)	Sulfide Contained (oz)	Total Contained (oz)		
1	222,176	124,230	97,946	14,362	396	14,758	1.27	15.19
2	2,600,964	2,087,888	513,076	82,663	632	83,295	4.07	78.84
3	2,733,908	2,144,618	589,290	91,577	966	92,543	3.64	86.72
4	3,481,481	2,736,226	745,255	112,792	1,771	114,563	3.67	105.21
5	3,657,913	2,867,973	789,940	118,400	2,202	120,602	3.63	110.08
6	4,648,896	3,501,614	1,147,282	154,097	3,870	157,967	3.05	135.80
7	5,479,926	4,216,764	1,263,162	171,043	4,492	175,535	3.34	149.13
8	5,749,082	4,397,644	1,351,438	177,612	7,744	185,356	3.25	154.98
9	6,153,377	4,749,582	1,403,795	184,737	8,454	193,191	3.38	160.41
10	6,479,291	5,006,242	1,473,049	190,653	10,794	201,448	3.40	165.32
11	6,899,416	5,314,479	1,584,937	199,692	13,622	213,314	3.35	171.82
12	7,380,143	5,720,835	1,659,308	207,397	14,966	222,363	3.45	176.83
13	7,605,380	5,877,663	1,727,717	212,443	16,516	228,959	3.40	180.26
14	8,058,511	6,233,092	1,825,419	220,389	18,583	238,971	3.41	185.29
15	8,522,560	6,572,320	1,950,240	227,313	24,099	251,412	3.37	190.37
16	8,744,639	6,718,259	2,026,380	230,414	28,037	258,451	3.32	192.86
17	9,181,458	7,089,386	2,092,072	236,754	28,267	265,020	3.39	195.67
18	9,280,499	7,149,913	2,130,586	237,783	30,794	268,577	3.36	196.76
19	9,663,510	7,473,327	2,190,183	242,702	31,705	274,407	3.41	198.84
20	10,150,631	7,859,366	2,291,265	248,075	35,717	283,793	3.43	201.86
21	10,358,615	8,009,928	2,348,687	250,849	37,655	288,503	3.41	203.21
22	10,380,713	8,018,826	2,361,887	251,325	38,147	289,472	3.40	203.46
23	10,982,368	8,530,513	2,451,855	257,437	40,286	297,723	3.48	205.80
24	11,163,873	8,664,476	2,499,397	259,865	41,413	301,278	3.47	206.65
25	11,353,135	8,780,857	2,572,278	262,066	44,713	306,779	3.41	207.69
26	11,429,026	8,835,246	2,593,780	263,114	45,187	308,301	3.41	207.99
27	12,004,962	9,303,755	2,701,207	268,818	47,727	316,545	3.44	209.44
28	12,092,447	9,368,373	2,724,074	269,850	48,333	318,182	3.44	209.65
29	12,151,307	9,403,493	2,747,814	270,487	49,310	319,798	3.42	209.84
30	12,238,531	9,462,571	2,775,960	270,803	51,253	322,056	3.41	210.11
31	12,486,679	9,658,964	2,827,715	271,015	55,772	326,787	3.42	210.57
32	12,811,015	9,929,680	2,881,335	272,930	58,275	331,205	3.45	210.93
33	12,985,907	10,066,233	2,919,674	273,745	60,449	334,195	3.45	211.12
34	13,066,251	10,126,624	2,939,627	274,737	60,588	335,325	3.44	211.21
35	16,595,643	13,507,274	3,088,369	292,809	61,896	354,706	4.37	210.87
36	16,956,096	13,809,107	3,146,989	294,461	64,862	359,323	4.39	210.99
37	17,012,628	13,845,038	3,167,590	294,810	65,837	360,647	4.37	210.99
38	17,131,410	13,939,615	3,191,795	295,903	66,114	362,017	4.37	210.96
39	17,867,161	14,633,304	3,233,857	299,220	67,152	366,372	4.53	210.52

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 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.

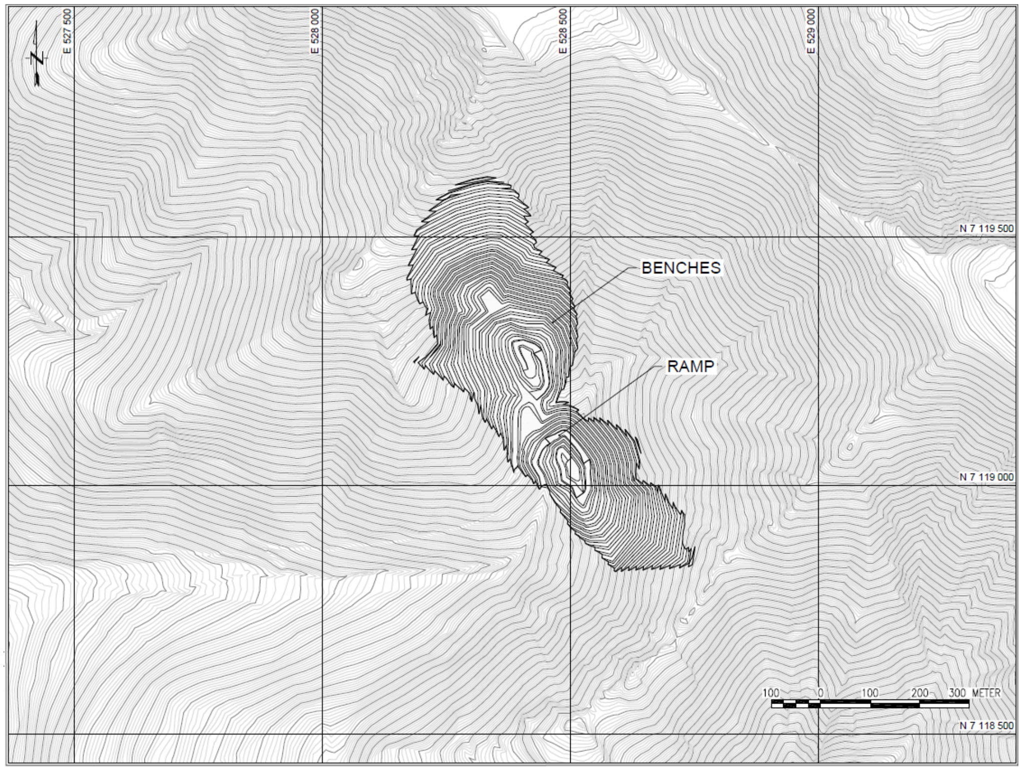
16.3.5 Pit Design Results

The final designed pit includes 2.24 Mt of oxide Mineral Resource and 0.69 Mt of sulphide Mineral Resource, with a LOM strip ratio of 4.9. A material summary for the final pit is provided in Table 16.4. Figure 16.2 shows a general view of the final pit.

Table 16.4 Pit Design Results

Material	Tonnage (t)	Au (g/t)
Mineralized Oxide to Mill	2,237,503	4.06
Mineralized Sulphide to Mill	693,036	2.99
Dilution	288,843	-
Waste (Rock and Low Grade Oxide/Sulphide)	15,628,071	-
Total Material Mined	**18,847,453**	-

Figure 16.2 Pit Design



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 sulphide material above the economic cut-off will be scheduled for processing. Oxide and sulphide material below the economic cut-off will be handled as waste. The developed production schedule maximizes the NPV of the Project by targeting higher-grade Mineral Resources earlier in the mine life. A cut-off grade policy was applied, where relatively low-grade material, in excess of the processing capacity in a particular production year, is stockpiled and reclaimed in later years when pit production of mineralized material is low. Relatively high-grade material will be sent directly to the primary crusher, located southwest of the pit. Low-grade stockpile material will be stored 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 eight years, including one year of pre-stripping followed by seven years of mill production. During the last production year, mill feed will come from the low-grade stockpile. The production schedule is shown in Table 16.5 and Figure 16.3. Over the eight-year mine life, the pit will produce 3.2 Mt of mineralized material and 15.6 Mt of waste rock. The LOM average gold grade of oxide and sulphide material is 4.06 g/t and 2.99 g/t, respectively. The LOM stripping ratio (defined as waste material mined divided by mineralized material mined) is 4.9. Figure 16.4 shows the status of mining activity at the end of mine life.

Table 16.5 Production Schedule

Year	Tonnage Mined								Material Processed						Mill Feed Balance			
	Total (kt)	Oxide (kt)	Sulphide (kt)	OB (kt)	Volcanic (kt)	Waste Rock (kt)	Total Waste Material (kt)	Strip Ratio	Total Tonnage (kt)	Oxide		Sulfide		Dilution (kt)	Mine to Mill (kt)	Mine to SP (kt)	SP to Mill (kt)	In SP (kt)
										kt	g/t Au	kt	g/t Au					
-1	2,600	136	3	134	1	2,325	2,499	24.8	-	-	-	-	-	-	-	101	-	101
1	3,600	416	-	79	302	2,803	3,344	13.1	357	305	5.20	3	4.26	49	256	-	101	-
2	3,600	1,007	37	90	143	2,323	2,856	3.8	548	502	5.83	8	3.38	38	548	196	-	196
3	3,600	405	71	116	292	2,715	3,244	9.1	548	447	3.45	31	3.27	70	356	-	191	5
4	2,850	1,280	149	125	79	1,217	2,300	4.2	547	488	3.22	17	1.39	43	542	7	5	7
5	1,400	468	502	31	89	311	828	1.4	548	264	2.97	240	3.23	44	548	24	-	32
6	1,197	316	744	1	3	134	556	0.9	548	193	3.27	319	3.08	36	548	94	-	125
7	-	-	-	-	-	-	-	-	125	40	1.43	75	2.05	10	-	-	125	-
LOM	18,847	4,029	1,507	576	909	11,828	15,628	4.9	3,219	2,238	4.06	693	2.99	289	2,797	422	422	-

Note: SP – stockpile; OB - overburden

Figure 16.3 Production Schedule

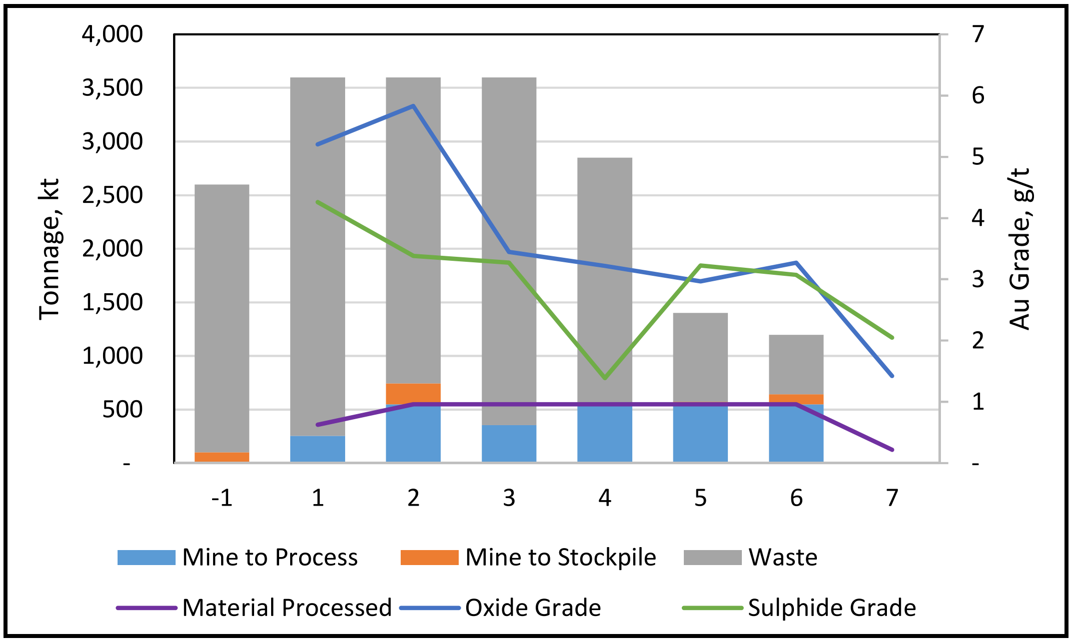
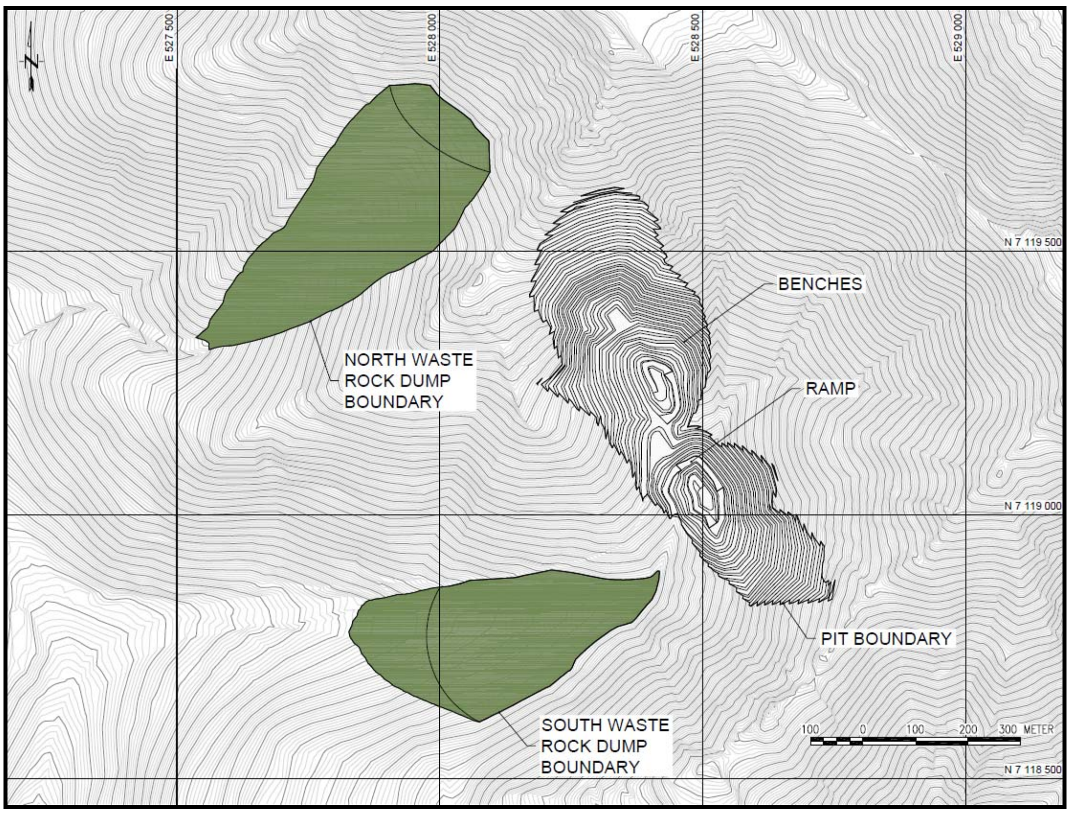


Figure 16.4 LOM Mine Status Map



16.5 MINE WASTE ROCK AND STOCKPILE MANAGEMENT

Sulphide material below the cut-off limit will be encapsulated and stored within the TMF. The remaining waste rock and low-grade oxide material below the economic cut-off will be stored in two waste dumps located northwest and southwest of the pit. LOM waste material mined will be 15.6 Mt; approximately 5.4 Mt will be utilized in the construction of the TMF and the remaining, 10.2 Mt will be stored in the two waste dumps. The northwest waste dump was designed with an overall slope of 2H:1V and the southwest dump was designed with a slope of 2.5H:1V.

Low-grade oxide and sulphide material above the economic cut-off will be stored close to the primary crusher at the southwest side of the pit. Stockpile material will be progressively reclaimed during the seven-year production life and will be completely reclaimed by the end of the seventh production period.

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 16.6, the expected delays per shift are 197 minutes.

Table 16.6 Operational Delays per Shift

Delay	Time (min)
Weather	59
Breaks	60
Shift Change	15
Blasting	30
Communication	2
Training	1
Fuel, Equipment Moves, Other	30
Total	**197**

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 16.7.

Table 16.7 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 16.8.

Table 16.8 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 takes into account 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 also considers the specific characteristics of the Project. The LOM support and ancillary equipment requirements are listed in Table 16.9.

Table 16.9 Support and Ancillary Equipment Requirements

Equipment	Maximum Fleet Size
Track Dozer 9.8 feet (2.9 m)	1
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 16.10 shows the maximum salaried staff requirements during the LOM. The hourly mining operator and maintenance labour on payroll is shown in Table 16.11.

Table 16.10 LOM Maximum Salaried Staff Requirement

Position	Maximum Number of Employees
Technical Services Staff	9
Operations Staff	5
Total	14

Table 16.11 Operator and Maintenance Staff on Payroll

Production Year	Operators	Maintenance	Total
-1	37	22	59
1	41	23	64
2	42	23	65
3	41	23	64
4	33	20	63
5	25	17	42
6	24	17	41

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 processes is proposed for the Project. The process 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 process 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.

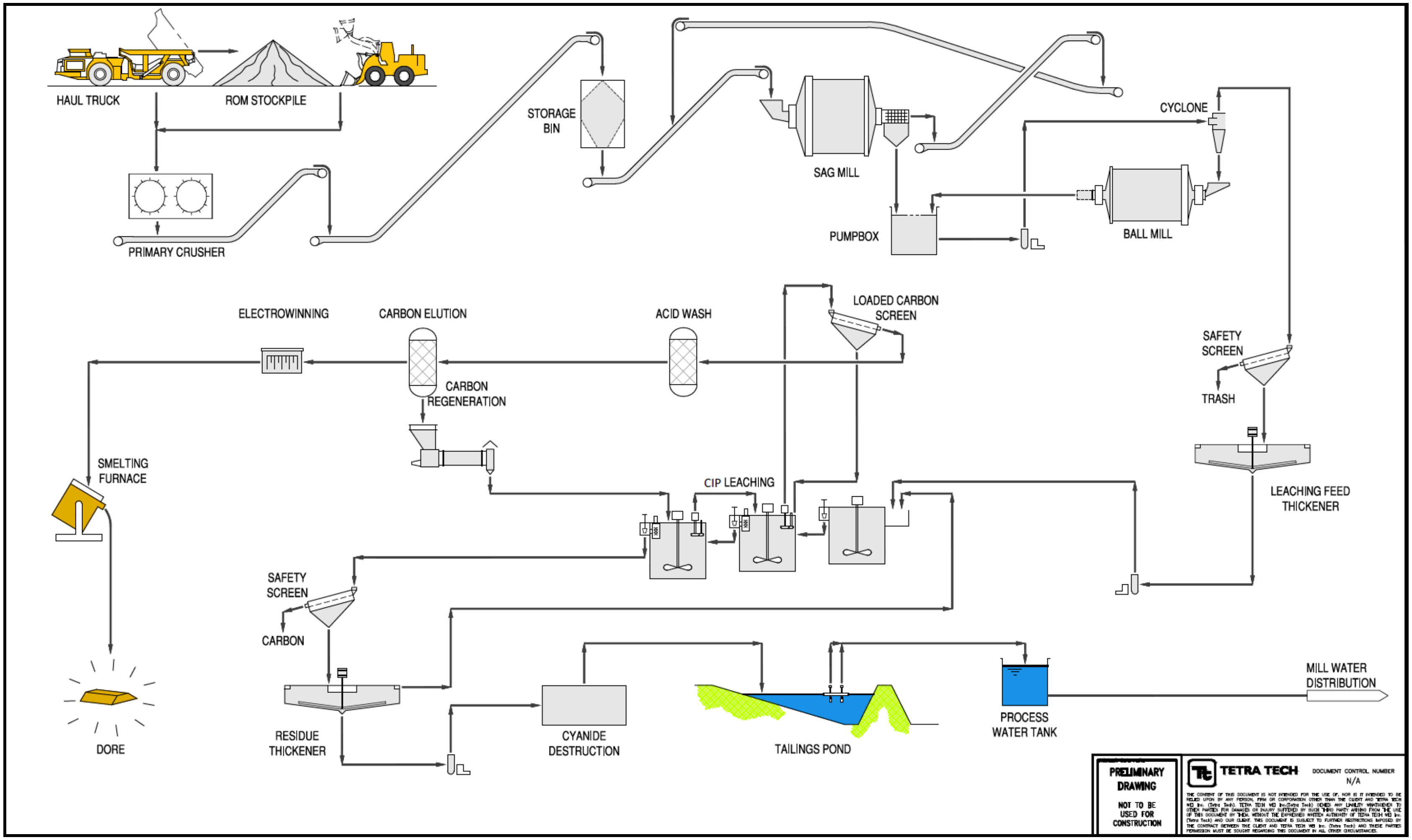
The hydrocyclone overflow from the primary grinding circuit will flow by gravity to a thickener where the slurry will be thickened for downstream cyanidation. The underflow of the thickener will be diluted with process water to the optimum solid density and be cyanide leached in a CIP circuit to recover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

As discussed in Section 13.0, previous heap leach test work showed promising results from oxide mineralization and some extensive engineering studies were previously conducted using a hybrid heap and tank leach processing system to extract the gold from the mineralization. These studies, however, revealed numerous potential technical risks and challenges associated with the heap leach option, including:

- potential agglomerate strength issues and the effect of the freeze-thaw cycle on agglomerate strength
- paucity of suitable sites near the proposed project site
- concerns over the ability to run the heap year round
- possible environmental issues
- concerns over the suitability of the sulphide resource to heap leaching.

To better recover the gold from both oxide and sulphide mineralization, simplify the processing operation, and improve project economics, 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 process 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	%	84
Gold Production, Average	oz/a	~50,000
Ore Characteristics		
Head Gold Grade , Average	g/t Au	3.5
Specific Gravity - Oxide	-	2.38
- Sulphide	-	3.38
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

table continues...

Criteria	Unit	Value
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.

Steel balls will be manually added into the mills on a batch basis as grinding media. 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 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 to protect operators.

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 secure facility with security entrances and monitored by 24-hour 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. The 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 ensure that spills do not occur during operation. Appropriate ventilation and fire 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.

All of the solid reagents (including hydrated lime, sodium hydroxide, sodium cyanide, copper sulphate, and sodium metabisulfite) will be mixed with fresh water to 10 to 25% solution strength in respective mixing tanks, and stored in separate holding tanks before being added to various addition points by 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 areas as well.

Flocculant will be received in solid form and will be prepared in a packaged preparation system, including a screw feeder, a flocculant eductor and mixing devices. The flocculant mixing system will run automatically based on solution level of the holding tank. Mixed solution will be transferred and stored in an agitated flocculant holding tank. Flocculant will be made up to a 0.2% solution strength and added via metering pumps to the leach feed thickener and the leach residue thickener after further dilution.

17.3.11 WATER SUPPLY

Two separate water supply systems will be provided to support the process operations—one fresh water system and one process water system for various process circuits. Both the process and fresh water tanks 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 fresh water tanks will be full at all times and will provide at least 2 h of firewater in an emergency.

The potable water from boreholes will be treated (chlorination and filtration) 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 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. The overflow of the residue thickener will be directly used to dilute 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 analysis, 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 when possible.

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

In addition to the plant control system, a closed-circuit television (CCTV) system will be installed at various locations throughout the plant 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.

For the protection of operating staff, cyanide monitoring and alarm systems will be installed at the cyanide preparation, leaching and destruction areas.

17.4 Yearly Metallurgical Performance Projection

According to the test work results 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*

Year	Oxides to Mill		Sulphides to Mill		Recovery		
	Tonnage (t)	Grade (g/t Au)	Tonnage (t)	Grade (g/t Au)	Oxides (%)	Sulphides (%)	Total (%)
1	305,209	5.20	2,891	4.26	90.9	66.7	90.7
2	501,510	5.83	8,486	3.38	91.5	61.3	91.2
3	446,541	3.45	30,551	3.27	89.5	59.7	87.7
4	487,862	3.22	16,694	1.39	89.2	55.1	88.7
5	263,950	2.97	239,750	3.23	89.0	58.2	73.7
6	192,501	3.27	319,357	3.08	89.4	57.4	69.9
7	39,930	1.43	75,307	2.05	87.4	55.1	63.8
Total	2,237,503	4.06	693,036	2.99	90.3	57.7	84.2

Note: *excluding a total of 288,843 t of mining dilution materials

18.0 PROJECT INFRASTRUCTURE

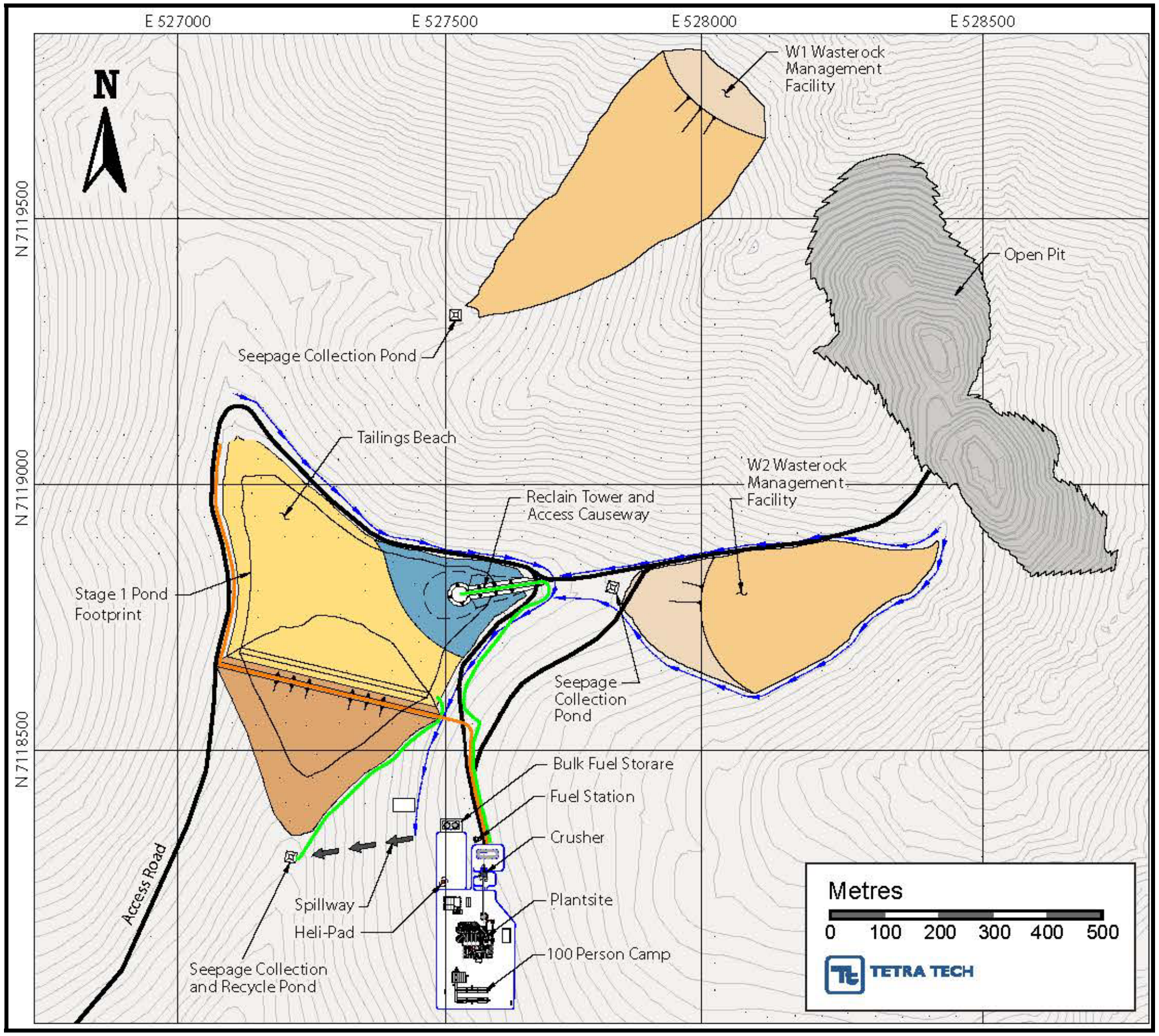
18.1 SITE LAYOUT

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
- a laydown area
- 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
- 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 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-metre 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

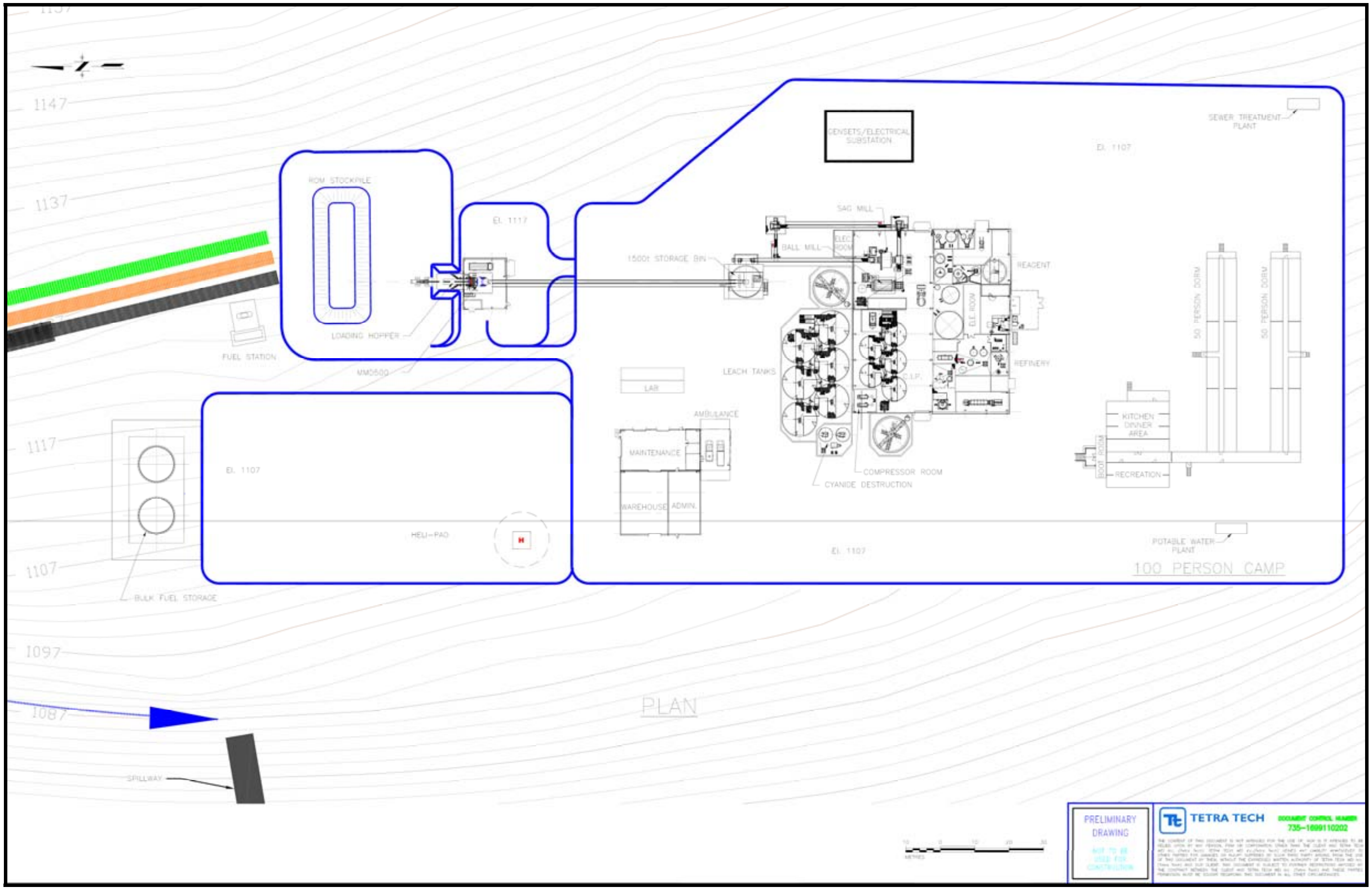
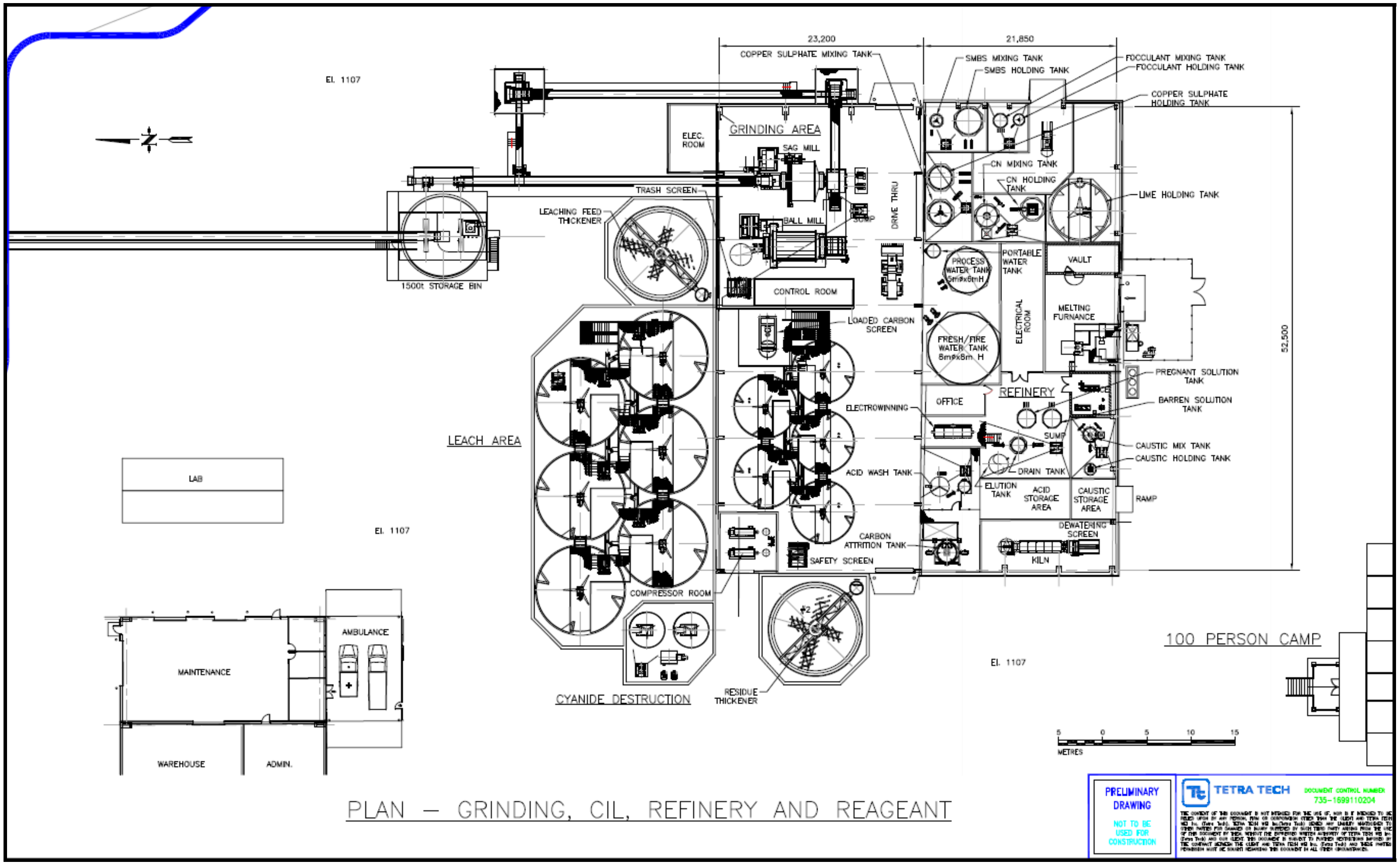


Figure 18.3 Grinding, Leaching and Recovery Area Layout



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 centres (MCCs) and power distribution centres 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

Knight Piésold developed the basic design criteria for the TMF and WRMFs which are summarized in Table 18.1.

Table 18.1 Design Criteria Summary

Parameter	Units	Value
Average Mill Throughput	t/d	1,500
Design Life	a	7
Total Tonnes of Tailings (Design)	Mt	3.2
Year 1 – Tailings Tonnage	t	356,928
Years 2 to 6 – Tailings Tonnage	t	547,500
Year 7 – Tailings Tonnage	t	125,094
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 and Freeboard)	m	5
2 Year Starter Tailings Tonnage	t	904,428
Waste Rock Density	t/m^3	2
Waste Rock Tonnage	Mt	15.6
Waste Rock Volume	Mm^3	7.8
Overall Waste Rock Storage Facility Slopes	-	2H:1V

The following assumptions were 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; therefore the TMF will be fully lined with an engineered liner system in the basin area and extended up the embankment face 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 takes 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
- inclusion of monitoring features for all aspects of the facility to ensure performance goals are achieved and design criteria and assumptions are met
- staged development of the TMF over the life of the Project.

TMF Design Features

The general site layout is shown in Figure 18.1. The TMF has the following specific features for tailings and water management:

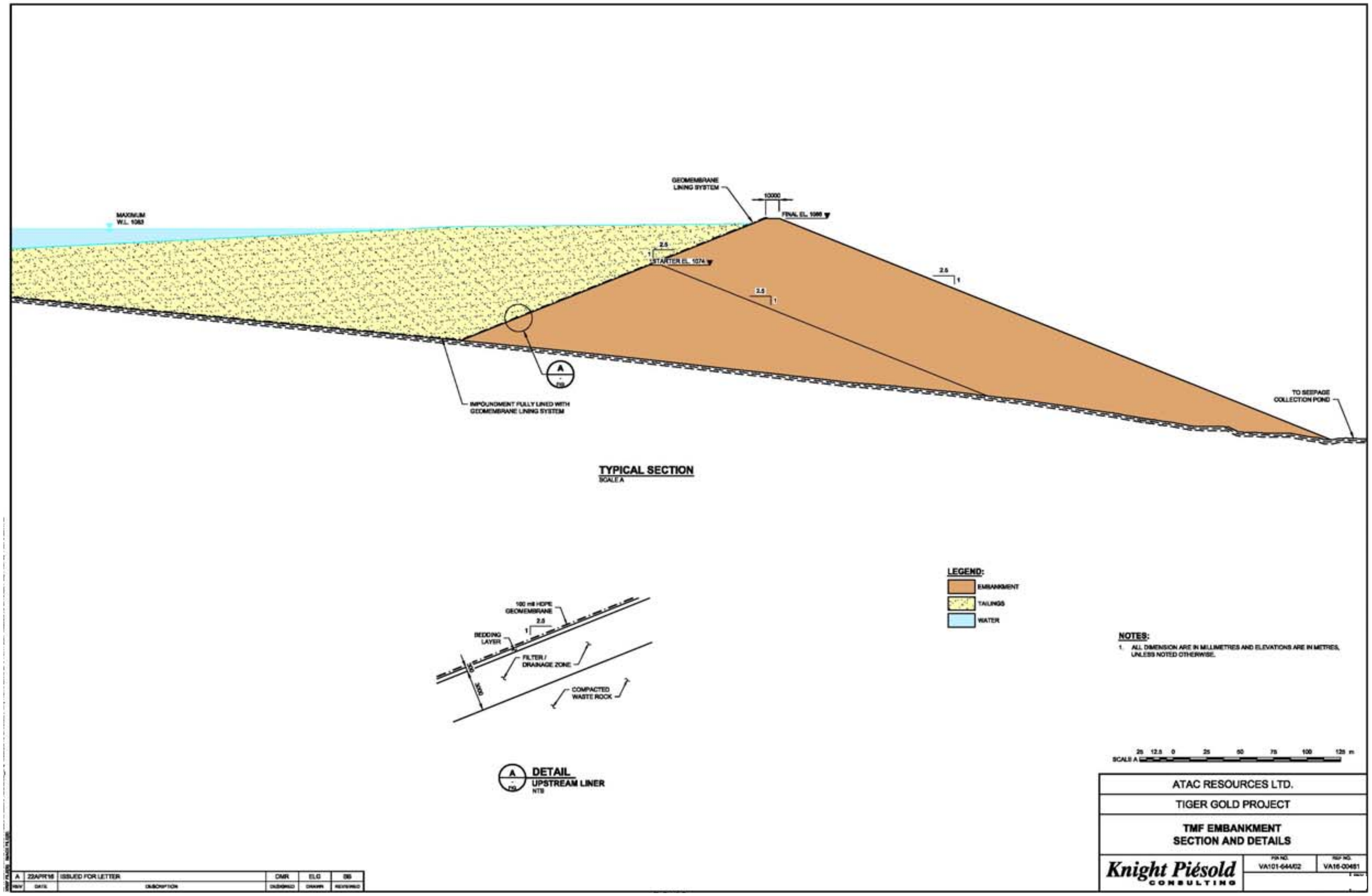
- zoned embankment constructed from mine waste rock and processed filter/drainage zones
- fully lined impoundment to minimize seepage losses
- basin drainage system
- foundation drainage system
- tailings beach
- tailings distribution system
- reclaim water system
- diversion ditches.

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 geosynthetic liner to minimize seepage of tailings and water into the surrounding area. The filter zones provide a bedding surface for the liner to prevent the migration of fines downstream in the event of liner damage.

Expansion of the TMF would be through the downstream method using waste rock and processed materials. An initial starter dam would be constructed to contain the first two years of tailings production, in order to minimize up front capital expenditure. The dam would be raised twice over the LOM to increase the storage capacity and maintain a minimum of 5 m freeboard at all times. A cross section of the embankment is shown in Figure 18.4.

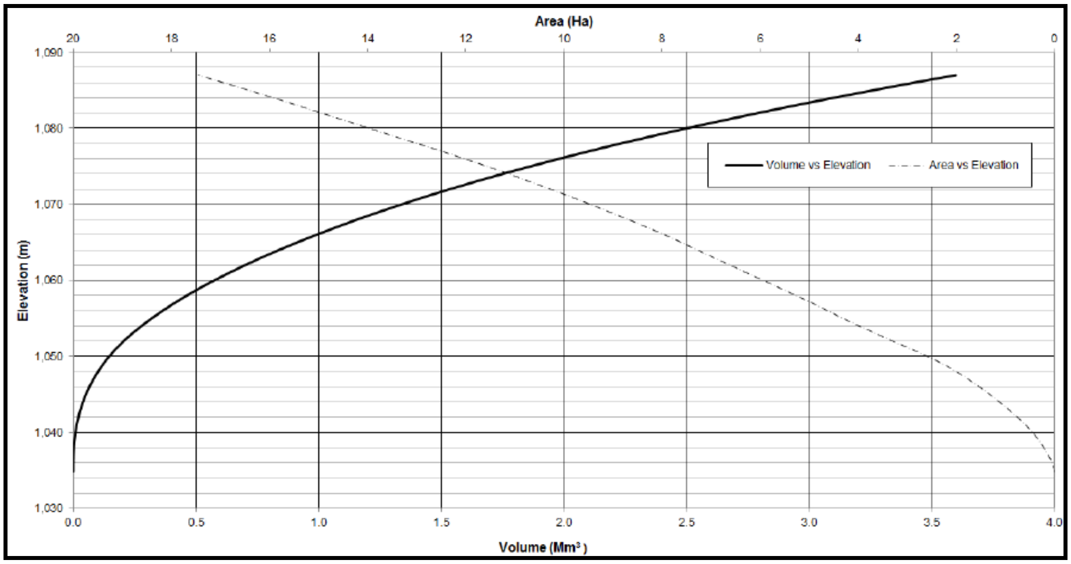
Figure 18.4 TMF Embankment Section



TMF Capacity and Filing Schedule

The slurry tailings option developed for the TMF provides storage capacity for almost 3 Mm³ for tailings, process water, storm storage, and freeboard to an elevation of 1,088 m. This will provide storage for seven years of mine operations. The depth/area/capacity (DAC) relationship for the site to an elevation of 1,088 m is shown on Figure 18.5.

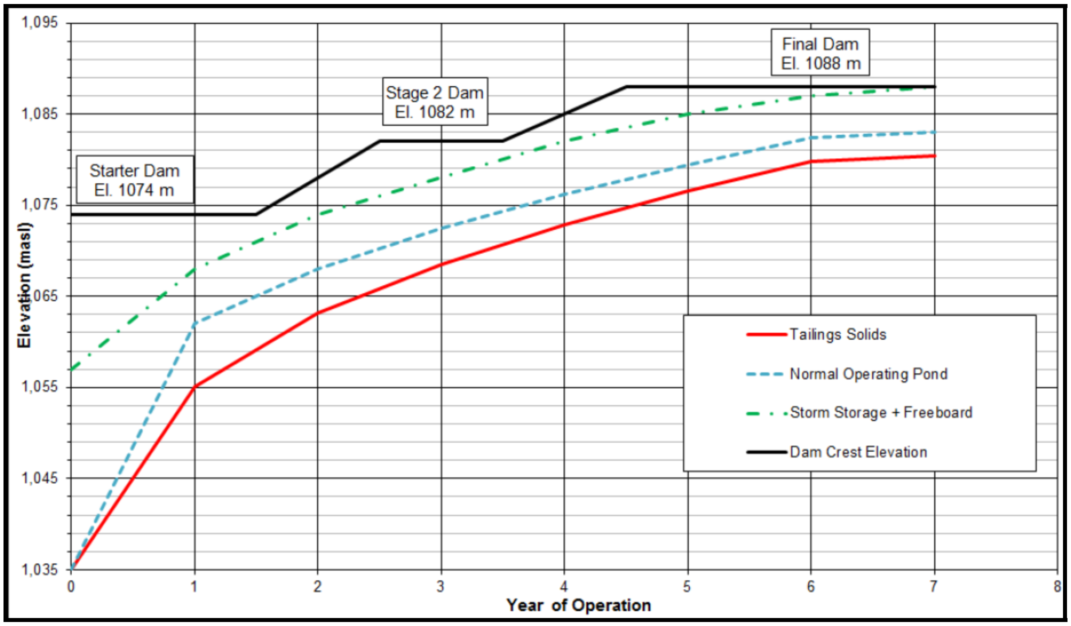
Figure 18.5 TMF DAC Relationship



The facility will have sufficient freeboard to manage runoff, storm storage, and process water. Reclaim water will be recirculated from the supernatant pond back to the mill and used as process water. The reclaim water will be pumped to the mill using a submersible pump installed in a wet well surrounded by coarse rockfill to serve as an access causeway. This allows water to be reclaimed beneath the ice and snow cover.

The staged filling schedule for the TMF is shown in 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 Embankment. 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%. Finer tailings particles will travel farther and settle at a flatter 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 embankments.

Water Reclaim System

The water reclaim system serves the following:

- 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 established at the northeast of the facility on top of the geosynthetic liner to allow process water to be reclaimed throughout the operation. The reclaim tower will be raised over the life of the facility and will be accessed via an access causeway constructed from 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 constructed around the wet well to prevent fines migration.

Basin Underdrainage System

A basin underdrainage system is included in the design to drain the tailings mass and promote consolidation during early years of operation. Poor consolidation of tailings has relevant influence on the design, as it reduces the achievable density and therefore reduces storage efficiency of the TMF.

The drains will work most effectively during initial deposition, although they will lose some efficiency as the facility is developed. The underdrainage is most important during the early stages of TMF development, when the tailings have higher moisture content and it is difficult to control particle segregation. The drain efficiency reducing with time is not critical, as the function of placing wide depositional beaches and controlling tailings discharge using spigots allows greater control of the water during later stages.

Seepage Collection and Recycle Ponds

Seepage to the embankment drainage zones will be largely controlled by the geosynthetic liner and tailings beach. Seepage that is intercepted in the embankment will be routed to the seepage collection and recycle ponds located at the topographic low point along 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 also be collected in the pond. Water collected in pond will be continuously monitored and pumped back into the TMF via a recovery pipeline.

Emergency Spillway Design

The TMF would 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 location of the sites are based on topography and proximity to the proposed pit location and identified as sites W1 and W1 shown in 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 will be pushed out across the gully in 50-m increments. This WRMF is designated W1 and provides capacity for 8 Mt of waste rock storage between elevations 1,220 to 1,550 m stacked at an overall slope of 2H:1V in the gully. Waste rock to W1 would be developed in stages depending on the mining sequence of the hill slope. Site W2 is utilized later in the mining operation for storage of up to 7.4 Mt of waste rock between elevations 1,110 to 1,200 m within the drainage. The bulk of the waste rock to this site is delivered from lower elevation excavation of the pit. 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
- WRMFs W1 and W2
- mill (including fresh and process water tanks)
- 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.

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 ENVIRONMENTAL SETTING

The Property is located within 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. Outcrop is most abundant near ridge crests and along 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. 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 understorey. 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 understorey 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 terrains 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 within the Yukon Territory ranging from lows of -62°C to highs of +36°C. Higher terrains experience 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).

ATAC is preparing to permit a 67 to 69 km, all-season, gated and restricted access road to support advanced exploration at the Tiger Deposit and elsewhere along the Rau Trend. Public consultation commenced during spring 2016 and is ongoing in advance of a formal YESAB submittal. The assessment and permitting for the proposed access road is independent of the Project.

20.2 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. Environmental evaluations, permitting, and licensing regimes for Yukon's mining projects are described in Section 20.4.

The following overview, presented in Table 20.1, represents the current status and principle investigators for relevant environmental studies on the Property, along with identified areas where further studies will be necessary, as the Project progresses.

Table 20.1 Environmental Studies Overview

Field	Source	Adequacy Assessment	Network Coverage	Data Frequency	Important Issues Identified	Comments
Water Quality-Surface Water	J. Gibson Environmental Consulting	Good quality assessment conducted on 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 will be added to the network during Type A Water Use Licensing.	Good	Minor exceedance of CCME Guidelines for Aquatic Life for selenium in five of twelve water quality stations; some iron, aluminum and zinc exceedance in stations.	None of the five stations with high baseline levels for metals are directly affected by the Tiger Deposit.
Hydrology – Surface Water and Stream Gauging	J. Gibson Environmental Consulting	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 if possible.	Good for stage of project, with the possibility that additional water quantity license monitoring and compliance points will be added to the network during Type A Water Use Licensing.	Year-round continuous monitoring recommended where possible at key sites (Rau 11, Rau 12, Rau 9).	None identified	Continue monthly program; 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.
Hydrology – Subsurface Groundwater Quality and Characterization	N/A	Gap Identified. Detailed understanding of nature and characteristics of groundwater will be required to support advanced mine development licensing.	No data collected to date.	N/A	N/A	A subsurface hydrological investigation will need to be undertaken prior to future stages of study. Data should be collected and analyzed so as to provide an accurate characterization of groundwater depth, flow and quality, where potentially affected by pit, waste rock disposal areas, plant site, and TMF.

table continues...

Field	Source	Adequacy Assessment	Network Coverage	Data Frequency	Important Issues Identified	Comments
Wildlife	Laberge Environmental Services	Very Good	Good	N/A	None identified	
Vegetation	Laberge Environmental Services	Very Good	Good	N/A	None identified	No rare plant species noted
Benthic Invertebrates	Laberge Environmental Services	Very Good	Good	N/A	None identified	With the exception of Cameron Creek, all sites have robust, diverse benthic invertebrate populations with good representation of pollution sensitive organisms.
Stream Sediment Characterization	Laberge Environmental Services	Very Good	Good	N/A	Exceedances of CCME ISQG* and PEL* ubiquitous throughout study area for arsenic, cadmium, and zinc, some exceedances for lead and occasional exceedances of ISQG for chromium and copper.	Baseline levels for metals in sediment detected at stations within the direct project footprint including Rau 11, Rau 12, Rau 9, and Rau 4.
Fisheries	Laberge Environmental Services	Very Good	Good	N/A	Fish habitat limited to the lower reaches of Camp Creek, fish tissue studies show no significant elevation of metal levels.	Project direct impacts to streams (Camp Creek and tributaries) should not directly impact fish habitat; habitat impact mitigation and compensation plans should not be required.

table continues...

Field	Source	Adequacy Assessment	Network Coverage	Data Frequency	Important Issues Identified	Comments
Heritage	Matrix Research	HRIA* and HROA* completed for access road, airstrip: good for stage of project	HRIA was conducted on 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.	N/A	Heritage impact terms and conditions will be part of the Mining Land Use licensing conditions. Completion of an updated HRIA is recommended for the unassessed portions of the proposed road corridor.	Evaluate the existing HRIA work to determine any requirement for additional site work (HROA identified several areas of high potential along the access route, but nothing at proposed development locations). HRIA avoidance areas can be used for future development planning and design.
Geochemical Characterization	N/A	Gap identified: Currently, available data includes 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.	N/A	N/A	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 be commenced prior to future study, to 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. Geochemical characterization may be required for borrow sources and overburden stripping areas.

table continues...

Field	Source	Adequacy Assessment	Network Coverage	Data Frequency	Important Issues Identified	Comments
Permafrost	EBA	Good for stage of project; detailed design phase will require more in-depth study. The project will require accurate and complete understanding of project area permafrost regime to support YESAB submission, major licensing, and final design.	Good; more robust analysis (drilling and installation of monitoring instrumentation) and fill-in coverage necessary to plan for development of major site infrastructure.	N/A	Current preliminary work notes that Tiger Deposit production area is located in a region of widespread discontinuous permafrost, and there is a moderate to high probability of permafrost occurrence.	Construction of major site infrastructure will require a robust understanding of site geotechnical and permafrost conditions. A geotechnical drilling program will be required to determine sub-surface conditions to support the detailed design phase of site development.
Climate and Weather	J. Gibson Environmental Consulting	Good for stage or project with continuous monitoring data available from 2013 and 2014 at RAU airstrip; ongoing continuous weather station data collection is advised to support senior permitting	Weather monitoring data collection from the project site (higher elevation than RAU airstrip) may be required to support detailed design and project assessment and senior permitting.	N/A	N/A	Localized temperature, precipitation, and wind data collection (including snowpack surveys) should be continued. Detailed localized climate data analysis will be required for senior permitting.

Note: *CCME (1999) interim freshwater sediment quality guidelines (ISQG) and Probable Effects Levels (PEL)
CCME – Canadian Council of Ministers of the Environment; HRIA - Heritage Resource Impact Assessment; HROA – Heritage Resource Overview Assessment; ABA – acid-based accounting; ARD – acid rock drainage; ML – metal leaching

20.3 TAILINGS MANAGEMENT

The basic design criteria for the TMF includes the storage of 3.2 Mt of tailings over a seven-year deposition schedule. The preliminary TMF design and deposition schedule was developed by Knight Piésold, with details presented in Section 18.3. Construction of the TMF embankments will use waste rock and waste rock processed materials, and the preliminary design specifies that the TMF will be fully lined with an engineered liner system in the basin area to prevent leachate loss. The TMF dam will be constructed in three phases over the life of the project. Sequencing was designed to provide necessary freeboard allowances to manage runoff, storm storage, and process water.

Tailings will be delivered to the TMF from the mill via gravity in a tailings pipeline and discharged from a series of valved off takes located along the TMF dam crest and perimeter road. Tailings beaches will be developed with the intent to maximize storage volume and to control the location of the supernatant pond away from the dam embankment.

A basin underdrainage system will be placed below 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 intercepted in the embankment drainage zones will be routed to the seepage collection and recycle pond located at the toe of the TMF embankment. Water collected in the pond will be monitored and can be pumped back into the TMF via a recycle pipeline if required. An emergency spillway will be incorporated into the embankment abutment as a contingency to convey excess water safely from the TMF in case of emergency.

20.4 WASTE ROCK MANAGEMENT

All major mine components, including the WRMFs will be designed and operated with progressive reclamation techniques so as to afford the most efficient closure scenario possible.

The preliminary WRMF size, locations, and construction sequencing were developed by Knight Piésold, with details presented in Section 18.4. Waste rock storage totaling 10.2 Mt, as per Section 16.5, will be located in two gullies to the northwest (W1) and south (W2) of the deposit. W1 will be utilized initially followed by W2 and developed in stages depending on the mining sequence of the hillslope and pit excavation elevations. The concept for WRMFs involves construction by first selectively placing large boulders of limestone waste rock as the foundation layer. This carbonate layer will act as a French drain to permit spring snowmelt to run through the base of the facility undiminished in rate of flow. After construction of this initial drainage layer, waste rock will then be hauled along contour and lobes pushed out across the gully in increments.

Diversion ditches will be constructed to divert clean runoff around the WRMFs and contact water will be collected at the base of the storage areas and directed to the TMF.

20.5 WATER MANAGEMENT

Solution balance and water management for the process plant are discussed in Section 17.0. A conceptual overall site water management was developed by Knight Piésold, with key details presented in Section 18.5. Key concepts include 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 be conveyed away from project contact areas via appropriate diversions to prevent surface water runoff from entering the process system. Groundwater wells will be installed for fresh water supply and potable water.

Sediment traps will be constructed at intermediate locations to allow surface water runoff that drains from disturbed areas to settle prior to naturally overflowing to the environment.

Future work will include rinse and neutralization tests to determine the rinse cycle required for spent leach materials, as well as to what degree rinsing will affect the water balance and lead to further refinement of the overall site water balance.

20.6 PROJECT PERMITTING REQUIREMENTS

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

- Senior 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.
- Minor 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.

The discussion presented in this report will only be concerned with an assessment of the senior permits and licenses, as it is assumed that the numerous minor permits will be secured as necessary to support the mine development time schedule. Prior to production, the Project will require the following senior authorizations, shown in Table 20.2.

Table 20.2 Senior Authorizations Required

Mine criteria trigger	Authorization Required	Issuing Agency	Legislation
>100 t/d gold mine	YESAA 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

As indicated in Table 20.2, the first step in mine permitting in Yukon is the environmental and socio-economic assessment conducted under the YESAA. While development of the Project will require more senior review from the Executive Committee as noted above, it is nevertheless instructive to view the conclusion reached by the Mayo Designated Office of the YESAB in 2014 in their evaluation of the current mining land use permit:

> *Based on the comments submitted and other relevant considerations, three valued components were identified: Wildlife and Wildlife Habitat, Other Land users and Environmental quality. The Mayo Designated Office has determined that the Project will have significant adverse effects on the above-mentioned valued components. The application of existing legislation, the Proponent's mitigations (Appendix A), as well as the recommended mitigation measures (listed below) are adequate to mitigate the significant adverse effects of the Project.*

YESAB evaluation reports typically provide recommendations for the development of adaptive management plans for specific components of the mine, which will then be incorporated either into the Quartz Mining License (terrestrial effects mitigation) or the Water License (aquatic effects mitigation). Management plans will therefore include, at a minimum:

- fish and fish habitat management plans, including habitat impact mitigation and compensation plans that satisfy section 35(2) of the *Fisheries Act* (if necessary)
- site and access road management plan, including traffic management, maintenance and safety on site roads and on the construction site
- waste rock management plan
- tailings management plan

- TMF operations maintenance and surveillance plan
- ARD/ML prediction, prevention, and management plan
- water management plan
- air emissions and fugitive dust management plan
- materials handling and management plan
- soil management plan
- adaptive management plan
- hazardous goods storage and management plan
- explosives management plan
- erosion control and sediment control plan
- vegetation management plan
- wildlife management plan
- spill contingency and emergency response plan
- domestic and industrial solid waste management plan
- airport and aircraft management plan
- archaeological and heritage site protection plan
- construction plan, including provision for environmental supervision.

Preliminary stages of mine development can be and typically are authorized by early stage mine permitting, such as a Type B Water Use License for construction of permanent water crossings (supported by a Designated Office level YESAB assessment) and a Class IV Mining Land Use Authorization for construction of various mine components (again supported by a Designated Office level YESAB evaluation).

The authorizations in Table 20.2, listed in the order in which they will be acquired, will be issued for the full LOM period as described within this 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 Metal Mine Effluent Regulations under the federal *Environment Act*, which will set monitoring requirements and criteria for all discharges emanating from the mine and its various infrastructures (e.g. pit, heap, tailings pond, etc.).

Approval of a Detailed Decommissioning and Reclamation Plan will be a requirement of the Quartz Mining License. This document will be used to set security requirements, which must be met prior to receiving authorization for the commencement of commercial production. Conceptual closure measures are outlined in Section 20.7 of this report, and have been used as a basis to calculate an estimate of the security bonding.

Due to the nature of the Project and geology of the deposit, environmental assessment and permitting is anticipated to proceed without significant problems. 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 project design strength.

20.6.1 Current Permits

There is an existing current Class 3 Exploration Permit (Mining Land Use Regulations) for the Property, including the Tiger Deposit, which expires in August 2019.

ATAC is preparing to assess and permit a 67 to 69 km, all-season, gated, and restricted access road into the Rau Trend at the western end of the Project for use in advanced exploration and prefeasibility work. Public consultation is ongoing in advance of a formal YESAB submittal, scheduled for spring 2016. The assessment and permitting for the proposed access road is independent of the Project but is anticipated to facilitate development of the Tiger Deposit.

20.7 Socioeconomic, Community Engagements

The Project is located within the Traditional Territory of the NNDFN, whose people have lived a subsistence lifestyle off the land for centuries. Since concluding a Land Claim Agreement with the Government of Canada the NNDFN have developed the capacity to provide skilled personnel and a broad range of services to mining projects.

In recognition of these facts, ATAC has developed a good working relationship with 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.

In addition to hosting mining operations within their traditional lands for nearly a hundred years (primarily by the former United Keno Hill Mines Ltd.), the NNDFN has also been directly involved in modern mining and mine development projects such as Alexco Resource Corp's Bellekeno Mine, Victoria Gold Corp.'s Eagle Gold Project, and numerous other public mining companies whose projects are at the exploration stage. This historical familiarity, enhanced significantly by their recent involvement in the post-land claim era, has resulted in the NNDFN growing in capacity and sophistication as service providers of their own (e.g. fuel supply, water/sewage, personnel transport etc.) or as joint venture partners with larger specialized contractors (e.g. camp catering, underground contract mining, etc.). Skilled NNDFN personnel have filled employment roles as camp cooks, water treatment plant operators, mill workers, underground miners, road construction heavy equipment operators, environmental monitors, administration staff, exploration technicians, etc. It is expected that ATAC will continue to benefit from this enhanced local capacity as the Project advances.

Documentation of formalized socio-economic consultation is a requirement for a 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.

The community of Mayo, situated approximately 98 km southwest of the Project, is historically and currently supportive of mining and another good potential source of employees and service providers.

20.8 Mine Closure/Reclamation Requirements and Costs

All major mine components such as the WRMFs and TMF will be designed for closure. Engineering design and construction techniques will be developed and implemented so as to afford the most efficient closure scenario possible. For example, the WRMFs, are to be situated in the upper valleys of ephemeral alpine streams and will be constructed by first selectively placing large boulders of limestone waste rock as the foundation layer. This carbonate layer will act as a French drain to permit spring snowmelt to run through the base of the facility undiminished in rate of flow.

The most significant long-term closure consideration for the Project is most likely the TMF. In order to enhance long-term geochemical and geotechnical stability, water quality, and reduce long term monitoring costs and liability, the following TMF conceptual closure objectives and respective rationales are presented in Table 20.3.

Table 20.3 TMF Closure Concepts

Closure Objective	Rationale
Primary Objective	
Declassification of dam/reduction of dam classification	Improvement of long term geochemical and geotechnical safety and stability, reduction in monitoring costs
Secondary Objectives	
Reroute drainage course	Reestablishment of natural drainage around TMF
Dewatering and compaction of tailings	Geochemical and geotechnical stability, suitability for recontouring
Recontour tailings, dam, water storage area	Promote shedding of water and prevent inflow and infiltration
Perforate liner/confining layer	Prevention of water retention

20.8.1 Conceptual Closure/Reclamation Measures and Cost Estimates

The following closure and reclamation measures, shown in Table 20.4, are estimations of the activities that will be required as part of the Detailed Decommissioning and Reclamation Plan that will be prescribed within the issuance of a Quartz Mining License. These closure/reclamation measures and cost estimations are conceptual in nature. All closure measures are provisional and estimated; determination of final measures will require public, government, and First Nations consultation to determine closure objectives prior to the selection of measures to meet those objectives.

Despite the early developmental and design stages of the Project, the closure costs present within this document are believed to contain reasonable assumptions from which a security bond estimation could be completed by the Yukon Government Department of Energy, Mines and Resources, prior to the commencement of production.

A site decommissioning and reclamation plan will be required as part of the design and project proposal submission. It is the expectation that most, if not all, facilities will be removed from the site and that the associated surface disturbances will be reclaimed to minimize the impact upon wildlife and other land users. Part of the Project's design process will be to minimize the infrastructure's physical footprint as much as possible. This will be coupled with ongoing progressive reclamation that will be conducted concurrently with mining operations. Facilities that have reached their end of use will be demolished, with materials being reused/recycled or salvaged where possible, and their footprints regraded or reclaimed.

Water quality and site monitoring of conditions will continue for a period of time after the completion of operations and the removal/reclamation of facilities and their footprints.

The proponent will be required to file an annual report stating what progressive reclamation has been accomplished and the results of environmental monitoring programs. The proponent will be required to monitor and determine the effectiveness of the mitigation measures and progressive reclamation and closure work completed.

Table 20.4 Conceptual Liability Components and Closure/Reclamation Measures

Mine Component	Conceptual Closure/Reclamation Measures
Open pit	Construct earthen safety berm from geochemically benign waste rock above high-angle wall.
WRMFs, ROM stockpile	All WRMFs to have 1-m thick evapotranspiration granular cover and vegetation established. WRMFs have a designed-for-closure French drain constructed from boulder-sized carbonate rocks that are placed as an initial layer along the bottom of the WRMF creek channels. Very little/no rock to be moved at closure. Scarifying/grading to be performed if necessary to establish natural surface runoff drainage patterns. ROM pad to be demolished/buried. Soil to be tested for contamination, where needed, and removed as required. Waste rock tonnage and composition to be better understood during future studies.

table continues...

Mine Component	Conceptual Closure/Reclamation Measures
Process plant	Disturbed surface area is approximately 4.5 ha. Remove concrete footprints, remove salvageable components/steel as appropriate, bury refuse in place, scarify/regrade surface disturbances if necessary to establish natural surface runoff drainage patterns.
TMF	Closure measures for the TMF include dismantling and removal of tailings and reclaim delivery systems, the re-establishment of drainage, repurposing/modification of TMF to declassify/reduce dam classification, tailings dewatering and stabilization and recontouring, perforation of liner system, and covering with a 1-m thick evapotranspiration cover.
Fuel farm	Remove/salvage as appropriate, bury non-contaminated refuse in place, test soil for hydrocarbon spills where appropriate, establish/operate local land treatment facility if required. Scarify/regrade surface disturbances if necessary to establish natural surface runoff drainage patterns.
Site roads	Remove of culverts/safety berms and restore cross drainage. Allow natural revegetation to occur. Scarify/regrade surface disturbances if necessary to establish natural surface runoff drainage patterns.
Camp (incl. septic)	Remove camp trailers, demolish walkways/power lines etc. Scarify/regrade surface disturbances if necessary to establish natural surface runoff drainage patterns and revegetate.
Site Infrastructure (e.g. blasting magazine, sediment ponds etc.)	Remove/salvage as appropriate, bury non-contaminated refuse, test soil for contamination where appropriate and treat as necessary. Scarify/regrade surface disturbances if necessary to establish natural surface runoff drainage patterns.
Compliance monitoring and reporting	Water quality monitoring and reporting to occur at regular frequency during the immediate post-closure period with reduced frequency over time. Engineering inspection/reporting required for major site infrastructure post-closure. Personnel assumed to fly-in after Year 2.
Revegetation of surface disturbances (except open pit).	Application of locally developed seed and/or fertilizer source for revegetation.
Project Management	Includes requirements as per Contaminated Site Assessment (*Environment Act*). Supervision of contractors will be required for first 2 seasons.
Contingency environmental monitoring/water treatment	Enhanced environmental monitoring likely required for limited post-closure period with the establishment of a passive water treatment system and/or achievement of geochemical stability. Bioreactors/construction of wetlands may be required. Closure scenarios for waste rock/TMF within site drainages and long-term potential impacts to be refined and developed.

Notes:
1. Closure measures are based on conceptual mine components as currently envisaged, which may change as the Project is further developed.
2. Final costs will be developed using third-party Yukon contractor heavy equipment rates, and as such will not consider potential reductions in costs that may be obtained if undertaken by mine owner.
3. Area calculations for mine component infrastructure and surface disturbances as per project site drawings developed by Tetra Tech and Knight Piésold.
4. Assumed natural revegetation on all linear disturbances (e.g. roads,) and assisted revegetation on aerial disturbances (e.g. WRMFs).

20.8.2 Closure Cost Estimate and Security Bonding Requirements

Undertaking the measures outlined in Table 20.4, is estimated to cost approximately $4,400,000, which includes a 15% contingency of approximately $575,000. The final figure will be agreed upon by ATAC and the Government of Yukon after a third-party technical review is conducted. It is expected that the physical/earthworks required to complete the measures found in Table 20.4 can be completed within two seasons following the cessation of production. Site monitoring is expected to continue for 15 years, post closure.

Financial assurance (security bonding) will be posted to Yukon Government to secure the closure/reclamation works. A determination of the potential outstanding mine closure and reclamation liabilities associated with the Project will be sealed by a professional engineer licensed to practice in the Yukon.

Security bonding to address the estimated closure costs will be based on public liability, as determined by Yukon Government, and is typically required to be submitted in a phased manner, concomitant with estimated existing public liability. Payment of the initial tranche of security is therefore likely to be approximately 50% prior to legal "commencement of production" (a defined set of circumstances, as set out in the Quartz Mining License), followed by 25% in Year 1, and the final 25% tranche in Year 2; with revisions of the security set out in the Quartz Mining License, and based on actual circumstances at site.

The percentage of closure costs to be covered by the initial security bonding tranche will be set by Yukon Government to address the estimated liability at site, prior to mining. Repayment of the security bond will be based upon agreed inspection procedures, and sign off, by Yukon Government. Upon signing, Yukon Government will confirm that the measures and activities undertaken on site as adequate, and that the preliminary implementations have been successfully demonstrated. Yukon Government will issue a Certificate of Closure once all obligations have been met. Provisions for temporary closure, along with criteria that trigger the initiation of temporary closure, will also be terms and conditions of the Quartz Mining License.

Yukon will determine the amount and form of security to be provided by the proponent. The Yukon Government will also ensure that the security is maintained and held in trust at all times. The proponent will have to provide an initial security payment, prior to the commencement of any developmental activities to Yukon. Yukon may make periodic adjustment to the security bond to ensure that full security is held for outstanding reclamation and closure liabilities throughout the development, operations, closure and post closure of the mine site. Ongoing progressive reclamation may reduce the amount of financial security required by Yukon.

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 ($ million)	Average Unit Cost ($/t milled)
Initial Capital Costs	109.4	-
Sustaining Capital for LOM	8.3	-
LOM On-site Operating Costs	214.4	66.59

All costs are reflected in 2016 Q1/Q2 Canadian Dollars unless otherwise specified. The expected accuracy range of the cost estimates is +40%/-25%. When required, costs in this report have been converted from US Dollars to Canadian Dollars using a currency exchange rate of CAD1.00:USD0.78.

21.2 CAPITAL COST ESTIMATE

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $109.4 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 Capital Cost Summary

Description	Cost ($000)
Overall Site	3,076
Open Pit Mining	13,050
Mine Dewatering	144
Materials Crushing and Handling	1,986
Process	29,683
TMF	7,893
On-Site Infrastructure	5,026
External Access Roads	11,063
Project Indirect Costs	19,818
Owner's Costs	1,197
Contingencies	16,464
Total Initial Capital Cost	$109,400

21.2.1 Class of Estimate

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

21.2.2 Estimate Base Date and Validity Period

This estimate was prepared with a base date of Q1/Q2 2016 and does not include any escalation beyond this date. The quotations used for this PEA estimate were obtained in Q1/Q2 2016 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. When required, quotations received from vendors were converted to Canadian Dollars using a currency exchange rate of CAD1.00:USD0.78. 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, mills, and permanent buildings.

The total direct cost for the Project is estimated to be $71.9 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 $19.8 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 $1.2 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 $16.5 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 six years mine life. The estimated salvage values are summarized in Table 21.3.

Table 21.3 Salvage Value

Area	Value ($000)
Primary Crushing	250
Grinding	1,291
Leaching	1,700
ADR	973
Reagent	224
Metallurgical Laboratory	327
Control System	98
Gensets (Lease to Own)	1,117
Total	5,980

21.3 OPERATING COST ESTIMATE

21.3.1 SUMMARY

On average, the LOM on-site operating costs for the Project were estimated to be $66.59/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 Q1/Q2 2016, or based on the information from the database of the consulting firms involved in the cost estimates. When required, costs in this estimate have been converted from US Dollars to Canadian Dollars using the average currency exchange rate in Q1/Q2 2016. The expected accuracy range of the operating cost estimate is +40%/-25%.

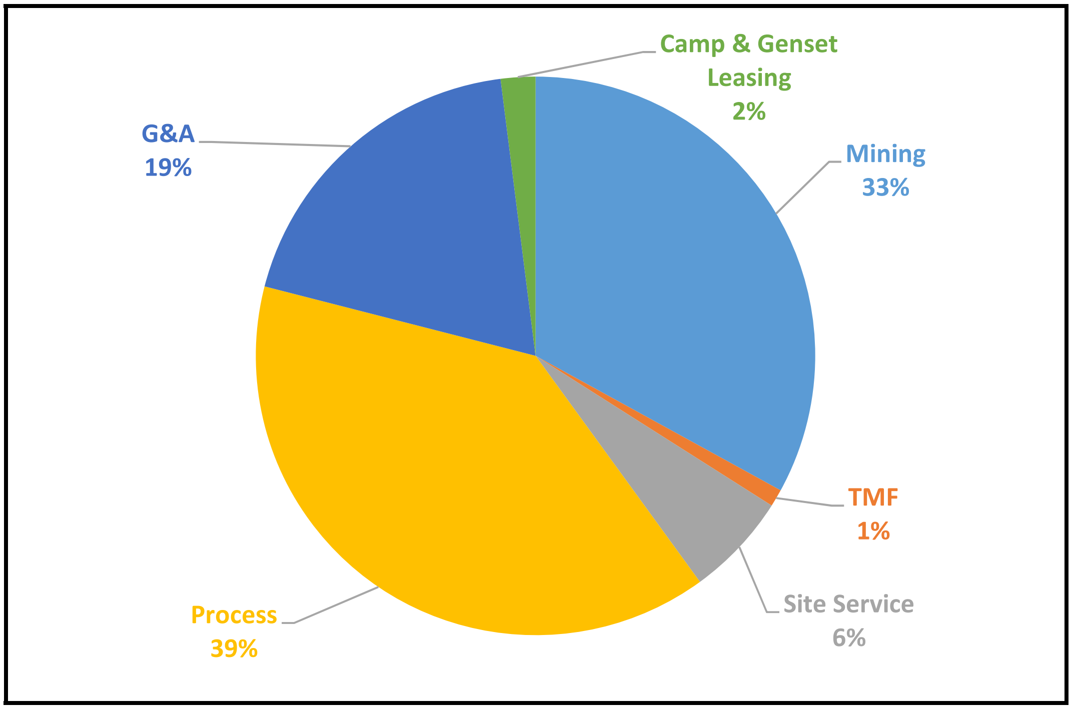
Table 21.4 LOM Average Operating Cost Summary

Area	Cost ($/t milled)
Mining	21.75
Process	26.26
TMF	0.72
G&A	12.38
Site Service	3.80
Camp and Genset Leasing Cost	1.68
Total Operating Cost	66.59

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 2015”. 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.

RELEVANT CONSUMABLES PRICES

Table 21.5 shows the consumables pricing used in the calculation of mining operating costs.

Table 21.5 Relevant Consumables Prices

Description	Unit	Price
Fuel	$/L	0.85
Lube	$/L	5.17
Emulsion	$/t	1,005
ANFO	$/t	980

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. The fixed cost of this service is estimated at $0.05/t mined, and does not include consumables. 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 21.6 summarizes the mining operating costs per activity for the pre-production period, the production period, and the LOM.

Table 21.6 Mining Operating Cost Summary

	Pre-production		Production		LOM	
Description	Cost ($ million)	Cost ($/t mined)	Cost ($ million)	Cost ($/t mined)	Cost ($ million)	Cost ($/t mined)
Drilling	0.54	0.209	3.21	0.198	3.75	0.199
Blasting	1.05	0.406	5.64	0.347	6.70	0.355
Loading	0.61	0.233	3.93	0.242	4.53	0.241
Hauling	1.68	0.645	7.75	0.477	9.43	0.500
Support Equipment	0.99	0.379	5.91	0.364	6.90	0.366
Ancillary Equipment	0.53	0.203	3.11	0.191	3.64	0.193
Dewatering	0.13	0.050	0.81	0.050	0.94	0.050
Labour	7.02	2.701	38.88	2.393	45.90	2.435
Other	0.01	0.005	0.77	0.048	0.79	0.042
Total Mining Operating Costs	12.56	4.830	70.02	4.309	82.57	4.381

21.3.3 PROCESSING

PROCESS OPERATING COSTS

The average LOM unit process operating cost was estimated to be $26.26/t milled, or $25.66/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 21.7.

Table 21.7 Process Operating Cost Summary

Description	Unit Cost ($/t milled)
Manpower (56 persons)	9.55
Metal Consumables	1.02
Reagent Consumables	4.90
Maintenance Supplies	1.96
Operating Supplies	0.44
Power Supply	7.79
Total Process Operating Cost	25.66

The process operating cost estimate includes:

- personnel requirements, including supervision, operation and maintenance; salary/wage levels, including burdens, based on the estimated 2016 Q1/Q2 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 Q1/Q2 2016 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 $0.258/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 $9.55/t milled. The projected process personnel requirement is 56 persons, including:

- 9 staff for management and technical support
- 22 operators servicing for overall operations from crushing to doré melting
- 25 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 2016 Q1/Q2 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 $8.32/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 $5.92/t milled. Most of the consumable prices were quoted from potential suppliers in Q1/Q2 2016.

The cost for maintenance/operation supplies was estimated at $2.40/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 $7.79/t milled. The electricity will be supplied by gensets on site. The power unit cost used in the estimate was $0.258/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 $0.72/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 costs were estimated to be $11.43/t milled for G&A and $3.58/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 2016 Q1/Q2 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 G&A and site service cost estimates is shown in Table 21.8. The costs for management and service personnel were estimated to be \$3.08/t milled for G&A and \$1.89/t milled for the site services. The estimated other costs for G&A and site services are \$8.35/t milled and \$1.69/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 \$3.0 million per year.

Table 21.8 G&A and Site Service Cost Estimates

Description	Manpower	Annual Cost (\$/a)	Unit Cost (\$/t milled)
G&A			
Labour	18	1,686,000	3.08
Other Costs	-	4,572,000	8.35
Subtotal	**18**	**6,258,000**	**11.43**
Site Services			
Labour	12	1,036,000	1.89
Other Costs	-	924,000	1.69
Subtotal	**12**	**1,960,000**	**3.58**

21.3.6 CAMP AND GENSET 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. According to indicative leasing terms from the potential vendors, the average LOM operating cost for leasing is estimated to be \$1.68/t milled.

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 USD1,250/oz and an exchange rate of USD0.78:CAD1.00 (all currency units are Canadian dollars unless otherwise specified):

- 34.8% IRR
- 1.85-year payback on $109.4 million initial capital
- $106.6 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 22.5 for further details).

The following post-tax financial results were calculated:

- 28.2% IRR
- 1.93 year payback on $109.4 million initial capital
- $75.7 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
Total Oxide Tonnes to Mill	t	2,237,503
Total Sulphide Tonnes to Mill	t	693,036
Annual Oxide Tonnes to Mill	t	360,888
Annual Sulphide Tonnes to Mill	t	111,780
Average Grade		
Oxide Head Grade	g/t	4.063
Sulphide Head Grade	g/t	2.992
Total Production		
Gold	oz	302,307
Average Annual Production	-	-
Gold	oz	48,759

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.

Working capital is assumed to be two months of the annual on-site operating cost and fluctuates from year to year based on the annual operating cost. The working capital will be recovered at the end of the LOM.

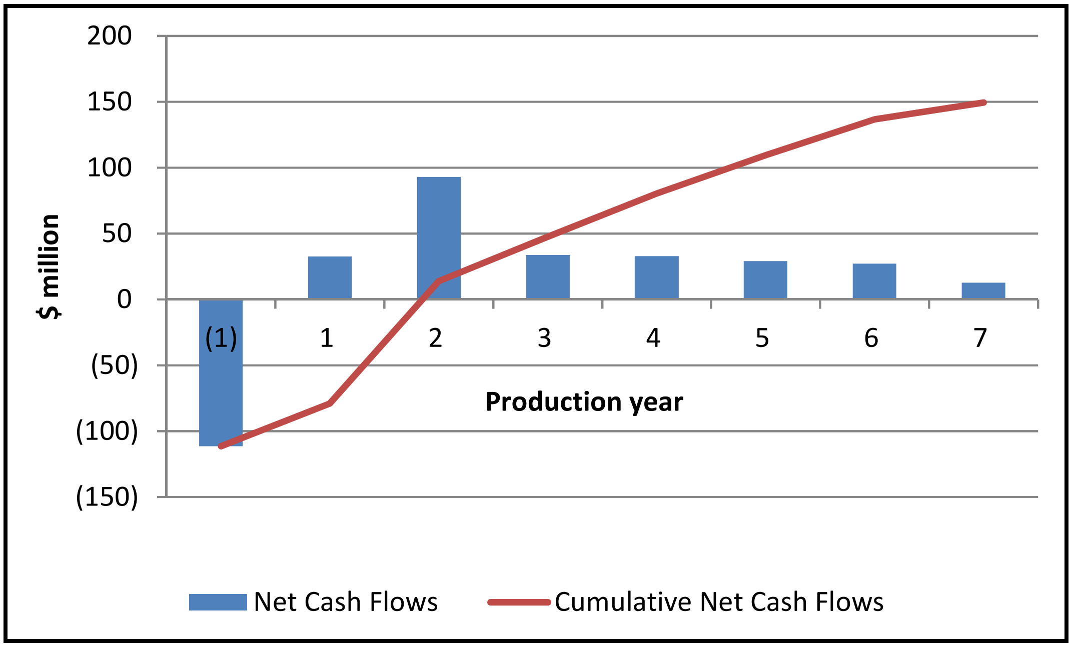
Initial and sustaining capital costs over the LOM were estimated at $109.4 million and $8.3 million, respectively.

Mine closure and reclamation costs were estimated at $4.2 million with costs bonded at 50% in Year -1, 25% in Year 1, and 25% in Year 2.

As indicated in Section 21.0, salvage value of the processing plant was estimated at $6.0 million.

The 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



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	1,250.00
	CAD/oz	1,602.56
Exchange Rate	USD/CAD	0.78
Mine/Mill Production		
Waste	t	15,628,071
Ore	t	3,219,382
Stripping Ratio	-	4.9
Milling Period	years	7.00
Gold Grade, Oxide	g/t	4.06
Gold Grade, Sulphide	g/t	2.99
Contained Gold	oz	358,974
Recovered Gold	oz	302,307
Recovered Gold Value	$000	484,466
Operating Costs		
On-site		
Mining Cost	$000	70,015
Process Cost	$000	86,866
G&A Cost	$000	39,870
Surface Services Cost	$000	12,247
Total On-site Operating Costs	$000	214,406
Off-site		
Deductions	$000	2,422
Refining/Smelting	$000	386
Doré Transportation and Insurance	$000	1,938
Royalty	$000	0
Total Off-site Operating Costs	$000	4,746
Operating Cash Flows	$000	265,314
Capital Costs		
Initial Capital	$000	109,400
Sustaining Capital	$000	8,329
Salvage value	$000	-6,000
Closure/Environmental Bonding	$000	4,198
Total Capital Cost	$000	115,926
Cash/Total Cost Net of Gold Credit		
Cash Cost (LOM)	$/oz Au recovered	725
Capital Cost (LOM)	$/oz Au recovered	383
Total Cost	$/oz Au recovered	1,108
Pre-tax Financial results		
NCF	$000	149,388
Discounted Cash Flow NPV @ 3%	$000	122,126

table continues…

Item	Units	Value
Discounted Cash Flow NPV @ 5%	$000	106,580
Discounted Cash Flow NPV @ 8%	$000	86,505
Payback	years	1.85
IRR	%	34.8

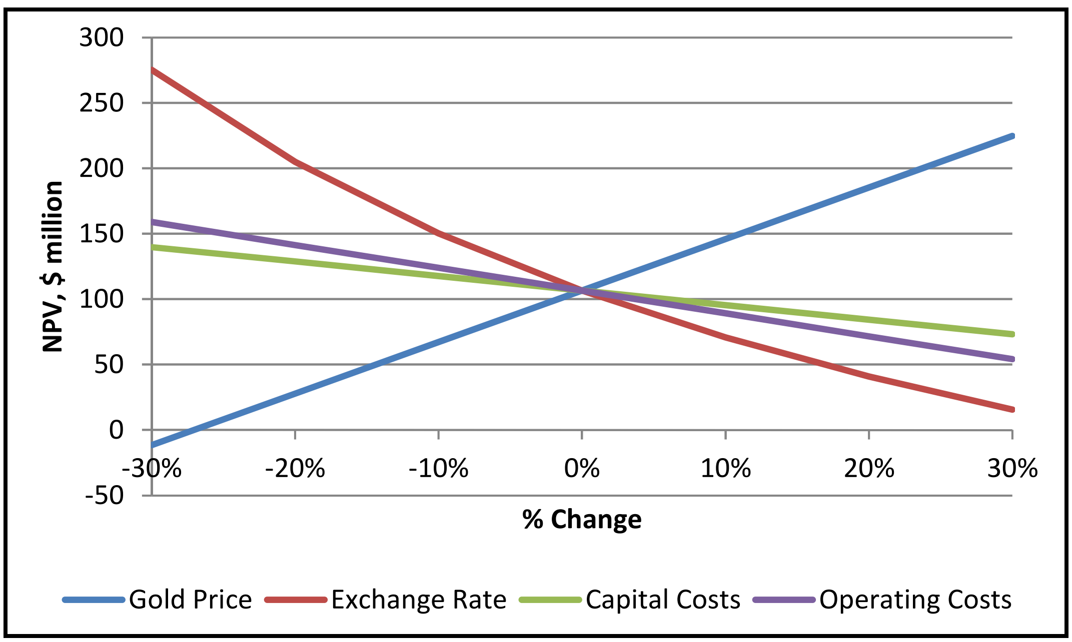
22.3 SENSITIVITY ANALYSES

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

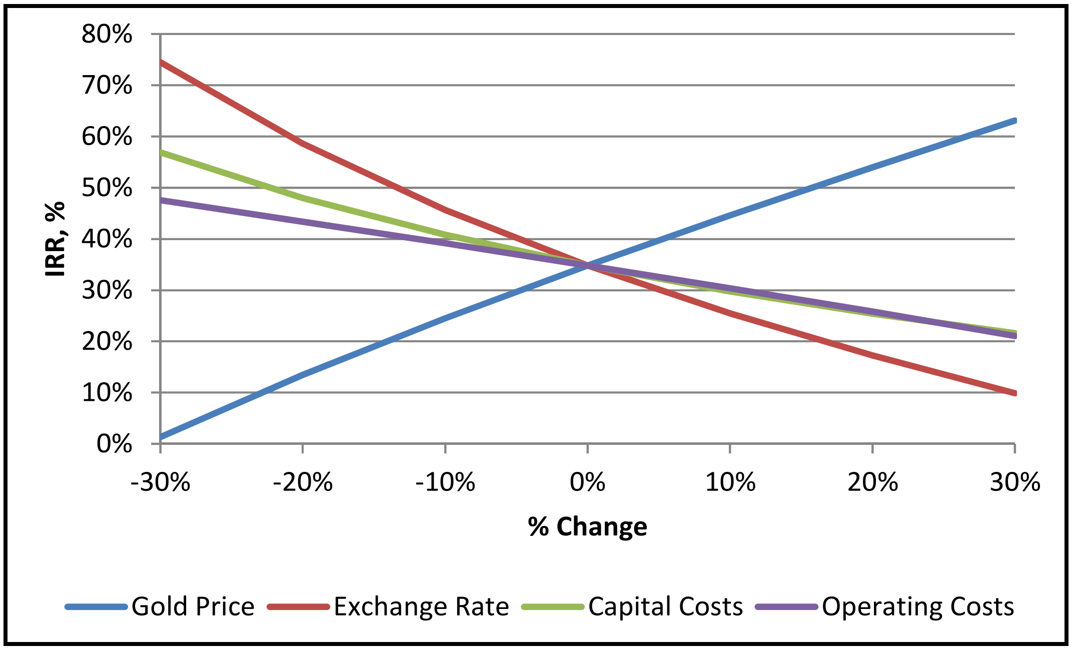
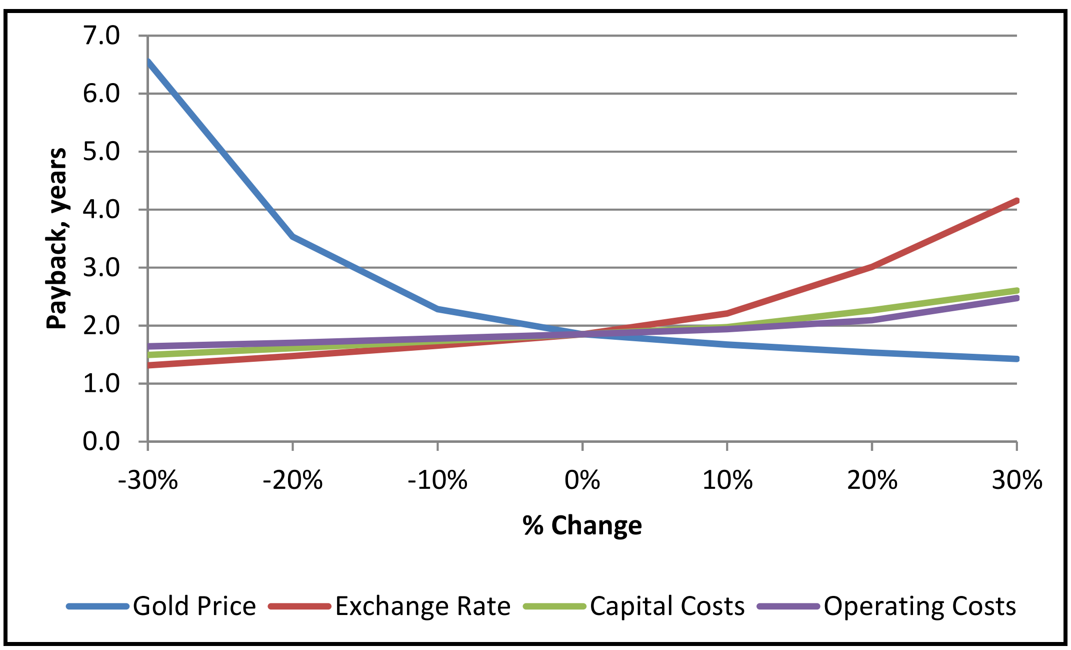


Figure 22.4 shows that the payback period is most sensitive to gold price followed by exchange rate, capital costs and on-site operating costs.

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 (USD0.78:CAD1.00)

	Gold Price (USD/oz)		
	1,200	1,250	1,300
Pre-tax CNCF ($ million	130.1	149.4	168.7
Pre-tax NPV @ 5% discount rate ($ million)	90.8	106.6	122.3
Pre-tax IRR (%)	30.8	34.8	38.8

Table 22.4 Summary of Exchange Rate Sensitivity ($1,250/oz Gold)

	Exchange Rate (USD:CAD1.00)		
	0.76	0.78	0.80
Pre-tax CNCF ($ million)	162.0	149.4	137.4
Pre-tax NPV @ 5% discount rate ($ million)	116.9	106.6	96.8
Pre-tax IRR (%)	37.4	34.8	32.3

22.4 Post-tax Financial Analysis

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.

For purposes of tax calculation, 2017 was considered the base year of project start. 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 15% 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.

Resource property acquisition costs and the costs of mine shafts, main haulage ways, and other underground workings are accumulated in the Canadian Development Expense (CDE) pool. CDE is amortized against income at 30% on a declining balance basis.

Until recently, most other pre-production mine development and exploration expenditures were accumulated in the Canadian Exploration Expense (CEE) pool. CEE was generally amortized at 100%, to the extent of taxable income from the mine. However, after the federal budget in 2013, pre-production mine development expenses will now be included as CDE rather than CEE. The transition from CDE to CDE treatment will be phased in, with pre-production mine development expenses allocated proportionally to CEE and CDE based on the calendar year in which the expenses are incurred (Table 22.5).

Table 22.5 CEE and CDE (%)

Phase-in Period	2013 to 2014	2015	2016	2017	After 2017
CEE Portion	100	80	60	30	0
CDE Portion	0	20	40	70	100

Fixed assets purchased prior to the commencement of production are accumulated in an Undepreciated Capital Cost pool (Class 41.2). Until recently, generally 100% of this pool could be amortized to the extent of taxable income from the mine. However, since the 2013 federal budget amortization is limited to 25% of the pool, additional depreciation is permitted to reduce taxable income based on the percentage allowed during the phase-out period (Table 22.6).

Table 22.6 Percentage Allowed During the Phase-out Period (%)

Phase-out Period	2013 to 2016	2017	2018	2019	2020	After 2020
Percentage	100	90	80	60	30	0

Fixed assets purchased after the commencement of production are accumulated in Class 41(b) and are amortized at 25% on a declining balance basis.

22.4.2 Yukon Mineral Tax Regime

Under the *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 (Table 22.7).

Table 22.7 Yukon Mineral Tax

	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.

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 LOM.

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.

22.4.3 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 $39.53 million over the seven-year production life. The components of the various taxes that will be payable for the base case are shown in Table 22.8.

Table 22.8 Components of the Various Taxes

Item	Value
Gold Price (USD/oz)	1,250
Exchange Rate (USD:CAD)	0.78:1.00
Income Tax Payable ($ million)	29.73
Yukon Mining Tax Payable ($ million)	9,80
Total Taxes ($ million)	39.53

The base case post-tax financial results are summarized in Table 22.9.

Table 22.9 Summary of Post-tax Financial Results

Description	Base Case
Gold Price (USD/t)	1,250
Exchange Rate (USD:CAD)	0.78:1.00
NCF ($ million)	109.86
Discounted Cash Flow NPV ($ million) at 3%	88.12
Discounted Cash Flow NPV ($ million) at 5%	75.71
Discounted Cash Flow NPV ($ million) at 8%	59.67
Payback (years)	1.93
IRR (%)	28.2

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 was 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 1,741 km^2 RGP, while the Nadaleen Trend forms 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 Project. 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 Project, 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, is located 33 km southwest of the Project and 6 km from the proposed tote-road route. MinQuest Limited has the right to earn up to a 75% interest in the Marg Project from Golden Predator Corp. 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 Project. 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 6,222 assays taken from 150 diamond drillholes, totalling 26,844 m. The effective date of this Mineral Resource estimate is October 28, 2015. A 3D solid model was constructed to constrain oxide and sulphide mineralization.

Gold distribution, within the mineralized solids, were examined using a lognormal cumulative frequency plot to determine appropriate capping levels. Three metre composites were formed, honouring solid boundaries, using the capped assay data.

The Tiger Deposit Mineral Resource was estimated by Gary Giroux, P.Eng, MASc. OK was used to interpolate gold and silver values into a block model. The search parameters were based on variography.

Mineral Resources are reported at a 0.5 g/t cut-off in oxides and 1.0 g/t cut-off in sulphides in Table 25.1. These cut-off grades were selected based on comparison to other analogous deposits.

Table 25.1 Combined Oxides and Sulphide Resource

Type	Classification	Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off		Contained Metal	
				Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
Oxides	Measured	0.50	2,600,000	3.10	4.77	259,100	398,700
	Indicated	0.50	1,720,000	2.47	4.10	136,300	226,700
Sulphides	Indicated	1.00	1,360,000	2.07	0.56	90,300	24,500
Total	**M+I**	-	**5,680,000**	**2.66**	**3.56**	**485,700**	**649,900**
Oxides	Inferred	0.50	280,000	1.52	5.67	13,700	51,000
Sulphides	Inferred	1.00	2,950,000	1.84	0.47	174,800	44,600
Total	**Inferred**	-	**3,230,000**	**1.81**	**0.92**	**188,500**	**95,600**

Note: CIM definition standards (CIM 2014) were used for the Mineral Resource.

The author is not aware of any known environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other similar factors that could materially affect the stated Mineral Resource estimate.

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 responded well to gold cyanidation at both coarse particle sizes (approximately 80% passing 10 mm or coarser) and fine particle 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 showed the oxide material to be very soft (BWi = 8.5 kWh/t) to ball mill grinding. For the sulphide mineralization, it appears that gold occurs both as solid solution in sulphides and as discrete mineralization. In the samples studied, 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.

Gravity concentration tests suggest that both types of mineralization contain little coarse discrete 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 eight years, including one year of pre-stripping followed by seven years of mill production. Over the eight-year LOM, the pit will produce 3.2 Mt of mineralized material and 15.6 Mt of waste rock. The LOM average gold grade of oxide and sulphide material is 4.06 g/t and 2.99 g/t, respectively. The LOM stripping ratio (defined as waste material mined divided by mineralized material mined) is 4.9.

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
- a laydown area
- 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 $109.4 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	Cost ($000)
Overall Site	3,076
Open Pit Mining	13,050
Mine Dewatering	144
Materials Crushing and Handling	1,986
Process	29,683
TMF	7,893
On-Site Infrastructure	5,026
External Access Roads	11,063
Project Indirect Costs	19,818
Owner's Costs	1,197
Contingencies	16,464
Total Initial Capital Cost	109,400

On average, the LOM on-site operating costs for the Project were estimated to be $66.59/t of material processed. The operating costs are defined as the direct operating costs including mining, processing, surface services, G&A, and freight costs (Table 25.3).

Table 25.3 LOM Average Operating Cost Summary

Description	Cost ($/t milled)
Mining	21.75
Process	26.26
TMF	0.72
G&A	12.38
Site Service	3.80
Camp and Genset Leasing Cost	1.68
Total Operating Cost	66.59

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 USD1,250.00/oz and an exchange rate of USD0.78:CAD1.00 (all currency units are Canadian dollars unless otherwise specified):

- 34.8% IRR
- 1.85-year payback on the $109.4 million initial capital
- $106.6 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 22.5 for further details).

The following post-tax financial results were calculated:

- 28.2% IRR
- 1.93 year payback on $109.4 million initial capital costs
- $75.7 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 $4.65 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	900,000
Geotechnical and Hydrogeological	1,150,000
Mineral Processing and Metallurgical Testing	200,000
Mining	400,000
Infrastructure	1,500,000
Environmental	500,000
Total	4,650,000

26.1 GEOLOGY AND MINERAL RESOURCES

Exploration at the Tiger Deposit has defined significant gold resources. Additional drilling and exploration should include the following:

- Access trails should be constructed along the south slope using an excavator. Mapping and sampling should be conducted along these newly constructed trails to better define the extents of oxide mineralization further up the hillside to the south.
- Infill diamond drilling within the sulphide zone in order to better define high-grade areas and to convert Inferred Mineral Resources to Indicated or better. Six holes totaling approximately 600 m are recommended to complete this.
- Infill diamond drilling within the oxide zone along strike to the southeast to convert Inferred Mineral Resources to Indicated or better. Five holes totaling 500 m are recommended to complete this.
- Update the Mineral Resource estimate upon completion of the drill program.
- Total cyanide solubility assays should be conducted on all mineralized intervals grading greater than 0.50 g/t gold.

The cost for the proposed work is estimated to be $900,000.

26.2 GEOTECHNICAL

The following recommendations are made by Golder:

- The VWPs should be read weakly during the 2016 field season. 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.

The cost for the proposed work is estimated to be $1,500,000.

26.3 MINERAL PROCESSING AND METALLURGICAL TESTING

Further test work is recommended to optimize the proposed flowsheet and better understand the metallurgical responses of both oxide and sulphide mineralization. This test work should include the determination of process design related parameters.

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
  - 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, 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 $400,000.

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.

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 $1,500,000.

26.7 ENVIRONMENTAL

Coregeo recommends the following environmental work:

- Resume a monthly hydrology monitoring program, with the addition of flow measurements where required to provide data necessary for metal loading calculations.

- A subsurface hydrological investigation will need to be undertaken prior to future stages of study. Data should be collected and analyzed so as to provide an accurate characterization of groundwater depth, flow, and quality, where potentially affected by the pit, WRMFs, plant site, and TMF.
- Geochemical characterization should be commenced prior to future study, to represent all proposed excavations, including all lithologies and waste products (e.g. tailings streams). Static testing should be adequate for feasibility level; kinetic testing will likely be required for advanced permitting. Geochemical characterization may be required for borrow sources and overburden stripping areas.
- As the mine plan/project footprint is refined, a HRIA should be conducted to determine if any heritage conflicts exist.
- Further site geotechnical and permafrost condition studies should be conducted to support a feasibility study. This will require a geotechnical drilling program in the vicinity of planned major site infrastructure developments to determine sub-surface and permafrost conditions and potential impact.
- A weather monitoring station with continuous monitoring should be established at the Project site (at a higher elevation than the Rau airstrip) in order to support a feasibility study and future project assessment and senior permitting.
- Preliminary draft management plans should be developed to assist feasibility design and future assessments and permitting.

The estimated cost for the recommended environmental work will be approximately $500,000.

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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 WEI Inc. with a business address at Suite 1000, 10th Fl., 885 Dunsmuir St., Vancouver, BC, V6B 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 May 31, 2016 (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 23 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, 1.12, 1.14, 2.0, 18.1, 18.2, 18.6, 18.7, 19.0, 21.0, 24.0, 25.5, 25.6, 26.6, and 28.1 of the Technical Report.
- 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 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 12th day of July, 2016 at Vancouver, British Columbia.

*"Original document signed and sealed by
Hassan Ghaffari, P.Eng."*

Hassan Ghaffari, P.Eng.
Director of Metallurgy
Tetra Tech WEI Inc.

28.2 SABRY ABDEL HAFEZ, PH.D., P.ENG.

I, Sabry Abdel Hafez, Ph.D., P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech WEI Inc. with a business address at Suite 1000, 10th Fl., 885 Dunsmuir St., Vancouver, BC, V6B 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 May 31, 2016 (the "Technical Report").
- I am a graduate of Assiut University (B.Sc. Mining Engineering, 1991; M.Sc. Mining Engineering, 1996; Ph.D. in Mineral Economics, 2000). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, (#34975). My relevant experience is mine evaluation, with more than 19 years of experience in the evaluation of mining projects, advanced financial analysis, and mine planning and optimization. My capabilities range from conventional mine planning and evaluation to the advanced simulation-based techniques that incorporate both market and geological uncertainties. 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 November 15th, 2013.
- I am responsible for Sections 1.8, 1.13, 3.0, 15.0, 16.0, 22.0, 25.3, 25.7, 26.4, 27.2, and 28.2 of the Technical Report.
- I am independent of ATAC Resources Ltd. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property includes acting as a Qualified Person for the 2014 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 July 23, 2014.
- 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 12th day of July, 2016 at Vancouver, British Columbia.

"Original document signed and sealed by Sabry Abdel Hafez, Ph.D., P.Eng."

Sabry Abdel Hafez, Ph.D., P.Eng.
Senior Mining Engineer
Tetra Tech WEI 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 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 May 31, 2016 (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, 11.0, 12.0, 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 12th day of July, 2016 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 GARY GIROUX, P.ENG.

I, Gary H. Giroux, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Consulting Geological Engineer with Giroux Consultants with a business address at 982 Broadview Drive, North Vancouver, British Columbia.
- 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 May 31, 2016 (the "Technical Report").
- I am a graduate of the University of British Columbia (BASc Geological Engineering, 1970; MASc Geological Engineering, 1984. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#8814). My relevant experience includes 40 years of experience estimating Mineral Resources. I have previously completed Mineral Resource estimations on a wide variety of precious metal deposits both in British Columbia and around the world, including carbonate hosted deposits at Ketza River and Miekle Mine. 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 August 30 to 31, 2011.
- I am responsible for Sections 1.6, 14.0, 18.5, and 28.5 of the Technical Report.
- I am independent of ATAC Resources Ltd. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property includes completing the Mineral Resource Estimate for this deposit in November 2011, also used in the PEA completed in 2014.
- 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 12th day of July, 2016 at Vancouver, British Columbia.

"Original document signed and sealed by Gary H. Giroux, P.Eng., MASc."

Gary H. Giroux, P.Eng. MASc.
Giroux Consultants

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 2T8.
- 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 May 31, 2016 (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 25 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 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 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 12th day of July, 2016 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 WEI Inc. with a business address at Suite 1000, 10th Fl., 885 Dunsmuir St., Vancouver, BC, V6B 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 May 31, 2016 (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, and 28.6 of the Technical Report.
- 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 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 12th day of July, 2016 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 WEI Inc.

28.7 ERI A. BOYE, P.GEO.

I, Eri A. Boye, P.Geo., of Whitehorse, Yukon Territory, do hereby certify:

- I am a Principal with Core Geoscience Services Inc. with a business address at 11 Dolly Varden Drive, Whitehorse, Yukon, Y1A 6A1.
- 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 May 31, 2016 (the "Technical Report").
- I am a graduate of the University of Victoria (B.Sc. Geotechnics, 2007) and the University of Iceland and the University of Akureyri, Iceland (M.Sc. Renewable Energy Sciences, 2011). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#37718). My relevant experience includes over 10 years of experience working in mining, mineral exploration, and related environmental impact assessment and permitting fields. 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 1.11, 20.0, 26.7, 27.3, and 28.7 of the Technical Report.
- 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 12th day of July, 2016 at Whitehorse, Yukon Territory.

"Original document signed and sealed by
Eri A. Boye, P.Geo."

Eri A. Boye, P.Geo.
Principal
Core Geoscience Services 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 May 31, 2016 (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 19 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, 27.4, and 28.8 of the Technical Report.
- 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 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 12th day of July, 2016 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.