APPENDIX BB

PRELIMINARY ECONOMIC ASSESSMENT

NOTE TO READER
APPENDIX BB

In April 2015, Treasury Metals submitted an Environmental Impact Statement (EIS) for the proposed Goliath Gold Project (the Project) to the Canadian Environmental Assessment Agency (the Agency) for consideration under the Canadian Environmental Assessment Act (CEAA), 2012. The Agency reviewed the submission and informed Treasury Metals that the requirements of the EIS Guidelines for the Project were met and that the Agency would begin its technical review of the submission. In June 2015, the Agency issued a series of information requests to Treasury Metals regarding the EIS and supporting appendices (referred to herein as the Round 1 information requests). The Round 1 information requests included questions from the Agency, other federal and provincial reviewers, and members of Indigenous communities, as well as interested stakeholders. As part of the Round 1 information request process, the Agency requested that Treasury Metals consolidate the responses to the information requests into a revised EIS for the Project.

Appendix BB to the revised EIS (Preliminary Economic Assessment) presents the analysis of the economics of the Project using indicated and inferred mineral resources. The information presented in this appendix was considered in the assessment of alternatives (Section 2.0 of the revised EIS), as well as supporting the economics assessment presented in Section 6.16 of the revised EIS. No changes have been made to this appendix from the original EIS issued in April 2015. It should be noted that since the submission of the original EIS Treasury Metals has released an updated Preliminary Economic Assessment to reflect the current market conditions at the time (March, 2017). The updated Preliminary Economic Assessment does not provide a substantial change to the overall layout or design of the project and the majority of the change comes from updated costs, metal prices and currency rates. The updated PEA provides an improved economic outlook to the Preliminary Economic Assessment presented in the original EIS. As such the revised EIS has elected to rely on the original Preliminary Economic Assessment for the EIS as it presents a more conservative view of the economic benefits of the project.

As part of the process to revise the EIS, Treasury Metals has undertaken a review of the status for the various appendices. The status of each appendix to the revised EIS has been classified as one of the following:

- **Unchanged**: The appendix remains unchanged from the original EIS, and has been re-issued as part revised EIS.
- **Minor Changes:** The appendix remains relatively unchanged from the original EIS, and has been re-issued with relevant clarification.
- **Major Revisions**: The appendix has been substantially changed from the original EIS. A re-written appendix has been issued as part of the revised EIS.

- **Superseded:** The appendix is no longer required to support the EIS. The information in the original appendix has been replaced by information provided in a new appendix prepared to support the revised EIS.
- **New**: This is a new appendix prepared to support the revised EIS.

The following table provides a listing of the appendices to the revised EIS, along with a listing of the status of each appendix and their description.

List of Appendices to the Revised EIS		
Appendix	Status	Description
Appendix A	Major Revisions	Table of Concordance
Appendix B	Unchanged	Optimization Study
Appendix C	Unchanged	Mining Study
Appendix D	Major Revisions	Tailings Storage Facility
Appendix E	Minor Changes	Traffic Study
Appendix F	Major Revisions	Water Management Plan
Appendix G	Superseded	Environmental Baseline
Appendix H	Minor Changes	Acoustic Environment Study
Appendix I	Unchanged	Light Environment Study
Appendix J	Minor Changes	Air Quality Study
Appendix K	Minor Changes	Geochemistry
Appendix L	Superseded	Geochemical Modelling
Appendix M	Minor Changes	Hydrogeology
Appendix N	Unchanged	Surface Hydrology
Appendix O	Superseded	Hydrologic Modeling
Appendix P	Unchanged	Aquatics DST
Appendix Q	Major Revisions	Fisheries and Habitat
Appendix R	Major Revisions	Terrestrial
Appendix S	Major Revisions	Wetlands
Appendix T	Unchanged	Socio-Economic
Appendix U	Minor Changes	Heritage Resources
Appendix V	Major Revisions	Public Engagement
Appendix W	Unchanged	Screening Level Risk Assessment
Appendix X	Major Revisions	Alternatives Assessment Matrix
Appendix Y	Unchanged	EIS Guidelines
Appendix Z	Unchanged	TML Corporate Policies
Appendix AA	Major Revisions	List of Mineral Claims
Appendix BB	Unchanged	Preliminary Economic Assessment

List of Appendices to the Revised EIS		
Appendix	Status	Description
Appendix CC	Unchanged	Mining, Dynamic And Dependable For Ontario's Future
Appendix DD	Major Revisions	Indigenous Engagement Report
Appendix EE	Unchanged	Country Foods Assessment
Appendix FF	Unchanged	Photo Record Of The Goliath Gold Project
Appendix GG	Minor Changes	TSF Failure Modelling
Appendix HH	Unchanged	Failure Modes And Effects Analysis
Appendix II	Major Revisions	Draft Fisheries Compensation Strategy and Plans
Appendix JJ	New	Water Report
Appendix KK	New	Conceptual Closure Plan
Appendix LL	New	Impact Footprints and Effects

A.C.A. HOWE INTERNATIONAL LIMITED
Mining and Geological Consultants

PRELIMINARY ECONOMIC ANALYSIS

OF THE

GOLIATH GOLD PROJECT

KENORA MINING DIVISION
NORTHWESTERN ONTARIO, CANADA

FOR

TREASURY METALS INC.

Report No. 964

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Effective Date: July19, 2012
Signature Date: August 20, 2012

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PRELIMINARY ECONOMIC ANALYSIS
FOR THE
GOLIATH GOLD PROJECT
KENORA MINING DIVISION
NORTHWESTERN ONTARIO, CANADA

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1 EXECUTIVE SUMMARY

This technical report ("Report") was prepared by A.C.A. Howe International Limited ("Howe") at the request of Mr. Martin Walter, President & CEO of Treasury Metals Inc. ("Treasury" or the "Company"). This Report is specific to the standards dictated by National Instrument 43-101 (NI 43-101), companion policy NI 43-101CP and Form 43-101F (Standards of Disclosure for Mineral Projects) in respect to the Goliath Gold Project (the "Goliath Project" or "Project"). This Report:

- Re-states the NI 43-101 resource estimate reported in Howe's report #955 titled "*Technical Report and Mineral Resource Update on the Goliath Gold Project, Kenora Mining Division, northwestern Ontario, Canada*" and dated November 9th 2011" (Roy and Trinder, 2011); and
- Presents a Preliminary Economic Assessment ("PEA") of the Project based on the above mineral resource estimate for a proposed operation consisting of open pit and underground mining with on-site milling.

The PEA indicates that the proposed Project is of economic interest and recommends continued work by Treasury towards a pre-feasibility study of the Project.

1.1 PROPERTY LOCATION ACCESS AND DESCRIPTION

The Goliath Project, located in northwestern Ontario, lies about 125 kilometres east of the City of Kenora, 20 kilometres east of the City of Dryden, and 325 kilometres northwest of the port City of Thunder Bay, in the Kenora Mining Division, Ontario, Canada.

The Goliath Project consists of 137 contiguous unpatented mining claims (254 units – 4,064 hectares) and 19 patented land parcels (approximately 817 hectares) as detailed in Appendix A. The total area of the claim group is approximately 4,881 hectares (approximately 49 km^2) covering portions of Hartman and Zealand townships east of the City of Dryden. Treasury holds the Project 100%, subject to certain underlying royalties and payment obligations remaining on 13 of the 19 patented land parcels. All claims are currently active and in good standing with Ontario's Ministry of Northern Development, Mines and Forestry ("MNDMF").

1.2 PROPERTY HISTORY

There is only limited documentation of exploration activity conducted on the Project area prior to 1989. Previous exploration in the area was either regional in nature or focused mainly on the western portion of the Property. Reconnaissance investigation by Teck Exploration Ltd. (now Teck Resources Limited) geologists in 1989 identified a poorly exposed, broad area of weak mineralization and anomalous gold extending through parts of Lots 3 through 8 of Concession IV of Zealand Township. The discovery hole (TL-001) on the Main Zone of the Thunder Lake Deposit was drilled in October, 1990, intersecting multiple horizons of gold mineralization with intersections of 1.5 g/tonne Au over 22.2 metres, 0.9 g/tonne Au over 11.6 metres and 17.5 g/tonne Au over 2.6 metres (Page, 1995). Land acquisition, field surveys, drilling and underground bulk sampling were completed by Teck Resources Limited ("Teck") and its various partners between late 1989 and 1998; the Thunder Lake project was put on hold in 1999. Total diamond drilling on the Thunder Lake Property from 1990 to 1998 amounted to approximately 78,461.20 metres in 293 drill holes.

In 1998, as part of the underground sampling program, 4 bulk samples from the Main Zone (No. 1 and No.2 shoots) totalling 2,375 tonnes and grading >3.0 g/tonne Au were collected from the underground workings (Page et al., 1999b). The original bulk sample of 2,375 tonnes had an estimated overall grade of 9.07 g/tonne Au or 692 ounces of contained gold (Page et al., 1999b). Metallurgical results obtained on a composite sample of 24 kg from the No. 1 Shoot indicated that cyanidation achieved the best recoveries for gold at 98.7% (Corona, 2001; Hogg, 2002). Gravity and flotation resulted in recoveries of 97.3% Au and gravity alone recovered 69.1% Au (Corona, 2001; Hogg, 2002). Final gold recovery was calculated at 96.85% and silver recoveries were approximately 38% (Corona, 2001).

By 1999, surface and underground exploration and sampling led to the outlining of the Thunder Lake Deposit and the reporting of a historical Inferred Mineral Resource (non-compliant with NI 43-101) containing 2.974 million tonnes grading 6.47 g/tonne Au, using a cut-off of 3.0 g/tonne Au and a minimum thickness of 3.0 m (CAMH, 2007; Gray and Donkersloot, 1999). Howe considers all of the historical resource estimates to be non-compliant with National Instrument 43-101 standards and as such they should not be relied upon.

1.3 Geological Setting

The Goliath Project is located within the Wabigoon Subprovince of the Archaean Superior Province, northwestern Ontario, and is situated north of the Wabigoon Fault. Much of the Project area is underlain by the Thunder Lake Assemblage, an upper greenschist to lower amphibolite metamorphic grade volcanogenic-sedimentary complex of felsic metavolcanic rocks and clastic metasedimentary rocks (Beakhouse 2000). The assemblage comprises quartz-porphyritic felsic to intermediate metavolcanic rocks represented by biotite gneiss, mica schist, quartz-porphyritic mica schist, a variety of metasedimentary rocks and minor amphibolites. Compositional layering in metasedimentary rocks strikes ~70° to 90° and dips from 70° to 80° south-southeast. The Thunder River Mafic Metavolcanic rocks underlie the south part of the Property. The mafic rocks are generally massive flows but are pillowed locally and include amphibolite and mafic dykes, which are characterised as chlorite schists. Some rocks have been described as ultramafic in character (Hogg, 2002).

1.4 Mineralization

The main zones of mineralization (Thunder Lake Deposit) project to surface approximately 250-300 metres north of Norman Road. The Main Zone, Footwall Zone (B, C and D subzones), and Hangingwall Zone (H and H1 subzones) of the Thunder Lake Deposit strike approximately east-west, varying between 090° and 072°, with dips that are consistently 72°-78° toward the south or southeast. The main area of gold, silver and sulphide mineralization and alteration occurs up to a maximum drill-tested depth of ~805 metres (TL135) below the surface, over a strike-length of approximately 2,300 metres within the current defined resource area. The historic drilling of Teck and its various partners confirmed that anomalous gold mineralization extends over a strike length of at least 3,500 metres (Corona, 1998) and work by Treasury has shown this anomalous gold mineralization and alteration to extend over a strike length of +5,000 metres.

The mineralized zones are tabular composite units defined on the basis of anomalous to strongly elevated gold concentrations, increased sulphide content and distinctive altered rock units and are concordant to the local stratigraphic units. Stratigraphically, gold mineralization is contained in an approximately 100 to 150 metre wide central zone composed of intensely altered felsic metavolcanic rocks (quartz-sericite and biotite-muscovite schist) with minor metasedimentary rocks. Overlying hangingwall rocks consist of altered felsic metavolcanic rocks (sericite schist, biotite-muscovite schist and metasedimentary rocks), with the footwall comprising metasedimentary rocks with minor porphyries, felsic gneiss and schist. Gold within the central unit is concentrated in a pyritic alteration zone, consisting of quartz-sericite schist (MSS), quartz-eye gneiss and quartz-feldspar gneiss (Corona, 2001).

The Treasury drilling programs primarily targeted the Main Zone, but the Hangingwall Zone was intersected as was the Footwall Zone by deeper drill holes. Drilling has intersected the Main Zone over a strike length of approximately 2,300 metres and a thickness of 5 to 30 metres. The Main Zone is composed of well-defined pyritic quartz-sericite schist (MSS) separated by less-altered biotite-feldspar schist (BMS). Sulphide mineralization and local visible gold (VG) occurs mainly within the leucocratic bands, but occasionally it is localized in the melanocratic bands enriched with biotite and chlorite. The sulphide content of the mineralized zone is generally 3-5% but locally is up to 15%. Highest gold and silver values are associated with very strong pervasive quartz-sericite alteration. It appears that gold content does not directly correlate with pyrite content, but generally an increase in the gold and silver correlates with an increase in the pyrite and more specifically, the sphalerite content. An increase in chalcopyrite and galena content has a lower correlation to an increase in gold values. Low grade Au-Ag mineralization is pervasive in the Main Zone, Hangingwall Zone and in the Footwall Zone, whereas high-grade gold mineralization (>3 g/tonne) is concentrated in several steeply dipping, steep west-plunging shoots with relatively short strike-lengths (up to 50 metres) and considerable down-plunge continuity. These higher-grade shoots are separated by rock containing lower grade gold mineralization.

The high-grade shoots are interpreted to be the result of tight folding of the mineralized horizon (gold concentrated in fold noses) and appear to occur at regular intervals (Corona, 1998). Very rare flakes of aquamarine green mica (fuchsite: Cr muscovite) occur in the strongly altered sericite alteration with high-grade gold. Usually, mineralized intervals are narrow (up to 0.5 metres) zones enriched with 3-10% visible sulphides (pyrite, sphalerite, galena, chalcopyrite ± arsenopyrite, ± dark grey needles of stibnite) within a wider quartz-sericite or biotite-feldspar sections with fine-grained disseminated pyrite located in the foliation planes.

1.5 EXPLORATION

Prior to Treasury’s 2008 exploration program, no exploration work had been completed on the Thunder Lake Property (Thunder Lake East and West) or the Laramide Property since 1999 and 1994, respectively (Sills, 2007). Treasury’s 2008 exploration program comprised a property wide airborne magnetic survey, ground IP, and geological surveys over the Thunder Lake deposit area, trenching and diamond drilling totalling 13,203.6 metres. Treasury’s 2009 exploration program comprised reconnaissance prospecting, outcrop channel sampling, and diamond drilling totalling 4,612.6 metres. Treasury’s 2010 exploration program comprised reconnaissance prospecting, trenching, and diamond drilling totalling 10,228 metres. Treasury’s 2011 and 2012 (to June 6, 2012) exploration programs consisted exclusively of diamond drilling totalling 49,926.5 metres and 15,635 metres respectively. Additionally the 2012 drilling included the re-entry (re-drilling) and extension of 5 historical Teck Resources Inc. diamond drill holes for a total of 473 metres.

1.6 MINERAL RESOURCE ESTIMATE

This Report re-states the mineral resource estimate for the Goliath Project prepared by Howe in November, 2011 (Howe Report #955 titled "*Technical Report and Mineral Resource Update on the Goliath Gold Project, Kenora Mining Division, northwestern Ontario, Canada*" and dated November 9th 2011 (Roy and Trinder, 2011)). Howe prepared the mineral resource estimate for the Project based on a combination of historical drill holes and holes drilled by Treasury up to Hole TL11228 that was drilled during 2011.

The mineral resource estimate for the Project is reported at a block cut-off grade of 0.3 g/tonne for surface resources (less than 150 metres deep) and 1.5 g/tonne for underground resources.

Non-diluted Indicated Mineral Resources (surface plus underground), located within the Main Zone and C-Zone, total 9.1 million tonnes with an average gold grade of 2.6 g/tonne and an average silver grade of 10.4 g/tonne, for 810,000 ounces of gold and gold equivalent.

Non-diluted Inferred Mineral Resources (surface plus underground), from all zones, total 15.9 million tonnes with an average gold grade of 1.7 g/tonne and an average silver grade of 3.9 g/tonne, for 900,000 ounces of gold and gold equivalent.

Category	Surface or Underground	Cut-Off Grade (g/tonne)	Tonnes	Gold Grade (g/tonne)	Silver Grade (g/tonne)	Gold Ounces	Silver Ounces	Gold Equivalent Ounces (of Silver)	Ounces Gold Plus Gold Equivalent
Indicated	Surface	0.30	6,002,000	1.8	7.1	326,000	1,257,000	22,000	348,000
Indicated	Underground	1.50	3,136,000	4.3	18.0	433,000	1,812,000	32,000	465,000
Total Indicated (Rounded)			**9,140,000**	**2.6**	**10.4**	**760,000**	**3,070,000**	**54,000**	**810,000**
Inferred	Surface	0.30	11,093,000	1.0	3.3	352,000	1,184,000	21,000	374,000
Inferred	Underground	1.50	4,789,000	3.3	5.2	514,000	807,000	14,000	528,000
Total Inferred (Rounded)			**15,900,000**	**1.7**	**3.9**	**870,000**	**1,990,000**	**35,000**	**900,000**

Notes for Resource Estimate:

1. Cut-off grade for mineralized zone interpretation was 0.5 g/tonne.
2. Block cut-off grade for surface resources (less than 150 metres deep) was 0.3 g/tonne.
3. Block cut-off grade for underground resources (more than 150 metres deep) was 1.5 g/tonne.
4. Gold price was US$ 1,500 per troy ounce.
5. Zones extended up to 150 metres down-dip from last intercept. Along strike, zones extended halfway to the next cross-section.
6. Minimum width was 2 metres.
7. Non-diluted.
8. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
9. Resource estimate prepared by Doug Roy, M.A.Sc, P.Eng.
10. A specific gravity (bulk density) value of 2.75 was applied to all blocks (based on 194 samples).
11. Non-cut. Top-cut analysis of sample data suggested no top cut was needed because of the absence of high-grade outliers.
12. 1 ounce gold = 57 ounces silver. Silver equivalency parameters: Metallurgical recovery: Gold 95%, Silver 72%; Price: Gold $1500 per ounce, Silver $35 per ounce.

This Report quotes estimates for mineral resources only. There are no mineral reserves prepared or reported in this technical report.

1.7 PROPOSED OPERATION

Howe has reviewed the Goliath Project at the level of a Preliminary Economic Assessment (PEA). The reader is cautioned that this PEA uses Indicated and Inferred Mineral Resources.

NI 43-101 Part 2, Section 2.3(1)(b) and Companion Policy 43-101CP, Part 2, Section 2.3(1) Restricted Disclosure, prohibits the disclosure of the results of an economic analysis that includes or is based on inferred mineral resources, an historical estimate, or an exploration target. However, under NI 43-101, Part 2, Section 2.3(3) and Companion Policy 43-101CP, Part 2 section 2.3(3), the use inferred mineral resources is allowed in a Preliminary Economic Assessment in order to inform investors of the potential of the property.

This PEA is preliminary in nature, it 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, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The proposed operation considered in this PEA includes surface and underground mining of the Goliath Project mineralization and onsite milling. Mining will be by open pit methods initially, with the pit supplying feed to the mill for 4 to 4½ years while lower grade feed is stockpiled. The overall pit will have a generally oval shape with its long axis oriented along the east-west strike of the deposit. Early in Year 2, underground development would begin with underground production commencing in Year 3 supplemented by the low-grade stockpile from surface mining. Underground mining will last for eight years.

Pre-production stripping of overburden and waste rock will take place during the final year of plant construction. The processing plant will then be fed from open pit and underground mining for 10½ years.

Treasury's targets for the proposed mining operation were:

- Capital costs of less than $100 million;
- A mill feed grade of 2 g/tonne or greater; and,
- A production rate of 90,000-100,000 ounces per year, at least for the first couple of years.

Preliminary mine planning and scheduling were carried out with the aim of achieving these targets or at least coming as close to the targets as possible. The proposed combined open pit and underground mining schedule is as follows:

Combined open pit and underground mining schedule.

		'000 tonnes												
Location		**Pre-Prod.**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Total**
Central Pit	Mill feed, t		875	64										939
Western Pit	Mill feed, t			567	512									1,079
Eastern Pit	Mill feed, t			244	144	292	49							729
Sub-Total, Open Pit	Mill feed, t		875	875	656	292	49							2,747
Underground	Mill feed, t				219	583	583	583	583	583	583	583	226	4,526
Stockpile to Mill	Mill feed, t						243	292	292	292	292	292	63	1,766
Total feed to Mill	Mill feed, t		875	875	875	875	875	875	875	875	875	875	289	9,039
Waste Stripping	t	1,800	11,740	10,300	9,480	7,500	1,210							42,030
Pit to Stockpile	Mill feed, t		767	509	386	88	15							1,766
Total Surface Material Moved	Tonnes	1,800	13,382	11,684	10,523	7,880	1,517	292	292	292	292	292	63	47,954

1.7.1 Surface Mining

A series of nested pits were optimised using the following parameters:

Pit optimisation parameters.

Item	**Value**
Exchange Rate	US$ 1.00 = C$ 1.02
Gold Price	Base Case US$ 1,375 per Ounce For Nested Pits, $875-1625 per Ounce in $50 Increments
Silver Price	US$ 26 per Ounce
Mill Throughput	2,500 tonnes per day
Unconsolidated Overburden Stripping	$4 per Cubic Metre
Mining	$3.15 per tonne (Mineralized Rock) $3.00 per tonne (Waste Rock)
SG	2.75 (Rock) 2.0 (Soil)
Processing (Gravity / Cyanide)	$15.65 per tonne Milled
G&A	$2 per tonne Milled (Added to the Processing Cost During Pit Optimisation)
Maximum Slope Angle	50° (Avg., Including Haul Roads)
Dilution	15% at 0.20 g/tonne Au, 4.3 g/tonne Ag *
Mining Recovery	90%
Milling Recovery	95% Gold 70% Silver
Smelter Return	99%
Smelter Treatment Charge / Selling Cost	1% of Base Case Price: Gold: $14 per ounce Silver: $0.26 per ounce
Tailings Disposal	(Included in Milling Cost)
Waste Rock Reclamation	$0.25 per tonne

The "US$1,175 pit shell" was selected for more detailed analysis partly because the present value of the operation steadily increases down to that pit depth. Deepening the pit beyond the US$1,175 shell does not improve the NPV. In fact, after a certain depth the NPV decreases. In other words, going deeper than the US$1,175 shell would not improve the project's value.

1.7.2 Surface Mining and Scheduling

Various scheduling scenarios were attempted before deciding on the following schedule.

Milling would be carried out at the rate of 2,500 tonnes per day.

Pre-production would consist of stripping 1,800,000 tonnes of waste rock and mining 150,000 tonnes of mineralized rock to produce an initial 60 day mill stockpile.

Open pit mining will use standard truck-and-shovel methods.

Mining would begin with the Central Pit and produce almost 90,000 ounces (gold + equivalent) in Year 1.

To meet Treasury's desired mill feed grade and yearly ounce production targets, lower grade material (between 0.5 g/tonne and 1.1 g/tonne) would be sent to a large low-grade stockpile. Rock with grades greater than 1.1 g/tonne would be sent directly to the mill stockpiles.

Because the Western Pit's average grade is slightly lower than the Central Pit's grade, the Eastern Pit (higher average grade) would be mined simultaneously with the Western Pit at a 30:70 ratio, respectively. The Western Pit would be exhausted in the Year 3 (and used for waste rock after mining is complete) with the Eastern Pit finishing at the start of Year 5.

After the end of active surface mining, rock from the low-grade stockpile would be fed into the mill at a rate of 830 tonnes per day to supplement underground production.

1.7.3 Underground Mining and Scheduling

During the second year of open pit production, a decline ramp will be sunk to provide access for underground mining. Sufficient development, including main levels and a ventilation raise, will be completed in time for the underground mine to provide some of the mill feed during the third year. Underground production will be supplemented by recovery of material from the low-grade stockpile.

The underground mining method will be longhole stoping with hydraulic backfill. The level interval is 45 metres vertically. The average stope width is 10.5 metres. Primary stopes will be 10 metres long and the backfill (classified mill tailings) will contain 5% Portland cement. Secondary stopes, 20 metres long, will be filled, but cement will not be required. This plan eliminates the need for rib pillars.

Stoping blocks were outlined at a cut-off grade of approximately 2.5 g/tonne (gold + equivalent). The majority of stopes were in the Main Zone, with other stopes in the B and C zones.

1.7.4 Milling and Recovery

The available metallurgical testwork indicates that the Goliath material is readily amenable to conventional processing and that gravity concentration followed by cyanidation can be used to obtain relatively high gold recovery.

For purposes of this PEA a flowsheet consisting of gravity concentration followed by cyanidation of the gravity tails via carbon-in-leach circuit (CIL) is selected. Selected design parameters for the study are as follows:

Selected design parameters.

Area	Parameter	Value	Units
Grinding	Bond ball mill index	11.1	kWh/t
	Grind (K_{80})	105.0	microns
Gravity	Concentrate	0.1	wt %
Cyanidation	Gold recovery (overall)	95.0	%
	Silver recovery (overall)	70.0	%
	Total cyanidation time	32.0	h

As proposed, crushed feed is ground to a K_{80} of 105 microns in a two stage grinding circuit at a rate of 2,500 tonnes per day or 912,500 tonnes per annum (2,747 tonnes per day at 91% availability). A gravity recovery circuit is incorporated within the grinding circuit for recovery of free gold. The gravity concentrate is leached separately and the product directed to the main gold recovery circuit.

Ground product from the grinding circuit is fed to a CIL circuit for gold extraction. A conventional carbon elution circuit recovers gold that is smelted to yield a doré[1] product.

1.8 ECONOMIC ANALYSIS

An Excel spreadsheet was used to model and analyse the Net Cash Flow (NCF) of the Goliath Project. The model calculates the pre-tax and post-tax NCF as well as the Internal Rate of Return (IRR) and the Net Present Value (NPV) at various discount rates. The repayment period, the minimum gold price required to breakeven, and the IRRs at higher and lower metal prices and operating and capital costs are also calculated.

1.8.1 Results

The Goliath Project returns an IRR of 32.4% on a post-tax basis and 39.3% on a pre-tax basis. The respective payback periods are 2.8 years and 2.2 years after the start of production. The "break even" price of gold is US$930 per ounce post tax and US$924 on a pre-tax basis where "break even" is the gold price required to produce a zero Net Cash Flow (i.e. all capital is paid back but no profit is incurred).

The project also generates a NCF of $249.8 million post-tax and $334.7 million pre-tax. At a 10% discount rate, the project's NPVs are $83.5 million post-tax and $119.9 million pre-tax.

The underlying assumptions and parameters used in Howe's model include:

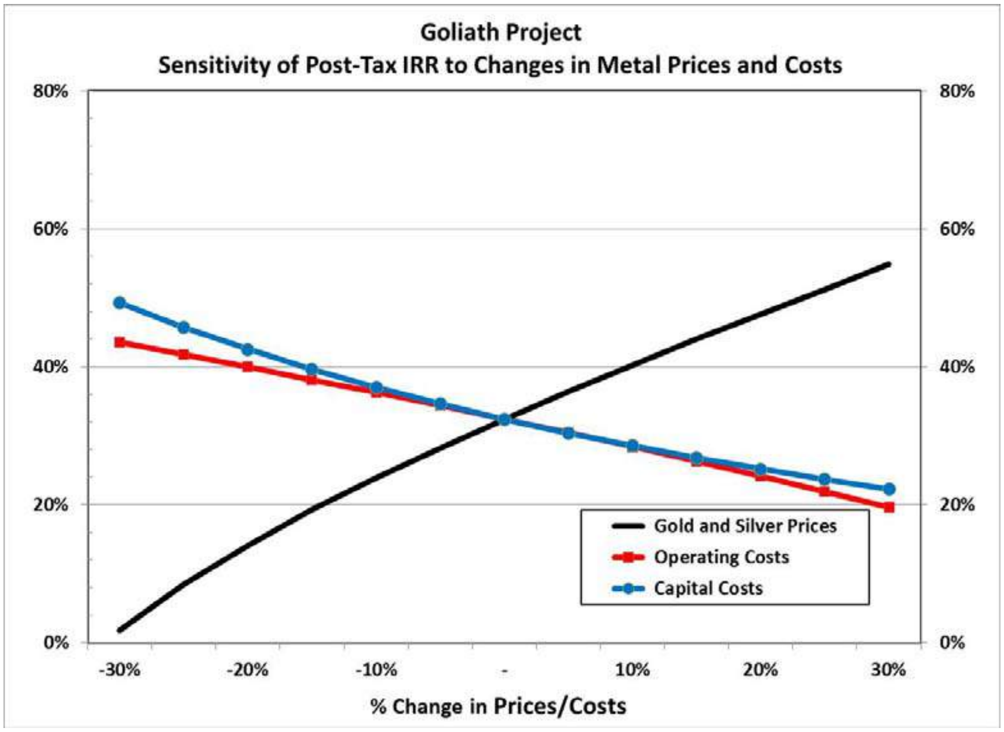
[1] A doré product is a semi-pure alloy of gold and silver created at the mine site and then transported to a refinery for further purification.

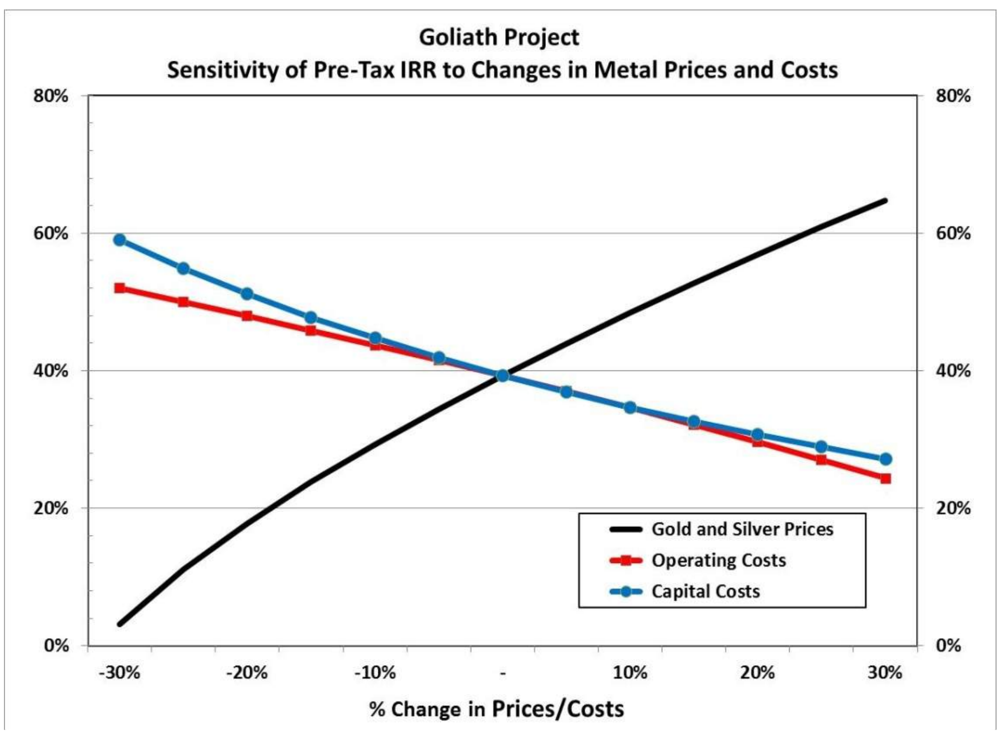
- All units of measurement are metric unless otherwise stated.
- All dollars are Canadian Dollars unless otherwise stated.
- The gold (US$ 1,375 per troy oz) and silver (US$ 26.00 per troy oz) prices are based on the average London 2nd Fixing for the last three years as of June 30, 2012.
- The United States: Canadian exchange rate (C$1.02: US$1.00) is based on the three year trailing average as of June 30, 2012.
- The model has assumed a four year pre-production period. This allows for two years to complete environmental studies, permitting, a final feasibility study and the time to put financing in place. In the second two years, the model assumes that the company will build the processing plant, supporting infrastructure and strip 1.8 million tonnes of waste.
- The production rate is designed to supply 2,500 tonnes per day (tpd) or 875,000 tonnes per annum of mineralized material to the mill. This generates an open pit life of 2 full years of production plus 3 partial years. In addition, the mine stockpiles 1,766,000 tonnes of lower grade material that is used to supplement the underground operation to satisfy mill feed requirements. The underground mine operates from year 3 to year 11 and produces a total of 4,526,000 tonnes of mineralized material. Thus the total mine life is 10.3 years
- 42,030,000 tonnes of waste are removed during the life of the open pit operation (including 1.8 million tonnes during development) for a waste: “ore” ratio of 9.3 (including stockpiled mineralized material)
- The Production schedule has been prepared by Messrs.’ Brady and Roy of Howe and includes waste and mineralized material tonnages and gold and silver grades for each production year as well by pit and underground.
- Mill recoveries are based on gravity concentration followed by cyanidation of the gravity tails via carbon-in-leach circuit (CIL) and are 95% and 70% for gold and silver respectively.
- Howe has estimated costs for gold and silver smelting and refining (including transportation and insurance) at $14.00 and $0.26 per ounce of gold and silver respectively produced by the proposed Goliath mill.
- There are a number of different royalties that apply to various areas of the Goliath property. These royalties are applied to the gold and silver revenues after deducting smelting and refining costs. The average royalty is 0.65% of Net Smelter Revenue (NSR) and at US$1,375 per oz for gold and $26.00 per oz for silver incurs a cost of $7.5 million over the life of the project.
- Capital costs have been developed by Howe and are shown in Section 21.
- Operating costs have been calculated by Howe and are shown in Section 21.
- The model calculates depreciation using the Units of Production (UOP) method. In this method the model calculated depreciation based on the amount of mineralized material milled each year.
- Working Capital is based on
  - Two weeks of precious metal inventory (at the NSR value).
  - Accounts Receivable as four weeks of metal production (at the NSR value).
  - Spare Parts and Supplies as $1.0 million.
  - Less: Accounts Payable as one half of four weeks of operating costs.
- The model calculates Federal and Ontario Corporate taxes and Ontario Mining Taxes. Basically, the Federal and Ontario Corporate taxes are based on net income as calculated for taxes.
- The Federal Income Tax base has been calculated as:
  - Earnings before Depreciation, Amortization and Taxes (EBITDA)
  - Less: Ontario Mining Taxes
  - Less: Capital Cost Allowance (CCA), i.e. depreciation where the two main forms are:
    - Class 41a, 100% Declining Balance (DB); applies to new mines.
    - Class 41b, 30% DB, most ongoing capital costs.

    - Less: Canadian Exploration Expenses (CEE), 100% DB; includes most pre-production exploration expenses plus waste stripping and mine excavations.
    - Less: Canadian Development Expense (CDE), 30% DB; resource acquisition costs as well as sinking mine shafts and major underground haulageways after coming into production.
    - Less: Interest Expense.
    - Equals Net Taxable Income.
    - Federal Corporate Tax is charged at 18% of Net Taxable Income.
    - Note that losses can currently be carried back three years and forward 20 years.
- Ontario Corporate Taxes are calculated on the same basis as Federal Corporate Taxes except:
    - There is a Ontario Resource Allowance Tax Credit equal to 25% of Net Corporate Tax.
    - The Ontario Corporate Tax Rate is 10% for mining operations.
- Ontario Mining Taxes are calculated as:
    - EBITDA.
    - Plus: Royalties payable to other stakeholders (except government royalties).
    - Less: Depreciation charged on New Mining Assets calculated on a Straight Line (SL) basis at 100%.
- Less: Depreciation on Ongoing Mining Assets calculated on a SL basis at 30%.
    - Less: Depreciation on Processing and Transportation Assets calculated on a SL basis at 15%.
    - Less: Depreciation Exploration and Development Expenses calculated on a DB basis at 100%.
    - Less: A Processing Allowance (PA) of 8% of processing and refining assets purchased and installed to date. The minimum PA is 15% of net income at this point with a maximum of 65% of net income at this point.
    - The first $10 million of net income at this point is tax free during the first three years of production.
    - The taxation rate is 10% of any net profits that exceed $500,000.
    - No deduction is allowed for interest expense or royalties paid to third parties.
    - Ontario Mining Tax is treated as a royalty rather than a tax as it is applied to the mine itself.

1.8.2 Sensitivity

Howe tested the sensitivity of the Goliath Project IRR to changes in metal prices, operating costs and capital costs. Metal prices and costs were varied up and down by 30%. As would be expected the IRR is more sensitive to changes in metal prices. The changes in operating and capital costs have approximately the same effect on the IRR. For instance, a drop in metal prices of 30%, leads to a post-tax IRR of 1.8% while an increase in metal prices of 30% raises the post-tax IRR to 54.9%. Similarly, an increase in operating costs of 30% drop in the post-tax IRR to 19.6% and a decrease in the operating costs of 30% raises the post-tax IRR to 43.6%.





1.9 CONCLUSIONS AND RECOMMENDATIONS

Howe's economic modelling and analysis of the Goliath Project reveals the Project could yield a post-tax IRR of 32.4% and a post-tax NPV, discounted at 7.5%, of C$109.9 million. In Howe's opinion the Goliath Project is a potentially very robust one and warrants Treasury's continued advancement of the Project towards an eventual pre-feasibility study.

To proceed with the assessment of the potential development of the Project, Howe recommends surface and underground bulk sampling, and pilot plant testing be undertaken.

For surface work, a portion of the Main Zone would be stripped-off. Geological mapping and sampling would be carried out. A bulk sample of at least 5,000 tonnes would be taken. The sample would be split down to 50-100 tonnes then shipped to a pilot plant laboratory facility.

For underground work, the existing exploration portal, decline, and underground workings could be rehabilitated and used as a starting point from which the B and C-Zones would eventually be accessed for bulk sampling purposes. As with the surface sample, this would be split down to 50-100 tonnes then shipped to a pilot plant laboratory facility.

In addition to the bulk samples, the lateral development and raising needed to collect the samples, plus any test stoping that would be carried out as well, would allow mining and processing parameters to be determined to a preliminary feasibility study level of accuracy (+/- 15-20%). Should the preliminary feasibility study yield positive results, mineral reserves can be identified for the Project.

The grand total budgetary cost for this work is estimated to be in the order of C$3.2 million.

TABLE of CONTENTS

LIST OF TABLES

LIST OF FIGURES

LIST OF APPENDICES

2 INTRODUCTION

2.1 GENERAL

This technical report ("Report") was prepared by A.C.A. Howe International Limited ("Howe") at the request of Mr. Martin Walter, MBA, B.Sc. (Geology), President & CEO of Treasury Metals Inc. ("Treasury"). This Report is specific to the standards dictated by National Instrument 43-101, companion policy NI43-101CP and Form 43-101F (Standards of Disclosure for Mineral Projects) in respect to the Goliath Gold Project (the "Goliath Project" or "Project") and focuses on Howe's preliminary economic analysis for a potential mining operation.

The Goliath Project, located in north-western Ontario, lies about 20 kilometres east of the City of Dryden, 125 kilometres east of the City of Kenora, and 325 kilometres northwest of the port City of Thunder Bay, in the Kenora Mining Division, Ontario, Canada.

Treasury is a mineral exploration company incorporated in the province of Ontario, Canada, and is listed on the Toronto Stock Exchange (TSX) under the symbol "TML". Treasury was originally a subsidiary of Laramide Resources Ltd. (Laramide) and became listed as a public company on the TSX as of August 19th, 2008. It is focused on the acquisition and development of precious metal assets in Canada, with a focus on gold. The corporate head office is located at 130 King Street West, Suite 3680, Toronto, Ontario, Canada, M5X 1B1. Treasury's Goliath Project field office is located at 899 Tree Nursery Rd., Wabigoon, Ontario, approximately 4 kilometres south of the deposit area.

Howe is an international geological and mining consulting firm that was incorporated in the province of Ontario in 1966 and has continuously operated under a "Certificate of Authorization" to practice as Professional Engineers (Ontario) since 1970 and Professional Geoscientists (Ontario) since 2006. Howe provides a wide range of geological and mining consulting services to the international mining industry, including geological evaluation and valuation reports on mineral properties. The firm's services are provided through offices in Toronto and Halifax, Canada and London, U.K.

Neither Howe nor the authors of this Report (nor family members or associates) have a business relationship with Treasury or any associated company, nor with any company mentioned in this Report that is likely to materially influence the impartiality or create a perception that the credibility of this Report could be compromised or biased in any way. The views expressed herein are genuinely held and deemed independent of Treasury.

Moreover, neither Howe nor the authors of this Report (nor family members or associates) have any financial interest in the outcome of any transaction involving the property considered in this Report other than the payment of normal professional fees for the work undertaken in the preparation of this Report (which is based upon hourly charge-out rates and reimbursement of expenses). The payment of such fees is not dependent upon the content or conclusions of either this Report or consequences of any proposed transaction.

2.2 TERMS OF REFERENCE

This Report was prepared on behalf of Treasury for the purpose of exploring the scoping-level economics (i.e.: a preliminary economic analysis) for a potential mining operation that would exploit the Goliath gold deposit, with recommendations to allow Treasury and current or potential partners to reach informed decisions. This Report was prepared by Messrs. William Douglas Roy, M.A.Sc, P.Eng. Associate Mining Engineer with Howe; Ian D. Trinder, M.Sc., P.Geo, Senior Geologist with Howe; Bruce Brady, B.Eng., P.Eng, Associate Mining Engineer with Howe; Gordon Watts, B.A.Sc., P.Eng, Senior Associate Mineral Economist with Howe; and Mr. Alfred S. Hayden, B.A.Sc., P.Eng, Associate Metallurgical Engineer and president of EHA Engineering Ltd.

Mr. Roy is a mining engineer with fifteen years experience in the mining industry. He has participated in numerous resource estimates and feasibility studies for precious metals and base metals projects and has authored or co-authored numerous OSC-2A and NI 43-101-compliant reports. Mr. Trinder has over 25 years experience in the mining industry with a background in international precious and base metals mineral exploration including project evaluation and management. Mr. Brady has 20 years of experience in operations, engineering, and management in underground and open pit mining operations in Africa and Canada, and 20 years of consulting experience. Mr. Watts has over 42 years experience in mining exploration, mine operation, mine engineering, project evaluation, feasibility studies and financial evaluation. Mr. Hayden has been a consultant metallurgical engineer for A.C.A. Howe International Limited since 2002. Mr. Hayden is a metallurgical engineer with over 45 years of experience in operations, engineering, and management in underground and open pit operations in Africa and Canada as well as consulting work.

Mr. Trinder visited the Project during the period September 14th to 16th, 2008, as part of due diligence in the preparation of Howe's 2008 technical report. During the property visit, Mr. Trinder met with Dr. Scott Jobin-Bevans, then president of Treasury, and Caracle Creek International Consulting Inc. (CCIC) field personnel Mr. R. Krocker, Ms. A. Tremblay and Mr. T. Loney, to examine the project area and discuss Treasury's exploration activities, methodologies, findings and interpretations. Mr. Trinder conducted a review of available data at Treasury's field office in Dryden, Ontario, and an inspection of surface outcrops and workings at several areas of the Project area, including a recent trench. Selected drill core from Treasury's drill holes was examined at its secure core logging and storage facility in Dryden. Prior to the site visit, Mr. Trinder reviewed the Company's most recent work, compilation reports and data as well as historical information.

Mr. Roy subsequently visited the Project during the period November 25th to 27th, 2011, as part of due diligence in the preparation for a technical report and resource estimate update (Roy and Trinder, 2011). During the property visit, Mr. Roy met with Treasury representatives, Rory Krocker and Ash Martin, to examine the project area and discuss Treasury's exploration activities, methodologies, findings and interpretations. Mr. Roy conducted a review of available data at Treasury's field office in Dryden, Ontario, and an inspection of several areas of the Project. Selected drill core from Treasury's drill holes was examined at its secure core logging and storage facility in Dryden.

Treasury has accepted that the qualifications, expertise, experience, competence and professional reputation of Howe's Principals and Associate Geologists and Engineers are appropriate and relevant for the preparation of this Report. Treasury has also accepted that Howe's Principals and Associates are members of professional bodies that are appropriate and relevant for the preparation of this Report.

Treasury has warranted that full disclosure of all material information in its possession or control at the time of writing has been made to Howe, and that it is complete, accurate, true and not misleading. Treasury has also provided Howe with an indemnity in relation to the information provided by it, since Howe has relied on Treasury's information while preparing this Report. Treasury has agreed that neither it nor its associates or affiliates will make any claim against Howe to recover any loss or damage suffered as a result of Howe's reliance upon that information in the preparation of this Report. Treasury has also indemnified Howe against any claim arising out of the assignment to prepare this Report, except where the claim arises out of any proven wilful misconduct or negligence on the part of Howe. This indemnity is also applied to any consequential extension of work through queries, questions, public hearings or additional work required arising out of the engagement.

Previously, during September-November 2008, Howe completed a resource estimate for Treasury's Thunder Lake Deposit using historical third party drilling and Treasury drilling current to drill hole TL0845. As part of a Preliminary Economic Assessment, Treasury then commissioned Howe to update the resource estimate utilizing additional data from drilling that was carried out during 2008 and 2009. That resource estimate update, completed during March-May 2010 and released in July 2010 was based on information known to Howe as of January 26, 2010 and included assay data for 293 historic Teck and Corona Gold Corp. ("Corona") diamond drill holes and 86 Treasury diamond drill holes (TL0801 to TL0986) completed in 2008 and 2009.

Howe "locked" the resource database on October 27, 2011 to initiate the resource estimate update that was carried out during Fall, 2011 (Roy and Trinder, 2011). That report was based on assay data available to Howe as of that date and included the previous data and an additional 144 new Treasury drill holes totalling approximately 60,000 metres. This took into account two in-fill focused drilling programs: approximately 10,000 metres completed in 2010 (TL1086 to 10118) and approximately 50,000 metres in 2011 (TL11119 to 11229). The assay results of one hole, TL11229, were not available as of October 27. This hole was collared northeast of the current resource area and therefore would not have been included in the resource estimate update.

Treasury has subsequently completed additional drilling which is not included in the 2011 resource estimate update. From January 25 to June 6, 2012, Treasury Metals completed 48 diamond drill holes (TL12230 to TL12277) totaling 15,635 metres in two phases. Additionally, 5 historical Teck Resources Inc. diamond drill holes were re-entered and extended for a total of 473 metres.

Historical mineral resources figures contained in the Report, including any underlying assumptions, parameters and classifications, are quoted "as is" from the source. Howe confirms that its estimated resource is in compliance with National Instrument 43-101, companion policy

NI 43-101CP and Form 43-101F (Standards of Disclosure for Mineral Projects) and the definitions and guidelines of the CIM Standards on Mineral Resources and Reserves.

The authors believe that the data presented by Treasury are a reasonable and accurate representation of the Goliath Project.

The effective date of this report is July 19, 2012.

Only the target areas within the Project area and those visited by Howe are discussed in any detail in this Report. Howe reserves the right, but will not be obligated to revise this Report and conclusions if additional information becomes known to Howe subsequent to the date of this Report.

Treasury reviewed draft copies of this Report for factual errors. Any changes made as a result of these reviews did not include alterations to the conclusions made. Therefore the statement and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Report.

2.3 SOURCES OF INFORMATION

The information, conclusions and recommendations contained herein are based on a review of the diamond drill hole database, other digital and hard copy data, geological reports, maps, miscellaneous technical papers, company letters, memoranda and other information made available by Treasury, discussions with representatives and consultants of Treasury who are familiar with the Project and the area in general as well as various published geological reports and other public and private information as listed in Section 27 of this Report. Howe has assumed that all of the information and technical documents reviewed are accurate and complete in all material aspects.

Howe has only reviewed the land tenure in a preliminary fashion, and has not independently verified the legal status or ownership of the property or the underlying agreements.

In addition, Howe carried out discussions with the management, consultants, and technical personnel of Treasury. Howe's extensive experience in Archaean lode gold and volcanogenic massive sulphide deposits was also drawn upon.

2.4 UNITS AND CURRENCY

The Metric System or SI System is the primary system of measure and length used in this Report and is generally expressed in kilometres, metres and centimetres; volume is expressed as cubic metres, mass expressed as metric tonnes, area as hectares, and zinc, copper and lead grades as percent or parts per million. The precious metal grades are generally expressed as grams/tonne but may also be in parts per billion or parts per million. Conversions from the SI or Metric System to the Imperial System are provided below and quoted where practical. Many of the geologic publications and more recent work assessment files now use the SI system but older work assessment files almost exclusively refer to the Imperial System.

Metals and minerals acronyms in this report conform to mineral industry accepted usage and the reader is directed to an online source at www.maden.hacettepe.edu.tr/dmmrt/index.html.

Conversion factors utilized in this report include:

- 1 troy ounce/ton (short ton) = 34.2857 grams/tonne (metric tonne)
- 1 gram/tonne = 0.0292 troy ounces/ton
- 1 troy ounce = 31.1035 grams
- 1 gram = 0.0322 troy ounces
- 1 pound = 0.4536 kilograms
- 1 foot = 0.3048 metres
- 1 mile = 1.609 kilometres
- 1 acre = 0.4047 hectares
- 1 square mile = 2.590 square kilometres

The term gram/tonne or g/t is expressed as “gram per tonne” where 1 gram/tonne = 1 ppm (part per million) = 1,000 ppb (part per billion).

Other abbreviations include ppb = parts per billion; ppm = parts per million; oz/t or opt = ounce per short ton; Moz = million ounces; Mt = million tonne; t = tonne (1,000 kilograms); SG = specific gravity; lb/t = pound/ton; and, st = short ton (2,000 pounds).

Dollars are expressed in Canadian currency (C$) unless otherwise noted. Zinc, copper, and lead prices are stated as US$ per tonne (US$/t) whereas gold and silver prices are stated in US$ per troy ounce (US$/oz).

Unless otherwise noted, Universal Transverse Mercator (“UTM”) coordinates are Zone 15 North, NAD83 Datum.

3 RELIANCE ON OTHER EXPERTS

Howe has relied on information provided by Treasury regarding land tenure, underlying agreements and technical information not in the public domain. While Howe has not independently verified the legal status or ownership of the property or any of the underlying agreements, all of the information appears to be of sound quality. Howe has also reviewed the mineral dispositions as posted on the MNDMF website:

(www.mndmf.gov.on.ca/mines/claimaps_e.asp).

4 PROPERTY LOCATION AND DESCRIPTION

4.1 LOCATION

The Goliath Project is located in the Kenora Mining Division in north-western Ontario, 20 kilometres east of the City of Dryden, 125 kilometres east of the City of Kenora, and 325 kilometres northwest of the port City of Thunder Bay (Figure 4-1 and Figure 4-2). The area is covered by National Topographic System ("NTS") map sheets 52F/09, 10, 15 and 16 and straddles Zealand and Hartman townships. The Property is centred at approximately UTM 532441mE and 5511624mN (NAD83 Zone 15N; 49°45'22" N, 92°32'58" W).

4.2 DESCRIPTION AND OWNERSHIP

The Goliath Project consists of 137 contiguous unpatented mining claims (254 units – 4,064 hectares), 19 patented land parcels (approximately 817 hectares) as detailed in Appendix A. The total area of the claim group is approximately 4,881 hectares (approximately 49 km^2) covering portions of Hartman and Zealand townships east of the City of Dryden, Kenora Mining Division (Figure 4-3). All claims are currently active and in good standing with Ontario's MNDMF.

The Goliath Project comprises two historic properties that are now consolidated: the larger Thunder Lake Property, purchased from Teck and Corona and the Laramide Property. The land acquisition agreements are described in Section 4.2. The Goliath Project has been expanded from its original size through:

- Additional staking and acquisition of 21 unpatented mining claims (131 units – 2,096 hectares);
- An option agreement pursuant to which Treasury has the right to acquire a 100% interest in the mining rights (only) of certain patented lands (the Brisson Property – 40.8711 hectares) located immediately west and contiguous to the Goliath Project;
- The acquisition of a 100% interest in the surface and mineral rights of a parcel of private land (MNR Tree Nursery) totalling 117.492 Ha.
- In addition, in 2011, Treasury made final payment on an option to purchase a 16 Ha surface rights only patent within the Project area (LeClerc - Parcel 34303).

The Project is held 100% by the Company, subject to certain underlying royalties and payment obligations on 13 of the 19 patented land parcels, totalling approximately $103,500 per year and an option on one patented land parcel to earn in 100% as described for the Brisson Mineral Property (Section 4.2.2.2 and Table 4-1).

Table 4-1. Option and royalty obligations, patented land parcels, Goliath Project.

PARTY	PARCEL	ADVANCE ROYALTY (per year)	DUE	OPTION (per year)	NSR (%
Lundmark[1]	41941	C$50,000 *	January 1st	-	2.0
Collins[1]	17395	-		-	2.0
Sheridan[1]	21374	-		-	1.0
Johnson[1]	15401	-		-	2.0
Hudak[1]	21609	US$3,500 *	January 1st	-	2.0
Fraser[1]	15395	C$50,000	January 1st	-	2.0
Delk[2]	24724	-		-	2.5
Davenport[2]	19088	-		-	2.0
Jones[3]	41215	-		-	2.5
Nemeth[2]	6556	-		-	2.0
Sterling[4]	4822	-		-	2.0
Medlee[4]	21553	-		-	2.5
Schultz[4]	13492	-		-	2.0
Brisson				$45,000***	
	TOTAL C$:	**$100,000**		**$45,000**	
	TOTAL US$:	**$3,500**			

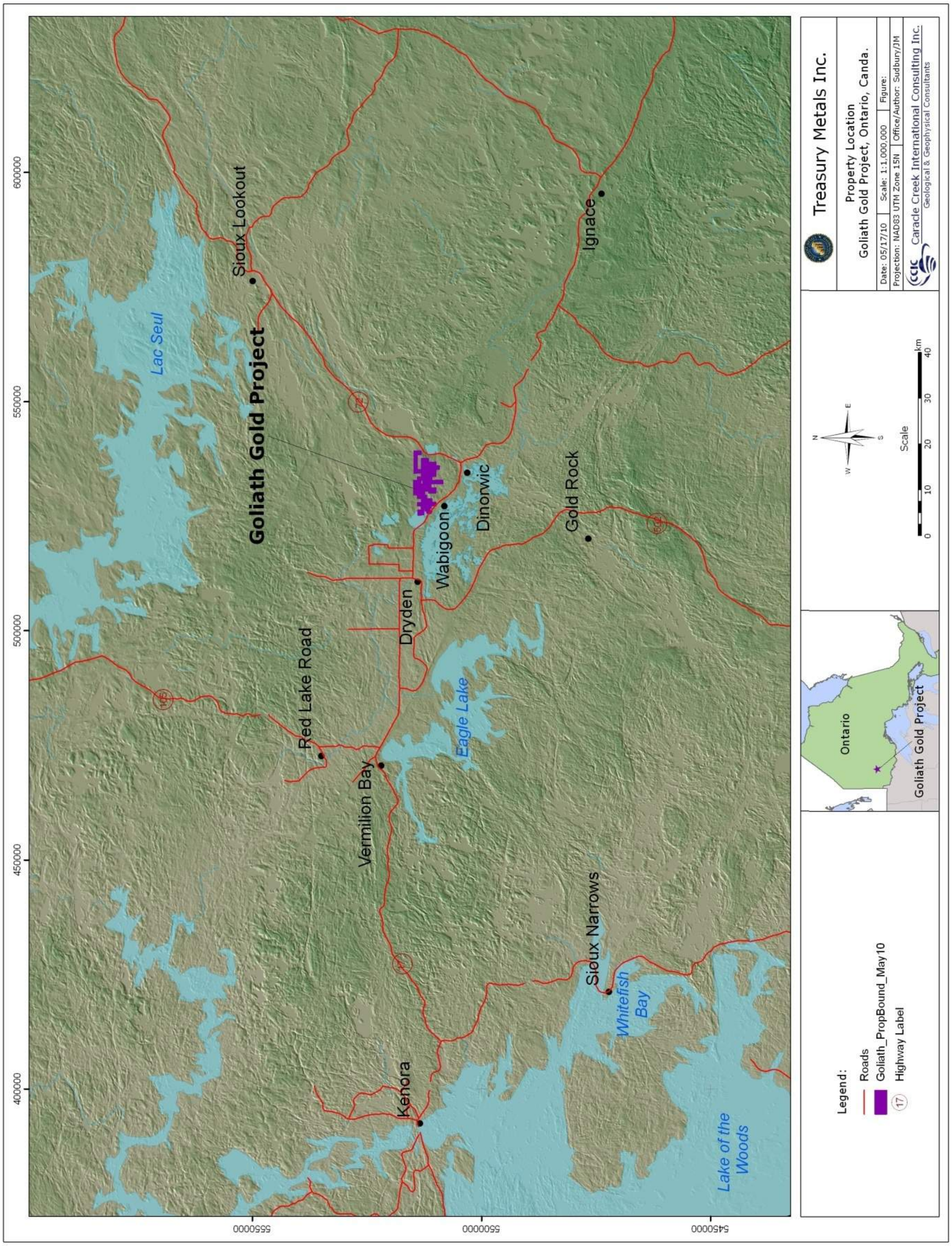
[1]Thunder Lake West; [2]Thunder Lake East; [3]Jones Property, [4]Laramide Property
*subject to withholding tax;
*** Option payments vary according to anniversary – See Table 4-3

The Project is bound by two provincial parks: Lola Lake Provincial Reserve located at the northern boundary; and, Aaron Provincial Park at the western boundary on the south shore of Thunder Lake (Figure 4-3). Lola Lake was designated a nature reserve class park in 1985, whereas Aaron is a serviced recreation-class park, operated in co-operation with the City of Dryden.

Treasury warrants that it possesses all permits required to execute exploration activities it has undertaken to date on the property. Treasury is conducting ongoing community consultations including discussions with the local First Nation communities. The effect of these discussions on future access, title or the right or ability to perform the work recommended in this report on the Project area is not known at this time.

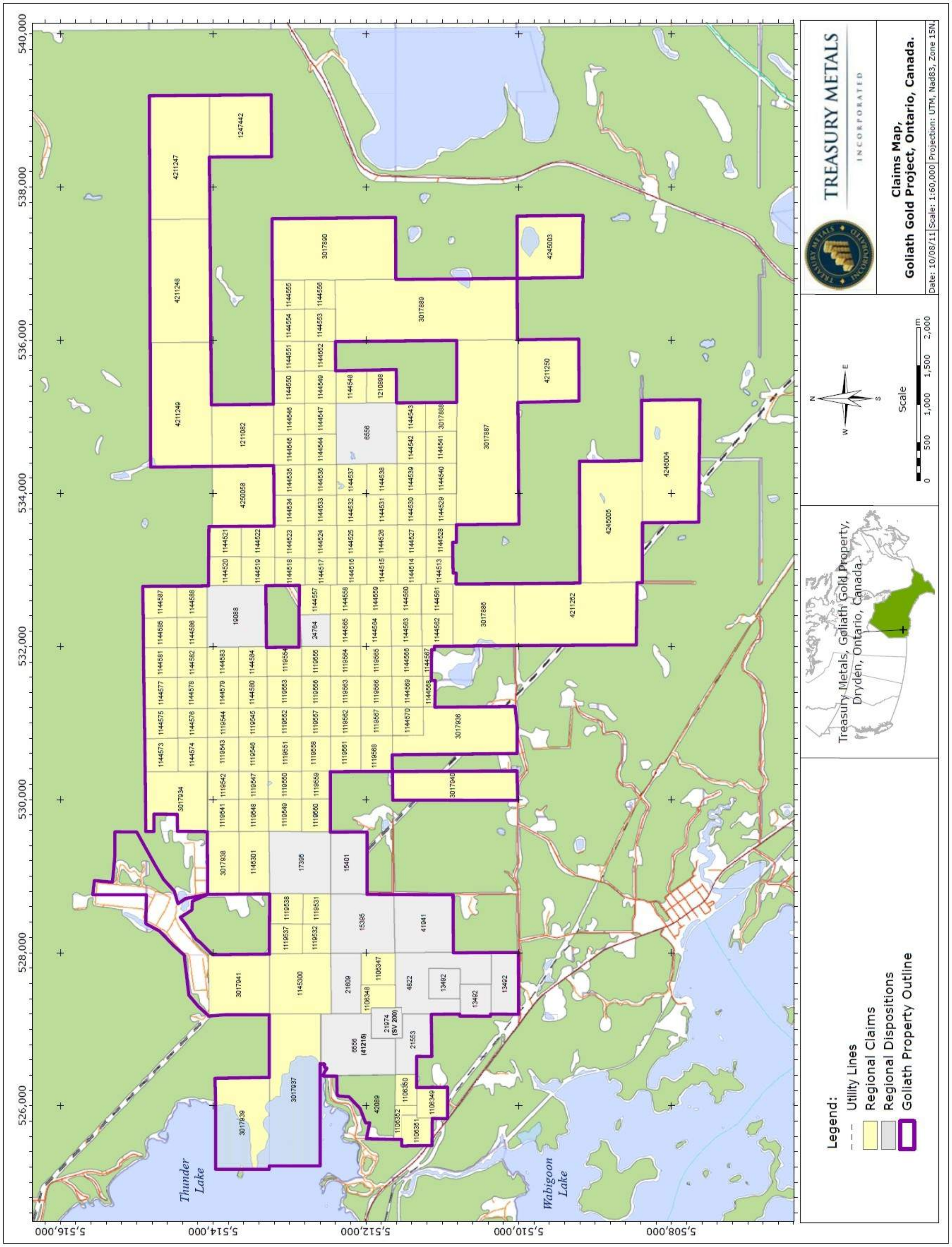
Source: CCIC, 2008

Figure 4-1. Location of the Goliath Project, northwestern Ontario.



Source: CCIC 2010

Figure 4-2. Location of the Goliath Project (red), northwestern Ontario.



Source: Treasury 2011

Figure 4-3. Land tenure of the Goliath Project.

4.2.1 Property Purchase Transaction

Treasury Metals Inc., a subsidiary of Laramide Resources Ltd. ("Laramide"), was "spun-out" of Laramide as a dividend to Laramide's shareholders and to hold its non-uranium assets. Treasury was listed and began trading on the TSX exchange on August 19th, 2008 under the trade symbol "TML".

4.2.1.1 Thunder Lake Property

As announced in April 2007 (Laramide Press Release: April 3, 2007), Laramide closed its purchase transaction of the Thunder Lake Property as of October 2007 (Laramide Press Release: October 4, 2007). Laramide purchased, through its wholly owned subsidiary, Divine Lake Exploration Corp. (now "Treasury Metals Inc."), 100% of Corona's (82%) and Teck's (18%) respective interests in the Thunder Lake Property. On closing, Corona received from Laramide cash consideration of $5,000,000 and under the terms of the agreement Corona received from Laramide aggregate cash payments of $10,000,000 and a 10% interest in Treasury after it became a public company. Teck received cash consideration of approximately $1,137,299 at closing and received from Laramide aggregate cash payment of $2,274,598 and a 2.27% interest in Treasury. The balance of consideration for the Properties was payable as follows:

- Cash payment of $6,137,229 sixty (60) days after the closing date;
- Cash payment of $6,137,229 one hundred and twenty (120) days after the closing date;
- 12.27% of the common shares of Treasury issued and outstanding on completion of a transaction pursuant to which Treasury becomes a public company.

Treasury announced in an August 26, 2008 press release that it had completed the final instalment of the purchase price to Corona and Teck-Cominco pursuant to the purchase agreement. In accordance with the 2007 Purchase Agreement, Corona and Teck shall receive, for no additional consideration, that number of common shares sufficient for each of Corona and Teck to maintain their respective percentage interest in the Company of 10% and 2.27% until such time as the Company receives aggregate proceeds from the insurance of common shares of $7.5 million. This threshold has been reached. Laramide and Treasury have met all of the obligations to Teck and Corona.

4.2.1.2 Laramide Property

As part of the of the spin-out of Treasury, Laramide transferred to Treasury its Goliath Property (herein referred to as the Laramide Property) and certain of Laramide's other non-uranium assets. As of May 2010, Laramide held approximately 13.7% of the issued and outstanding Treasury common shares. Treasury owns the Laramide Property 100% subject to royalties as detailed in Table 4-1.

4.2.2 2009 Property Expansion

In 2009 the Goliath Project was expanded from its original size through the combined staking and acquisition of 18 unpatented mining claims and the signing of an option agreement pursuant to which Treasury has the right to acquire a 100% interest in the mining rights (only) of certain patented lands (the Brisson Property) contiguous to the Goliath Gold Project.

4.2.2.1 Unpatented Mining Claims

In 2009 the Company acquired and/or staked 18 additional unpatented mining claims (111 units) totalling 1,776 hectares as detailed in Table 4-2

Table 4-2. Unpatented Mining Claims added to Goliath Project in 2009.

Township/Area	Claim Number	Claim Recording Date	Claim Due Date	Claim Units	Area (ha)	Status
HARTMAN	1247442	2007-Aug-21	2010-Aug-21	4	64	A
HARTMAN	3017886	2009-Jul-10	2011-Jul-10	4	64	A
HARTMAN	3017887	2009-Jul-10	2011-Jul-10	12	192	A
HARTMAN	3017888	2009-Jul-10	2011-Jul-10	1	16	A
HARTMAN	3017889	2009-Jul-10	2011-Jul-10	12	192	A
HARTMAN	3017890	2009-Jul-10	2011-Jul-10	8	128	A
HARTMAN	4211247	2007-Aug-21	2010-Aug-21	8	128	A
HARTMAN	4211248	2007-Aug-21	2010-Aug-21	8	128	A
HARTMAN	4211249	2007-Aug-21	2010-Aug-21	8	128	A
HARTMAN	4211250	2007-Aug-21	2010-Aug-21	4	64	A
ZEALAND	3017934	2008-May-21	2010-May-21	4	64	A
ZEALAND	3017936	2008-May-21	2010-May-21	5	80	A
ZEALAND	3017937	2008-May-21	2010-May-21	9	144	A
ZEALAND	3017938	2008-May-26	2010-May-26	2	32	A
ZEALAND	3017939	2008-Jul-04	2010-Jul-04	6	96	A
ZEALAND	3017940	2008-Sep-10	2010-Sep-10	4	64	A
ZEALAND	3017941	2008-Oct-10	2010-Oct-10	4	64	A
ZEALAND	4211252	2007-Sep-06	2010-Sep-06	8	128	A
TOTAL:	**18**			**111**	**1,776**	

4.2.2.2 Brisson Property

On December 11, 2009 the Company entered into an option agreement to acquire a 100% interest in the mining rights (only) of certain patented lands (40.8711 Ha) from Edward Henry Brisson (the Brisson Property) located immediately west and contiguous to the Goliath Gold Project. Under the terms of the agreement, the Company is to make option payments totalling $100,000 and issue common shares of the Company equal to $100,000 based on the market price of the date issue as outlined in Table 4-3.

Table 4-3. Brisson Property Option Payments.

		Cash Payment	Common Share Delivery
1	On or before Effective Date (Dec. 11, 2009)	$25,000	A number of common shares in the capital of the Optionee equal to the quotient obtained by dividing $25,000 by the Market Price on the date the cash payment (1) is made.
2	On or before 1st anniversary of Effective Date (Dec. 11, 2010)	$20,000	A number of common shares in the capital of the Optionee equal to the quotient obtained by dividing $25,000 by the Market Price on the date the cash payment (2) is made.
3	On or before 2nd anniversary of Effective Date (Dec.11, 2011)	$20,000	A number of common shares in the capital of the Optionee equal to the quotient obtained by dividing $25,000 by the Market Price on the date the cash payment (3) is made.
4	On or before 3rd anniversary of Effective Date (Dec. 11, 2012)	$35,000	A number of common shares in the capital of the Optionee equal to the quotient obtained by dividing $25,000 by the Market Price on the date the cash payment (4) is made.

As at December 31, 2010, the Company had paid $45,000 and issued common shares of the Company with a market value of $50,000.

4.2.3 2010-2011 Property Expansion

In 2010 and 2011 the Goliath Project was further expanded through the staking of 3 unpatented mining claims; the final option payment and acquisition of a 100% interest in the surface rights (only) patent of LeClerc (Parcel 34303, 16.59 ha) and; the acquisition of a 100% interest in the surface and mineral rights of the historic Dryden Tree Nursery (101 ha).

4.2.3.1 Unpatented Mining Claims

In 2011 the Company staked 3 additional unpatented mining claims (20 units) totalling 320 hectares as detailed in Table 4-2

Table 4-4. Unpatented Mining Claims added to Goliath Project in 2011.

Township/Area	Claim Number	Claim Recording Date	Claim Due Date	Claim Units	Area (ha)	Status
HARTMAN	4245003	2011-Feb-28	2013-Feb-28	4	64	A
HARTMAN	4245004	2011-Feb-28	2013-Feb-28	8	128	A
HARTMAN	4245005	2011-Feb-28	2013-Feb-28	8	128	A
TOTAL:	**3**			**20**	**320**	

4.2.3.2 Dryden Tree Nursery Area

On November 5, 2010 the Company acquired a 100% interest in the surface and mineral rights of certain private lands (117.492 Ha) formerly known as the Dryden Tree Nursery located immediately northwest and contiguous to the Goliath Gold Project.

4.2.3.3 Additional Surface Rights

On April 12, 2011 the Company completed the final payment on the option to purchase the LeClerc surface rights (only) patent (Parcel 34303, 16.59 ha) located immediately east of the Thunder Lake Deposit within the Goliath Gold Project area.

4.3 ESTABLISHING MINERAL RIGHTS IN ONTARIO

In Ontario, Crown lands are available to licensed prospectors for the purposes of mineral exploration. A licensed prospector must first stake an unpatented mining claim to gain the exclusive right to prospect on Crown land. Claim staking is governed by the Ontario Mining Act and is administered through the Provincial Mining Recorder and Mining Lands offices of the MNDMF.

An unpatented mining claim is a square or rectangular area of open Crown land or Crown mineral rights that a licensed prospector marks out with a series of claim posts and blazed lines. Mining claims can be staked either in a single unit or in a block consisting of several single units. In un-surveyed territory, a single unit claim is laid out to form a 16 hectare (40 acre) square with boundary lines running 400 metres (1,320 feet) astronomic north, south, east and west. Multiples of single units, up to a maximum of 16 units (256 hectares), may be staked with only a perimeter boundary as one block claim but must be staked in a square or rectangular configuration.

Upon completion of staking, and within 31 days of the completion date, a recording application form is filed with payment to the Provincial Recording Office. Staking completion time takes priority, meaning that if two licensees file applications to record the staking of all, or part of the same lands, then the applicant with the earliest completion time will have priority. Where the time limited for any proceeding or for the completion of said proceeding in an office of a mining recorder or an office of the Commissioner or an office of the Minister or Deputy Minister expires or falls upon a day on which the relevant office is closed, the time so limited extends to and the recording may be done on the day next following the day on which the relevant office was closed. All claims are liable for inspection at any time by the Ministry and may be cancelled for irregularities or fraud in the staking process. Disputes of mining claims by third parties will not be accepted after 1 year of the recording date or after the first unit of assessment work has been filed and approved.

A claim remains valid as long as the claim holder properly completes and files the assessment work as required by the Mining Act and the Minister approves the assessment work. A claim holder is not required to complete any assessment work within the first year of recording a mining claim. In order to keep an unpatented mining claim current the mining claim holder must perform $400 worth of approved assessment work per mining claim unit, per year; immediately following the initial staking date, the claim holder has two (2) years to file one year worth of assessment work. Claims are forfeited if the assessment work is not done.

A claimholder may prospect or carry out mineral exploration on the land under the claim. However, the land covered by these claims must be converted to leases before any development work or mining can be performed. Mining leases are issued for twenty-one year terms and may be renewed for further 21-year periods. Leases can be issued for surface and mining rights, mining rights only or surface rights only. Once issued, the lessee pays an annual rent to the province. Furthermore, prior to bringing a mine into production, the lessee must comply with all applicable federal and provincial legislation.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Goliath Project is located 20 kilometres east of the City of Dryden and is accessible from the Trans-Canada Highway 17 and various secondary roads that extend north of the highway from the village of Wabigoon including Tree Nursery Road, along the north–south boundary of Zealand and Hartman townships, and Nelson Road which runs east–west between Concession III and Concession IV in Zealand Township (Figure 4-3). Fieldwork can be completed year-round with summer conditions between April and October and winter's freezing conditions between November and March; the latter allowing for improved access for heavy machinery such as diamond drill rigs to wet areas of the Property.

5.2 CLIMATE

The Goliath Project lies in a region that experiences typical northern Canadian climate conditions. Annual temperatures range from 27°C to -26°C with an average rainfall between 60 and 80 centimetres and average snowfall between 1.3 and 2.3 metres.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

All major industrial services and supplies are available in Dryden and the Dryden Airport services the region. The local economy is based on the forestry and tourism industry and since amalgamating with the village of Barclay in 1998; Dryden has a population of about 8,195 persons (2006, Statistics Canada). The Domtar pulp and paper mill is the major employer in the area with approximately 250 mill employees and 200 woodland contractors. Dryden's location in northwestern Ontario, on Wabigoon Lake and Wabigoon River also supports an outdoor tourism economy (fishing, snowmobiling, etc.).

The Goliath Project is located about 325 kilometres northwest of the City of Thunder Bay, a major economic centre along the Trans-Canada Highway and port at the northwest head of the St. Lawrence Seaway (Lake Superior). Major and minor hydro transmission lines cross portions of the Project area. Howe has not contacted power authorities to determine if these lines have available capacity to support a mine operation. The Canadian Pacific Railway line is located approximately 2 kilometres to the southwest, parallel to Hwy 17. The Trans-Canada natural gas pipeline crosses portions of the Property. The closest centre of active mining operations is in the Red Lake area, approximately 155 kilometres northwest of the Project, however, northwestern Ontario generally possesses the necessary labour and infrastructure to support new exploration and mining operations.

At this time it appears that Treasury holds sufficient surface rights necessary for any potential future mining operations including tailings storage areas, waste disposal areas and a processing plant.

5.4 PHYSIOGRAPHY

A discontinuous mantle of Quaternary surficial deposits overlies the Archaean bedrock. Three main terrain types dominate the landscape: rolling glaciolacustrine plains composed of varved clay and bedrock knobs; rolling rocky uplands of bedrock which may be bare or thinly covered with patches of till and/or varved clay; and complex, moraine-like features commonly capped with beach sand and gravel. Extensive plains of glaciofluvial outwash make up almost 70% of the overburden (as sandy glacial till) overlying the Goliath Project area. Alluvial terrain is mainly organic and accounts for the abundance of peat and swampy areas in the low-lying poorly drained areas of Hartman Township (Roed, 1980).

Maximum relief is about 30 to 40 metres and occurs in the area of Lot 3 Concession IV of Zealand Township. Swamps and a lack of outcrops characterize Hartman Township and Zealand Township is well wooded with second growth poplar and fir trees and areas of shallow swamps.

6 HISTORY

There is very limited documentation of exploration activity conducted in the Project area prior to 1989 (assessment files, government mapping, etc.). Previous exploration in the area was either regional in nature or focused mainly on the western portion of the Thunder Lake Property. Historic exploration targeted zinc in 1956 (G.L. Pidgeon); iron in 1956-57 and 1966-68 (Compton-Wabigoon and Algoma Steel); base metals in 1971 (INCO); and, gold in 1983 (Jalna Resources) (Ontario Geological Survey, 1991). None of these previous exploration programs identified the mineralization now known as the Thunder Lake Deposit, discovered by Teck Exploration Ltd. (now Teck Resources Ltd. or "Teck") geologists in 1989.

6.1 THUNDER LAKE PROPERTY

Land acquisition, field surveys, drilling and underground bulk sampling were completed by Teck and its various partners between late 1989 and 1998; exploration at the Thunder Lake Property was put on hold in 1999. Total diamond drilling on the Thunder Lake Property from 1990 to 1998 amounted to approximately 78,461.20 metres in 293 drill holes. Expenditures during the period 1994 to 1999 (Teck-Corona joint venture) amounted to $11.3 million at Thunder Lake West and $1.2 million at Thunder Lake East ($12.5 million total expenditures). The exploration history of the Thunder Lake Property is described in several reports to Teck (e.g., Page, 1994 and 1995; Page, Waqué, and Galway 1999; Stewart, 1996; Stewart et al., 1997).

6.1.1 1989-1993: Teck Exploration Ltd. (now Teck Resources Ltd.)

Reconnaissance investigation by Teck Exploration Ltd. geologists in 1989 identified a poorly exposed, broad area of weak mineralization and anomalous gold extending through parts of Lots 3 through 8 of Concession IV of Zealand Township. The reconnaissance was part of the "Quest Project" (Stewart, 1996), a generative program designed to identify Hemlo-type mineralization and led by Richard Page.

At this time, the Thunder Lake Property consisted of the Thunder Lake East and Thunder Lake West properties. From 1989 to 1993, exploration over the Thunder Lake West property included line-cutting, geological mapping, geophysical surveys, outcrop stripping and sampling, and diamond drilling of 44 holes totalling 11,100 metres (Page, 1995). The original exploration grid baseline on the Teck Thunder Lake East and West properties was along Nelson Road, which runs east to west, along the border of Concessions III and IV (the boundary between the Laramide Property and the Thunder Lake Property). The baseline locator for L0+00 was located on the southeast corner of Lot 6, Concession III in Zealand Township (Hogg, 2002; Sills, 2007).

In 1990-91, Teck completed stripping and diamond drilling, concentrating in Lots 6 through 8 of Concession IV, Zealand Township. At this time, the general configuration of the West, East, and Main Zones of the Thunder Lake Deposit were established, extending over a strike length of about 1,500 metres. The discovery hole (TL-001) for the Thunder Lake Deposit (Main Zone) was drilled in October, 1990, intersecting multiple horizons of gold mineralization with intersections of 1.5 g/tonne Au over 22.2 metres, 0.9 g/tonne Au over 11.6 metres and 17.5 g/tonne Au over 2.6 metres (Page, 1995).

In 1993, under option by Cameco Corporation, 10 diamond drill holes totalling 1,848.5 metres were completed on the Thunder Lake East portion of the Property (Page, 1993). Although some anomalous gold concentrations were intersected, the results overall were not considered encouraging and subsequent exploration turned to the Thunder Lake West property.

6.1.2 1994-1999: Teck -Corona Gold

The Property was optioned to Corona under the terms of an agreement dated January 3, 1994. Corona met its obligations of the option by July 1996 and a joint venture was formed. Teck was project Operator and the work was largely funded by Corona. At this time, the Thunder Lake East and Thunder Lake West properties became known as the Thunder Lake Property. As of December 31, 1998, Teck owned 18% and Corona owned 82% interest in the Property.

In 1994, a high grade zone (Main Zone) of 1.0 opt Au was partially delineated and appeared to be continuous from surface to a vertical depth of 150 metres depth. A second mineralized zone, lower in grade but thicker than the high-grade area, was partially defined. Drilling for the remainder of 1994 traced the high grade mineralized zone (Main Zone) down plunge with varying continuity to a vertical depth of 525 metres. A zone of strong alteration with anomalous and potentially significant Au concentration was outlined within a 1,300-metre strike length to the east and west.

By 1995, most of the Thunder Lake West and East properties had been gridded, geologically mapped and surveyed with magnetic and VLF-EM geophysics. Drilling during the winter of 1995-1996, eight (8) drill holes (BQ size; 4,142 metres) extended the Main Zone to a vertical depth of 450 metres (Stewart, 1996). In 1996, exploration work consisted of induced polarization geophysical survey and stripping of deep overburden (22 trenches) over portions of the Main Zone and detailed mapping and sampling of the exposed mineralization. At this time, 9,669 metres of drilling was completed, comprising 10 drill holes (NQ size; 6,596 metres), 7 wedges from 3 of the drill holes (434 metres), 20 wedges from 7 previous drill holes (1,156 metres) and the deepening of 9 holes (1,483 metres) (Stewart et al., 1997).

At the Thunder Lake East property, the 1996 exploration program consisted of geological mapping and sampling, and diamond drilling of 21 holes totalling 5,750.2 metres (NQ size). Drilling encountered weakly anomalous gold concentrations over most widths, suggesting some promise for future exploration in the northeast region of the Property (Page et al., 1995).

In 1997, Teck carried out a program of aggressive resource delineation, which delineated the No. 3 Shoot from surface to a 600 metres vertical depth and 50 to 175 metres strike length and the No. 1 Shoot to a depth of 250 metres for a strike length of 50 to 100 metres, with data from 64 diamond drill holes in 21,984 metres (Page and Waqué, 1998).

In 1998, the underground bulk-sampling program was complemented by a drilling program consisting of 64 holes and one wedge totalling 21,984 metres (Page and Waqué, 1998). Also at this time, drilling was carried out in the west and east extensions of the mineralized zone, confirming that the mineralization tapers along strike to the west and with depth: overall gold values and alteration weaken and the extensions are characterized by alternating units of quartz ±

feldspar-porphyry and metasedimentary rocks that contain little alteration or veining (Page, Waqué and Galway, 1999).

6.1.3 Underground Exploration

In 1998, an underground exploration program was initiated to determine the nature and continuity of gold mineralization; to determine the structural control of the high grade shoots by detailed underground mapping; and, to establish the true grade of gold mineralization. A 27 metre long inclined trench, required to provide a 9 metre high face suitable for the portal collar, was subcontracted by J.S. Redpath Limited (North Bay) to Superior Drilling and Blasting. The portal and decline was completed by Redpath; standard 2.4 metre rock bolts with metal screening were the only ground support required in the portal, rock face and adjacent area (Page et al., 1999b).

The decline, at a grade of 15%, was driven north (356°) toward the Main Zone of gold mineralization with the portal located just north of Nelson Road and the north boundary of the Laramide Property (Figure 6-1). The decline was 4.0 metres high by 4.5 metres wide and approximately 275 metres in length, extending past the Main Zone for vehicle turn around and installation of the sump (Page et al., 1999b). The main mineralized zone was intersected at a distance of approximately 250 metres from the opening and at a depth of approximately 35 metres vertical (-38 metres floor elevation).

Ground conditions encountered throughout the ramp were excellent, requiring only standard 1.8 metre mechanical rock bolts on a 1.2 metre by 1.2 metre pattern. Water inflow was minimal in the ramp and also generally throughout the entire underground program (Page et al., 1999b).

Drifting along the Main Zone was controlled by following identifiable (narrow) units of strongly altered schists with weak to strong mineralization. A total of 220 metres of lateral drifting (3.0 metre by 3.0 metre cross section) were completed along the No. 1 Shoot and No. 2 Shoot of the Main Zone (Page et al., 1999b). Lateral development was completed 34 days after drifting was initiated and the entire underground and bulk sample processing program, from initial surface excavations through final closure plan, took 4 months (May 15 to September 15, 1998). The length of the underground workings totalled approximately 496 metres and a total of 23,035 tonnes of rock was excavated (Page et al., 1999b).

Figure 6-1. Portal/decline access to the Thunder Lake Deposit Main Zone gold mineralization for bulk sampling; 1998 Teck Cominco and Corona Gold joint venture (MNDM, 1998).

Results of the underground mapping and sampling included (Page et al., 1999b):

- Recognition of new rock variety (dark coloured intermediate quartz porphyry) spatially associated with silicified and mineralized regions;
- Nine (9) documented occurrences of coarse visible gold/electrum;
- Definition of the Main Zone No. 1 Shoot mineralization, which was found to have limited lateral continuity restricted to a strike length of about 22 metres.

The limited distribution of coarse gold/electrum in the deposit and the limited continuity of mineralization along strike resulted in lower gold grades and reduced tonnage in the re-calculated resource (see Section 6.2).

6.1.3.1 Bulk Sample

In 1998, as part of the underground sampling program, four (4) bulk samples from the Main Zone, totalling 2,375 tonnes and grading >3.0 g/tonne Au, were collected from various areas of the underground workings (Page et al., 1999b). A total of 1,737 tonnes of material was collected from the No. 1 Shoot (A-East and TDB) and 638 tonnes of material from the No. 2 Shoot (B Zone); approximately 0.08% of the material was lost through the initial crushing (Page et al., 1999b). Face sample data indicated that two of the bulk samples were relatively low in grade (3.0 to 6.0 g/tonne Au) while the other two samples were of higher grade (>20 g/tonne Au). The bulk samples were processed through a crushing plant, reduced in volume through a sampling tower to a total of 384 kilograms and the representative sample tower splits were shipped for processing and analysis at Lakefield Research Ltd., Lakefield, Ontario where the samples were further processed and analysed for gold concentration (Page et al., 1999b). In 1999, the remaining material, approximately 2,336 tonnes, was transported to and processed at the Stock Mine mill of St. Andrew Goldfields Ltd., Timmins, Ontario. Further discussion on the bulk sampling is provided in Section 13.

6.1.3.2 Remediation

Environmental permitting, sampling, and monitoring were sub-contracted to NAR Environmental Consultants (Sudbury, Ontario). Baseline water quality and biological surveys were completed in 1997 and sampling was continued in 1998 (Page et al., 1999b). After the program was complete, the area was contoured and reseeded and fully remediated in late 1999 (Figure 6-2).

6.1.4 Historical Drilling

Much of the historic exploration on the Thunder Lake Property centered on diamond drilling programs with the most drilling having been completed in the area north of the Laramide Property (Figure 6-3); there was minimal drilling on the former Thunder Lake East property (Hartman Township). From 1990 to 1998, a total of approximately 78,461.20 metres in 293 drill holes were completed on the entire Thunder Lake Property (Table 6-1; Figure 6-3); this includes all surface, underground and wedge drill holes. Teck geologists supervised the drilling programs and conducted all drill core logging and sampling.

Table 6-1. Summary of historical drilling on the Thunder Lake Property.

Property	Year	No. Drill Holes	Length (m)
Thunder Lake West	1990-1998	248	69,131.10
Thunder Lake East	1993 & 1998	31	7,598.70
Jones Property	1990 & 1998	14	1,731.40
	Total:	**293**	**78,461.20**

Figure 6-2. Site of reclaimed portal/decline and 2008 drill road - looking north (September 15, 2008; viewed from approximately the same position as Figure 6-1).

Table 6-2 summarizes the twenty (20) highest-grade intersections from the historic drilling of Teck and Corona on the Thunder Lake Property. All three mineralized zones are represented in this summary demonstrating that all three zones contain exceptional intercepts.

Table 6-2. Twenty (20) highest-grade intersections from historic Teck drilling (Sills, 2007).

DDH	From (m)	To (m)	Length (m)	Au (g/t)	Au (opt)	Zone
TL-073	25.0	26.5	1.5	17.00	0.50	Main
TL-193	54.5	56.0	1.5	13.36	0.39	Main
TL-114	60.2	61.7	1.5	31.16	0.91	Main
TL-077	64.0	65.5	1.5	45.55	1.33	Main
TL-117	66.7	68.2	1.5	19.08	0.56	West
TL-023	129.3	130.8	1.5	41.17	1.20	West
TL-049	185.0	186.5	1.5	15.40	0.45	Main
TL-029	254.0	255.6	1.6	40.97	1.19	Main
TL-128	402.0	403.5	1.5	21.38	0.62	West
TL-125	421.8	423.3	1.5	126.30	3.68	Main
TL-129W3	466.7	468.2	1.5	26.84	0.78	Main
TL-129W1	471.2	472.7	1.5	16.34	0.48	Main
TL-044	543.4	544.9	1.5	109.50	3.19	Main
TL-118	87.2	88.7	1.5	53.24	1.55	West
TL-176	109.0	110.5	1.5	15.66	0.46	East
TL-180	150.0	151.5	1.5	44.29	1.29	East
TL-147	189.5	191.0	1.5	24.67	0.72	East
TL-200	292.8	294.3	1.5	13.71	0.40	East
TL-151	450.2	452.0	1.8	128.20	3.74	East
TL-208	532.5	534.0	1.5	45.37	1.32	East Step-out

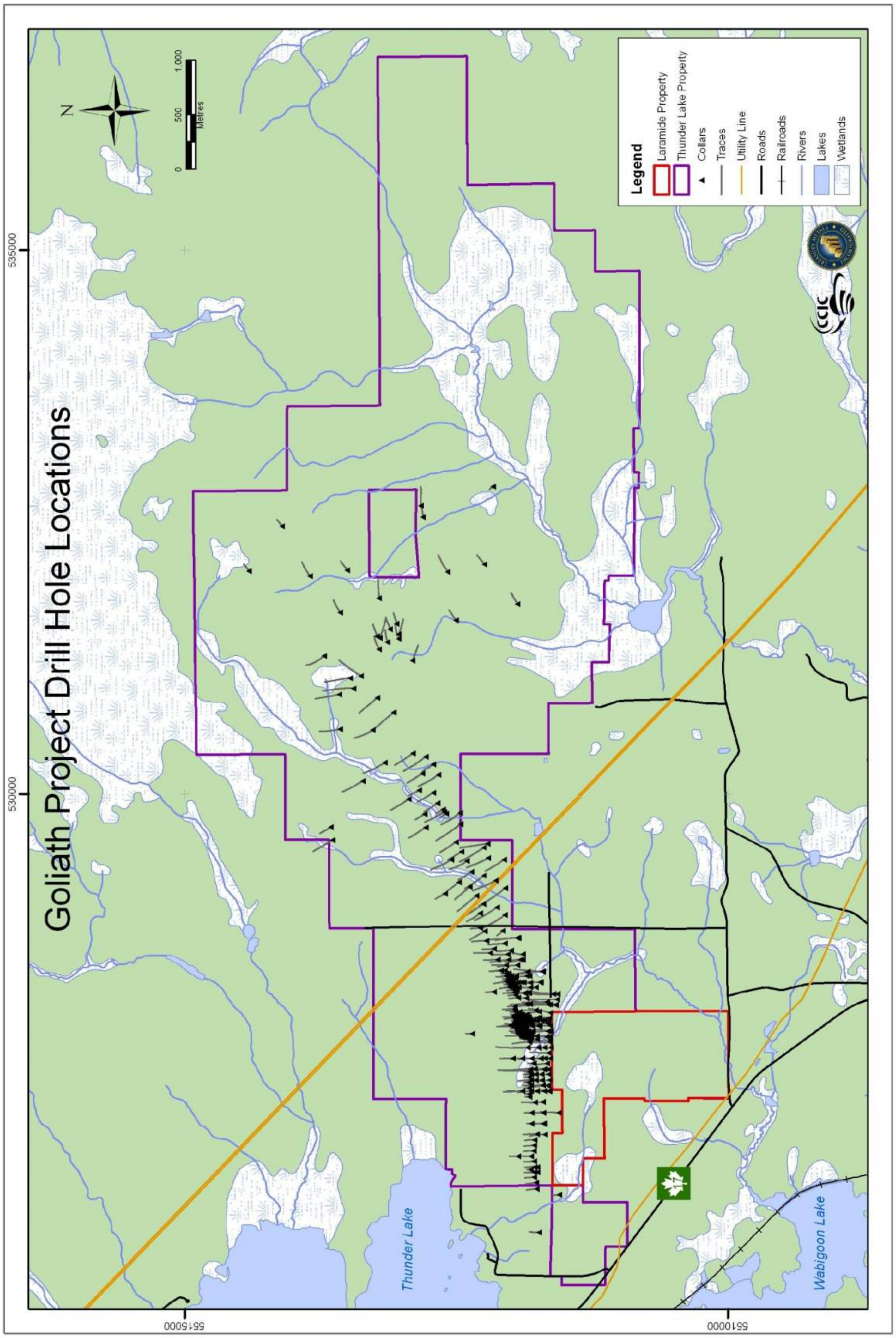


Figure 6-3. Location of drill collars and projections from Teck Cominco and Corona Gold joint venture (1989-1998).

6.2 LARAMIDE PROPERTY

The Laramide Property, historically referred to as the Goliath Gold Property, is located immediately south of the western portion of the Thunder Lake Property in parts of Lots 7 and 8 of Concession III in Zealand Township (Figure 4-1; Appendix A - Table A-2). Laramide Resources' interest by in this area was brought about by the Hemlo gold discovery near Wawa, Ontario and the discovery of the Thunder Lake mineralization by Teck in 1989-1990. Treasury has earned a 100% interest these parcels of land, including surface and mineral rights, totalling approximately 411 acres.

The exploration history of the Laramide Property is described in a number of reports to Laramide Resources Ltd. (Hogg, 1996; Hogg, 2002; Sills, 2007) and is summarized in Table 6-3. No work has yet been completed on land parcel 13492, acquired by Laramide in 2002. Hogg (2002) suggested that this parcel is underlain by metasedimentary rocks similar in character to those in the northern part of the Laramide Property and that rocks on this parcel could host altered gold-bearing zones.

Table 6-3. Summary of exploration completed on the Laramide Property.

Year	Parcel	Work Type	Comments/Results
1994	4822, 21553	Exploration Grid Geological Mapping Geophysical Survey (Mag/IP) 9 trenches and 10 pits with sampling	Geophysics completed by Rayan Exploration Ltd. Trenching/Sampling by I.M. Watson
1996	4822, 21553	8 NQ size diamond drill holes totalling 1,622 m	Drilled north at -45°; tested to maximum vertical depth of ~223 metres

Anomalous gold values were reported from surface sampling, 480 ppb Au within a narrow zone of quartz veining and pyritization within biotite schist (Hogg, 2002). Eight (8) shallow exploratory diamond drill holes (NQ size) were completed in October 1996, totalling 1,622 metres (Hogg, 2002). The same rock sequence that returned anomalous gold concentrations at surface was intersected in drill hole G-2 at 80 metres depth, with a grade of 675 ppb Au over 6.0 metres (Hogg, 2002); drill holes G-1 and G-3, located 100 metres to the east and west of G-2 also reported anomalous gold concentrations in the same rock type, suggesting some lateral and vertical continuity.

The northern boundary of the Laramide Property lies about 250 metres south of the Thunder Lake Deposit and is situated at what appears to be the down-dip extension of the Thunder Lake Deposit. Hogg (2002) noted that a press release from Teck-Corona drilling (hole TL-129) reported an intersection of 10.5 metres grading 4.48 g/tonne Au (0.13 opt Au) at a depth of about 450 metres and that this intersection lies approximately 50 metres north of the Laramide Property boundary. Hogg (2002) observed that while the plunge of the zones of the Thunder Lake Deposit is uncertain, it is clear that the mineralized system will dip onto the Laramide Property at a depth of between 600 and 800 metres. Hogg (2002) also described metasedimentary rocks and alteration on the Laramide Property that is similar in character to those on the Thunder Lake Property.

6.3 HISTORICAL MINERAL RESOURCES AND RESERVE ESTIMATES

Historical estimates of resources within the Thunder Lake gold deposits were reported following major annual exploration drilling programs. Estimates were determined using results from surface drilling and underground sampling obtained for the Main Zone and C-Zone only (Table 6-4; Page et al., 1999a, 1999b).

A qualified person has not done sufficient work to classify the historical estimates as current mineral resources. Treasury is not treating the historical estimates as current mineral resources. Howe considers all of the historical resource estimates to be non-compliant with NI 43-101 standards and as such they should not be relied upon.

The calculation of mineral resources at the end of 1996 was determined from drill hole data available at the time, and this estimate was later revised by Teck using additional data available at the end of 1997 (Table 6-5). In 1996, an Inferred Resource of 3.65 million tonnes grading 7.28 g/tonne Au was calculated (Corona, 1997) and with new data from diamond drilling in 1997 was adjusted to 3.78 million tonnes grading 7.02 g/tonne Au (Page and Waqué, 1998). The calculations were carried out using the polygonal method (polygons obtained by half-distances between drill holes) and based on a cut-off grade of 3.0 g/tonne Au, a specific gravity of 2.7 g/cm^3 and a minimum thickness of 3.0 metres (Page and Waqué, 1998).

Table 6-4. Historical Mineral Resource Estimates - Thunder Lake Deposit Main Zone.

Year	Au (oz)	Estimated Resource
1996	854,000	3.65 million tonnes grading 7.28 g/t Au (Corona, 1997 and 2001)
1997	853,000	3.78 million tons grading 7.02 g/t Au (Corona, 1997 and 2001)
1998	618,700	2.974 million tonnes grading 6.47 g/t Au (Corona, 1999 and 2001)

Note: Resources are based on cut-off grade of 3.0 g/t Au and minimum thickness of 3.0 metres.
Historic resources are non-compliant with NI 43-101 standards and as such they should not be relied upon.

The most recent historic non-NI 43-101 compliant resource estimate is based on all drilling and surface work done to 1998, including underground face sampling, bulk sampling and surface diamond drilling. A total of 678 underground samples and 219 diamond drill holes from within the resource area were involved in the calculation. The calculations, completed using computer generated three-dimensional solid models of the Main Zone and C-Zone quartz-sericite schist units, used block sizes of 3 metres thick x 10 metres height x 10 metres strike length and utilized the Ordinary Kriging method for grade interpolation (Page et al., 1999a). The Inferred Resources, estimated by Teck geologists in 1999 (Gray and Donkersloot, 1999) are provided in Table 6-5 at varying cut-off grades (Corona, 2001).

Table 6-5. Teck Cominco historic Mineral Resource Estimate based on results of all drilling and sampling to 1998.

Main Zone	Tonnes	Grade (g/t Au)	Total Au (oz)	C-Zone	Tonnes	Grade (g/t Au)	Total Au (oz)
Cut-off (g/t Au)				Cut-off (g/t Au)			
10.0	439,000	15.12	214,000	--	--	--	--
5.0	1,390,000	9.56	427,000	--	--	--	--
3.0	**2,925,000**	**6.52**	**613,000**	**3.0**	**49,000**	**3.71**	**6,000**
2.0	4,676,000	5.00	751,000	2.0	339,000	2.50	27,000
1.0	9,927,000	3.09	986,000	1.0	1,860,000	1.56	93,000

Note: Historic resources are non-compliant with NI 43-101 standards and as such they should not be relied upon.

The calculations in Table 6-5 provide the most current estimate of historic (non NI 43-101 compliant) Inferred Mineral Resources. Using a cut-off grade of 3.0 g/tonne Au, the historic resources are 2.974 million tonnes grading 6.47 g/tonne gold (3,277,000 tons grading 0.189 opt Au) which represents approximately 618,700 ounces of gold. This calculation includes 2.925 million tonnes of 6.52 g/tonne Au (0.190 opt Au) present in the Main Zone and 49,000 tonnes grading 3.71 g/tonne Au (0.108 opt Au) in the C-Zone (Page et al., 1999a; Corona, 1999 and 2001).

6.4 PREVIOUS HOWE MINERAL RESOURCE ESTIMATES

Howe previously completed a mineral resource estimate for the Thunder Lake Deposit in 2008 and a mineral resource estimate update in 2010; both completed in accordance with CIM Standards on Mineral Resources and Reserves. These estimates are superceded by Howe's current 2011 mineral resource update (Roy and Trinder, 2011) restated in Section 14.

6.4.1 2008 Resource Estimate

During 2008 Howe carried out a NI 43-101 compliant mineral resource estimate (Roy and Trinder, 2008) for the Thunder Lake Deposit using historical drill hole data and Treasury drill hole data up to drill hole TL0845, completed in 2008. The Resource Estimate was prepared by Doug Roy, M.A.Sc, P.Eng., Associate Mining Engineer with Howe. Micromine resource modelling software was used to facilitate the resource estimating process. The resource estimate was completed in accordance with CIM Standards on Mineral Resources and Reserves. Only Mineral Resources were estimated – no Reserves were defined.

No top-cut was applied. The specific gravity was 2.78.

Non-diluted mineral resources were determined using a block cut-off grade of 3 g/tonne Au, as follows:

Category		Cut-off Grade (g/tonne)	Tonnes Above Cut-off	Average Grade (g/tonne)	Ounces
Indicated - Main Zone		3.0	560,000	5.9	110,000
Inferred:					
	H1	3.0	-	-	-
	H	3.0	480,000	4.7	70,000
	Main	3.0	2,520,000	6.4	520,000
	B	3.0	130,000	4.2	18,000
	C	3.0	90,000	4.0	12,000
	D	3.0	50,000	3.2	5,000
Total Inferred		3.0	3,300,000	5.9	625,000

6.4.2 2010 Resource Estimate Update

During March to May, 2010 Howe carried out a NI 43-101 compliant mineral resource estimate update (Roy et al, 2010) for the Thunder Lake Deposit using historical drill hole data and Treasury drill hole data up to drill hole TL0986, completed in 2009. The Resource Estimate was prepared by Doug Roy, M.A.Sc, P.Eng., Associate Mining Engineer with Howe. Micromine resource modelling software (Version 11.0.4) was used to facilitate the resource estimating process. The resource estimate was completed in accordance with CIM Standards on Mineral Resources and Reserves. Only Mineral Resources were estimated – no Reserves were defined.

No top-cut was applied. The specific gravity was 2.78.

Resources were defined using a block cut-off grade of 0.5 g/tonne Au for surface resources (<100 metres deep) and 2.0 g/tonne Au for underground resources (>100 metres deep).

Non-diluted Surface plus Underground Indicated Resources total 3.4 million tonnes with an average gold grade of 2.5 g/tonne, for 270,000 ounces. Non-diluted Surface plus Underground Inferred Resources total 10.6 million tonnes with an average gold grade of 2.7 g/tonne, for 930,000 ounces. The Main Zone contains the majority of resources from both categories.

Zone	Cut-off Grade (g/tonne)	Tonnes Above Cut-off	Average Gold Grade (g/tonne)	Ounces	Average Silver Grade (g/tonne)	Average Copper Grade (g/tonne)	Average Lead Grade (g/tonne)	Average Zinc Grade (g/tonne)
Indicated								
Surface	0.5	2,900,000	1.9	180,000	5.4	86	820	1,700
Underground	2.0	490,000	5.7	90,000	13.8	100	710	1,500
Subtotal, Indicated		3,400,000	2.5	270,000	6.6	88	800	1,670
Inferred								
Surface	0.5	5,400,000	1.1	190,000	2.5	72	360	880
Underground	2.0	5,200,000	4.4	740,000	14.7	90	630	1,220
Subtotal, Inferred		10,600,000	2.7	930,000	8.5	81	490	1,050

Notes:

1. Cut-off grade for mineralised zone interpretation was 0.5 g/tonne.
2. Block cut-off grade for surface resources (less than 100 metres deep) was 0.5 g/tonne.
3. Block cut-off grade for underground resources (more than 100 metres deep) was 2 g/tonne.
4. Gold price was $US 850 per troy ounce.
5. Zones extended up to 150 metres down-dip from last intercept. Along strike, zones extended halfway to the next cross-section.
6. Minimum width was 2 metres.
7. Non-diluted.
8. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
9. Resource estimate prepared by Doug Roy, M.A.Sc., P.Eng.
10. A specific gravity (bulk density) value of 2.78 was applied to all blocks (based on 30 samples).
11. Un-cut. Top-cut analysis of sample data suggested no top cut was needed and removal of high grade outliers would not materially affect the global block model grade.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Goliath Project is located within the Wabigoon Subprovince of the Archaean Superior Province, a 150 kilometre-wide volcano-plutonic domain (greenstone belt) that has an exposed strike extent of 700 kilometres and extends an unknown distance beneath Palaeozoic strata at either end (Beakhouse et al., 1995). The Property is located north of the Wabigoon Fault, a major regional structure within the Wabigoon Subprovince that separates a northern domain characterized by generally southward-facing, alternating panels of metavolcanic and metasedimentary rocks, from a southern domain of generally northward-facing, volcanic rocks (Figure 7-1) (Beakhouse, 2000). The trace of the Wabigoon Fault occurs just south of the village of Wabigoon.

7.2 Property Geology

The most recent investigations of the Goliath Project area geology were carried out by the Ontario Geological Survey from 2000 to 2005. Detailed descriptions were published by Beakhouse and Pigeon (2003) on the geology of Zealand Township. Berger (1990) had earlier described the geology of Laval and Hartman townships.

The following description of the Goliath Project area geology is an integration of the historic mapping and 2008 geological mapping by CCIC personnel on behalf of Treasury. Major lithological units were identified on the basis of visual examination of rock type in outcrops, drill core, and trenches. The rocks have been grouped into the Thunder Lake Assemblage and the Thunder River Mafic Metavolcanic rocks.

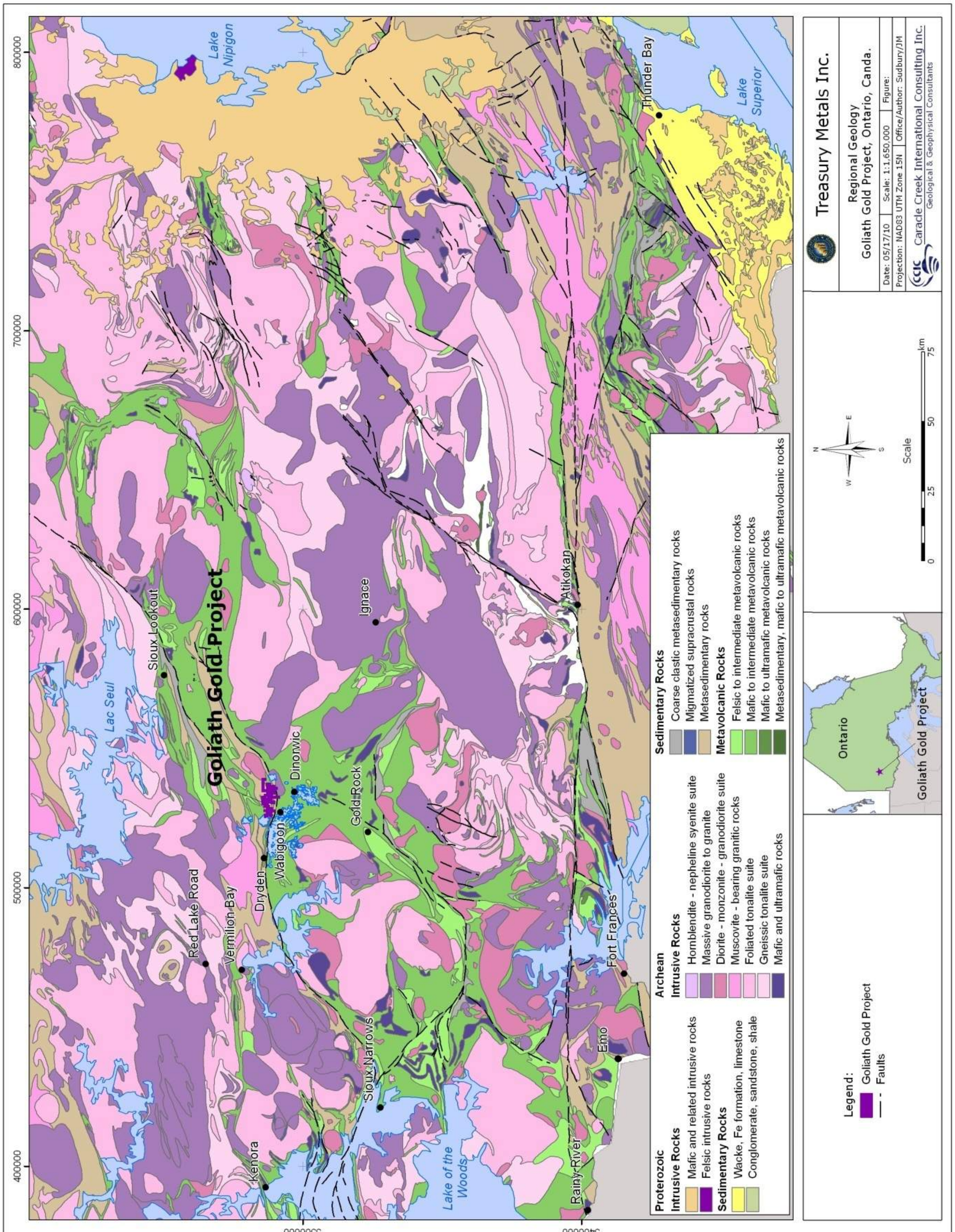


Figure 7-1. Location of the Goliath Project and regional geology of northwestern Ontario (from Percival and Easton, 2007a).

7.2.1 Thunder Lake Assemblage (Beakhouse, 2000)

The Thunder Lake Assemblage, an upper greenschist to lower amphibolite metamorphic grade volcanogenic-sedimentary complex of felsic metavolcanic rocks and clastic metasedimentary rocks (Beakhouse 2000), underlies much of the Project area (Figure 7-2). The assemblage comprises quartz-porphyritic felsic to intermediate metavolcanic rocks represented by biotite gneiss, mica schist, quartz-porphyritic mica schist, a variety of metasedimentary rocks and minor amphibolites (Table 7-1).

Beakhouse (2001) described the main sedimentary unit as dominated by biotite-muscovite and biotite schist (greywackes) with subordinate inter-layered metasediment (probably pyroclastic siltstone and arkose sandstone) which exhibits highly strained and well-preserved primary sedimentary structures such as graded bedding, scour, rip-up clasts etc. This sedimentary unit, known as the Thunder Lake Sediments includes ink blue magnetite layers that are closely associated with distinctive garnet-rich layers and calc-silicate rock, shown in earlier publications as Iron Formation (Satterly, 1941).

The Project area is also underlain by a unit dominated by felsic metavolcanic rocks conformably inter-layered with wacke-siltstone. The lenses of metasedimentary rocks that occur within the felsic unit are similar to those making up the main sedimentary unit.

Compositional layering in metasedimentary rocks strikes 90° and dips from 70° to 80° south-southeast. Schistosity is commonly developed within both the metasedimentary rocks and volcanic rocks and exhibits a similar orientation (Hogg, 2002).

All of the rocks have been subjected to folding and moderate to intense shearing with local hydrothermal alteration, quartz veining, and sulphide mineralization.

7.2.2 Thunder River Mafic Metavolcanics

The Thunder River Mafic Metavolcanic rocks (Table 7-2) underlie the south part of the Property. The mafic rocks are generally massive but are pillowed locally and include amphibolite and mafic dykes, which are characterised as chlorite schists. Some rocks have been described as ultramafic in character (Hogg, 2002).

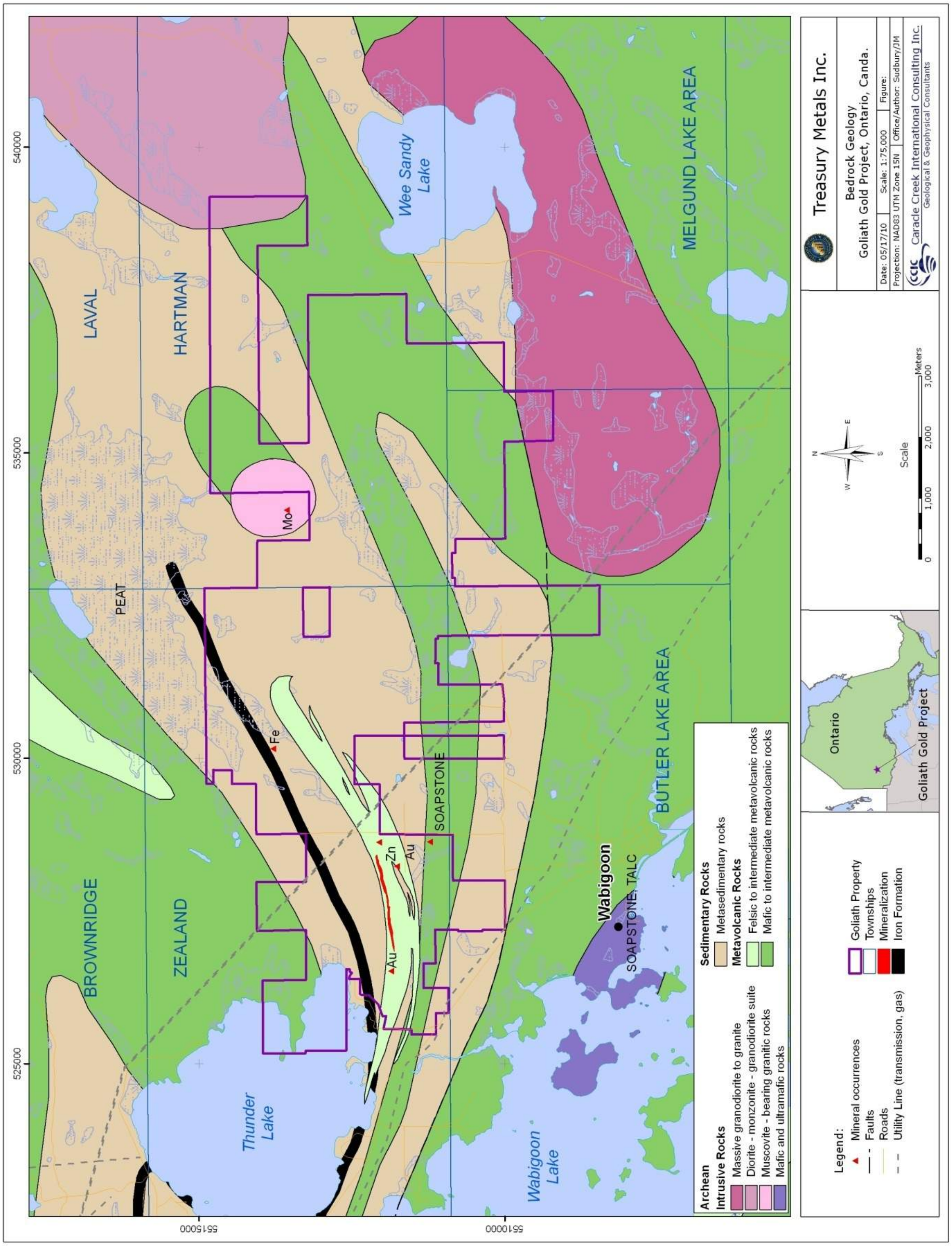


Figure 7-2. Bedrock geology in the area of the Goliath Project area, northwestern Ontario (after Beakhouse and Idziszek, 2006; Percival and Easton, 2007a).

Table 7-1. Thunder Lake Assemblage Rock Descriptions.

Rock Type	Description
Biotite muscovite schist (BMS)	Dark grey to grey, fine to medium grained mica schist. Usually it consists of intercalated leucocratic and melanocratic bands. This unit contains a high number of grey to milky white quartz veins. Most of the veins are 1-15 cm wide, parallel or crosscutting the foliation. Some veins are associated with highly chloritized and silicified intervals with tourmaline and sulphides.
Muscovite sericite schist (MSS)	Light grey to beige grey, fine to medium grained quartz- sericite schist. It is variably siliceous, commonly contains interbedded, dark grey biotite-muscovite bands and grey to milky white quartz veins. It is characterized by the presence of moderate to strong pervasive sericite alteration and gold and silver bearing disseminated sulphides.
Iron formation (IF)	Dark greenish grey calc-silicate metamorphic rocks, which include coarse to medium grained gneiss, biotite schist, 10 to 15 cm wide distinctive layers enriched with garnet, chlorite and narrow ink blue magnetite bands. The rock unit is magnetic and contains disseminated pyrite.
Metasediment (MSED)	Grey to dark grey-green medium grained massive unit, which consists of biotite, feldspar, quartz, muscovite with a weak patchy potassium and sericite alteration and rare hematite (rusty brown) alteration. Foliation is poorly developed but more prominent in contact and altered areas. Quartz veins, parallel or crosscutting the foliation are very common. This unit can be distinguished by presence of numerous “quartz eyes” or quartz porphyroblast. (identified as “arkose metasediment” or “quartz feldspar porphyry” in Teck/Corona drill logs and historic reports). This unit may contain 1-5% bleb-finely disseminated pyrite and chalcopyrite.
Biotite schist (BS)	Dark grey to black, fine to medium grained, slightly to well foliated schist. Locally contains disseminated pyrite in the foliation planes and fractures. It was referred to as pelites or greywackes in the historical reports
Chloritic-Biotite schist (Chl-BS)	Dark grey to greenish grey medium grained, slightly to well foliated schist. Locally it contains disseminated pyrite along foliation planes and fractures. Referred to as pelites or greywackes in the historical reports.

Table 7-2. Thunder River Mafic Metavolcanic Rock Descriptions.

Rock Type	Description
Mafic dyke (MD)	Usually narrow dark green to almost black massive or slightly foliated fine to medium grained biotite-chlorite schist. The width of the layers can reach up to 5m. The dykes can be either parallel to or crosscut the foliation.
Amphibolite (Amf)	Coarse to medium grained, dark green to black to green units, which consist mainly of 30-50% amphibole (hornblende and actinolite), 30-40% feldspar and pyroxene with rare post genetic quartz veins and layers of chlorite schist. It has typical “salt and pepper” appearance and nematoblastic texture.
Green schist	Usually dark green to almost black foliated fine to medium grained schist, which consists mainly of chlorite, biotite, feldspar, amphibole. The width of the layers can reach up to 5m.

7.2.3 Thunder Lake Deposit Area Geology

For the purpose of the exploration and development, three major rock groupings are consistently recognized from south to north at the Goliath Project's Thunder Lake deposit (Page, 1994):

- A hangingwall unit of quartz ± feldspar-porphyry intrusive rocks and metasedimentary rocks;
- A central unit of approximately 100-150 m true thickness, which hosts the most significant gold concentrations and consists of intensely deformed and variably altered felsic, fine to medium grained, quartz-feldspar-sericite schist (MSS) and biotite-quartz-feldspar-sericite schist (BMS) with minor metasedimentary rocks (MSED); and,
- A footwall unit of predominantly metasedimentary rocks (BMS and IF) with some porphyritic units and minor felsic gneiss and schist.

7.3 Structural Geology

The Property is within the Wabigoon Sub-Province and north of the Wabigoon Fault. The key structural features have been described and interpreted by Page (1994), Beakhouse (2001), Ravnaas et al. (2002) and Wetherup (2008). Three different deformation events and three related generations of folds and fault have been interpreted in the area. Structures and veins observed in the area of the Thunder Lake deposit have been interpreted within and relative to this basic framework (Table 7-3). CCIC personnel collected additional structural data during Treasury's 2008 mapping and drilling programs.

Table 7-3. Summary of structural features observed on the Thunder Lake Property (Wetherup, 2008)

Event	Structure	Deformation	Vein	Description
D_0	S_0	Compositional layering of metavolcanic and meta-sedimentary rocks; argillic alteration zones (?)	V_0	Greyish, highly deformed, S_1 foliation parallel quartz-sulphide ribbons and silicification hosted by quartz-sericite schist
D_1	F_1 S_1	Isoclinal folding F_1 axial planar and layer parallel foliation/schistosity	V_1	White, deformed, locally cross-cutting quartz+/-tourmaline+/-sulphide veins
D_2	F_2	Closed (60°) folds; axial planes ~045/90; discrete, 5-40 m spaced, axial planes	V_2	Weakly deformed white quartz+/-sulphide veins along F_2 axial planes & at 45° to F_2 axial planes.
D_3	NW Fault	Brittle faults/fractures dip moderately NNE	V_3	Un-deformed white, non-planar quartz veins dip moderately NNE and cross-cut or follow foliation locally

The deformation features observed in the outcrops and the drill core are listed below:

D_0 pre-deformation structures developed during the rock formation and are a result of possibly transposed bedding and/or alteration zones. Alternating leucocratic quartz-sericite and melanocratic biotite-feldspar layers represents compositional layering within felsic metavolcanic and metasedimentary rocks. The width of the layers varies from 0.5 to 10 centimetres, but locally forms larger units interbedded with layers of metasediments. Larger zones (up to 40 metres wide) of dominantly quartz–sericite schist locally contain greyish, very fine-grained layers or "ribbons" of quartz, V_0 veins, which are usually associated with sulphide mineralization. The association of almost pure very fine-grained quartz layers within the center of a larger zone of quartz-sericite schist could represent transposed and metamorphosed sericite alteration around quartz veins within the felsic metavolcanic rocks. Sulphide minerals observed in drill core commonly occur along S_1 foliation planes and appear to have been remobilized.

Contacts between the lithostratigraphic units were measured in the outcrops and in the core. Within the felsic volcanic rocks the contacts between the sericite schist and the biotite-muscovite schist is transitional. More noticeable is the contact between the felsic volcanic rocks and the metasedimentary rocks. Usually it is marked by a very small angular discordance and is almost parallel to the primary bedding. The strike and dip are approximately 90°/70°S, but can change from 68°/72°S to 90°/80S. Treasury interprets that primary syngenetic gold and silver mineralization was deposited during this event because the mineralization is contained within the sericite schist and/or biotite-muscovite schist.

D_1 deformation is represented by well-developed foliation S_1 and isoclinal folds F_1 within the felsic metavolcanic (BMS, MSS) and metasedimentary rocks (BS, IF). The foliation and the axes of the folds were measured in the outcrops, in the trench and during the orientation drilling of holes TL0822 to TL0837. The foliation is approximately 074°/70°S, but it can vary from 064°/62°S to 090°/80°S. It is suppressed in the mafic metavolcanic units and in many cases the texture of the mafic rocks is almost massive.

F_1 folds were observed in the outcrops and in the core. The folds are isoclinal and the fold axes are parallel to the F_1 foliation (Figure 7-3). The dip and strike of the axial planes are approximately 090°/70° but it can change from 080°/68°S to 100°/78°S. In most cases the hinges/fold noses display evidence of distension where continuing compressional deformation has stretched the hinge and its limbs are highly attenuated and thinned (Figure 7-4). These fold noses are often completely “decapitated” from their limbs and generally only hook shaped or quartz lenses remain which suggests that some of the boudinage or quartz lenses, observed in the felsic metavolcanic rocks may be more complicated. Deformed, white, coarse grained quartz veins ± tourmaline, ± stringers or porphyroblasts of sulphides, 1 to 10 centimetres wide occur dispersed throughout the felsic metavolcanic and metasedimentary rocks (Figure 7-5). White, coarse-grained veins are not localized to certain pre-deformational “stratigraphy” and are interpreted to be syn-tectonic quartz veins (V_1) as they are affected by D_1 deformation and occur in all rock types. They can be parallel to, but usually crosscut the foliation. The assay results do not show a direct correlation between the quartz veins and the elevated gold and silver concentrations.

Figure 7-3. A small outcrop with quartz lenses and F_1 fold structure in highly altered biotite-muscovite schist, (11+00W, 6+10N, UTM 527654E, 5511244N).

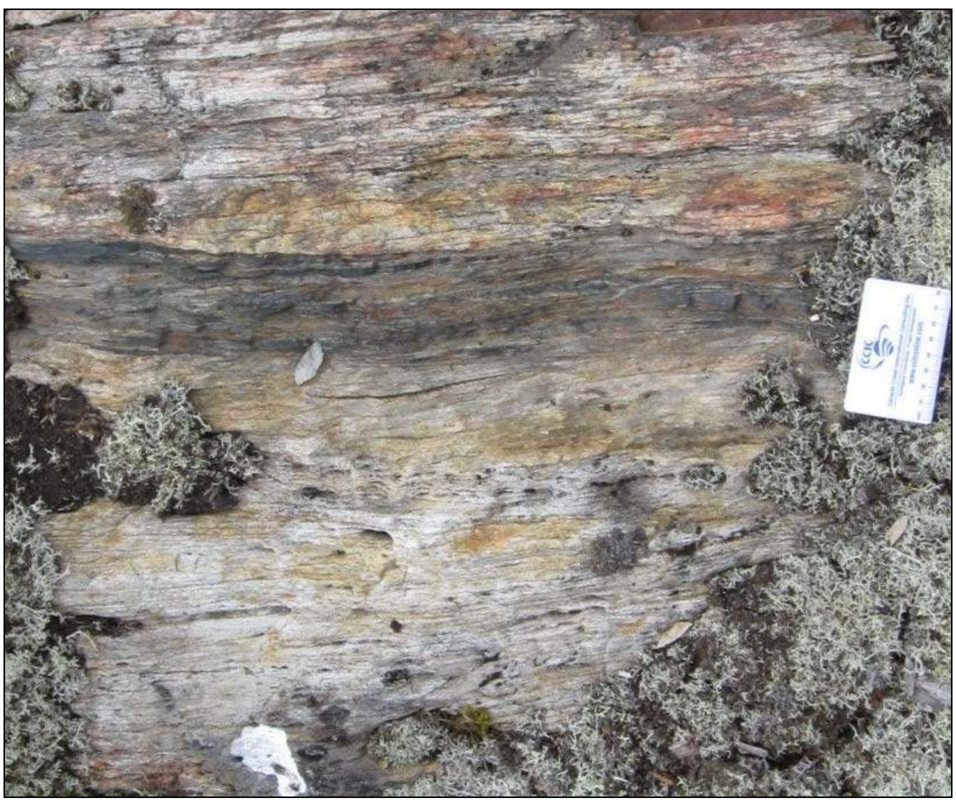


Figure 7-4. Small outcrop of highly foliated and altered MSS, (line 8+55W 1+01N, UTM 527917E, 5511753N, Zone 15, NAD 83). Structures - S_1 foliation and V_1 quartz veinlets (ribbons)

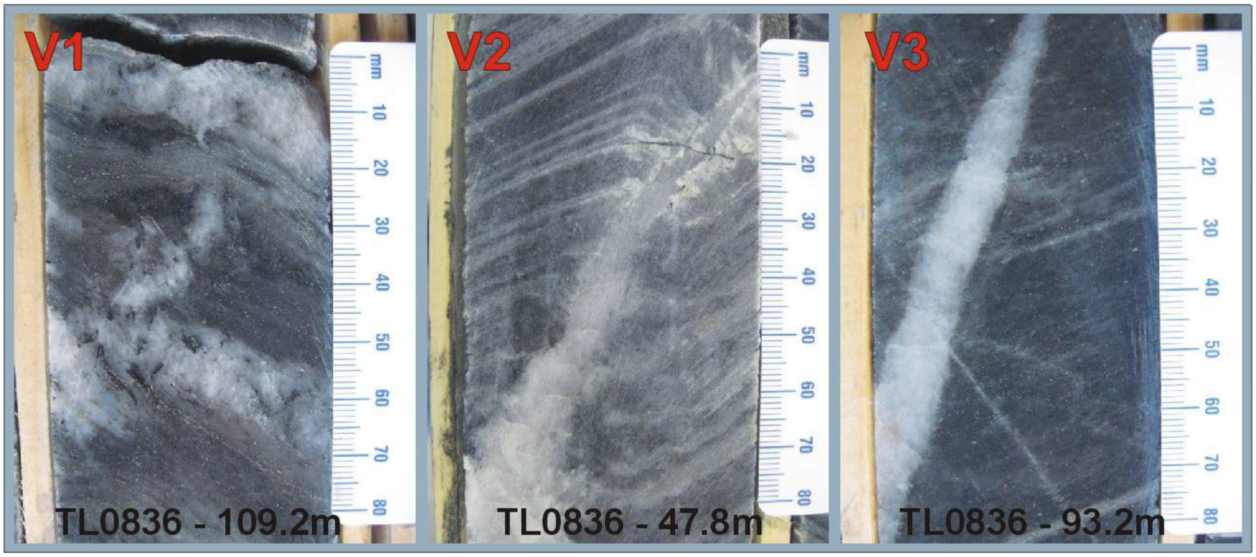


Figure 7-5. Examples of V1, V2 and V3 vein types from drill hole TL0836.

D_2 deformation is observed as zones of disturbed foliation related to closed F_2 folds and V_2 quartz veins. Rare F_2 fold hinges are observed in the outcrops. They are several cm in scale and affect the position of the felsic volcanic package that hosts mineralization on the Goliath Project. Where F_2 fold axes and fold noses occur within the gold-silver mineralized zones in the felsic metavolcanic rocks, gold and silver values are commonly 10 to 100 times higher than in the adjacent intervals. In some cases they contain coarse-grained visible gold (VG) or electrum, but even the very fine-grained mineralization returns higher gold or silver concentrations. Throughout the mapping program the orientation of the F_2 fold axes were measured in the outcropping rocks. The strike of the F_2 plane is approximately 220° to 230° and dips 85-90° southward. As demonstrated in the mineralization block model, the F_2 fold axes are almost vertical and the intersections of the F2 fold axes and the mineralization plunge steeply westward. Overall, discrete F_2 fold zones are narrow (up to 10-15 centimetres wide), widely spaced (5 to 25 metres) and locally carry significant gold mineralization. Determining where F_2 folds are likely to be located will identify the location of potential high-grade mineralization. S and Z folded F_1 foliation, V_0 and V_1 quartz veins, and undeformed crosscutting V_2 veins are all features attributed to the D_2 deformational event. The veins are differentiated on the basis of mineralogy, texture and amount of strain (Figure 7-5 as described in Table 7-3)

The D_3 deformational event is represented by brittle faults and fractures filled in with quartz, chlorite, feldspar, carbonate or/and gouge. Local shear zones and local faults are exposed in the outcrops and old trenches.

The first fault system is almost vertical and strikes 220 to 240°. The system consists of almost parallel microfaults with dextral displacement on a centimetre scale. Very often it is accompanied with a 1.0 to 1.5 metre wide sericite alteration.

The second fault system, exposed in the outcrops has almost N-S direction. The azimuth is 352 to 008° and the dip is 85 to 90°. Usually the fault zone consists of 2-3 microfaults located within

0.5 to 1 metres. It affects all rock units including clastic metasedimentary, felsic volcanic and mafic volcanic rocks. Commonly the surrounding area is highly fractured (Figure 7-6).

Figure 7-6. Chloritic biotite schist with 13cm wide fault zone, 352°/85, 16cm dextral displacement of feldspar vein. (8+85W, 4+30S, UTM 527879E, 5511200N, zone 15, NAD83).

The most significant feature found in the drill holes that can be related to D_3 deformation is what Teck-Corona described as the NW Fault. This is a brittle structure which strikes W to WNW and dips shallowly northward. It was intersected in most of the deeper holes (Figure 7-7). Drill section interpretation by Teck-Corona shows very little dip-slip movement along this structure (approximately 5 to 10 metres - hangingwall up). Most shallow dipping structures are dip-slip in nature but since this is such a prevalent feature there may be a significant component of strike-slip motion since dip-slip offset is minor. A third generation of white, coarse-grained quartz veins (V_3) are formed during the D_3 event. These veins occur in all rock units and typically crosscut the foliation obliquely with sharp margins (Figure 7-5). No deformation appears to have occurred in these veins, which can also cut D_2 structures. V_3 veins are hematized on the surface and where sampled, they have not returned any significant gold or silver values. D_3 deformation isn't related to the gold-silver mineralization but the NW fault, appears to offset the mineralized zone. Wetherup (2008) demonstrated that high-grade mineralization occurs along the steeply SW plunging intersections of F1-F2 fold axes and that these shoots are offset by the NW fault.

Figure 7-7. Tectonically brecciated muscovite sericite schist and "NW fault" zone intersected in the drill hole TL0815 at 148 m. The fault is filled in with white-greenish clay/gouge.

7.4 MINERALIZATION

The main zones of mineralization (Thunder Lake Deposit) project to surface approximately 250 to 300 metres north of Norman Road, which is the base line of the exploration grid (Figure 9-1). The Main Zone, Footwall Zone (B, C and D subzones), and Hangingwall Zone (H and H1 subzones) of the Thunder Lake Deposit strike approximately east-west, varying between 090° and 072°, with dips that are consistently 72°-78° toward the south or southeast. The main area of gold, silver and sulphide mineralization and alteration occurs up to a maximum drill-tested vertical depth of ~805 metres (TL135) below the surface, over a drill-tested strike-length of approximately 2,300 metres within the current defined resource area. The historic Teck-Corona drilling confirmed that anomalous gold mineralization extends over a strike length of at least 3,500 metres (Corona, 1998) and work by Treasury has shown this anomalous gold mineralization and alteration to extend over a strike length of +5,000 metres.

The mineralized zones are tabular composite units defined on the basis of anomalous to strongly elevated gold concentrations, increased sulphide content and distinctive altered rock units and are concordant to the local stratigraphic units (Figure 7-8). Stratigraphically, gold mineralization is contained in an approximately 100 to 150 metre wide central zone composed of intensely altered felsic metavolcanic rocks (quartz-sericite and biotite- muscovite schist) with minor metasedimentary rocks. Overlying hangingwall rocks consist of altered felsic metavolcanic rocks (sericite schist, biotite-muscovite schist and metasedimentary rocks) and the footwall rocks comprise metasedimentary rocks with minor porphyries, felsic gneiss and schist. Gold within the central unit is concentrated in a pyritic alteration zone, consisting of quartz-sericite schist (MSS), quartz-eye gneiss, and quartz-feldspar gneiss (Corona, 2001).

Schematic geological block models of the mineral zones as logged from drill core were developed by Howe from the Teck drill hole database and the Treasury database. The lithological units follow and define the main trend of mineralization for the Thunder Lake Deposit, including the Main Zone, Footwall, and the Hangingwall Zone.

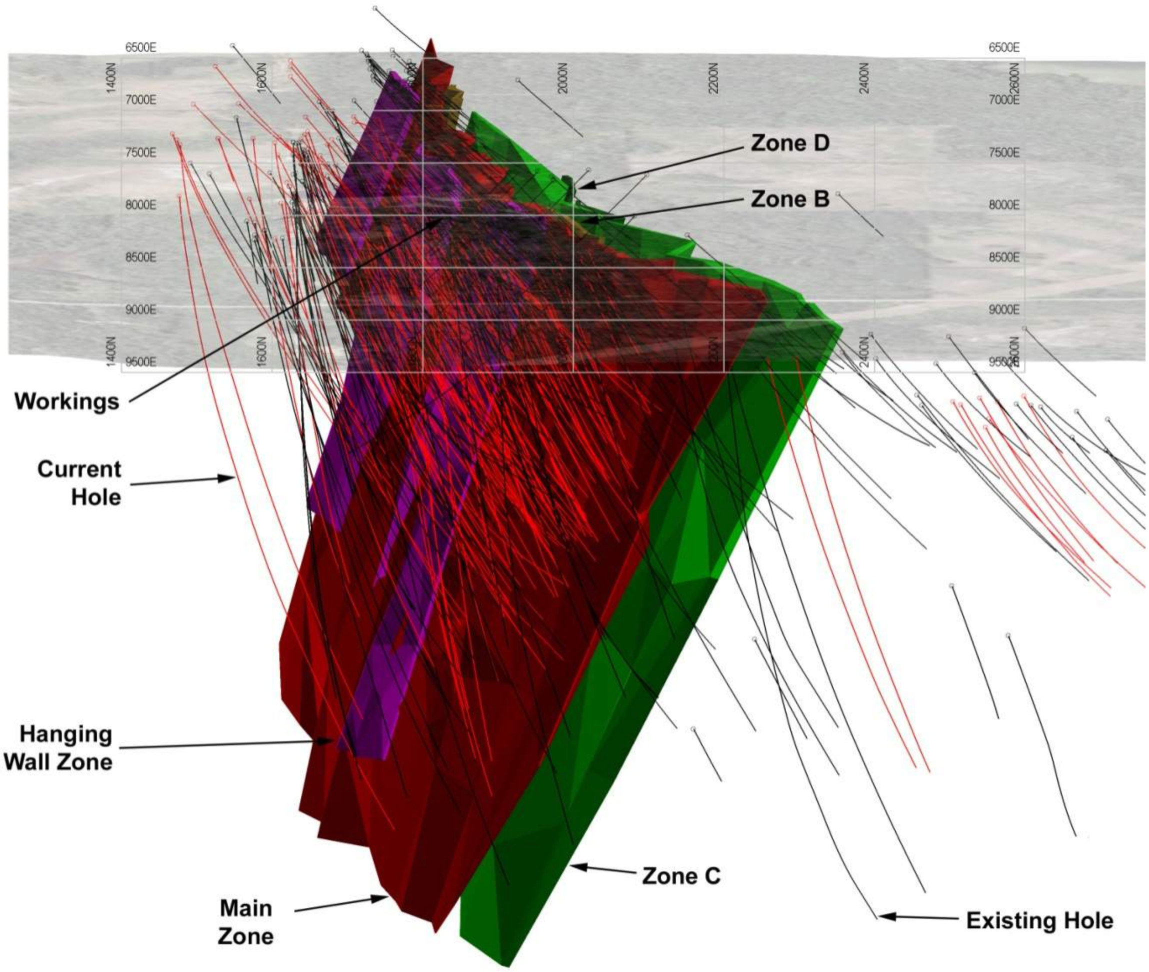


Figure 7-8. 3D view of interpreted mineralized zones of the Thunder Lake Deposit looking west - Main Zone, Footwall Zone (B, C and D subzones) and, Hangingwall Zone

The high-grade central part of the Main Zone was discovered in 1990 and partially delineated by 1994. The Treasury 2008 and 2009 drilling programs primarily targeted the Main Zone, but the Hangingwall Zone was intersected, as was the Footwall Zone by deeper drill holes. Drilling has intersected the Main Zone over a strike length of approximately 2,300 metres and a thickness of 5 to 30 metres. The Main Zone is composed of well-defined pyritic quartz-sericite schist (MSS) separated by less-altered biotite-feldspar schist (BMS). Sulphide mineralization and local visible gold (VG) occurs mainly within the leucocratic bands, but occasionally it is localized in the melanocratic bands enriched with biotite and chlorite. The sulphide content of the mineralized zone is generally 3-5% but locally is up to 15%. Highest gold and silver values are associated with very strong pervasive quartz-sericite alteration. It appears that gold content does not directly correlate with pyrite content, but generally an increase in the gold and silver correlates with an increase in the pyrite and more specifically, the sphalerite content. An increase in chalcopyrite and galena content has a lower correlation to an increase in gold values (Figure 7-9 to Figure 7-11).

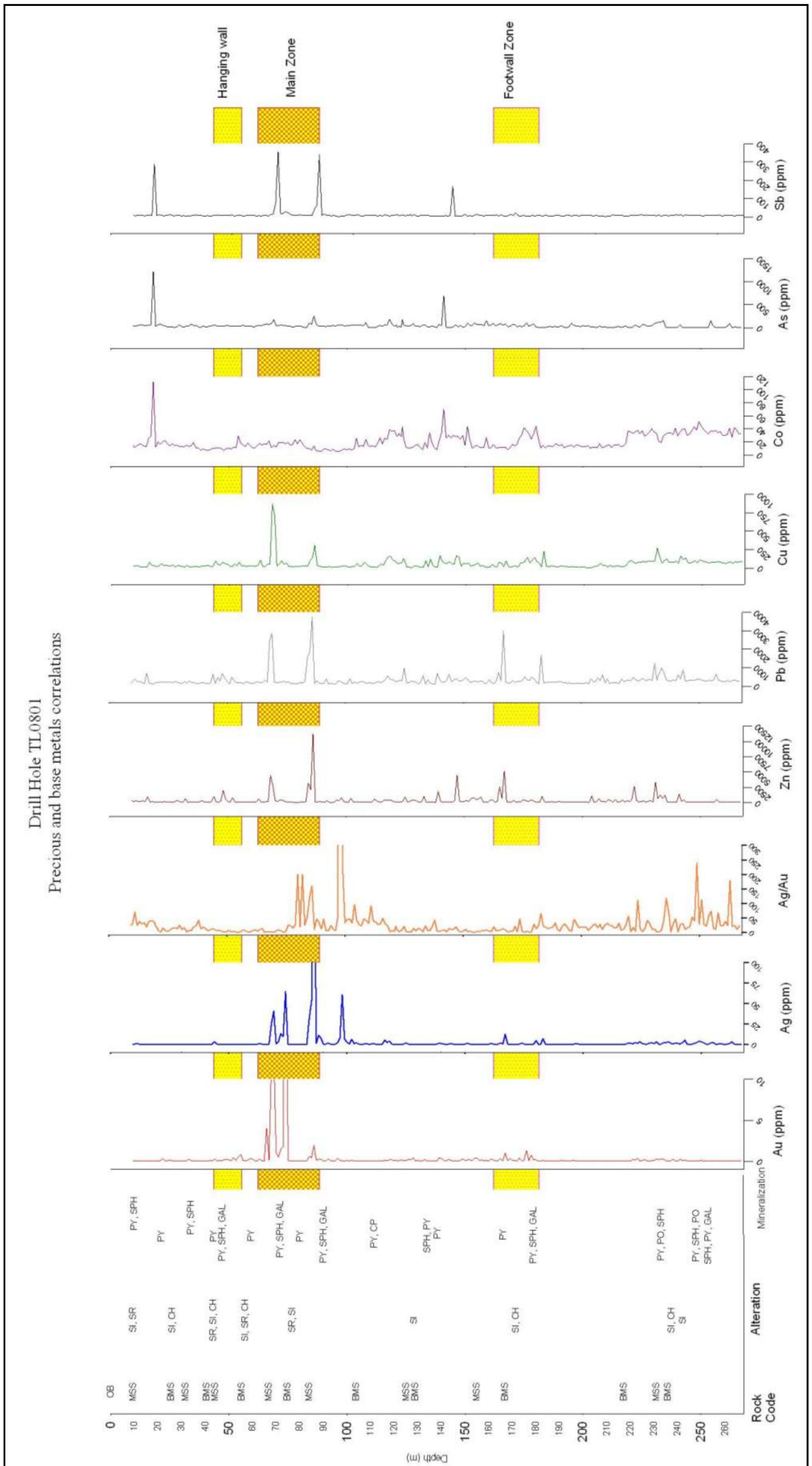


Figure 7-9. Correlation between precious and base metals in drill hole TL0801.

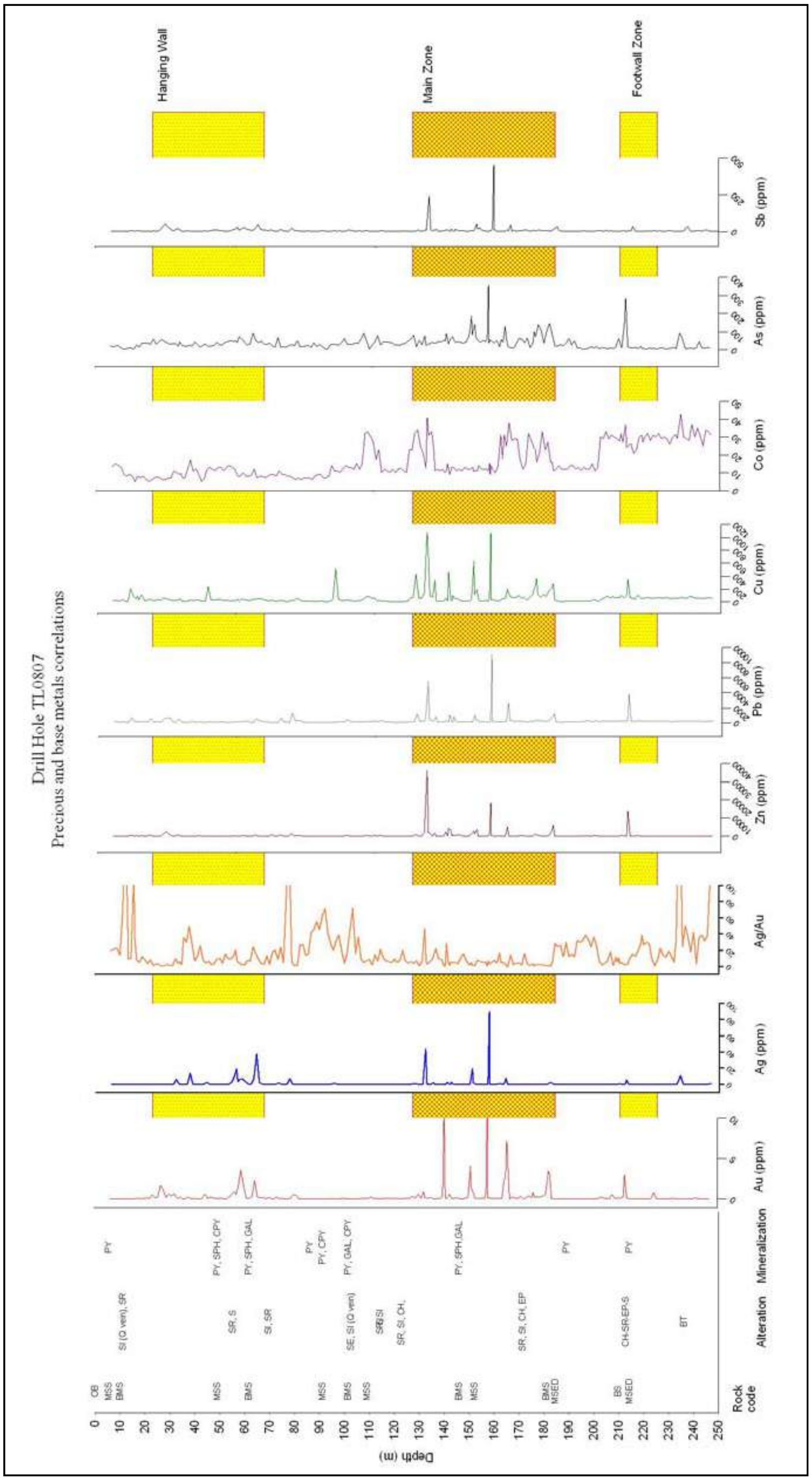


Figure 7-10. Correlation between precious and base metals in drill hole TL0807.

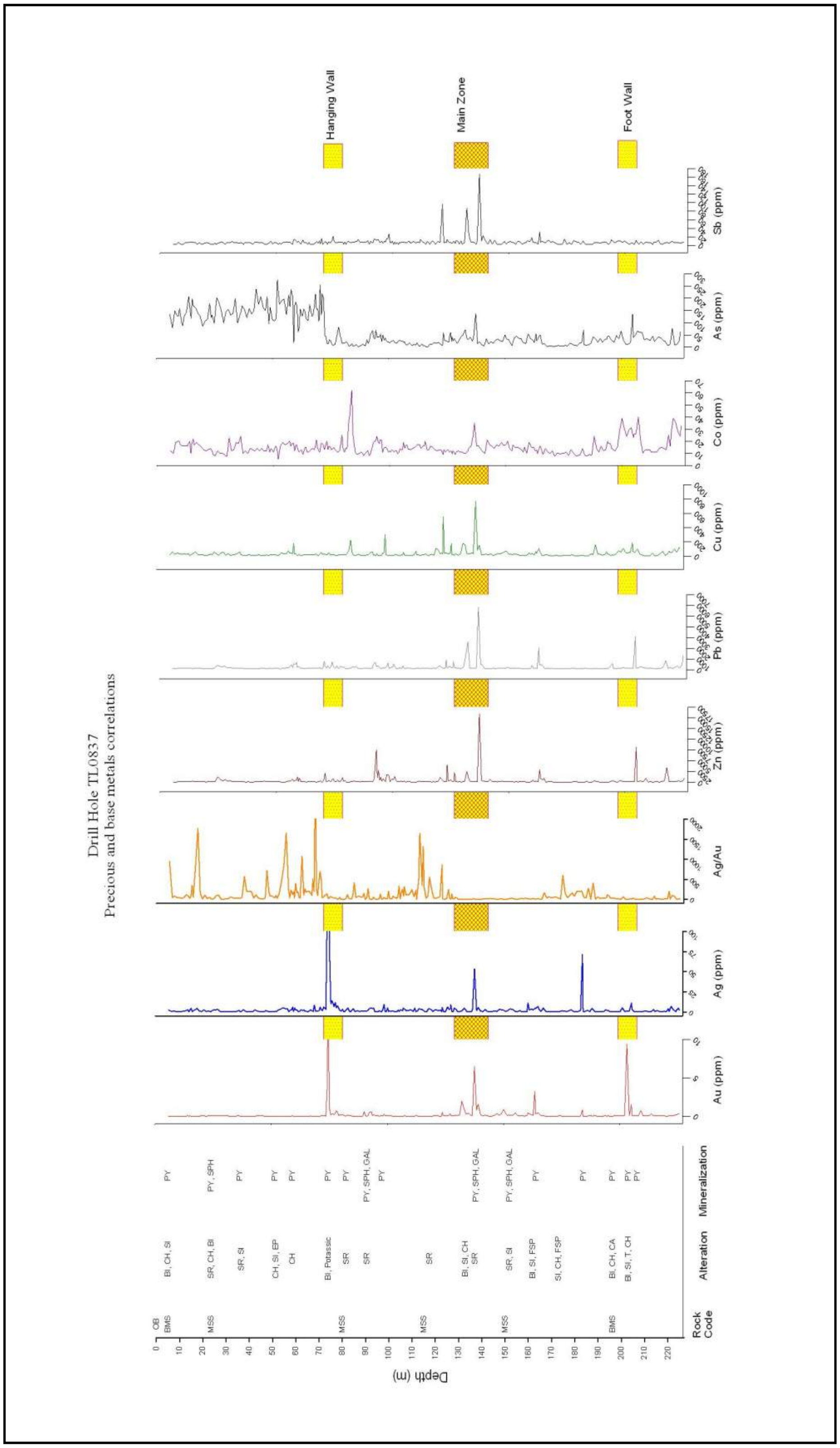


Figure 7-11. Correlation between precious and base metals in drill hole TL0837.

Both the metal concentration data and the whole rock analyses provide further insights into the nature of the Thunder Lake Deposit. CCIC calculated metal ratios in order to group the elements that were in the initial hydrothermal solution. Native gold and silver (VG and electrum) are associated with finely disseminated sulphides and coarser grained pyrite. The main sulphide phases are pyrite, sphalerite, galena, pyrrhotite, minor chalcopyrite and arsenopyrite in decreasing order of abundance. Two distinct types of pyrite are recognized: disseminated fine grained cubic euhedral crystals occurring in the foliation planes; and disseminated subhedral to irregular grains and stringers, with inclusions of galena, occurring in quartz veins and along the margins of the veins. The second type is commonly associated with other base metal sulphides.

Silver to gold ratios are generally random. Possibly during the syngenetic mineralization event, more silver than the gold was contained in the hydrothermal solutions (ratio Ag/Au>>1), but during the epigenetic mineralization event, some of the gold was redistributed and there was enrichment in structurally induced zones of enhanced porosity and permeability. A similar relationship of gold to base metals is observed. For this reasons the ratios Ag/Au, Au/Pb or Au/Zn didn't give a clear vector of the mineralization and reliable geochemical targets. An illustration of the above observations and interpretations is the high-grade section in hole TL0815. At 50.8 metres the mineralization is represented by very rare specks of visible gold (VG) (Figure 7-12, circled in red), fine-grained disseminated pyrite in the foliation planes, blebs, stringers and veinlets of pyrite. The base metals sulphides are concentrated in blebs and stringers of sphalerite, cubic fine-grained galena and chalcopyrite (Figure 7-12).

Low grade Au-Ag mineralization is pervasive in the Main Zone, Hangingwall Zone and in the Footwall Zone, whereas high-grade gold mineralization (>3 g/tonne) is concentrated in several steeply dipping, steep west-plunging shoots with relatively short strike-lengths (up to 50 metres) and considerable down-plunge continuity. Corona (1998) interpreted the high-grade shoots to be the result of tight folding of the mineralized horizon (gold concentrated in fold noses) and appear to occur at regular intervals however this remains to be confirmed. Very rare flakes of aquamarine green mica (fuchsite: Cr muscovite) occur in the strongly altered sericite alteration with high-grade gold. Usually mineralized intervals are narrow (up to 0.5 metres) zones enriched with 3 to 10% visible sulphides (pyrite, sphalerite, galena, chalcopyrite ± arsenopyrite, ± dark grey needles of stibnite) within a wider quartz- sericite or biotite-feldspar sections with fine grained disseminated pyrite located in the foliation planes.

Figure 7-12. High-grade gold mineralization with flakes of visible gold (VG) in a strongly altered section in felsic metavolcanic (biotite-muscovite schist) from the Main Zone (TL08-15, 50.8 m).

The Footwall Zone consists of 3 subzones: B, C and D. The Footwall Zone is well developed north of the Main Zone and has been drill intersected over a strike length of approximately 2,000 metres and is up to 25 metres thick. It has not been systematically drilled in that not all drill holes targeting the Main Zone have continued to intersect the Footwall Zone, therefore the discontinuity of the subzone intercepts may be more apparent than real. The Footwall Zone is thicker but lower in grade than the Main Zone and is located ~15-50 metres north of the Main Zone (Figure 7-8). The sulphides make up usually 2-4% of the mineralized section. Gold and silver are hosted within the highly altered quartz-sericite intervals, associated with fine grained disseminated pyrite, blebs, stringers and veinlets of pyrite, pyrrhotite, sphalerite, fine-grained galena and chalcopyrite in fractures and along the margins of quartz veins. Some coarse-grained visible gold was observed in hole TL0817 at 129.2 metres.

The Hangingwall Zone is located 25 to 50 metres south of the Main Zone. It is approximately 1,500 metres long, up to approximately 6 metres wide and is open along strike in both directions and to depth. It consists of two subzones: H and H1. Sulphides make up usually 3-5% of the whole section. Gold and silver are probably included in the pyrite or around the pyrite micro grains. Only few flakes of coarse-grained gold or electrum were visible in the core or in the grab samples. Most of the sulphides are located mainly in blebs or stringers parallel to the foliation planes (Figure 7-12). Usually blebs, stringers and veinlets of pyrite are associated with the stringers of sphalerite, cubic fine-grained galena, chalcopyrite and pyrrhotite. Very often they fill in small fractures in the host rock or are along margins of quartz veins.

7.5 UNDERGROUND EXPLORATION (TECK COMINCO - 1998)

The 1998 underground exploration and bulk sampling program provided insight into the structure and mineralization intersected during the historic Teck surface drilling programs. Page et al. (1999) reported the following observations from the underground program:

- More significant mineralized areas are in contact with units of dark coloured intermediate quartz porphyry.
- The Central Unit hosts the most significant gold concentrations and consists of intensely deformed and variably altered felsic gneiss and schist with minor metasedimentary rocks.
- Strongest gold mineralization is localized in siliceous quartz-sericite schist containing disseminated sulphides, sulphide veins, and sulphide mineralized quartz veins with rare coarse gold/electrum.
- Most of the gold is free and occurs in visible specks, and the "nugget effect" is pronounced confirmed by the results of wedge drilling (i.e. widely differing gold concentrations between original intersections and those from wedge intersections only a few feet from the original).
- Where investigated underground, the distribution of gold in the Main Zone is erratic and unpredictable.

7.6 ALTERATION

The western part of the Goliath Project area is underlain by hydrothermally altered felsic metavolcanics and metasediments and include an approximately 5 kilometre long zone of alteration and deformation with anomalous gold mineralization. Historic exploration established silicification and sericitization as the primary and most extensive alteration styles on the Property. Sericitic alteration is present in all rock types; quartz-sericite schist (MSS) units are derived from the quartz-eye gneiss and the metasedimentary rocks. Page (1995) correlated the sericitic alteration with moderate potassium enrichment and significant sodium depletion. Historic exploration and Treasury's 2008 exploration work show that the main alteration zone is defined by anomalous to strongly elevated gold and or silver concentrations, increased sulphide content (2-3% pyrite plus trace to 3% "sphalerite + galena ± chalcopyrite ± pyrrhotite ± arsenopyrite") and the presence of characteristic rock units (MSS and BMS) known to be prospective for gold and silver mineralization.

Treasury's detailed core logging, outcrop and trench mapping and examination of the geochemical data (assay and whole rock analyses) confirm that significant gold and silver mineralization on the Goliath Project is closely associated with fine grained sericite and K-feldspar-sericite-silica rocks (some exhibiting intensely bleached intervals). The hydrothermal alteration involved introduction of H_2O, S, K, CO_2 and the introduction or redistribution of silica, Ag, Au, Zn, Pb, As, Sb. The wall rock alteration tends to decrease in intensity with increasing the distance away from the central gold-silver mineralization. The strongly altered units occur within larger aureoles of sericitic-potassic and calc-silicate alteration which have approximate true thicknesses of ≥100 m and >300 m respectively, in the area of the 2008 exploration program

Treasury has classified alteration in drill core primarily by the visual observation and whole rock geochemistry results. Early in the 2008 drilling program, 756 samples were submitted for whole rock analyses, including all samples, top to bottom, from holes TL0801, TL0802, TL0807 and TL0808 and samples from 429 m to 441 m in hole TL0823.

Based on visual observations and whole rock analyses, sericitization, silicification, and chloritization are the most prominent and common alteration styles in all rock types in the Thunder Lake deposit area. Chlorite alteration is very widespread and frequently it is related to sulphide-bearing quartz veins, which parallel or crosscut foliation.

Treasury calculated Sericite and Chlorite Indices that were then plotted with gold and whole rock analyses on down hole plots in an attempt to determine a relationship between the gold bearing sulphide mineralization and alteration (Figure 7-13 to Figure 7-14). Description of the Sericite and Chlorite alteration discrimination indices are presented in Table 7-4.

Table 7-4. Various alteration discrimination indices.

Alteration Index	Element Ratios	Alteration Process	Source
Sericite Index	K20/(K20 + Na20)	replacement of feldspar by sericite	Saeki &Date, 1980
Chlorite Index	MgO + Fe2O3/ (MgO + Fe2O3 +2CaO + 2Na2O)	addition of Fe and Mg as chlorite	Saeki &Date, 1980

The following relationships were observed.

- The intervals with significant gold and silver mineralization are very strongly altered. Very often extensive pervasive hydrothermal alteration obscures primary textural and structural features to the extent that it's not possible to identify the original rock type. The host rocks are totally transformed, almost bleached. The hydrothermal alteration commonly involves massive depletion of CaO and Na2O and addition of H2O, K, silica, and sulphur as quartz ribbons and sericite. The feldspar and biotite are totally replaced by sericite, quartz and disseminated pyrite. Most of the mineralized zones are hosted by fine to medium grained quartz-sericite schist or in patches of sericite alteration in biotite-muscovite schist. The highest gold and silver values occur in the very strong pervasive quartz-sericite (Q-Ser) alteration. It seems that gold is distributed independently of pyrite, but an increase in pyrite and sphalerite content generally leads to an increase in the gold and silver content. Chalcopyrite and galena content does not appear to have a major effect on gold content.
- The chlorite alteration is more intense in zones of fractured and brecciated host rocks. As a result of the depletion of CaO and Na2O from the feldspar and addition of MgO and Fe2O3, sulphur and silica, quartz-pyrite-chlorite-tourmaline veins were formed. Very often old fractures are filled in with chlorite and disseminated pyrite.
- Complex, overprinting alteration and metamorphic assemblages and diverse metal associations are interpreted to be the result of a overprinting of hydrothermal and metamorphic fluids, which were focused in the zones of structurally-induced porosity/permeability.

The pervasive nature of hydrothermal alteration at the Goliath Project indicates that the hydrothermal fluids had circulated for an extended period of time. The spatial and temporal relationships between the different types of alteration encountered in the 86 Treasury drill holes and the structural control of the high grade zones support the Magmatic Hydrothermal Archaean Lode Gold Deposit (ALGD) genetic model.

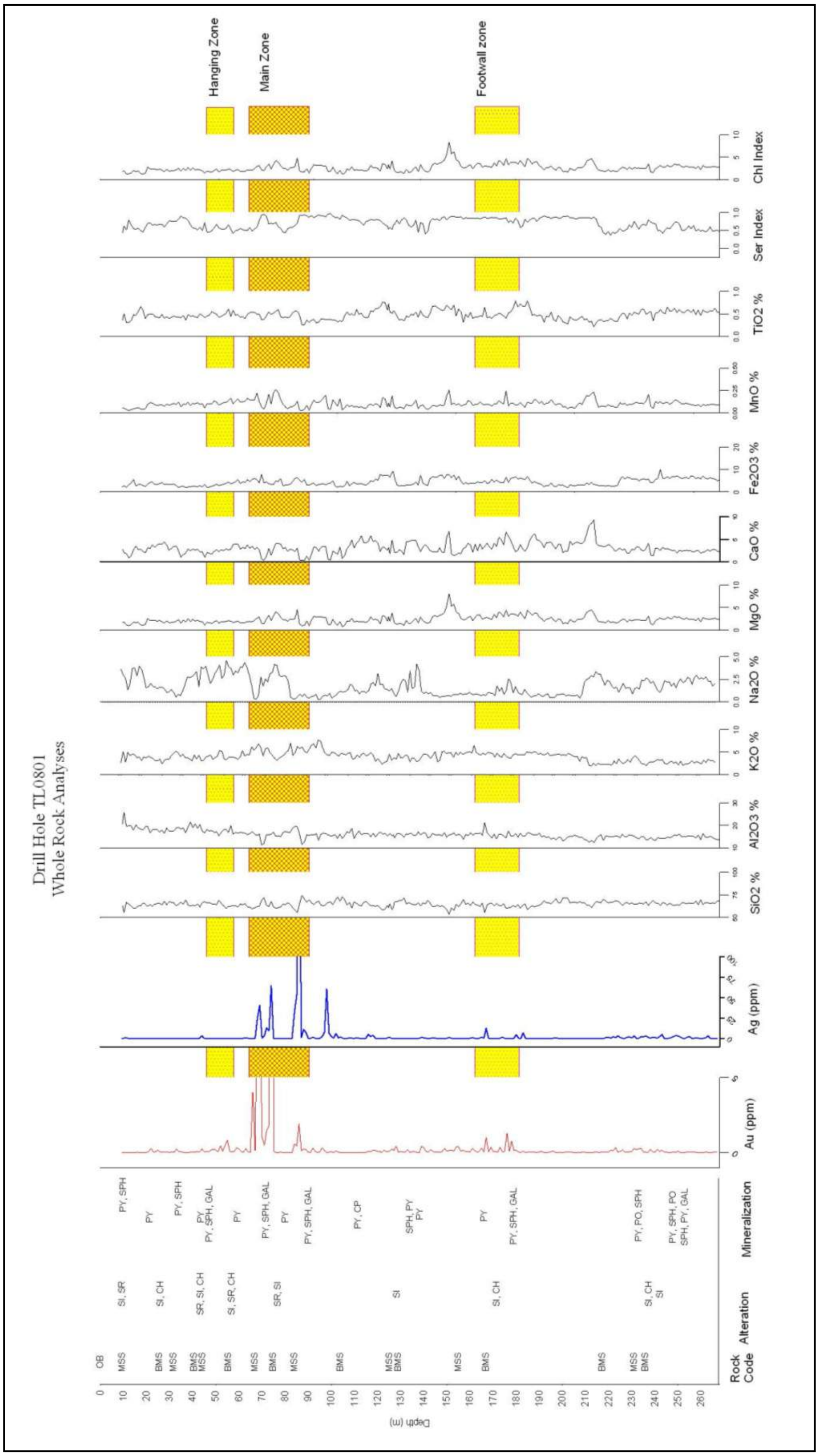


Figure 7-13. Whole rock analyses and the correlation with the Au-Ag mineralization for TL0801.

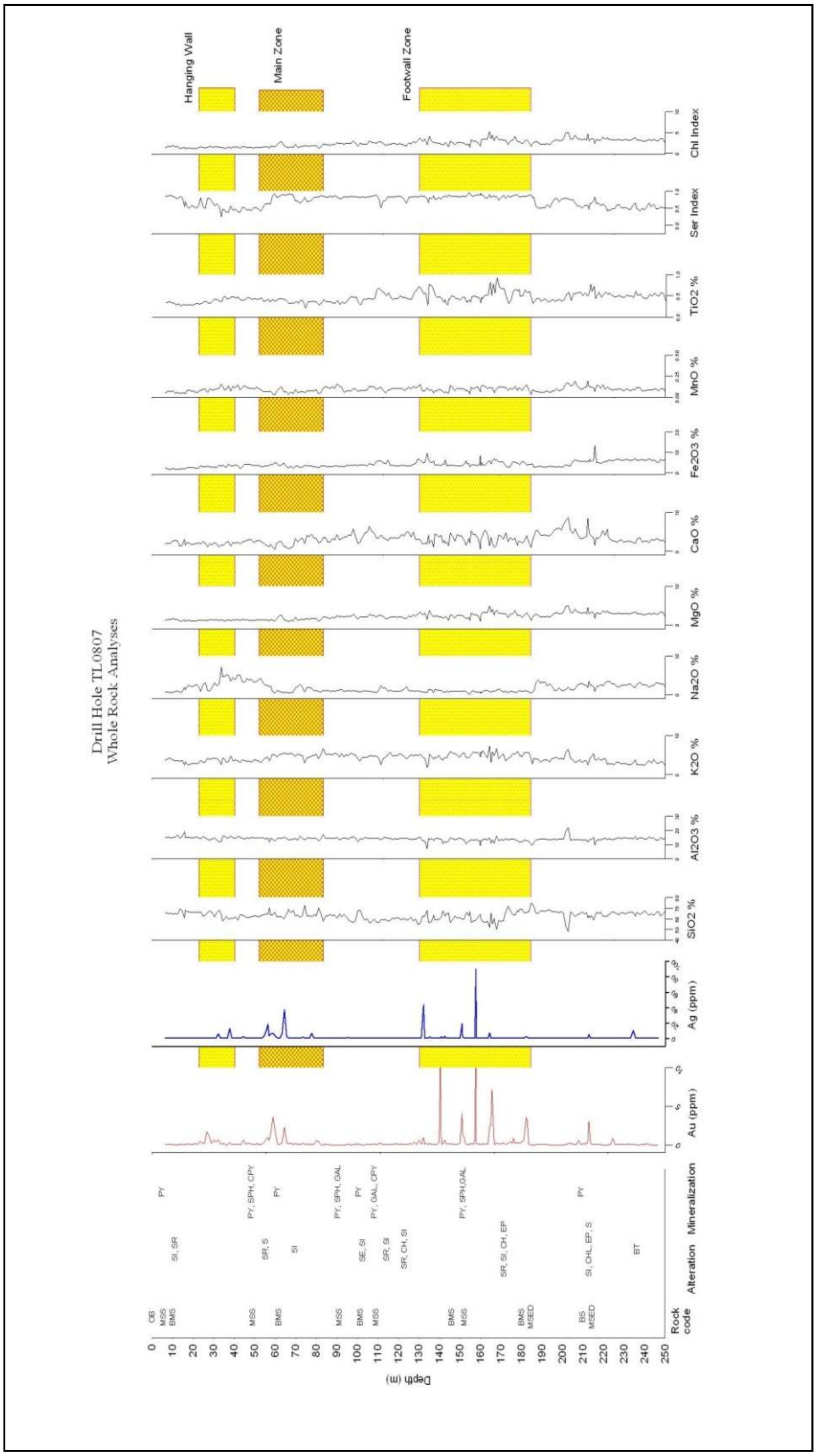


Figure 7-14. Whole rock analyses and the correlation with the Au-Ag mineralization for TL0807.

8 DEPOSIT TYPES

The Thunder Lake Deposit was described by Teck-Corona (2001) as a shear-hosted mesothermal gold deposit with structurally controlled gold mineralization related to local silica and sulphide replacements, and widespread, small, discordant to concordant quartz and sulphide veins. However, the deposit is missing most of the critical attributes of these types of deposits including the fact it is not hosted within a shear-zone, host rocks do not contain typical iron-carbonate alteration mineral assemblages, and gold is not commonly hosted by silicification and/or quartz veins (Beakhouse, 2002). Furthermore, the gold mineralization is generally associated with highly elevated silver (locally >100 g/tonne), zinc, copper, and lead. The gold mineralization is hosted by sulphide stringers and layers within felsic volcanic schist (Page, 1995), which is not common in shear-hosted mesothermal gold deposits.

Page (1995) describes the alteration of the host rocks in the area of the Thunder Lake Deposit as being enriched in potassium and depleted in sodium, which is a diagnostic feature peculiar to Volcanogenic Massive Sulphide ("VMS") deposits. On the basis of this "classic" alteration signature, along with the close association of gold with silver, copper, lead and zinc. Wetherup (2008) suggested that the Thunder Lake Deposit and other similar mineralization on the Thunder Lake Property might be part of a VMS system; specifically the Thunder Lake Deposit is better described as a preserved gold-rich VMS deposit, within a bimodal package of folded volcanic strata.

However, after considerable review of geochemical and geophysical data and field observations from the 2008 exploration program, and comparison of documented mineralogical, geochemical, and structural characteristics of well-explored deposits, Treasury's geological team favours the Magmatic Hydrothermal Archaean Lode Gold Deposit ("Magmatic Hydrothermal") model as the most promising genetic model to explain the geological features and mineralization of the Thunder Lake Deposit. Treasury notes that there is evidence for anomalous syngenetic gold (silver) mineralization that has been subsequently upgraded and overprinted by deformation and alteration events including the magmatic hydrothermal event. A short description of the Au-rich VMS Deposit Model is therefore also provided.

8.1 MAGMATIC HYDROTHERMAL ARCHAEAN LODE GOLD DEPOSIT MODEL

Treasury suggests that the most applicable genetic model for Thunder Lake Deposit is that of a magmatic-hydrothermal deposit, or a variation thereof, in which the ore metals were derived from temporally and genetically related intrusions. Large polyphase hydrothermal systems developed within and above genetically related intrusions and commonly interacted with meteoric fluids (and possibly seawater) on their tops and peripheries. Redistribution, and possibly further concentration of metals, occurred in some deposits during the late stages (Brimhall, 1980; Brimhall and Ghiorso, 1983).

Magmatic Hydrothermal Archaean Lode Gold Deposits (ALGD) are a variation of porphyry deposits temporally and spatially related to Archaean intermediate to felsic plutonic rocks. Magmatic Hydrothermal ALGDs developed exclusively in a post-arc setting and are typically distal from the magmatic systems that may be the source of the magmatic hydrothermal fluid (Figure 8-1). Although their geometry is quite variable, ALGDs tend to occur as veins or disseminated replacement style mineralization that defines a steeply dipping tabular or prolate elliptical geometry. ALGDs are characterized by diverse ore and alteration mineral assemblages, only a subset of which is similar to those characterizing Phanerozoic magmatic hydrothermal (porphyry) deposits. ALGDs occur in structures that are related to late, often regional scale, tectonic processes and not in pluton-centered hydrothermal breccia zones.

The Troilus disseminated gold and copper deposit in the Archaean Frotet-Evans greenstone belt of Quebec is an example of a Magmatic Hydrothermal Archaean Lode Gold Deposit. The host rocks consist predominately of mafic lavas and intrusives with lesser intermediate to felsic volcaniclastic metasediments intruded by numerous sills and dykes of felsic porphyries. Gold generally occurs as electrum and native gold. The gold occurs as discrete grains, from 20 to 4,000 microns in diameter, along sulphide grain boundaries, along fractures within the sulphides and along grain boundaries in small quartz veinlets. The mineralization contains two to three per cent sulphides. Sulphides are pyrite, chalcopyrite, pyrrhotite, and rare sphalerite. The sulphides form disseminations, tiny veinlets, and narrow semi-massive seams that are controlled by both foliation and fractures. The mineralization occurs within a zone of potassic altered in-situ brecciation at the margin of a mafic intrusive. Mineralization also occurs in felsic dykes cutting the zone.

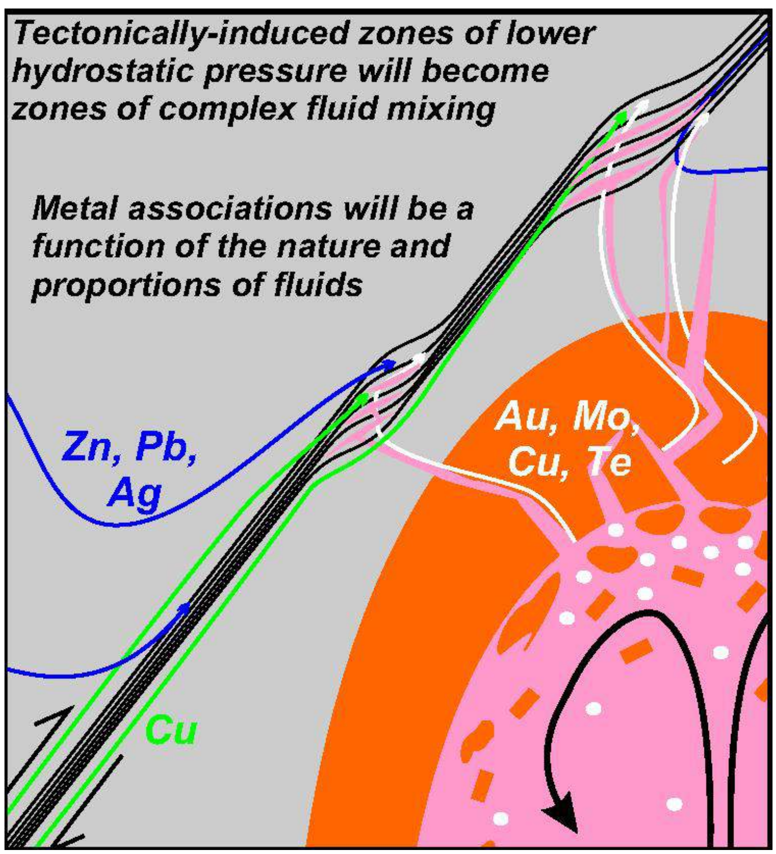


Figure 8-1. Idealized formation of magmatic hydrothermal Archaean lode gold deposit (after Burnham, 1979).

8.2 GOLD-RICH VOLCANOGENIC MASSIVE SULPHIDE MODEL

Gold-rich volcanogenic massive sulphide (VMS) deposits are a sub-type of both VMS and lode gold deposits (Dubé et al, 2006; Poulsen and Hannington, 1996; Hannington et al., 1999; Huston, 2000; Poulsen et al., 2000). Typical VMS deposits comprise a semi-massive to massive sulphide zone of concordant sulphide lenses underlain by a discordant stockwork system or feeder zone (Galley et al., 2007). An epigenetic gold-bearing event can be superimposed on this syngenetic VMS system resulting in gold-rich VMS mineralization (Dubé et al., 2006). Epigenetic gold-rich VMS deposits have gold grades exceeding the associated combined base metal grades. Distinct alteration features develop as a result of the epigenetic mineralizing event, including metamorphosed advanced argillic (aluminous) and silicic alteration, with this alteration focused in the region of the epigenetic stockwork. High-temperature (andalusite, kyanite, zinc-rich staurolite or Mn-garnet) or low-temperature (sericite, mica or chlorite) argillic minerals could be present, along with silicic alteration (quartz veins or quartz breccia zones). These alteration styles can be superimposed on the pre-existing syngenetic VMS alteration.

An example of gold-rich VMS deposits are the long producing world-class gold-rich VMS deposits of the Doyon-Bousquet-LaRonde district - Cadillac Mining Camp (e.g., Lapa Property and LaRonde Extension of Agnico-Eagle Mines Ltd.; Doyon Mine of IAMGOLD Corporation). Ravnaas et al. (2007) suggested that the “Zone 17 Gold Trend” of Rainy River Resources Ltd. is a potential example of this style of mineralization in northwestern Ontario.

Figure 8-2 provides a schematic section of the inferred crustal levels of formation of gold-rich VMS and shear-zone hosted environments and Figure 8-3 illustrates the geological setting and hydrothermal alteration associated with gold-rich (high-sulphidation) VMS hydrothermal systems.

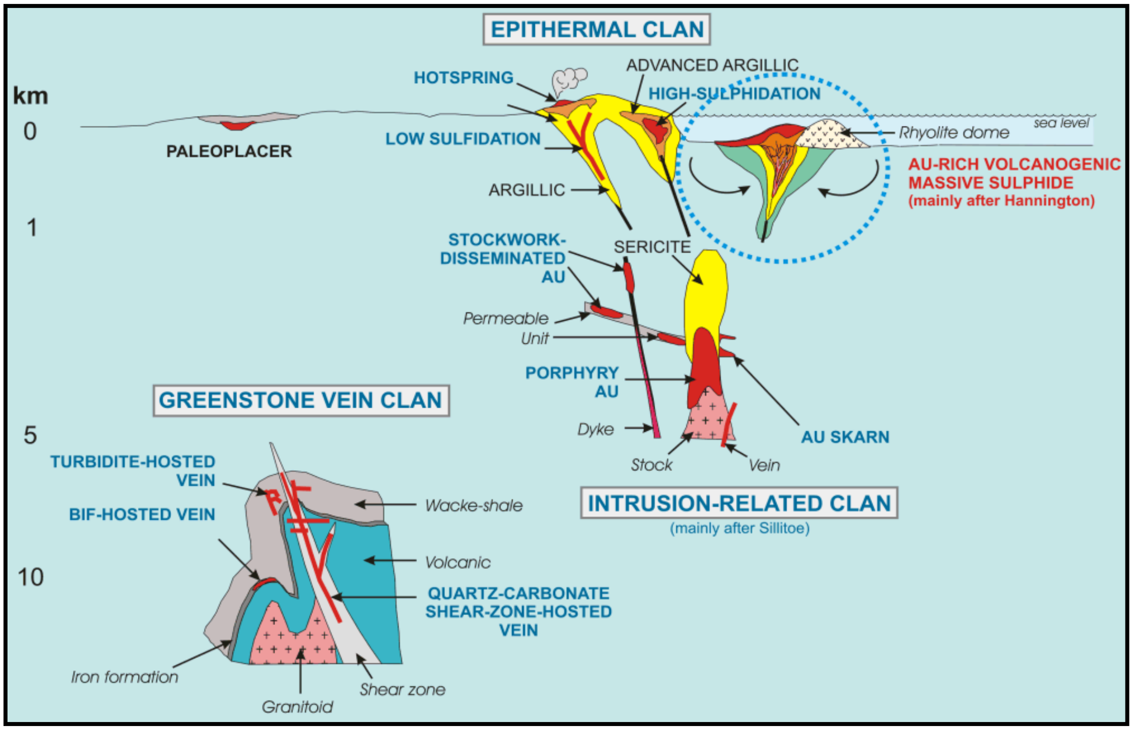


Figure 8-2. Various types of gold deposits and the inferred crustal levels of formation for gold-rich VMS deposits (Dubé et al., 2006).

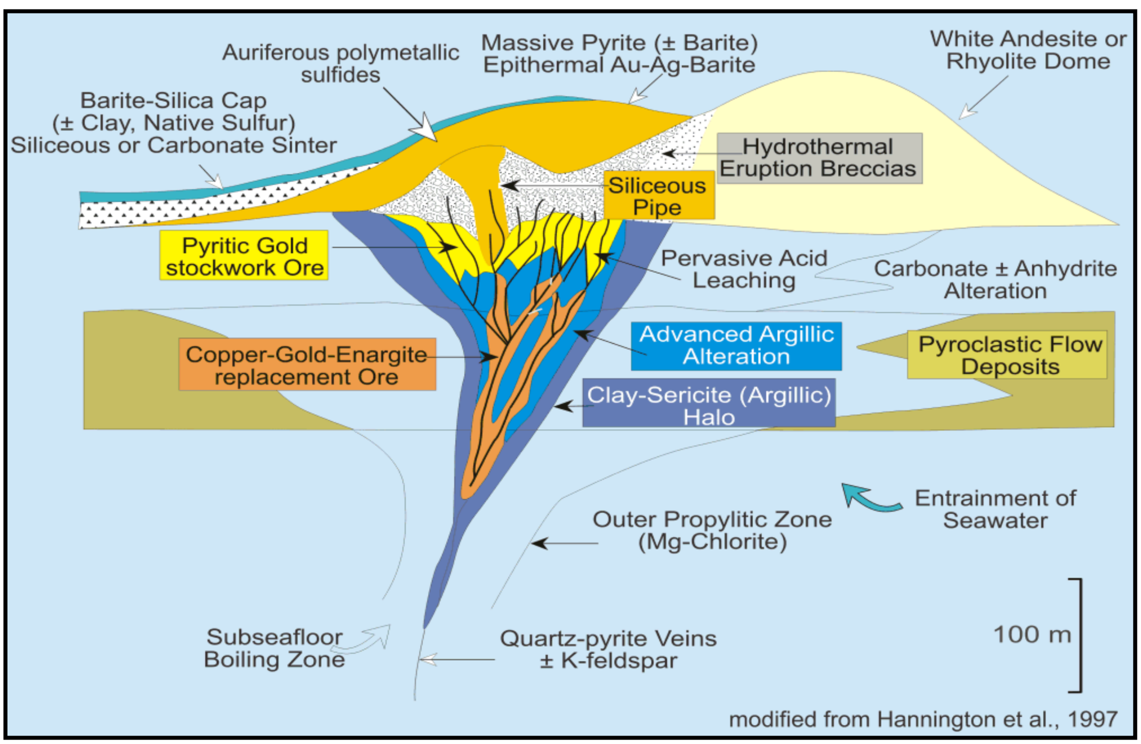


Figure 8-3. Geological setting and hydrothermal alteration associated with Au-rich high-sulphidation VMS hydrothermal systems (Dubé et al., 2006; Hannington et al., 1999).

9 EXPLORATION

Treasury's 2008, 2009 and 2010 exploration programs were conducted and managed by Caracle Creek International Consulting Inc. Canada (CCIC). Treasury personnel assumed field management of the Project's 2011 exploration program in February 2011.

9.1 HISTORIC CORE RECLAMATION

Drill core from Teck-Corona's previous exploration and resource delineation program and Laramide's exploration program was previously stored within a fenced and locked core compound directly across Highway 17 from the Pine Grove Motel in Wabigoon, Ontario, approximately 20 kilometres east of Dryden, Ontario. Approximately, one third of the Teck-Corona core (~8,000 boxes) was stored on metal racks and open to the elements. Many of the core boxes, stored outdoors and uncovered for over ten years, were rotted and required re-boxing before being moved or re-examined. The remaining core boxes (~16,000) were crossed-stacked on wooden pallets with approximately 100 core boxes per pallet. These stacks are poorly covered with core box lids and the boxes are in various states of decay. Despite the poor condition of the core boxes, almost no core had been spilled.

Treasury transferred the historic core intact to the Project office site/warehouses and much of the core should be available for re-logging and re-sampling.

9.2 2008 GEOLOGICAL MAPPING

Treasury's 2008 exploration program at the Goliath Project started in January with the establishment of a picket line grid to control mapping, sampling, trenching, and drilling. The mapping started in June 2008 and finished in August 2008. A base line was established along Norman Road, the border between the old Laramide and Teck properties. Cross lines were cut at intervals of 50 metres, 90° to the base line. Lines were chained and the picketed; picket stations were used by geologists, geophysical crews and drillers to locate, record and control their data. The 2008 grid consists of 30 lines at approximately 1,500 metres each, 11 lines at 1,225 metres, and 5 lines at 1,025 metres length. As work progressed at the Project, and as zones of particular interest were identified, those zones were mapped and examined in more detail. Geological mapping was done at 1:5,000 scale and the trench mapping and some of the outcrops were mapped at 1:200 scale.

Major lithological units were identified on the basis of visual identification of the rock type in outcrops, drill core and trenches (Figure 9-1). The rocks have been grouped into the Thunder Lake Assemblage and the Thunder River Mafic Metavolcanic rocks and are described in detail in Section 7.2.

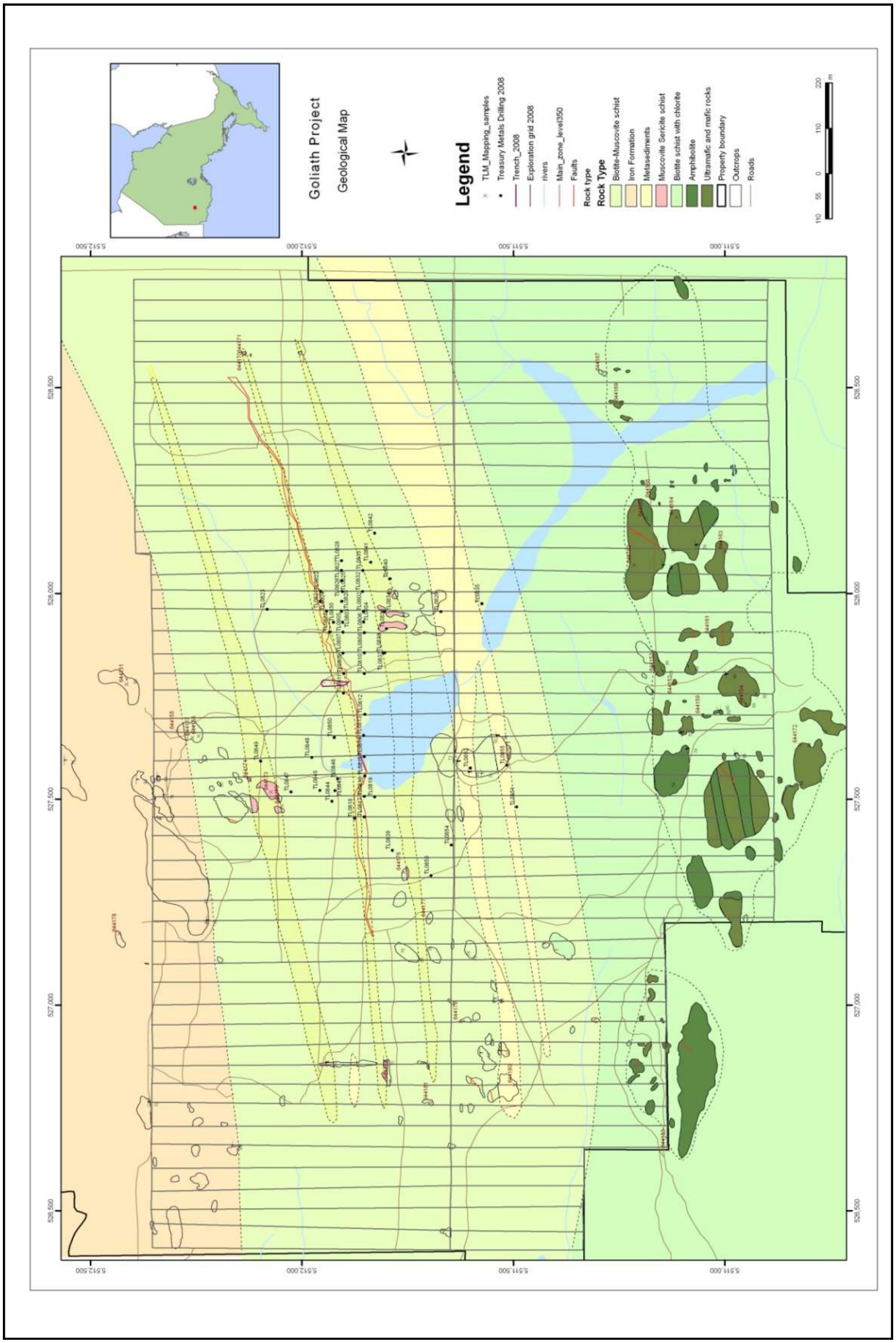


Figure 9-1. 2008 Geological Grid Map – Thunder Lake Deposit outlined in red.

A total of thirty-two (32) representative and grab samples were collected. Seventeen (17) samples were sent to Accurassay Laboratory in Thunder Bay, Ontario for Fire assay, whole rock and REE analyses. No significant precious or base metal contents were returned. Geological descriptions and analytical results are reported in Appendix B of Howe's 2008 Technical Report (Roy and Trinder, 2008).

9.3 2008 Structural Geology

The Project is within the Wabigoon Sub-Province and north of the Wabigoon Fault. Page (1994), Beakhouse (2001), Ravnaas et al. (2002) and Wetherup (2008) described three different generations of folds and three deformation events (Table 9-1). Structures observed in the Thunder Lake Deposit have been interpreted relative to this basic framework.

Additional structural data was collected during the 2008 drilling program. Oriented core drilling was implemented for the first time ever on this project for holes TL0822 to TL0837. The planar structures such as foliation, contacts, fault zones, and fold axes were measured using the EzyMark tool provided by Borinfo Ltd. The objectives of the oriented core drilling were to clarify the spatial relationships between the structural features and their influence on the mineralization.

Table 9-1. Summary of structural features observed on the Thunder Lake Property (Wetherup, 2008)

Event	Structure	Deformation	Vein	Description
D_0	S_0	Compositional layering of metavolcanic and meta-sedimentary rocks; argillic alteration zones (?)	V_0	Greyish, highly deformed, S_1 foliation parallel quartz-sulphide ribbons and silicification hosted by quartz-sericite schist
D_1	F_1 S_1	Isoclinal folding F_1 axial planar and layer parallel foliation/schistosity	V_1	White, deformed, locally cross-cutting quartz+/-tourmaline+/-sulphide veins
D_2	F_2	Closed (60°) folds; axial planes ~045/90; discrete, 5-40 m spaced, axial planes	V_2	Weakly deformed white quartz+/-sulphide veins along F_2 axial planes & at 45° to F_2 axial planes.
D_3	NW Fault	Brittle faults/fractures dip moderately NNE	V_3	Un-deformed white, non-planar quartz veins dip moderately NNE and cross-cut or follow foliation locally

9.4 2008 Exploration Trenching

In September 2008 a trench was excavated on the Property to expose the auriferous Thunder Lake Deposit "Main Zone" intersected by Treasury and historic drill holes. The objective was to cut a series of channel samples across the trench and obtain additional structural and geological information. The southern point of the trench is located at UTM 527782E 5511893N, NAD 83, Zone 15N. From this point the trench extends northward in an elongated oval shape. The trench is approximately 67 metres long and 14-15 metres wide at the surface and 5 m deep. The walls of the trench dip steeply inward and at the base of the trench the dimensions are approximately 46 metres long and 6-8 metres wide. A ramp was excavated at the southern end of the trench for easier access.

Two outcrops were successfully exposed; one, at the southern end of the trench, is approximately 12-13 metres long and 4-6 metres wide and the second, at the northern end of the trench, is approximately 4 metres long and 4 metres wide. A grid was established across the trench using rocks wrapped in labelled flagging tape. The base line of the grid runs north-south along the length of the trench with the 0+00N 0+00 BL origin point being located at the base of the decline at the southern edge of the trench; the origin point is located at UTM 527782E, 5511905N. From this point the grid was measured out in 2 metre increments towards the north and 2 metre increments east or west where necessary. The trench was then grid mapped at 1:200 scale and channel sampled.

A total of ten channel samples were cut across the two exposures and a total of 29 samples were collected from the channels. Seven channels were cut on the southern exposure and 23 samples were taken. On the northern exposure 3 channels were cut and 6 samples were taken. The channels were cut perpendicular to strike and were staggered sequentially from Channel 1 at the southern most exposed bedrock north to Channel 10 at the northern most point of the northern outcrop; Channel 1 began at coordinates UTM 527781E 5511905N (Appendix C map in Howe's 2008 Technical Report (Roy and Trinder, 2008).). The channels cut across all exposed outcrop within the trench. Each channel is approximately 4 to 5 centimetres wide and 5-6 centimetres deep. A blank or standard was inserted in alternating order at every tenth sample.

The mineralized zone was intersected in Channel 3 on the southern trench exposure. Sample 644111, located at the beginning of the channel, returned 1.15 g/tonne Au over 0.5 metres. Sample 644112 (0.65 metres) cut a "high grade zone "shoot" and yielded 27.55 g/tonne Au. A 1.5 metre lower-grade mineralized interval was also sampled in channel 5 where samples 644115, 644116 and 644117, each 0.5metres in length, returned 1.47 g/tonne Au, 2.74 g/tonne Au and 1.025 g/tonne Au respectively. A geological map, summary table of channel cuts and samples and table of assay results is presented in Appendix C of Howe's 2008 Technical Report (Roy and Trinder, 2008).

9.5 2008 AIRBORNE GEOPHYSICAL SURVEY

The airborne geophysical survey was designed to collect high-resolution magnetic data over the Goliath Project property. Flown in March 2008 by Firefly Geophysics, the survey totalled 309 line-kilometres over an area of 3064 hectares centred approximately 20 kilometres east of Dryden (Figure 9-2). Survey specifications are listed in Table 9-2.

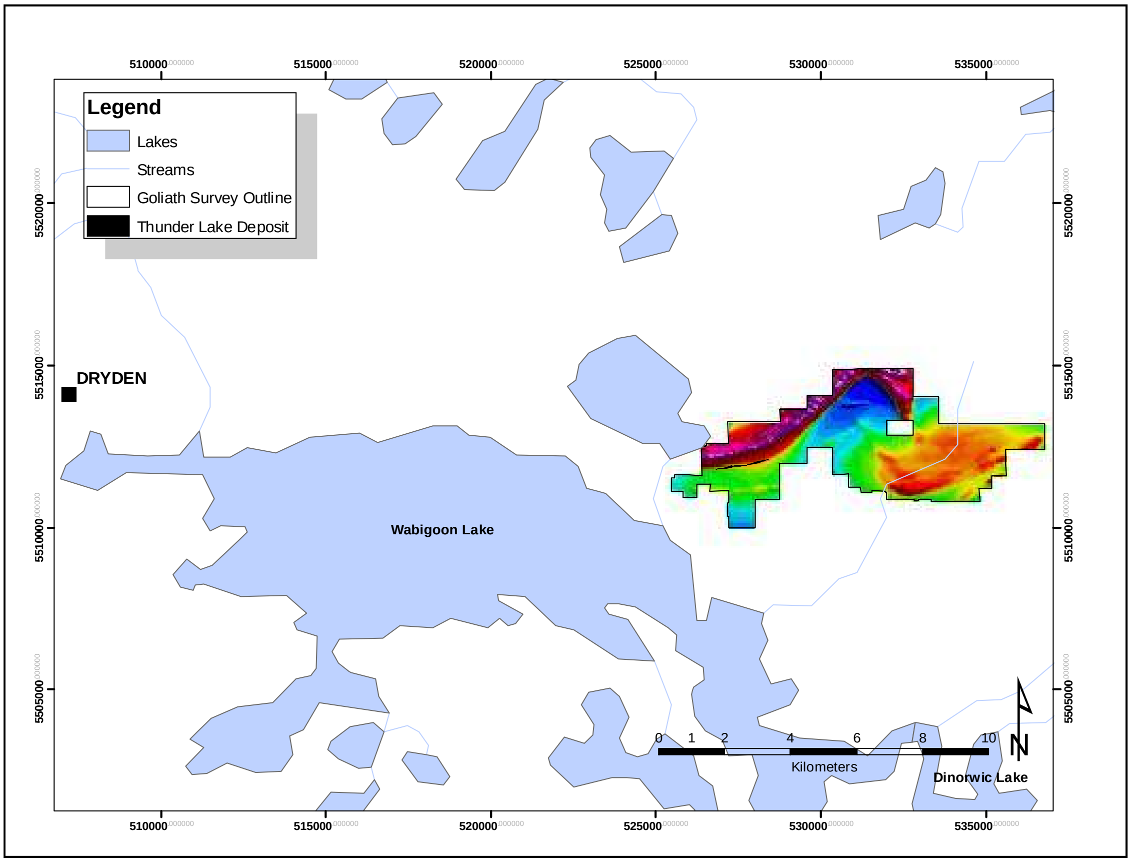


Figure 9-2. Goliath Survey Location Map.

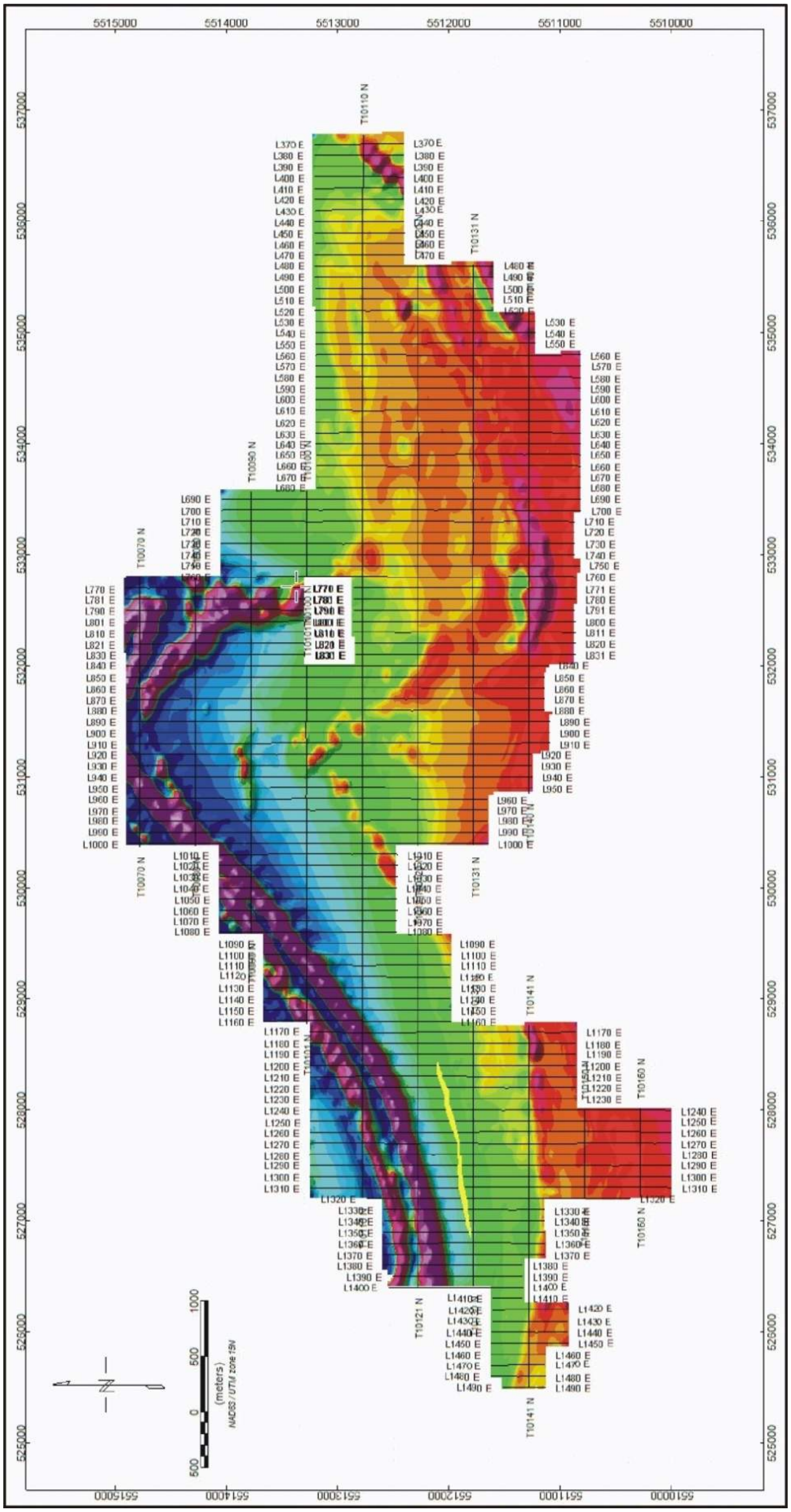
Table 9-2. Specifications of the Goliath Airborne Magnetic Survey.

Item	**Specification**
Company	Treasury Metals Inc.
Project	Goliath
Survey Name	Goliath Survey
Survey Type	Fixed Wing Magnetics
Platform	Single Sensor
Instrument	Geometrics G-822A Cesium Billingsley TFM-100G2 Fluxgate magnetometer
Flown By	Firefly Aviation
Aircraft	Piper Navajo PA-31
Date	March 2008
Line km	309 km
Area	3064 ha
Flight Height	60m
Sample Rate	10Hz
Nominal Speed	60m/s
Line Spacing	100m
Line Direction	000
Tie line Spacing	500m
Tie line Direction	090
Survey Base	Dryden, ON

Standard and enhanced gridding filters were applied to the Goliath survey data based on the calculated International Geomagnetic Reference Model (IGRF).

The surficial cover in the Goliath Project area is extensive with glacial deposit ranging from a few meters to over 40 metres thick. Glaciofluvial outwash covers approximately 70% of the Project area. Given this widespread surficial cover, magnetic data is of significant value to assist in identifying the regional bedrock geology and structure. The survey data exhibits the typical magnetic signature of a regional greenstone belt, which is expressed as a large, arcuate high/low magnetic sequence reflecting primary and secondary magnetite concentrations in the rocks and subsequent tectonic deformation, as can be seen in the north-west portion of the survey. The magnetic first vertical derivative image is shown below (Figure 9-3).

The Thunder Lake mineralized zone is not detected in the airborne magnetic data. Despite this, it is recommended to collect physical rock properties such as magnetic susceptibility and magnetic remnance and proceed with a constrained inversion of the data. This can help better understand the local geology and its relationship to the mineral deposit (Gordon, 2007).



Source (CCIC 2008)

Figure 9-3. First vertical derivative of the Goliath airborne magnetic survey, Thunder Lake Deposit is outlined in yellow.

9.6 2008 Ground Geophysical Surveys

Treasury contracted JVX Ltd. to complete a spectral induced polarisation (IP/Res) survey at the Project in March 2008. The survey coverage totalled 23 line-kilometres over 230 hectares, covering the Thunder Lake deposit and extending towards the west and south (Figure 9-4). The survey instrumentation consisted of a Scintrex IPC-7 (2.5 kW) transmitter and Scintrex IPR-12 receivers. This receiver system allows operators to access each reading independently and make adjustments when necessary to ensure that the chargeability data is repeatable and that the spectral parameters are calculated properly.

The survey employed the pole-dipole array method, which varies slightly from the dipole-dipole array. The pole-dipole method begins with a current separation of 25 metres and increases in spacing which results in higher currents in later dipoles, lowering the recorded noise. However the IP response is asymmetric. The array orientation must be taken into account during interpretation. The array separation collection ran from 1 to 8 (n=1 to 8). Although “deep cuts” (a=25 metres, n=9 to 16) were planned to image depths of 300 metres to 400 metres, time and weather constraints did not allow for the data collection.

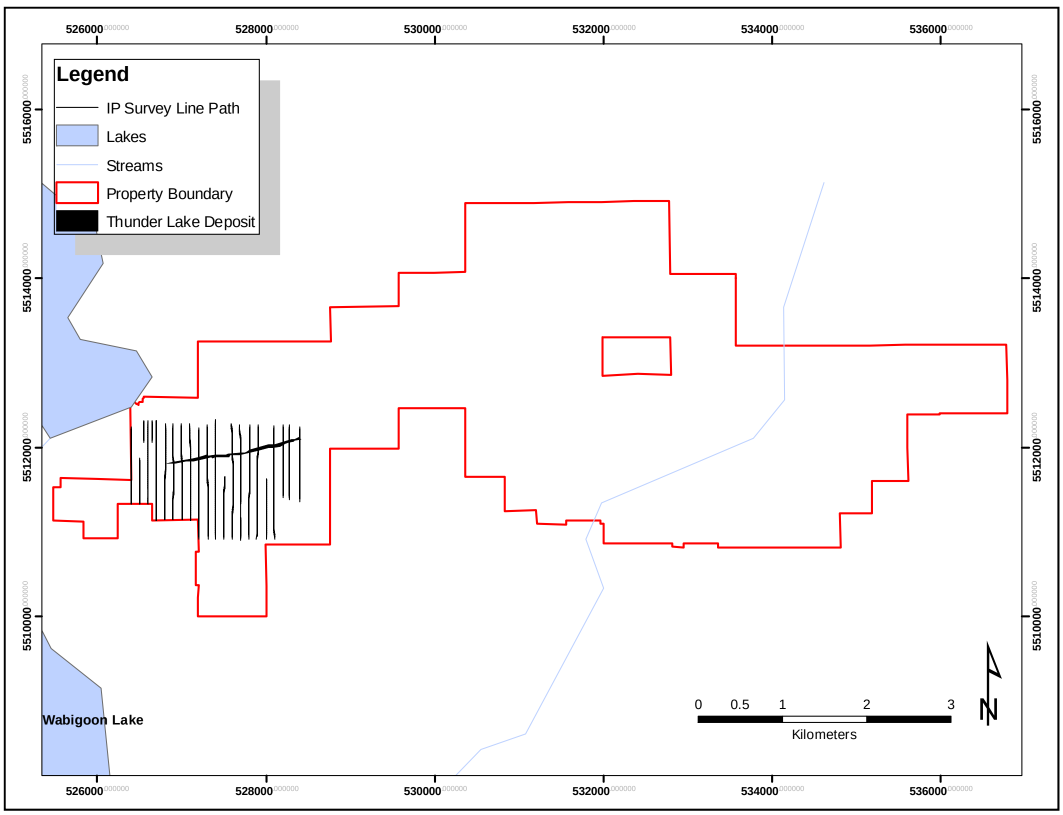
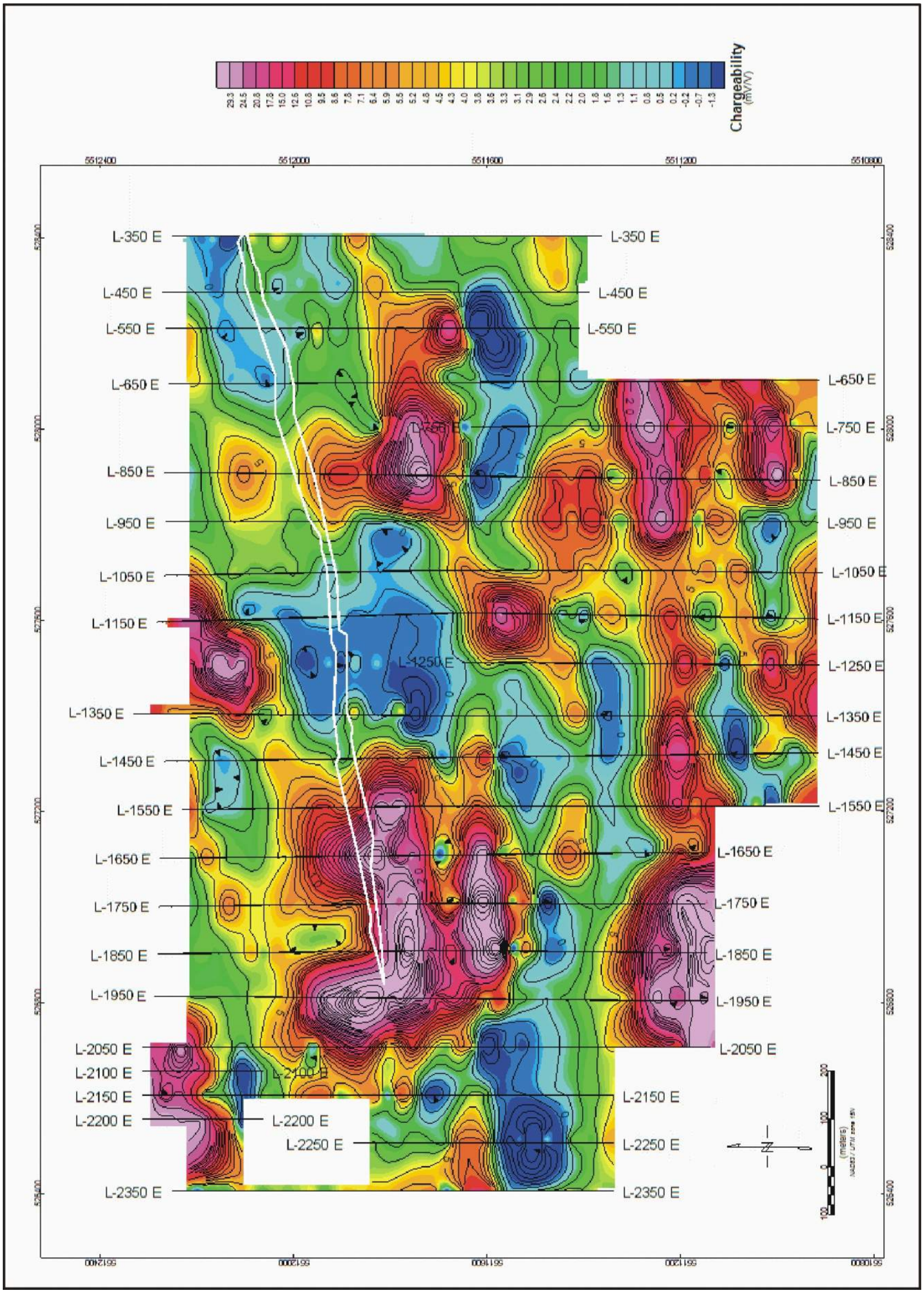
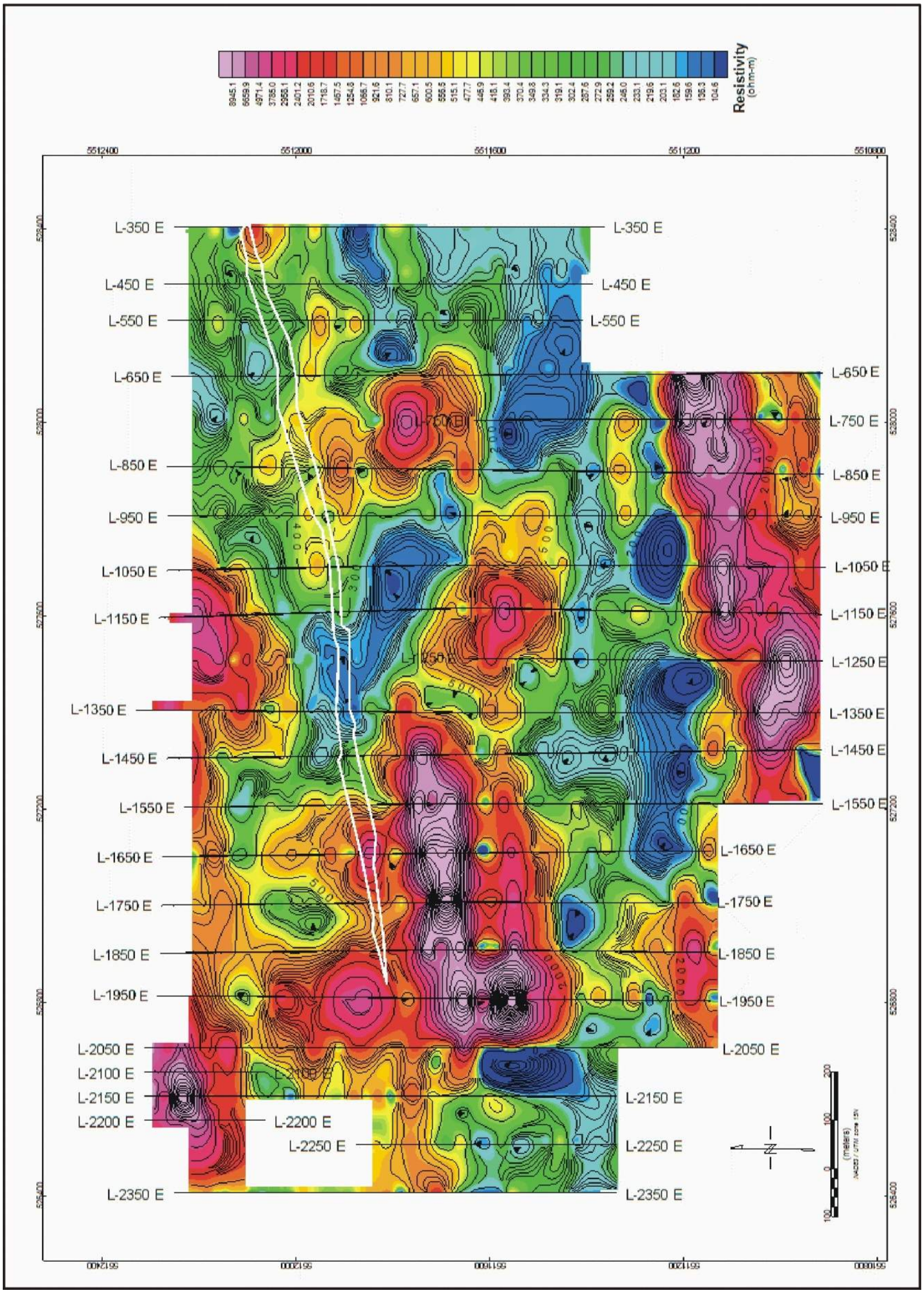


Figure 9-4. Location Map of IP survey on the Goliath Property.



(Source: CCIC 2008)

Figure 9-5. Chargeability (n:2) map, Thunder Lake Deposit outlined in white.



(Source: CCIC 2008)

Figure 9-6. Resistivity (n:2) map, Thunder Lake Deposit outlined in white.

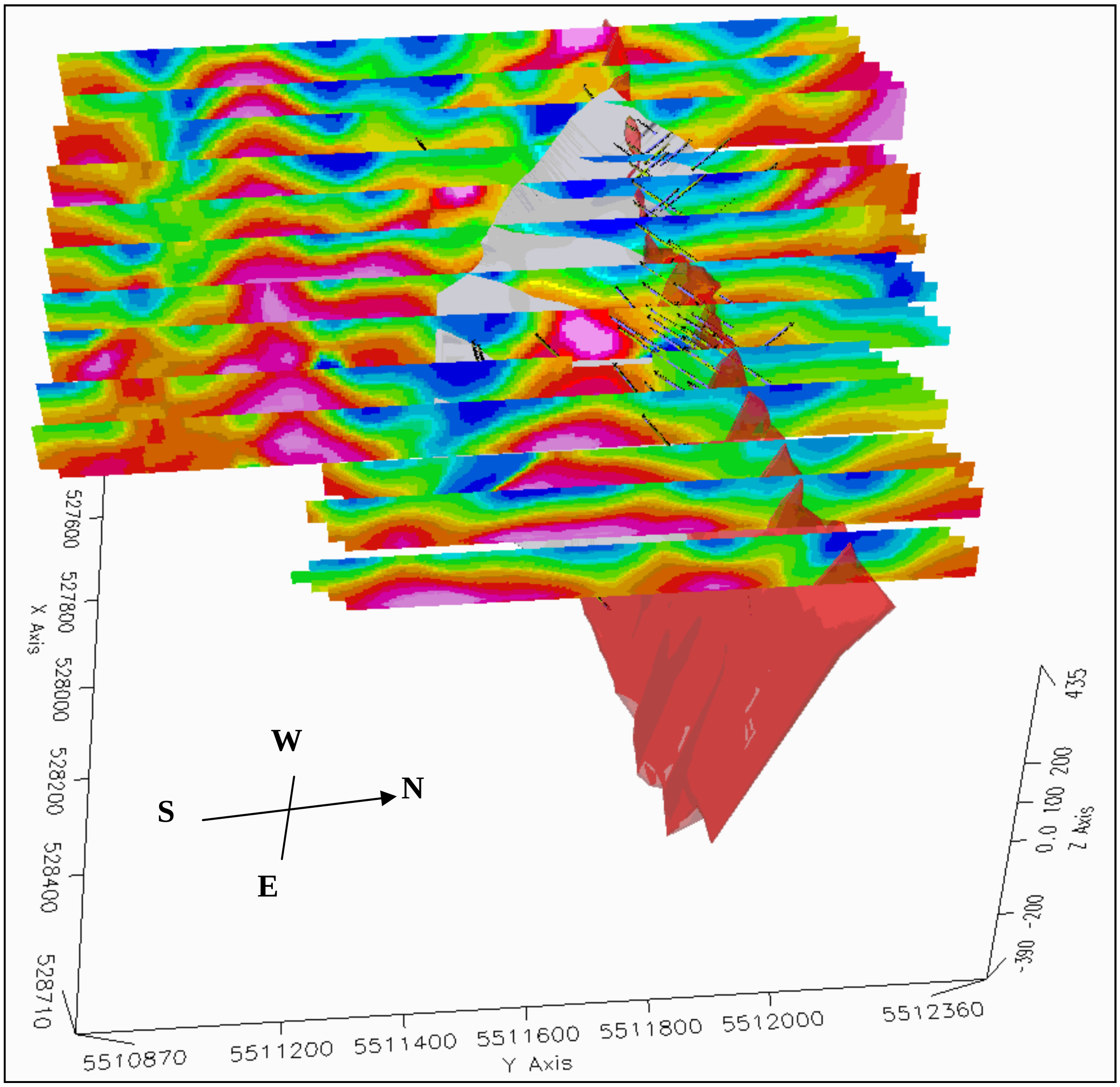
The survey detected extensive conductive surficial overburden, with 43% of the survey area at 250 Ωm or less. Conductive overburden can mask chargeable bodies, and thus a large volume of ore or high volume percentage of metallic sulphides must be present to overcome this problem. However, JVX noted that despite the conductive overburden responses detected, the overburden conductivity was not as high as initially anticipated (JVX, 2008).

The Thunder Lake deposit shows a weak resistivity high in isolated locations. It is likely that there is too much conductive overburden or the volume of mineralization and/or volume-percentage of conducting metallic sulphides may be low.

Four pseudosection products were generated by JVX and supplied as final products: chargeability ((Source: CCIC 2008) Figure 9-5), apparent resistivity ((Source: CCIC 2008) Figure 9-6), spectral MIP, and spectral tau. These products show a spatial coincidence between the northwest trending fault and low chargeability values. This appears to extend to the west-northwest ((Source: CCIC 2008) Figure 9-7. Treasury's and Teck's drilling indicates that alteration and gold mineralization extends in this direction. CCIC interpreted that the western extension of the Thunder Lake deposit may have been displaced to the west-northwest however it was not certain if the Main Zone has been intersected west of the resource and recommended that this area should be followed-up by several fences of diamond drilling to test the stratigraphy.

In addition to the potential target of a northwest fault-offset Thunder Lake deposit extension, that JVX identified, seven new exploration targets from the IP data as presented in Table 9-3. CCIC recommended that these targets also be drill tested.

Also, the IP data had not yet been inverted. CCIC recommended analysis of physical rock properties of both the mineralized and host rocks including resistivity, time-domain IP, and chargeability and, to then to proceed with a constrained inversion of the IP data. This would allow for a proper 3D integration of the data.



(Source: CCIC 2008)

Figure 9-7. 3D view of Chargeability sections, fault (grey) possibly extends to west-northwest. Mineralized zone in red

Table 9-3. Follow-up targets selected from 2008 Thunder Lake IP survey.

AnomalyID	Easting	Northing	Comments
TL_0001	526661	5512237	Cluster of strong IP anomalies at north end of lines 2050W, 2200W; Shallow; N1 resistivities are moderate to high; Short time constants - response of fine grained disseminated sulphides (+gold)
TL_0002	526908	5511224	Very strong, shallow IP anomalies 0 part of 300m long IP zone with weaker end members that may define an east/west IP zone that crosses entire grid; Coincident lower resistivities at depth may indicate a partial cause by bedrock conductors; Strong IP anomalies noted - masked by conductive cover - short time constants upgrading for gold target
TL_0003	527010	5511629	Stronger of two IP anomalies - lower resistivity at depth - possible bedrock conductor - time constant uniformly long
TL_0004	527009	5511705	Part of 400m long IP zone - may be on strike with Thunder Lake gold deposit; Moderate resistivity noted - possible bedrock conductor
TL_0005	527507	5512155	Two nearby strong, shallow IP anomalies 250m north of Thunder Lake. N1 resistivities are moderate. Some outcrop/subcrop and a prospecting history are likely. Time constants are long or mixed
TL_0006	528006	5511247	One of two strong IP anomalies south of the Thunder Lake deposit; Part of East-west trending IP/resistivity zones; Interpreted as probable bedrock conductors; This anomaly portion has short time constants and high resistivities - interesting for gold; N1 resistivity is high suggesting thin overburden
TL_0007	528006	5511021	One of two strong IP anomalies south of the Thunder Lake deposit; Part of East-west trending IP/resistivity zones; Interpreted as probable bedrock conductors

Coordinates: UTM Zone 15N – NAD83 Datum

9.7 2009 PROSPECTING AND SAMPLING

Between July 6 and August 4, 2009 a total of 19.5 days of general reconnaissance prospecting, outcrop sampling and channel sampling was completed by CCIC and Treasury personnel on the Project area. Work was completed on mining claims 4211250, 4211252, 3017936, 144570, 1119567, 1119562, 1119563, 1119564 and 1119555 and the Jones, Johnson and Wetelainen patents. Approximately 5 grab samples and 116 channel samples (34 channels) were collected and sent to Accurassay Laboratory in Thunder Bay, Ontario for fire assay and 32 element ICP analysis. Several channel samples returned encouraging assay results; in particular samples 59109 (20.519 g/tonne over 1 metre) and C156059 (2.138 g/tonne over 1 metre). Both samples are located several hundred meters from the defined resource area and may represent extensions of the mineralized zones defined in the resource area.

9.8 2010 GROUND GEOPHYSICAL SURVEYS

Treasury contracted CCIC to complete a downhole direct current induced polarization (DCIP/Resistivity) survey at the Project in the spring of 2010. The program was completed over twenty-four field days. The survey design consisted of sixty boreholes profiled for vertical resistivity/chargeability and ninety-four borehole-to-borehole tomography images between bores up to 150 m separation (Figure 9-4). There were four surface lines with twenty-one surface-to-borehole tomography pairings.

The survey instrumentation consisted of a new IP/resistivity technology, EarthProbe, which integrates surface, borehole, and borehole-to-borehole subsurface imaging into one system. The EarthProbe survey method deploys tightly spaced electrodes (5 m spacing) to a centralized data acquisition system that enables arbitrary selection of current and potential electrodes through relays. Rapid data acquisition and signal processing techniques allow for efficient use of conventional and non-conventional arrays and the removal of natural and cultural noise. The result is a high resolution DCIP system able to delineate both large resistivity/chargeability anomalies and narrow structural features down to depths of approximately 240 meters (CCIC, 2010a).

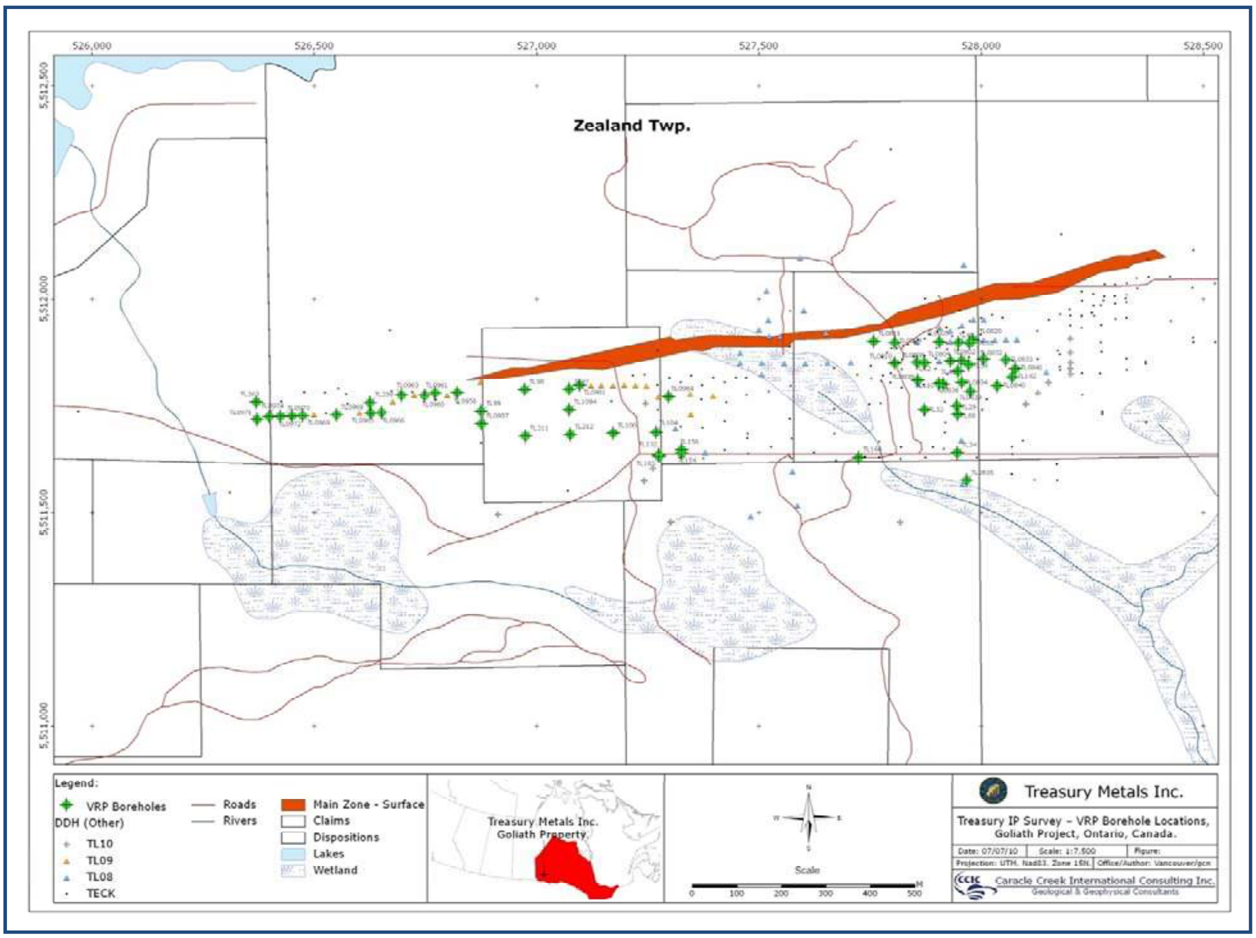


Figure 9-8. VRP (Vertical Resistivity Probe) and tomography locations.

Resistivity/Chargeability Correlations

CCIC (2010a) identified several resistivity/chargeability correlations from the DCIP survey:

- Mineralized zones exhibit low resistivity and high chargeability
- DCIP signatures differ between Main Zone and West Goliath extensional area
- Resistivity responses greater than 7,900 Ω.m (3.9 log Ω.m) reflect non-mineralized zones
- Resistivity responses less than 5,000 Ω.m (3.7 log Ω.m) reflect mineralized zones
- Chargeability responses less than 30 mV/V in the Main Zone and less than 50 mV/V in the West Goliath extensional area reflect non-mineralized zones
- Chargeability responses greater than 50 mV/V reflect mineralized zones
- There is an overlap of resistivity and chargeability response between the mineralized and non-mineralized zones in the Main Zone, suggesting that the occurrence of gold may be controlled by multiple factors (e.g. several alteration types) each having a unique IP signature.

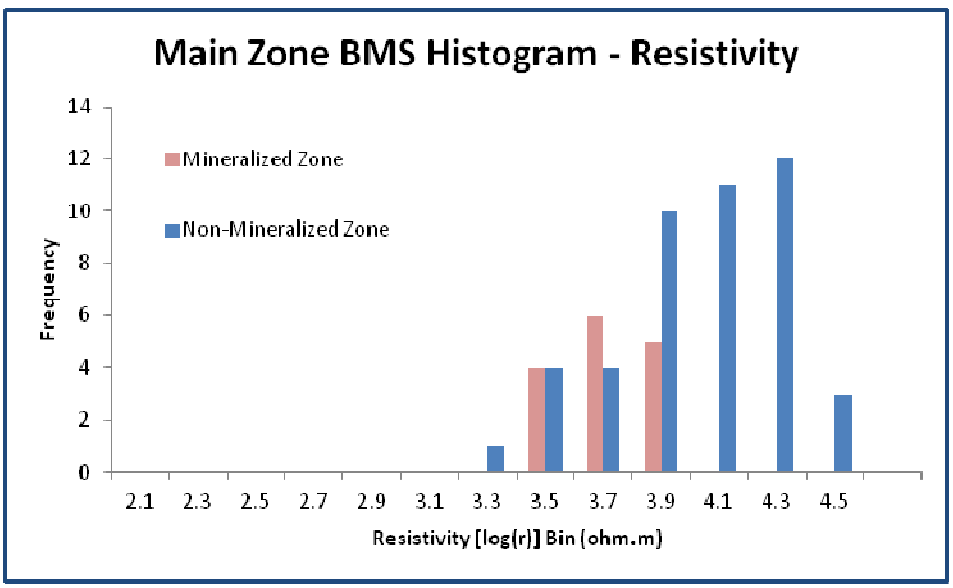


Figure 9-9. Mineralized vs. non-mineralized resistivity responses – Main Zone.

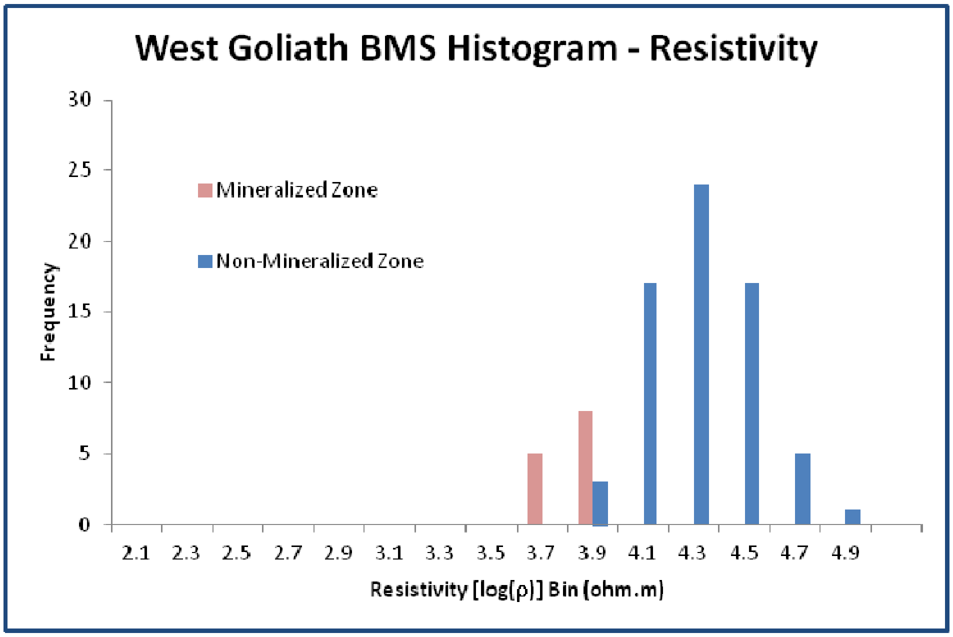


Figure 9-10. Mineralized vs. non-mineralized resistivity responses – West Goliath Ext.

<u>Mineralization Response</u>

CCIC (2010a) characterized three mineralization responses from the survey:

- Anomalous resistivity responses occur in association with mineralized zones that are greater than 4 m thick and exhibit a gold grade greater than 2 ppm.
- An anomalous resistivity response does not occur if the thickness of the mineralized zone is less than 2 m unless the intersection is in close proximity (less than 5 m) to a thicker mineralized zone.
- An anomalous resistivity response typically does not occur if the thickness of the mineralized zone is less than 4 m unless the gold grade exceeds 2 ppm and zinc exceeds 2000 ppm.

<u>Anomaly Summary</u>

CCIC summarized the anomaly findings as follows:

- Numerous in-hole and off-hole low resistivity responses were identified
- In the Main Zone:
    - A high level of electrical continuity existed between known gold intersections, suggesting mineralization continuity.
- Within the West Goliath extensional exploration area:
    - VRP and tomography results were well correlated with known mineralization zones, showing limited additional extent from previously drilled intersections. A

shallow conductor (50 – 70 m) was identified near TL0965, TL0966, TL0968, TL0969 and TL0972.
- Four low resistivity anomalies were identified from the surface survey. At least one of these anomalies was beyond the western extent of drilling at the time of the surveys.

Recommendations

The DCIP survey was not correlated to the alteration zones. CCIC recommended completing that correlation, as well as using the entire VRP and borehole assay dataset to characterize bulk resistivity/chargeability. CCIC suggested that it may be useful to put spatial resolution of resistivity responses into a format that can be overlain with the existing 3D model. CCIC also recommended drilling up to four IP anomalies in the West Goliath extensional area.

9.9 2010 TRENCHING

In summer 2010 a trench was excavated at the Project by CCIC at the Main Zone. Trenching exposed the Main Zone over a strike of approximately 42 metres centred at approximately UTM 527,800E 5,511,915N NAD 83 (Figure 9-11). The objectives were to cut a series of channel samples across the trench and obtain additional structural and geological information.

Four mappable units were identified within the trench based on the relative amounts of biotite-rich versus sericite-rich layers, quartz/silicification and the sulphide mineral content:

- Unit 1
  - Occurs on the southernmost exposures of the trench and consists almost entirely of ribbons of very fine grained quartz (almost cherty) with 1-2 mm wide layers of sericite; approx. 1% pyrite, increasing slightly with proximity to Unit 2.
- Unit 2
  - Contains significant sulphide minerals ~ 2-10% disseminated throughout and along foliation planes/parallel to layering and is darker in colour that Unit 1. Pervasive banding.
- Unit 3
  - Gradational unit between Units 2 and 4. Contains ~15 to 30% quartz-biotite-sericite schist layers with 2-3% disseminated pyrite
- Unit 4
  - Alternating layers of white and grey quartz-sericite schist layers and quartz-biotite-sericite schist layers with ~1-2% pyrite; the layering typically occurs as lenses or distended isoclinal fold hinges

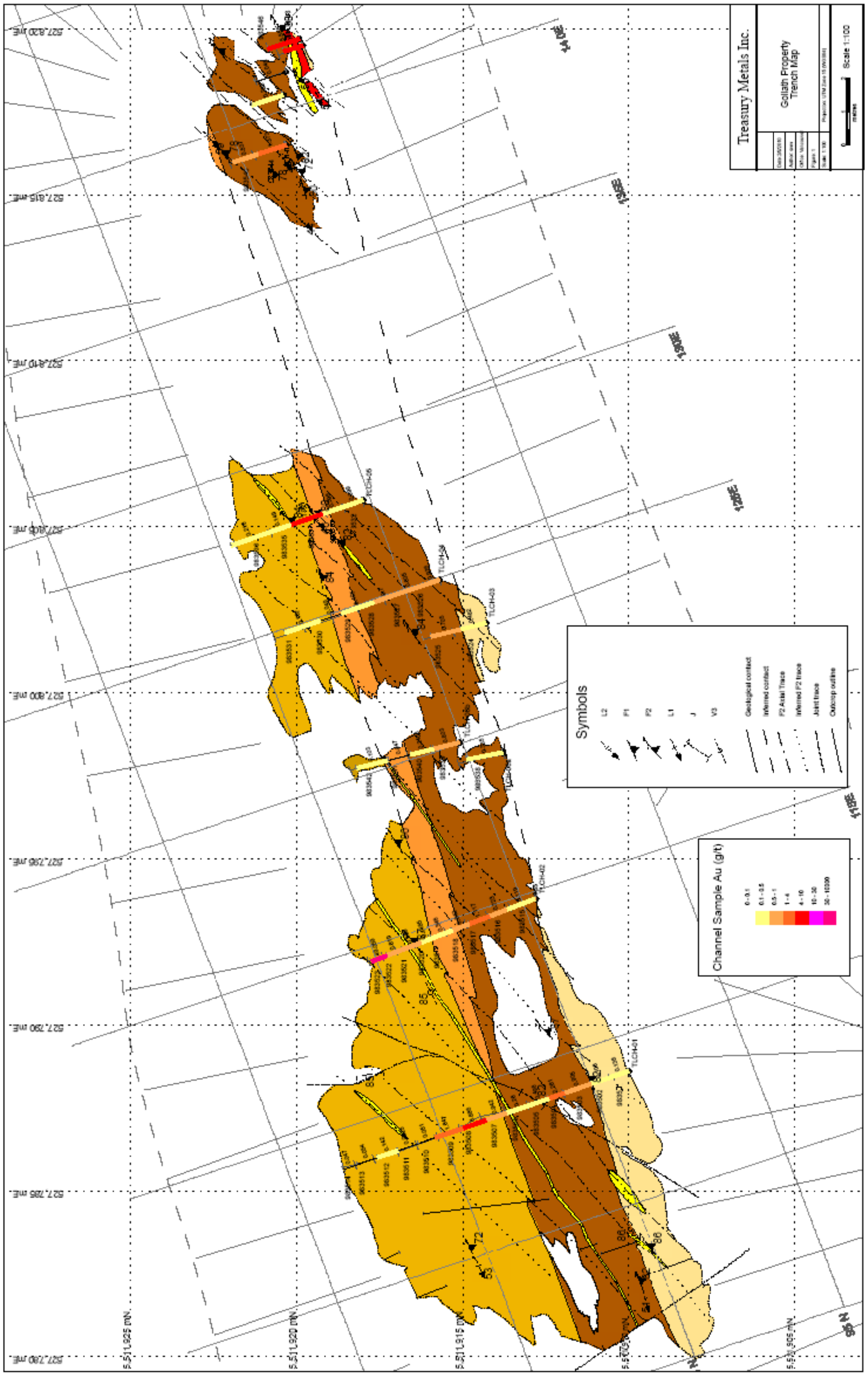


Figure 9-11. Geology and structural map of the 2010 Main Zone trench with gold assays.

Through detailed structural mapping, key controls on the gold mineralization were identified. The Company's structural geologist believes that the best potential for highest gold concentrations is likely at or near the F_1-F_2 intersections and in areas where there is an increased density of F_2 structures, resulting in the formation of high-grade shoots. Table 9-4 summarizes the structures observed in the Main Zone trench.

Table 9-4. Summary of structures in the Main Zone Trench (CCIC, 2010b).

Event	Structure	Description	Veins	Description
D_0	S_0	Compositional layering of meta-volcanic and meta-sedimentary rocks; argillic alteration zones (?)	V_0	White to grey, highly deformed, S_1 foliation parallel very fine grained qtz-sulphide ribbons and silicification with narrow sericite lamellae
D_1	F_1	Isoclinal folding	V_1	White coarse grained deformed, foliation parallel distended quartz lenses (rare)
	S_1	F_1 axial planar and layer parallel foliation/schistosity ~073/80		
	L_1	Stretching lineation, axis to isoclinal fold hinges; trend ~248, plunge 52^o		
D_2	F_2	Closed (interlimb angle 60^o) folds; axial planes ~052/83; discrete, 20 cm to 1.5 m spacing	V_2	Weakly deformed white qtz+/-sulphide lenses along F_2 axial planes.
	L_2	F_2 fold axes trend 228 and plunge 49^o		
D_3	J (?)	Brittle joints oriented ~162/81 and 032/82; possibly related to NW Fault	V_3	White un-deformed, planar crosscutting qtz-tourmaline+/-sulphide veins near vertical WSW striking.

The channel sample results demonstrate the erratic nature of high-grade gold zones within the Main Zone (CCIC, 2010b). There were forty-seven samples taken in total, plus two duplicate samples. Overall, samples from Unit 1 were generally low grade and average 0.19 g/tonne Au with a high value of 0.48g/tonne Au over 0.3 m. Unit 2 averaged 1.06 g/tonne Au with a high value of 5.55 g/tonne Au over 1 m, which came from a zone of semi-massive sulphide. Unit 3 averaged 2.11 g/tonne Au with a high value of 7.09 g/tonne Au over 1 m. Samples from Unit 4 averaged 2.99 g/tonne with a high value of 49.06 g/tonne over 0.55 m. Overall, Units 2, 3, and 4 average 1.9 g/tonne Au over the entire outcrop. The heterogeneity of gold within the trench is predicated upon the coincidence of higher grade lenses within the layering and F_2 fold axes. Both of these features plunge moderately westward and are the primary target for drilling to define higher-grade zones (CCIC, 2010b).

9.10 2011 Metallurgical Testwork

Treasury commenced initial metallurgical test work that followed up on the historical work performed by Teck Exploration Ltd. in 1998. This preliminary metallurgical program is presented in Section 13 (Mineral Processing and Metallurgical Testing).

9.11 2011 Airborne Geophysical Survey

A DIGHEM electromagnetic and magnetic airborne geophysical survey was carried out for Treasury over its Goliath property between July 14 and July 16, 2011. Total coverage of the survey amounted to 585.62 line-kilometres. The survey was conducted by Fugro Airborne Surveys of 2505 Meadowvale Boulevard, Mississauga, Ontario L5N 5S2. Survey specifications are listed in Table 9-5.

Table 9-5. Airborne geophysical survey specifications.

Item	Specification
Company	Treasury Metals Inc.
Project	Goliath
Survey Name	Goliath and Goldcliff Survey
Survey Type	Airborne Magnetic and DIGHEM Survey
Platform	Helicopter
Instrument	Fugro CF-1 cesium vapour magnetometer
Flown By	Great Slave Helicopters
Aircraft	A350-B3
Date	July 14-16, 2011
Line km	585.6 km
Flight Height	60m (helicopter) 35m (bird)
Sample Rate	10Hz
Nominal Speed	30m/s
Line Spacing	100m
Line Direction	**000**
Tie line Spacing	1,000m
Tie line Direction	**090**
Survey Base	Dryden Airport, ON

All the grids and maps were created with the Universal Transverse Mercator (UTM Zone 15N) coordinate system, NAD83 Datum.

Magnetics

Magnetic calculated vertical gradient (CVG) and horizontal gradient enhanced total magnetic intensity (HGETMI) maps clearly show define contacts of rock units and they are highly consistent with the known geological map. An iron formation with high magnetic responses shows as a banded belt in the western part of the property. The magnetic data also suggests that the assemblage units (including the iron formation) have been deformed.

The Thunder River Mafic Metavolcanic rocks, south of the metasediments, also show strong magnetic intensity in the southern parts of the property. Within the metavolcanic units there are bands of metasediments. These contacts are defined on the CVG map, in addition to other contacts not shown on the current geology map (Fugro, 2011).

Several breaks can be defined from the CVG and/or HGETMI maps based on the magnetic trend offsets or changes in strike direction. Two major alignments of the breaks in the Project area are NNW-SSE and NE-SW. Potential zones of structural deformation may warrant further investigation (Fugro, 2011).

Apparent Resistivity

Numerous cultural sources in the survey block had a detrimental effect on the apparent resistivity calculations, however 56kHz is not as severely affected as 7200Hz and 900Hz (Fugro, 2011). A NW-SE trending power line is better defined on the 7200Hz resistivity map and 900Hz data, than on the 56k Hz resistivity map.

Surficial resistive units in the area (UTM: 529500-532400E, 5512000-5514600N) are evident on the 56kHz map. However, the 7200Hz map and 900Hz resistivity data show more conductive features at depth and have better magnetic correlation in the same areas (Figure 9-12). This suggests that the deep conductive units are capped by superficial resistive units (Fugro, 2011). Similarly, in the east parts of the block (UTM: 534700-537000E, 5511100-5513400N) the 7200Hz map and 900Hz resistivity data show more conductive bedrock features.

Potential Bedrock Conductors

Fugro noted that potential exploration targets within the survey areas may be associated with quartz-rich units that contain moderate to no sulphide content, and which may be hosted by non-magnetic units that could be covered by either conductive overburden or resistive sand cover, therefore it is impractical to assess the relative merits of EM anomalies on the only on basis of conductance.

The majority of EM anomalous responses are of moderate to weak signal amplitude and they generally yield low conductance values of less than 5 Siemens (mhos). It should be noted that the calculated conductance values are based on the mid-frequency (5500 Hz) coaxial responses. These broad, poorly defined responses have generally been attributed to conductive overburden or flat-lying conductive metasedimentary layers, primarily on the southern half of the property (Fugro, 2011).

Numerous low resistivity zones were identified in the Project area. The 900 Hz resistivity deeper layer is generally more resistive than the surface layer, with the upper layer (56 kHz) showing larger variations, from conductive overburden and clay to resistive sand and gravel. Some of the resistive zones, however, are attributed to siliceous bedrock units near surface, or an absence of conductive cover (Fugro, 2011).

There were 987 anomalous EM responses detected in the survey block. Fugro (2011) indicated that nearly 69% of those responses are linked to conductive overburden or metasediments, about 7.5% are due to cultural sources, and approximately 23.5% are due to possible or probable bedrock sources.

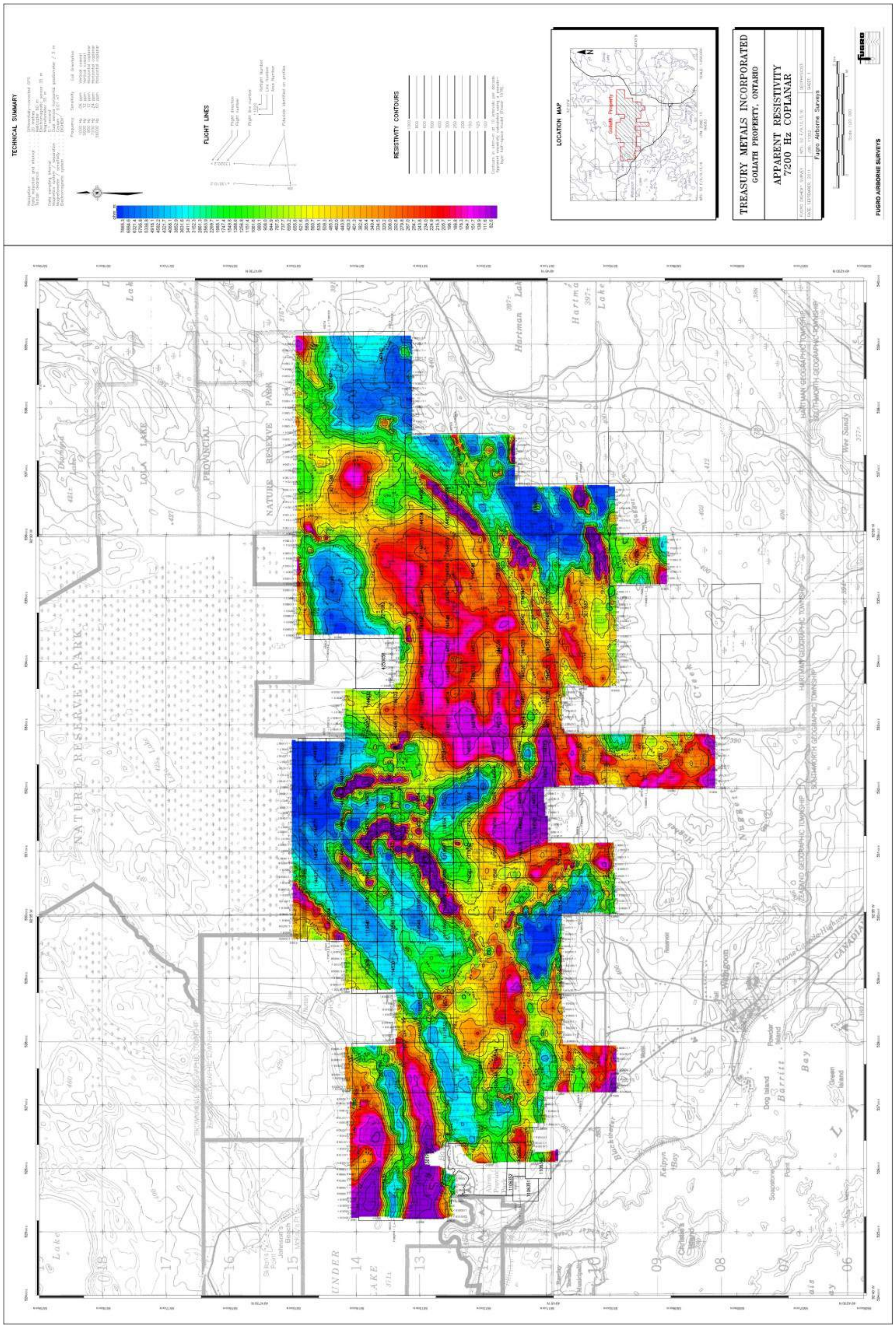


Figure 9-12. Apparent resistivity 7200Hz coplanar.

9.12 2008-2012 DIAMOND DRILL PROGRAMS

Treasury has completed five diamond drill programs on the Project. The programs are detailed in Section 10.

10 DRILLING

10.1 2008 DIAMOND DRILL PROGRAM

From 15 Feb 2008 to the end of September 2008, Treasury completed 55 diamond drill holes totalling 13,203.6 metres in its Phase 1 drilling program at the Goliath Project (TL0801 to TL0855). Diamond core drilling was NQ2 size only (50.6 millimetres). All holes were completed to their planned depths however hole TL08-35 required a restart after intersecting the Teck decline at a down-hole depth of 85m. Overall core recovery was excellent, averaging nearly 100%.

Diamond drill exploration efforts focussed on verifying results from historic Teck drilling, better definition of the lateral and down-dip extents of the Thunder Lake deposit and, in-filling areas of the historic mineral resource estimate.

G&O Drilling Ltd. drilled the first 37 holes and the rest were completed by North Star Drilling Ltd. CCIC personnel supervised the drill program. G&O Drilling Ltd. used a LY25 skid-mounted rig for all holes TL08-01 to TL08-37 except holes TL08-23, 29, and 35; a LY38 skid mounted rig was used for the latter. North Star Drilling Ltd. utilized a Zinex Mining Corp. A5 B20 skid mounted rig to complete holes TL08-38 to TL08-55. Both drill contractors operated two 12-hour shifts per day, seven days per week.

The drill contractors constructed drill access trails and drill pads. Drill water was supplied by pump from a local beaver pond.

Upon completion of drill holes, drill hole collar coordinates and elevations were surveyed by GPS instruments in UTM coordinates (NAD83). Early holes were surveyed utilizing a sub-metre Trimble survey instrument; later holes were surveyed using a handheld Garmin GPS. GPS coordinates were estimated to have an accuracy of approximately 4m or better. GPS coordinates of all collar locations were recorded and tied into the exploration grid. The drill contractors completed down-hole directional surveys on all diamond drill holes at approximately 50m intervals using a Reflex single shot digital survey tool.

The drill casing was left in each hole and capped to permit future down hole geophysical testing and/or deepening of the holes.

Core was retrieved from the drill string using conventional wireline techniques. Core was removed from the core tube by drilling personnel and carefully placed in core boxes. Filled core boxes were removed from the drill site at shift change by drilling personnel and brought to Treasury's secure core logging and sampling facility in Dryden. At the facility, the core was laid out on workbenches and cleaned prior to logging and sample interval marking.

Drill core and sample information were input into a digital database using portable computer workstations at the workbenches. The drill geologist logged the core and input a geotechnical core log including recovery and RQD and a descriptive log including rock type, alteration, structure, mineralization and vein density/percentage. Portable, hand-held XRF and magnetic susceptibility tests were conducted on core from selected drill holes. The geologist selected the sample intervals and marked the sample cut line on the core. Sample intervals were input into the drill database. The core was digitally photographed before sampling.

Oriented core drilling was implemented for holes TL0822 to TL0837. The planar structures such as foliation, contacts, fault zones and fold axes were measured using the EzyMark tool provided by Borinfo Ltd. The objective of the oriented core drilling was to clarify the spatial relationships between structural features and their influence on the mineralization.

Following core logging, the core was sampled as detailed in Section 11.3. At the completion of hole TL08-55, a total of approximately 11,808 core samples had been collected and sent to Accurassay laboratory in Thunder Bay, Ontario.

Digital assay files provided by Accurassay laboratory were merged with a “from” and “to” interval file created by CCIC, with the sample number linking the two files. This methodology limits data entry errors to sample numbering, as well as the “from” and “to” specifications; assay data re-entry errors are therefore be avoided. Sample numbering errors are identified during the merging process. Various mining and exploration software programs identify overlapping sample intervals.

10.2 2009 Diamond Drill Program

From October 19, 2009 to December 15, 2009, Treasury completed 31 diamond drill holes totalling 4,612.6 metres in its Phase 2 drilling program at the Goliath Project (TL0956 to TL0986). Diamond core drilling was NQ2 size only (50.6 millimetres). All holes were completed to their planned depths. Overall core recovery was excellent, averaging nearly 100%.

The Phase 2 program was designed to test shallow targets in the area west and along strike of Howe’s 2008 mineral resources in the Thunder Lake Gold Deposit. The majority of drill holes were collared at 25 metres intervals in order to target high-grade gold shoots and prepare for the expansion of mineral resources. Results from Phase 2 drilling confirmed the alteration and gold mineralized structure (moderate to strong quartz-sericite schist alteration with sulphide associated gold) extends for more than 650 metres west of the Thunder Lake Gold Deposit in an area where previous diamond drilling by Teck Resources Limited and Corona Gold Corp. intercepted anomalous (greater than or equal to 100 ppb Au) and higher grade gold concentrations. Twenty-two (22) of the thirty-one (31) drill holes intersected concentrations 3 g/tonne gold or higher

Distinctive Drilling Services Inc. of 2475 Dobbin Road #22, Ste. 706, Westbank, British Columbia V4T 2E9 was the drilling contractor. CCIC personnel supervised the drill program. Distinctive used two Zinex Mining Corp. A5 B20 skid-mounted rigs during the drill program. The drills were operated on two 10-hour shifts per day, seven days per week.

The drill contractor constructed drill access trails and drill pads. Drill water was supplied by pump from a local beaver pond.

Upon completion of drill holes, drill hole collar coordinates and elevations were surveyed by CCIC personnel in UTM coordinates (NAD83) utilizing a sub-metre Trimble survey instrument. Treasury’s 2008 drill hole collars were also resurveyed with the Trimble survey instrument. GPS coordinates of all collar locations were recorded and tied into the exploration grid. The drill contractor completed down-hole directional surveys on all diamond drill holes at approximately 50m intervals using a Reflex single shot digital survey tool.

The drill casing was left in each hole and capped to permit future down hole geophysical testing and/or deepening of the holes.

Core was retrieved from the drill string using conventional wireline techniques. Core was removed from the core tube by drilling personnel and carefully placed in core boxes. Filled core boxes were removed from the drill site at shift change by drilling personnel and brought to Treasury's secure core logging and sampling facility in Dryden. At the facility, the core was laid out on workbenches and cleaned prior to logging and sample interval marking.

Drill core and sample information were input into a digital database using portable computer workstations at the workbenches. The drill geologist logged the core and input a geotechnical core log including recovery and RQD and a descriptive log including rock type, alteration, structure, mineralization and vein density/percentage. The geologist selected the sample intervals and marked the sample cut line on the core. Sample intervals were input into the drill database. The core was digitally photographed before sampling.

Following core logging, the core was sampled as detailed in Section 11.3. At the completion of hole TL0986, a total of approximately 3,045 core samples (excluding QA-QC samples) had been collected and sent to Accurassay laboratory in Thunder Bay, Ontario.

Digital assay files provided by Accurassay laboratory were merged with a "from" and "to" interval file created by CCIC, with the sample number linking the two files. This methodology limits data entry errors to sample numbering, as well as the "from" and "to" specifications; assay data re-entry errors are therefore be avoided. Sample numbering errors are identified during the merging process. Various mining and exploration software programs identify overlapping sample intervals.

10.3 2010 DIAMOND DRILL PROGRAM

From February 20 to June 2, 2010, Treasury completed 27 diamond drill holes totalling 10,228 metres in its 2010 drilling program at the Goliath Project (TL1087 to TL11112). Diamond core drilling was NQ2 size only (50.6 millimetres). All holes were completed to their planned depths. Overall core recovery was excellent, averaging nearly 100%.

The diamond drill program was designed to test and delineate high-grade structures within the Main zone of the 2010 mineral resource area and further test the Western Extension. The program also assisted in verifying geophysical targets that were generated by surface and borehole induced-polarization surveys.

Distinctive Drilling Services Inc. of 2475 Dobbin Road #22, Ste. 706, Westbank, British Columbia V4T 2E9 was the drilling contractor. CCIC personnel supervised the drill program. Distinctive used two Zinex Mining Corp. A5 B20 skid-mounted rigs during the drill program. The drills were operated on two 10-hour shifts per day, seven days per week.

The drill contractor constructed drill access trails and drill pads. Drill water was supplied by pump from a local beaver pond.

Upon completion of drill holes, drill hole collar coordinates and elevations were surveyed by CCIC personnel in UTM coordinates (NAD83) utilizing a sub-metre Trimble survey instrument. GPS coordinates of all collar locations were recorded and tied into the exploration grid. The drill contractor completed down-hole directional surveys on all diamond drill holes at approximately 50m intervals using a Reflex single shot digital survey tool.

The drill casing was left in each hole and capped to permit future down hole geophysical testing and/or deepening of the holes.

Core handling, logging and sampling was completed as described in Section 10.2.

At the completion of hole TL10112, a total of approximately 2,957 core samples (excluding QA-QC samples) had been collected and sent to Accurassay laboratory in Thunder Bay, Ontario.

10.4 2011 Diamond Drill Program

From December 2 to 19, 2010 and January 17 to September 1, 2011, Treasury completed 117 diamond drill holes totalling 49,926.5 metres in its 2011 drilling program at the Goliath Project (TL10113 to TL11229). Diamond core drilling was NQ2 size only (50.6 millimetres). All holes were completed to their planned depths however 6 holes required restarting after the initial attempt failed. Overall core recovery was excellent, averaging nearly 100%.

The diamond drill program primarily focused on in-fill drilling to increase and upgrade a significant portion of the 2010 mineral resource from Inferred to the Indicated and Measured categories.

Distinctive Drilling Services Inc. of 2475 Dobbin Road #22, Ste. 706, Westbank, British Columbia V4T 2E9 was the drilling contractor. CCIC personnel supervised the drill program in December 2010 and January 2011 with Treasury personnel taking over supervision from February 2011 onwards. Distinctive used two to three Zinex Mining Corp. A5 B20 skid-mounted rigs during the drill program. The drills were operated on two 10-hour shifts per day, seven days per week.

The drill contractor constructed drill access trails and drill pads. Drill water was supplied by pump from a local beaver pond.

Upon completion of drill holes, drill hole collar coordinates and elevations were surveyed by Treasury personnel in UTM coordinates (NAD83) utilizing a sub-metre Trimble survey instrument. GPS coordinates of all collar locations were recorded and tied into the exploration grid. The drill contractor completed down-hole directional surveys on all diamond drill holes at approximately 50m intervals using a Reflex single shot digital survey tool.

The drill casing was left in each hole and capped to permit future down hole geophysical testing and/or deepening of the holes.

Core handling, logging and sampling was completed as described in Section 10.2.

At the completion of hole TL10228, a total of approximately 16,131 core samples (excluding QA-QC samples) had been collected and sent to Accurassay laboratory in Thunder Bay, Ontario. 16,313

10.5 Drill Data Utilized for the Mineral Resource Estimate

The Goliath Project drill hole database utilized in Howe's 2011 mineral resource estimate contains the results of Treasury's 2008 to 2011 diamond drill holes as summarized in Table 10-1.

Table 10-1. Goliath Project - Treasury 2008-2011 diamond drillholes in the mineral resource estimate.

Item	Value
Number of Drill Holes*	229
Total Length (m)	78,550
Average Drill Hole Length (m)	340
Maximum Drill Hole Length (m)	900
Minimum Drill Hole Length (m)	61
Maximum Drill Hole Inclination	87°
Minimum Drill Hole Inclination	45°
Average Drill Hole Inclination	59°
Holes With Down-Hole Surveys	229 (All)
Drill-Hole Assays	30,984

*Does not include aborted holes/failed first attempts

The Treasury drill holes intersected gold-bearing sulphide mineralization and returned significant assay results for gold, silver, zinc and lead. Drill holes were oriented at 360° or 180° azimuth, approximately perpendicular to the strike of the mineralized zone, which strikes approximately east-west. The mineralized zone dips 72° to 78° toward the south thus drill holes drilled at –45 to –87 degrees dip did not intersect perpendicular to the mineralization. True thicknesses of mineralization intersected are perhaps 75% to 90% of the apparent thickness of the mineralized core intercepts in shallower dipping holes and perhaps as low as 25% of the apparent thickness of the mineralized core intercepts in the steeply dipping near vertical holes.

The updated Howe resource estimation presented in Section 14 includes holes TL0801 to TL11228 of the Treasury's 2008 to 2011 drill programs in addition to historic Teck drill holes and underground samples.

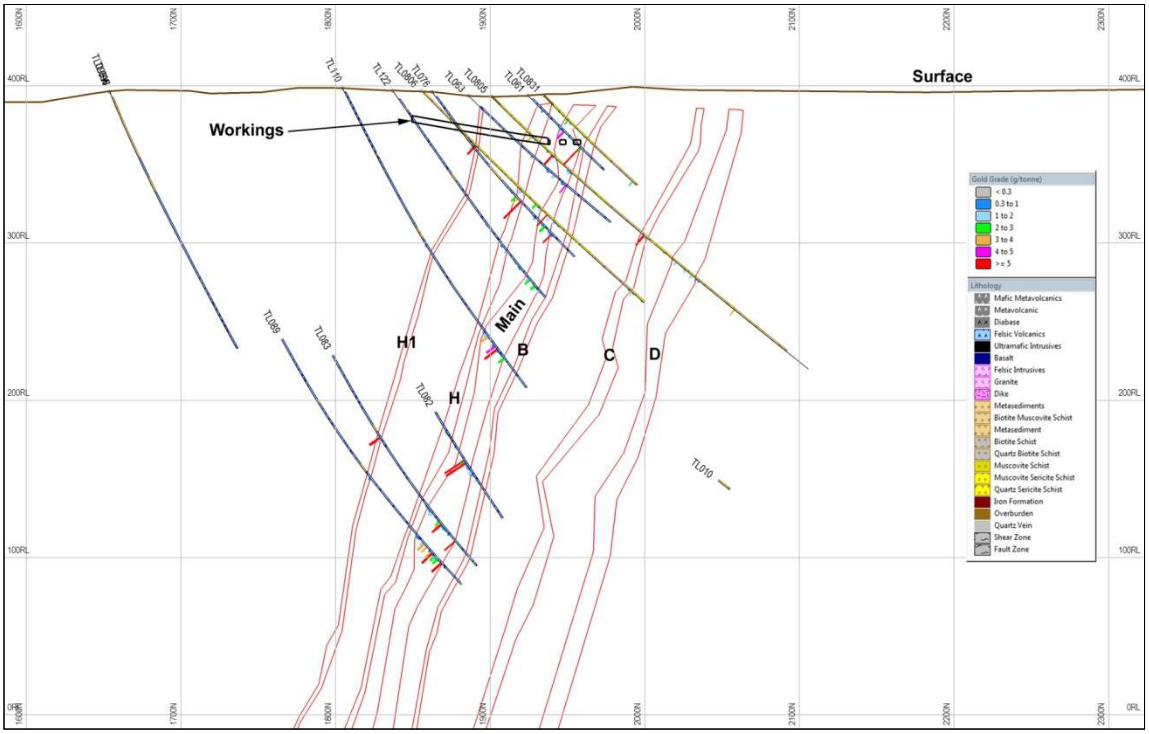


Figure 10-1. Cross-section 7900 m East – facing west.

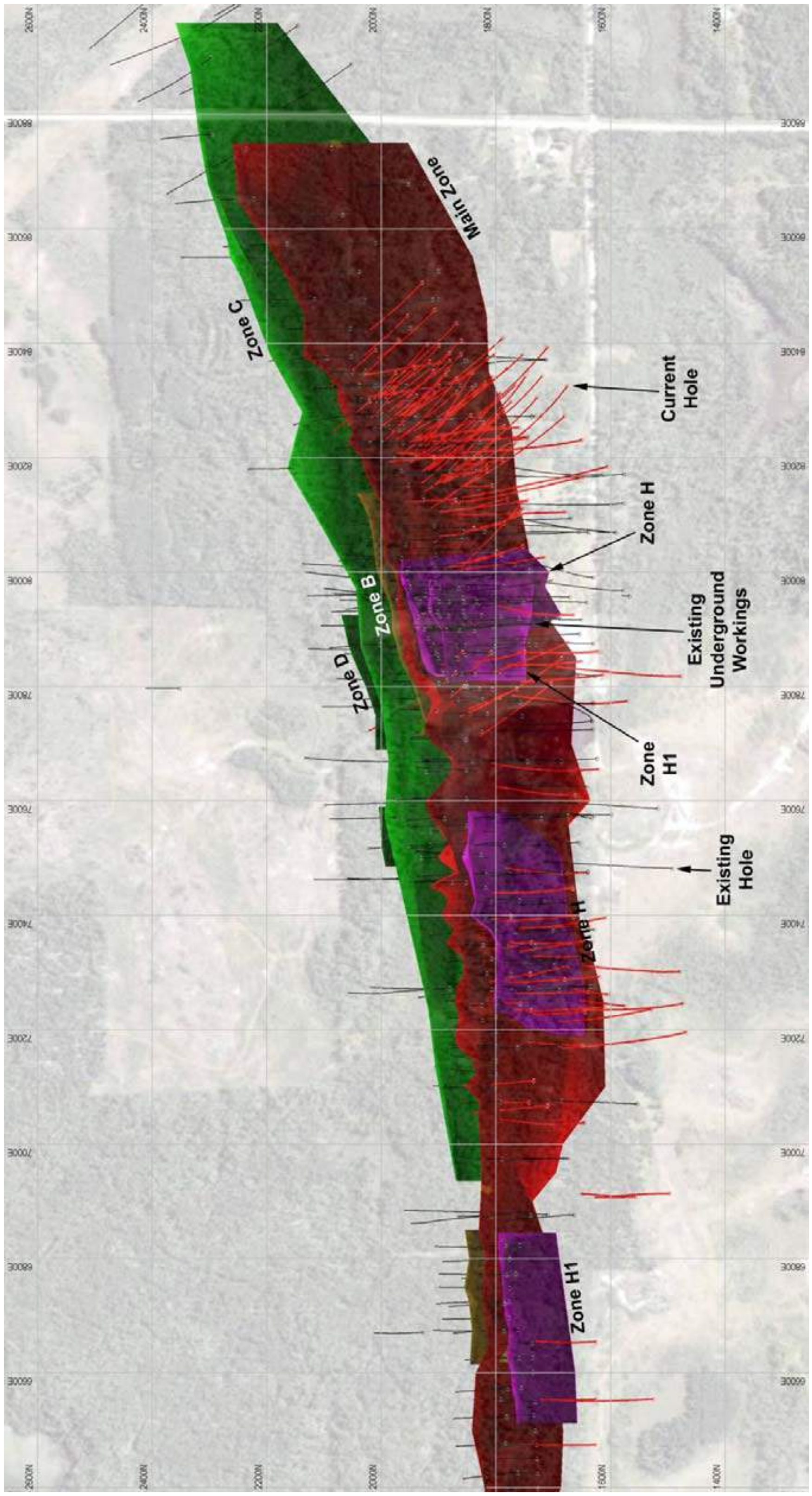
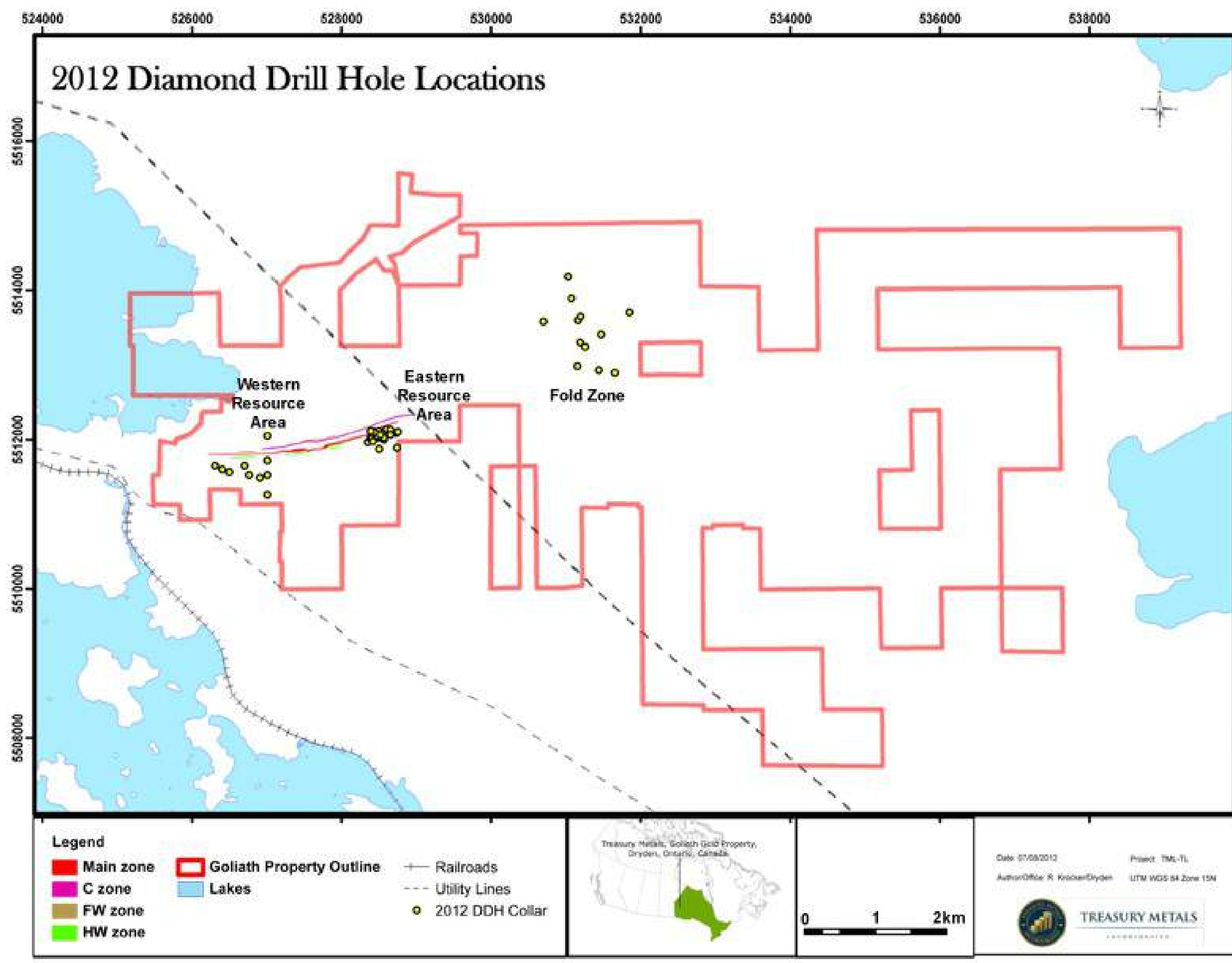


Figure 10-2. Plan view showing 2010-11 drill holes in red and older holes in black.

10.6 2012 DIAMOND DRILL PROGRAM

From January 25 to June 6, 2012, Treasury completed 48 diamond drill holes totalling 15,635 metres in two phases of its 2012 drilling program at the Goliath Project (TL12230 to TL12277). Additionally, 5 historical Teck Resources Inc. diamond drill holes were re-entered and extended for a total of 473 metres. The pre-existing Teck holes did not extend far enough to pass through the C-zone which was a target within this phase of drilling. Diamond core drilling was NQ2 size only (50.6 millimetres). All holes were completed to their planned depths however one planned re-entry hole was abandoned after it collapsed during the initial attempt. Overall core recovery was excellent, averaging nearly 100%.

The diamond drill program was designed to test a number of high-quality exploration targets outside of the already identified mineral resource. These include a possible new high grade shoot in the eastern resource area within the C-zone, the western resource area to add open pit grade material, and the NE fold zone to further explore where Teck's historic drilling has identified sporadic high grade gold mineralization (Figure 10-3).



Source: Treasury (2012)

Figure 10-3. Location of 2012 diamond drill holes

The reader should note that the 2012 drill data have not been incorporated into the mineral resource estimate upon which the economic analysis in this report is based (see Table 10-1). Selected result highlights from the 2012 program are shown below in Table 10-2.

As for the 2011 diamond drill program, Distinctive Drilling Services Inc. of 2475 Dobbin Road #22, Ste. 706, Westbank, British Columbia, V4T 2E9 was the drilling contractor. Treasury personnel supervised the program. Distinctive used two Zinex Mining Corp. A5 B20 skid-mounted rigs during the drill program. The drills were operated on two 10-hour shifts per day, seven days per week.

Upon completion of drill holes, drill hole collar coordinates and elevations were surveyed by Treasury personnel in UTM coordinates (NAD83) utilizing a sub-metre Trimble survey instrument. GPS coordinates of all collar locations were recorded and tied into the exploration grid. The drill contractor completed down-hole directional surveys on all diamond drill holes at approximately 50m intervals using a Reflex single shot digital survey tool.

The drill casing was left in each hole and capped to permit future down hole geophysical testing and/or deepening of the holes.

At the completion of hole TL12277, a total of approximately 7,972 core samples (excluding QA-QC samples) had been collected and sent to Accurassay laboratory in Thunder Bay, Ontario.

Table 10-2. Selected result highlights from the 2012 diamond drill program.

Western 'Resource' Area

Hole ID	Local Section	Zone	From	To	Intercept (m)	g/t
TL12-230	L17+50W	Main	119.2	124.18	4.98	1.82
TL12-232	L17+75W	C	440.5	442	1.50	3.83
TL12-234	L20+50W	HW	145.82	148.82	3.00	2.80
TL12-234	L20+50W	Main	172	177	5.00	0.92
TL12-235	L24+50W	C	199.18	202.5	3.32	1.05
TL12-236	L23+50W	HW	116	121.5	5.50	0.66
TL12-236	L23+75W	Main	229.5	234	4.50	0.45
TL12-237	L22+75W	HW	176	179	3.00	0.45
TL12-237	L22+75W	Main	298	300	2.00	1.08
TL12-238	L20+00W	Main	347.5	349.75	2.25	1.78
TL12-239	L18+50W	HW	317.28	319.94	2.66	0.73

Eastern 'Resource' Area

Hole ID	Local Section	Zone	From	To	Intercept (m)	g/t
TL12-240	L0+25W	FW	311	313	2.00	11.62
TL12-241	L0+25W	HW	157	165	8.00	0.49
TL12-241	L0+25W	C	461	463	2.00	3.72
TL12-242	L2+00W	C	289.3	293.4	4.10	0.76
TL12-243	L2+50W	HW	135	139	4.00	2.06
TL12-243	L2+75W	C	488	497.79	9.79	0.66
TL216-12RE	L3+75W	C	99	105.75	6.75	0.49
TL216-12RE	L3+75W	C	108	113	5.00	0.40
TL219-12RE	L4+25W	C	117.99	127.27	9.28	0.47
TL220-12RE	L4+50W	C	120.5	126.5	6.00	0.43
TL231-12RE	L4+00W	C	157	177	20.00	0.29
TL234-12RE	L4+75W	C	148.2	168.25	20.05	0.30
TL12-256	L4+00W	Main	146.6	155	8.40	0.74
TL12-256	L4+00W	C	228.75	243.5	14.75	0.40
TL12-257	L3+75W	C	186.29	194.5	8.21	0.55
TL12-258	L3+50W	Main	112.55	116.55	4.00	0.68
TL12-258	L3+50W	C	181.87	205.53	23.66	0.56
TL12-259	L3+25W	C	88.75	96.78	8.03	0.63
TL12-259	L3+25W	C	109.07	117.75	8.68	0.62
TL12-260	L3+00W	C	196.55	206.5	9.95	0.73
TL12-260	L3+00W	C	212.5	225.5	13.00	0.65
TL12-261	L2+75W	Main	146.34	150.75	4.41	5.40
TL12-261	L2+75W	C	211.5	226	14.50	0.63
TL12-263	L2+00W	C	158	168	10.00	0.32
TL12-264	L1+50W	C	159.9	162.6	2.70	0.76

Table 10-12. Selected result highlights from the 2012 diamond drill program cont'd.

TL12-265	L2+50W	C	115	123.5	8.50	0.72
TL12-265	L2+50W	C	142.55	145	2.45	0.50
TL12-267	L2+75W	HW	39.39	43.4	4.01	0.99
TL12-267	L2+75W	Main	106.5	111.25	4.75	1.94
TL12-267	L2+75W	C	219	226.35	7.35	1.39
TL12-268	L3+00W	C	113	120	7.00	3.44
TL12-269	L3+50W	Main	20	22.25	2.25	1.56
TL12-269	L3+50W	C	92.08	110.5	18.42	0.55
TL12-270	L3+25W	C	245.75	254.75	9.00	0.62
TL12-270	L3+25W	C	267	272.1	5.10	0.71
TL12-272	L0+00BL	FW BMS	240	241	1.00	2.76
TL12-273	L1+00W	Main	88.75	90.75	2.00	1.70
TL12-273	L1+00W	C	196.35	201.5	6.52	0.52
TL12-274	L1+00W	HW BMS	72	73	1.00	3.10
TL12-274	L1+00W	B	238.5	240.5	2.00	1.26
TL12-275	L1+75W	Main	178	181.5	3.50	0.59
TL12-275	L1+75W	C	278.3	286	7.70	0.52
TL12-276	L2+25W	Main	101.5	104	2.50	2.78
TL12-276	L2+25W	C	184.5	192.12	7.62	0.33
Fold Zone						
Hole ID	**Local Section**	**Zone**	**From**	**To**	**Intercept (m)**	**g/t**
TL12-244	Az 330 78	N/A	179.5	190	10.50	0.26
TL12-244	Az 330 77	N/A	399	400.5	1.50	2.95
TL12-245	Az 330 78	N/A	51	54	3.00	2.27
TL12-245	Az 330 79	N/A	201	204.4	3.40	1.50
TL12-246	Az 330 78	N/A	77.74	78.74	1.00	2.80
TL12-247	NE 77A	N/A	102	104	2.00	6.00
TL12-247	NE 77A	N/A	21	27	6.00	4.69
*includes	NE 77A	N/A	22.5	24	1.50	17.52
TL12-248	Az 330 72	N/A	171.85	180	8.15	0.39
TL12-248	Az 330 72	N/A	187.5	189	1.50	12.44
TL12-248	Az 330 72	N/A	191.5	200	8.50	0.33
TL12-249	Az 330 60	N/A	36	37.5	1.50	3.32
TL12-250	Az 330 33	N/A	85.45	86.45	1.00	5.86
TL12-251	NE PR F2	N/A	194	196	2.00	1.21
TL12-252	NE PR F2	N/A	52.5	58.5	6.00	0.34
TL12-253	NE PR F2	N/A	70.93	71.93	1.00	0.32
TL12-254	NE 93	N/A	118.5	120	1.50	3.04
TL12-254	NE 93	N/A	267	268.5	1.50	1.73
TL12-255	Az 330 75	N/A	36	39	3.00	0.49
TL12-255	Az 330 75	N/A	46.5	48	1.50	1.51
TL12-255	Az 330 75	N/A	262.5	267	4.50	0.52

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The majority of Treasury's sample data is from diamond drilling. Treasury also carried out trench sampling and surface rock chip sampling however these were not used in Howe's resource estimate. It is the Howe's opinion that all of the sampling was carried out according to industry standards and the samples are representative of mineralization at the Project.

It is presumed by Howe that all historic sampling was completed in a manner consistent with were then current industry standard sampling and assaying techniques.

11.1 OUTCROP SAMPLING

In 2008, representative continuous chip samples and grab samples were removed from the outcrop surface by chipping with a geological hammer. The focus of the sampling was mainly on altered rock and vein material. The rock samples were placed in sample bags and sealed. Each sample bag had the sample number written on the outside of the bag with black permanent marker and a sample tag was placed inside. GPS locations of the sample points were recorded utilizing a handheld GPS receiver. The sample number, description and location were then compiled in a Microsoft Excel spreadsheet.

In 2009, selected outcrops grid mapped and marked for channel sampling. Sample intervals generally varied from 0.5 to 1.0 metres. The channel samples were cut with a portable concrete saw perpendicular to strike. Two cuts approximately 5 centimetres deep were sawn 4 to 5 centimetres apart. The rock between the cuts was removed with a hammer and chisel, placed into sample bags and sealed. Each sample bag had the sample number written on the outside of the bag with black permanent marker and a sample tag was placed inside. GPS locations of the channels were recorded utilizing a handheld GPS receiver all data was then compiled in a Microsoft Excel spreadsheet.

11.2 TRENCH SAMPLING

Outcrop at the bottom of the trench walls were washed, grid mapped and marked for sampling. Sample intervals varied from 0.5 to 0.85 metres. The channel samples were cut with a portable concrete saw perpendicular to strike. Two cuts approximately 5 centimetres deep were sawn 4 to 5 centimetres apart. The rock between the cuts was removed with a hammer and chisel, placed into sample bags, and sealed. Each sample bag had the sample number written on the outside of the bag with black permanent marker and a sample tag was placed inside. All data was then compiled in a Microsoft Excel spreadsheet.

11.3 CORE SAMPLING

Core was retrieved from the drill string using standard wireline methods. Upon retrieval, the core was removed from the core tube and placed into core boxes in the order in which it was drilled.

The core was logged, split and stored in Treasury's field office and core shack in Dryden, Ontario under the on-site supervision of the CCIC/Treasury staff. After cleaning and logging, a sampling line was marked along the centerline of the core. Generally, sample lengths ranged between 0.2 and 1.5 metres with the majority of samples being 1.0 metre or less in length. Longer sample lengths were taken of strongly sheared core sections with poor core recoveries. Lengths were adjusted as necessary to reflect geological, alteration and/or mineralization contacts. Down-hole sample intervals were input directly into DHlogger software.

The core was sawn with two Husqvarna TS510 water-cooled masonry saws with 14-inch diamond blades and 230-volt 5hp motors. The core saws were located in an isolated and ventilated area of the core facility. Fresh water was used as a cooling/lubricating fluid; recycled water was not used.

The core was cut in half (50% split) with one half placed into labelled plastic sample bags and the other half returned to the core box for archive and future verification and testing (if required). Each sample bag had the sample number written on the outside of the bag with black permanent marker and a corresponding sample tag was placed inside. Core logging, sawing, sample bagging and sample shipment preparation was completed either by or under the onsite supervision of the CCIC/Treasury geologists. Certified reference materials (standards), sample blanks were inserted by CCIC/Treasury geologists into each sample batch submitted to the lab for the purpose of quality control. In the 2009 Phase 2, 2010 and 2011 drill programs, ¼ core duplicate samples were also inserted by CCIC/Treasury geologists into each sample batch.

After sampling was completed, the archived core boxes were labelled and placed on metal core racks assembled in the yard of Treasury's field office.

11.4 ANALYSES

In 2008, seventeen (17) outcrop samples, thirty-two (32) trench channel samples and 25 outcrop channel samples were sent to Accurassay Laboratories Ltd. in Thunder Bay, Ontario, for gold fire assay, ICP, whole rock and rare earth elements analyses. In 2008 and 2009 a total of approximately 14,853 core samples (excluding QA-QC samples) were assayed at Accurassay for gold, silver, copper, lead, zinc, and trace element geochemistry (a 26 element package). In 2008, whole rock analyses were performed on 852 core samples. In 2010 and 2011 a total of approximately 16,568 core samples (excluding QA-QC samples) were assayed at Accurassay for gold, silver, copper, lead, zinc, and trace element geochemistry (a 26 element package).

Accurassay is Treasury's primary analytical laboratory. Accurassay is accredited to international quality standards through the International Organization for Standardization/International Electrotechnical Commission to ISO/IEC 17025/2005. It is a Standards Council of Canada Accredited Laboratory (No. 434) and conforms to requirements of CAN−P−1579 (Mineral Analysis) and CAN−P−4E. Its scope of accreditation includes gold fire assay (FA) with AAS finish and Aqua Regia Digest with AAS finish for copper, nickel, cobalt, and zinc. A description of Accurassay Laboratories sample preparation and analytical techniques utilized on Treasury samples are presented in Sections 11.6 and 11.7.

It is the opinion of Howe that the sample preparation, security, and analytical procedures implemented have been adequate for the exploration conducted to date by Treasury. Treasury has implemented a QA-QC protocol as detailed in Section 11.10.

It is presumed by Howe that all historic sampling was completed in a manner consistent with then current industry standard sampling and assaying techniques.

11.5 Sample Security

All samples (rock and core) were bagged and sealed once collected. Samples were then placed in rice sacks and sealed. CCIC/Treasury personnel maintained possession of the samples in the secure core shack until pickup for delivery to the laboratory. When a sufficient quantity of samples had been collected, a local transport company, Courtesy Freight, delivered samples to Accurassay's lab in Thunder Bay Ontario. Laboratory pulps and rejects were backhauled to Dryden and stored in a locked garage at the Goliath Project.

11.6 Sample Preparation

Upon receipt at the lab, the samples were tagged with an Internal Sample Control Number and entered into Accurassay Laboratories' Local Information Management System (LIMS). Sample preparation consisted of conventional drying if required, in ovens with a temperature in the range of 110-120 C (230-250 F); crushing; splitting and; pulverizing. After drying, the sample was passed through a primary oscillating jaw crusher producing material of 90% passing an 8-mesh screen. A 250 to 500 gram sub-sample was split from the crushed material using a riffle splitter. This split was then ground to 90% passing a 150 mesh using a ring and puck pulveriser. Silica sand was used to clean the equipment between each sample to prevent cross contamination. Prepared sample pulps were matted to ensure homogeneity prior to analysis. The homogeneous sample was then sent to the fire assay laboratory or the wet chemistry laboratory depending on the analysis required.

11.7 Analytical Procedures

Treasury has utilized several analytical protocols throughout the drill program at the Goliath Project including gold fire assay with AAS finish; pulp metallic gold assay; silver, zinc and lead analysis by aqua regia digest and AAS finish; multi-element ICP analysis and; whole rock and REE analysis.

11.7.1 Multi-Element ICP scans

Accurassay's inductively coupled plasma atomic emission (ICPAES) analysis (analytical code ICPAR) utilizes an aqua-regia digestion of a 1-gram aliquot of sample followed multi-element ICPAES instrumental analysis. Aqua regia digestion's oxidizing properties make it suitable for dissolution of sulphide minerals and iron oxides. It is the weakest of the digestions. It quantitatively dissolves base metals for the majority of geological materials but major rock forming elements and more resistive metals are only partially dissolved. As such, the leach should be considered partial for most elements. The elements analysed and their detection ranges are presented in the following table:

Method code: ICPAR			
Elements and Ranges (ppm)			
Al (0.01% - 10%)			
As (2 - 8,000)	Cd (4 - 1,000)	Mn (0.01 – 10%)	Sn (10 - 10,000)
B (10 - 5,000)	Co (1 - 5,000)	Mo (1 - 8,000)	Sr (1 - 10,000)
Ba (1 - 5,000)	Cr (1 - 10,000)	Ni (1 - 5,000)	Ti (0.01% - 10%)
Be (1 – 1,000)	Fe (0.01% - 10%)	P (0.01 – 10%)	Tl (1 - 5,000)
Bi (5 - 5,000)	K (0.01% - 10%)	Sb (2 - 10,000)	V (2 - 10,000)
Ca (0.01% - 10%)	Mg (0.01% -10%)	Se (5 - 5,000)	W (10 - 10,000)

11.7.2 Precious Metal Analysis

For the analysis of gold, a 30gram charge of the sample is mixed with a lead based flux fused for one hour and fifteen minutes. Each sample has a silver solution added to it prior to fusion that allows each sample to produce a precious metal bead after cupellation. The fusing process results in lead buttons that contain all of the precious metals from the sample as well as the silver that was added. The button is then placed in a cupelling furnace where all of the lead is absorbed by the cupel and a silver bead, which contains any gold from the sample, is left in the cupel. The cupel is removed from the furnace and allowed to cool. Once the cupel has cooled sufficiently, the silver bead is placed in an appropriately labelled test tube and digested using aqua regia. The samples are bulked up with 1.0 ml of distilled de-ionized water and 1.0 ml of 1% digested lanthanum solution. The samples are allowed to cool and are mixed to ensure proper homogeneity of the solution. Once the samples have settled they are analysed for gold using atomic absorption spectroscopy. The atomic absorption spectroscopy unit is calibrated using appropriate ISO 9002 certified standards in an air-acetylene flame. All gold assays that are greater than 10 g/tonne are automatically re-assayed by fire assay with a gravimetric finish for better accuracy & reproducibility.

The atomic absorption results are checked by the technician. Using electronic transfer the results are forwarded to the database. A certificate is produced from the laboratory database system (LIMS). The Laboratory Manager checks the data, validates the certificates, and issues the results in digital and hardcopy format.

11.7.3 Pulp Metallic Gold Analyses

The rock samples are first entered into Accurassay Laboratories Local Information System (LIMS). The samples are dried, if necessary and then jaw crushed to –8mesh and the entire sample pulverized to approximately 90%-150 mesh. Non-silica based abrasive sand is used to clean out the pulverizing dishes between each sample to prevent cross contamination. The entire sample is screened through 106-micron mesh (150 mesh). Two sub-samples of the –150 mesh portion of the sample (the pulp) and the entire +150 mesh portion of the sample is fired. The sample is mixed with a lead based flux and fused for an appropriate length of time. The fusing process results is a lead button, which is then placed in a cupelling furnace where all of the lead is absorbed by the cupel and a silver bead, which contains any gold, is left in the cupel. The cupel is removed from the furnace and allowed to cool. Once the cupel has cooled sufficiently, the silver bead is placed in an appropriately labelled small test tube and digested using aqua regia. The samples are bulked up with 1.0 mL of distilled deionized water and 1.0 mL of gold blank solution. The total volume is 3.0 mL. For high grade samples the volume may be increased as necessary. The samples cool and are vortexed and the contents are allowed to settle. Once the samples have settled they are analyzed for gold using atomic absorption spectroscopy. The atomic absorption spectroscopy unit is calibrated for each element using the appropriate ISO 9002 certified standards in an air-acetylene flame.

The results for the atomic absorption are checked by the technician and then forwarded to data entry by means of electronic transfer and a certificate is produced. The Laboratory Manager checks the data and validates it if it is error free. The results are then forwarded to the client by email and hardcopy in the mail.

11.7.4 Base Metal Analysis

Samples analysed for base metals (lead, zinc, and silver) are weighed for a geochemical analysis and digested using aqua regia. The samples are bulked to a final volume and mixed. Once the samples have settled they are analysed for base metals using atomic absorption spectroscopy. The atomic absorption spectroscopy unit is calibrated for each element using the appropriate ISO 9002 certified standards in an air-acetylene flame. Any sample that contains a concentration of greater than 10,000 ppm of any element is sent back for an ore grade assay for that element. This assay is similar to the geochemical assay but requires a greater sample mass and final volume.

The atomic absorption results are checked by the technician and saved in the Laboratory database (LIMS). Using electronic transfer the results are forwarded to data entry terminal to produce a certificate. The Laboratory Manager checks the data, validates the certificates, and issues the results in digital and hardcopy format.

11.7.5 Whole Rock Analysis

Whole rock analysis (major oxides) is conducted using a lithium-metaborate fusion with an ICP finish. Performed with loss on ignition (LOI) analysis, a balanced composition of the rock is reported.

11.7.6 Accurassay Laboratories' Internal Quality Control

Accurassay Laboratories employs an internal quality control system that tracks certified reference materials and in-house quality assurance standards. Accurassay Laboratories uses a combination of reference materials, including reference materials purchased from CANMET, standards created in-house by Accurassay Laboratories and tested by round robin with laboratories across Canada, and ISO certified calibration standards purchased from suppliers. Should any of the standards fall outside the warning limits (±2 standard deviation), re-assays are performed on 10% of the samples analysed in the same batch and the re-assay values are compared with the original values. If the values from the re-assays match original assays the data is certified, if they do not match the entire batch is re-assayed. Should any of the standards fall outside the control limit (±3 standard deviation) all assay values are rejected and all of the samples in that batch are re-assayed.

11.8 2008 TREASURY QUALITY ASSURANCE/ QUALITY CONTROL

Treasury implemented quality assurance/quality control (QA/QC) procedures for the 2008 drill program that included the insertion of certified reference materials (standards) and sample blanks. The 2008 QA/QC program and results is presented in Howe's 2010 technical report (Roy et al, 2010).

11.9 2009 TREASURY QUALITY ASSURANCE/ QUALITY CONTROL

The 2009 external QA/QC procedure implemented by Treasury's consultant, CCIC, included insertion of certified reference materials (CRM) and blanks into the sample stream. Every tenth sample was a low-grade CRM, a medium-grade CRM, a high-grade CRM, or a blank. A quarter core duplicate was inserted every 20th sample.

In addition to the Company's QA/QC program, Accurassay Laboratories also inserted in-house standards, CANMET certified reference materials and blanks and analyzed duplicates. Accurassay provided details of the internal QC to the Company.

The 2009 QA/QC program and results are presented in Howe's 2010 technical report (Roy et al, 2010).

11.10 TREASURY 2010-2012 QUALITY ASSURANCE/ QUALITY CONTROL

The external QA/QC procedure originally implemented by CCIC and continued by Treasury Metals Inc. includes inserting certified reference materials (CRM) and blanks into the sample stream. Every tenth sample is a low-grade CRM, a medium-grade CRM, a high-grade CRM, or a blank (Table 11-1). Every 20^{th} sample is a quarter core duplicate.

Table 11-1. Example of how CRMs, blanks, and duplicates are inserted into the sample stream.

Sample #	Standard
10	low-grade CRM
20	blank
25	¼ core duplicate
30	medium-grade CRM
40	blank
45	¼ core duplicate
50	high-grade CRM
60	blank

This procedure was followed for holes found within or in close proximity to the main resource. For exploration drill holes in the north-east fold zone a standard or blank was inserted every 20th sample, there were no changes to the quarter core duplicate procedure.

In addition to the Company's QA/QC program, Accurassay Laboratories also inserted in-house standards, CANMET certified reference materials and blanks and analyzed duplicates. Accurassay provided details of the internal QC to CCIC.

11.10.1 Accuracy – 2010-2011

To monitor accuracy, certified reference materials (CRM) were inserted sequentially into the sample stream before shipment from the field at a rate of 1 in every 20 samples submitted.

Both higher grade, medium grade and low-grade gold standards were used in each sample shipment. All CRMs were obtained from CDN Resource Laboratories Ltd., Delta, BC; with exception to Oreas61D which was obtained from ASL Lab, Vancouver, BC. The standards were received prepared (pulverized to –200 mesh and blended) and pre-packaged in 50 to 60 gram packets.

Table 11-2. Summary of the certified reference materials, used in the QAQC for 2010-2011 drilling programs.

Standard Name	Recommended Au g/t	Standard Deviation
CDN-SE2	0.242	0.009
CDN-GS1F	1.16	0.065
CDN-GS5D	5.06	0.125
Oreas61d	4.76	0.14
CDN-CGS13	1.01	0.055
CDN-CM6	1.43	0.045
CDN-ME6	0.27	0.014

11.10.1.1 Acceptance Criteria for Routine Analyses – 2010-2011

To check the accuracy of the laboratory, control limits (CL) are established at accepted mean ±3σ (standard deviation) and warning limits (WL) at accepted mean ±2σ. Any single standard analysis beyond the upper (UCL) and lower (LCL) control limits is considered a "failure". In addition, three successive standard analyses outside of the upper (UWL) and lower (LWL) warning limits on the same side of the mean could also constitute a failure. Successive warning results may indicate laboratory bias and possibly incorrect calibration of the laboratory equipment.

11.10.1.2 Results of Routine Analyses – 2010-2011

The results from the QA/QC standards were plotted versus time for each standard (Figures 11-1 to 11-8). The minimum and maximum acceptable values and mean Au value (Au-g/tonne) for the QC sample are shown on each chart.

Most of the Certified Reference Material (CRM) inserted in the mineralized zone returned values within STD±3SD (standard deviations). In sample batches where the standard failed within or near significant mineralization (e.g. a sample with greater than 0.5g/tonne Au), CCIC/Treasury staff elected to re-analyse the pulps from the preceding 5 and following 5 samples in the batch at Accurassay. Results from the re-analysis were substituted into the assay database.

Not shown on the graph of CDN-SE2 (Figure 11-1) are 3 samples which are interpreted to be the result of mislabelled blanks and 1 sample which is interpreted to be the result of a mislabelled higher grade standard, possible CDN-GS1F.

Not shown on the graph of CDN-ME6 (Figure 11-2) is 1 sample that is interpreted to be the result of a mislabelled higher grade standard, possible CDN-GS1D or CDN-CGS13.

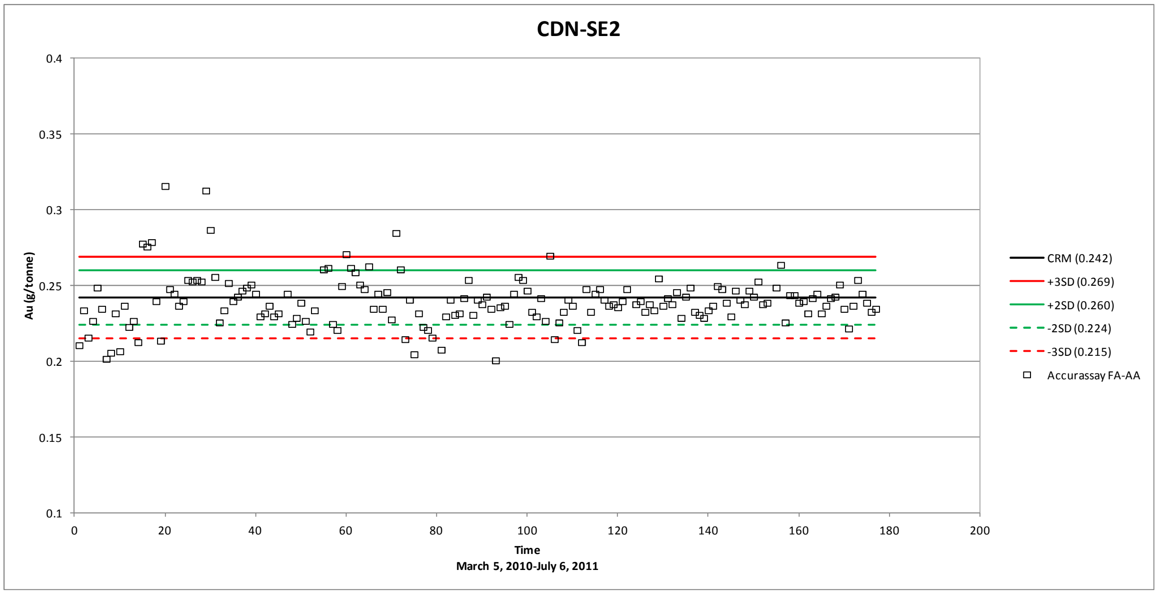


Figure 11-1. Standard CDN-SE2.

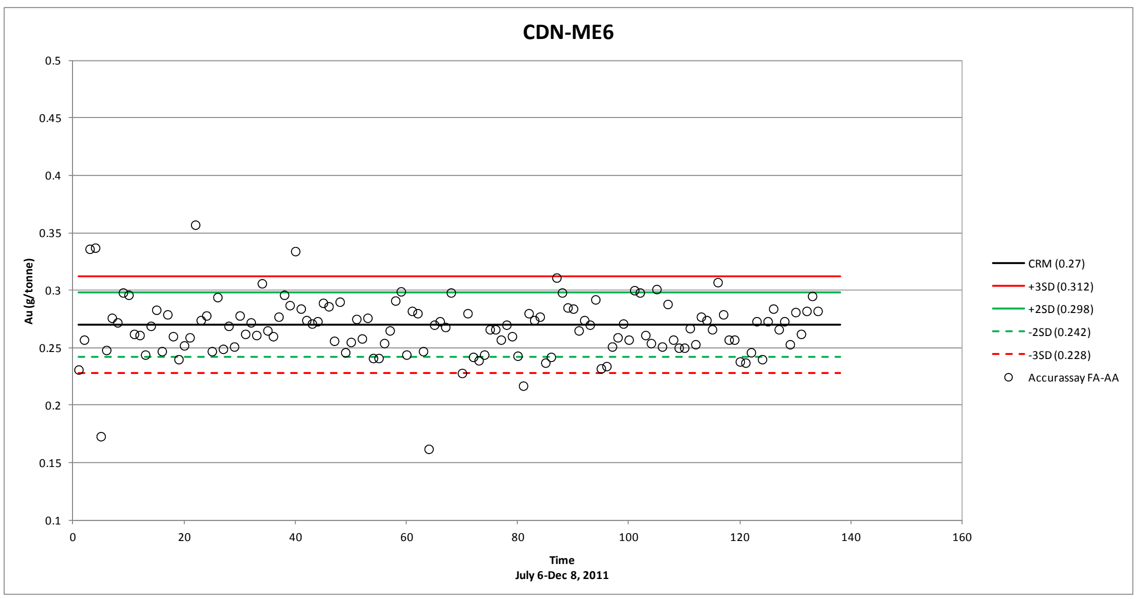


Figure 11-2. Standard CDN-ME6.

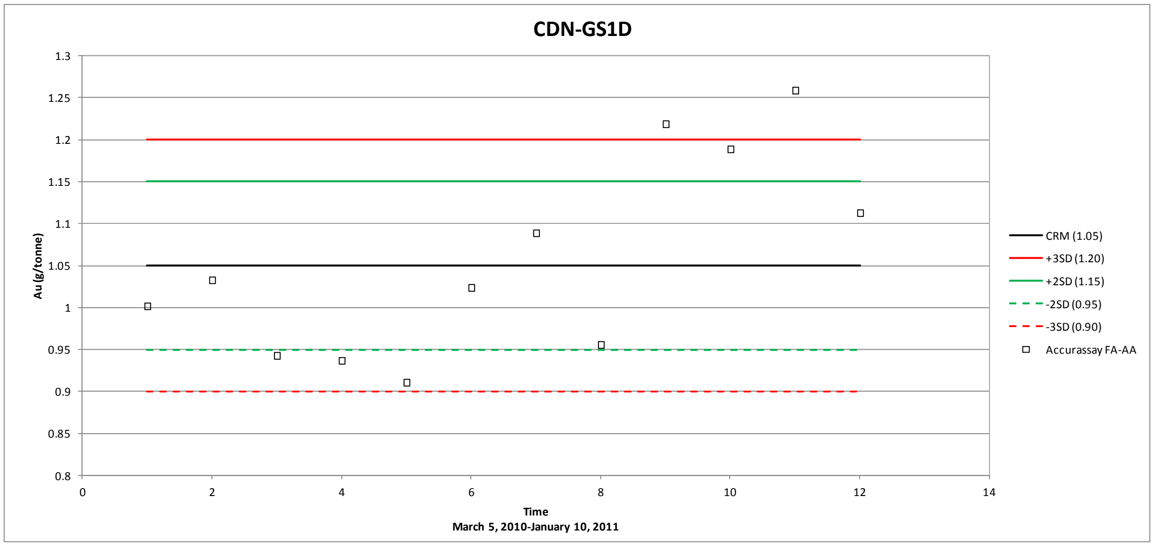


Figure 11-3. Standard CDN-GS1D.

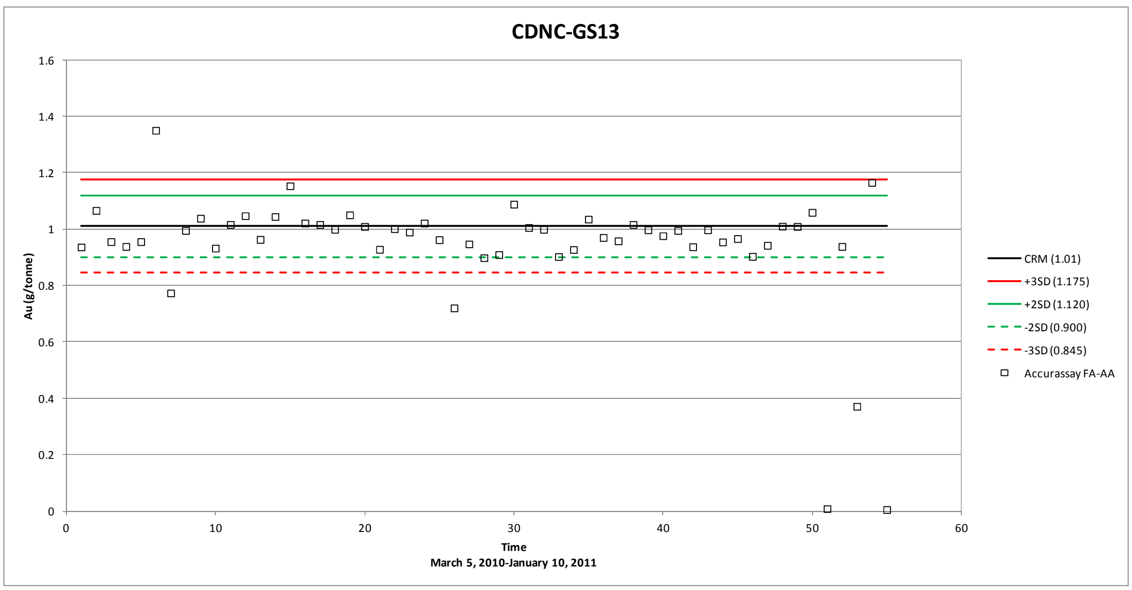


Figure 11-4. Standard CDN-CGS13.

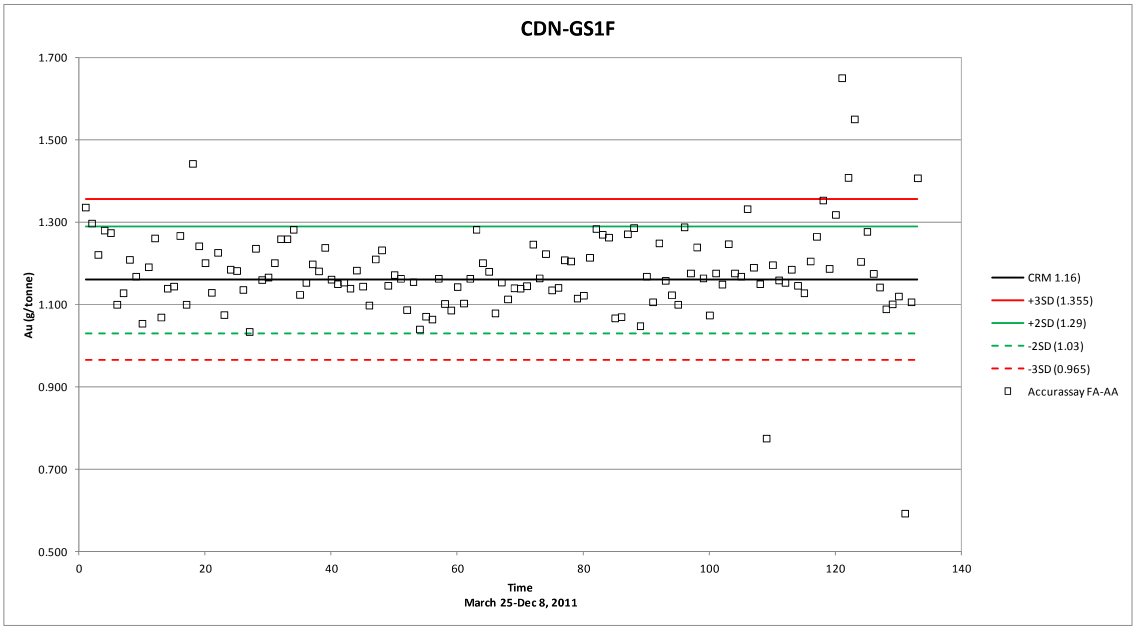


Figure 11-5. Standard CDNGS1F.

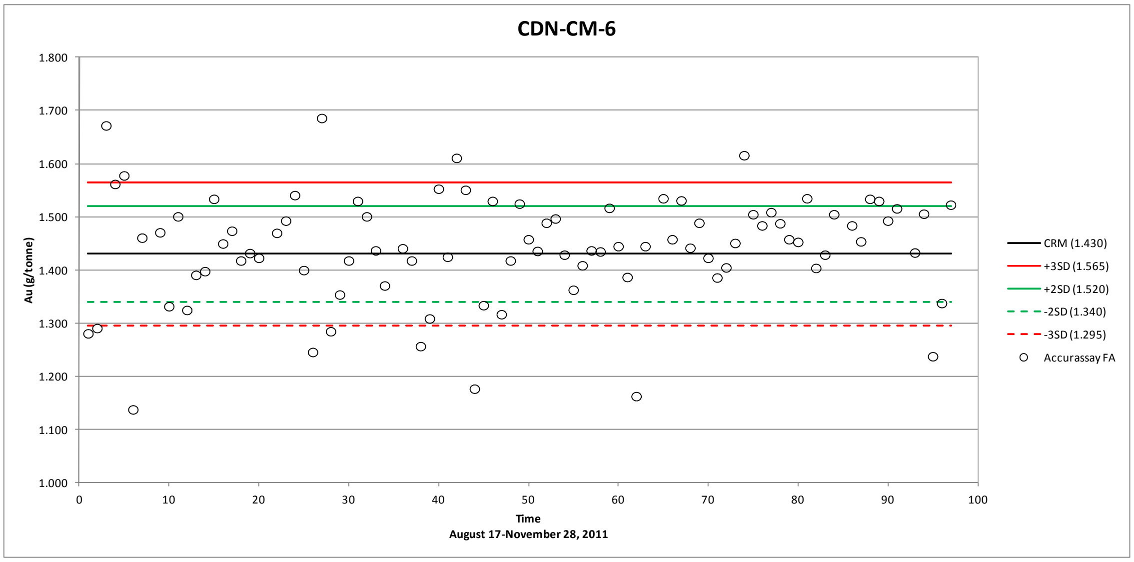


Figure 11-6. Standard CDN-CM6.

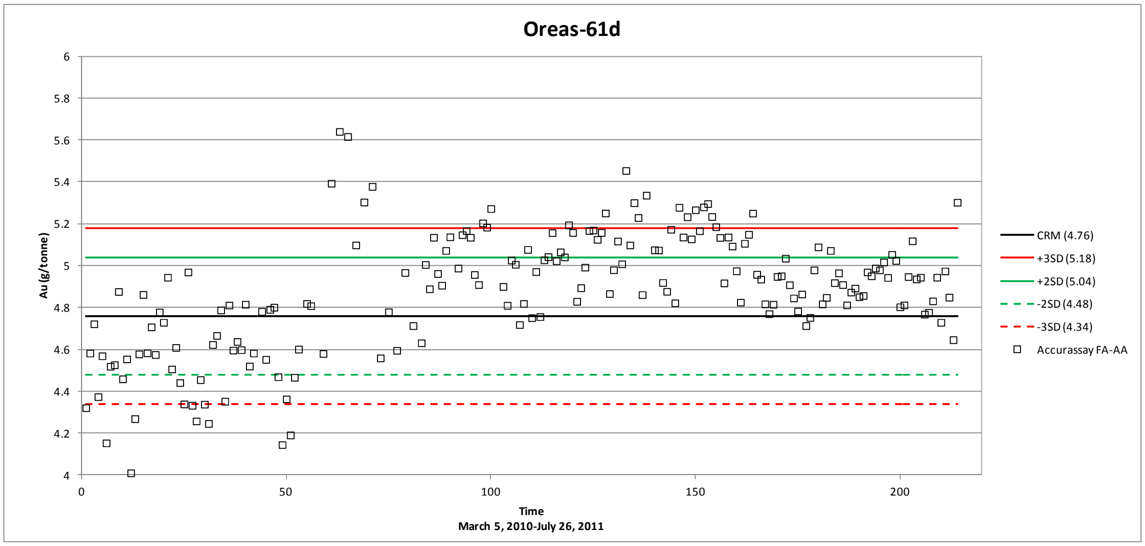


Figure 11-7. Standard Oreas-61D.

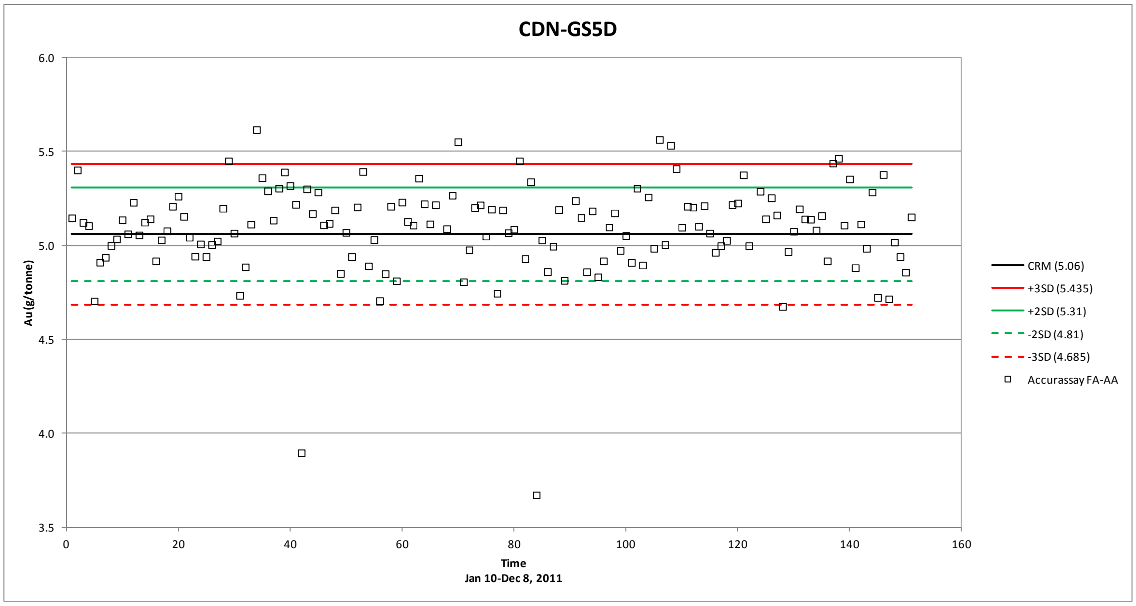


Figure 11-8. Standard CDN-GS5D.

11.10.2 Contamination – 2010-2011

CCIC/Treasury inserted blanks sequentially at least every 20th sample into the drill core samples before shipment. Instead of a coarse field blank, CCIC/Treasury used a prepared blank provided by Accurassay that was pulverized to –200 mesh, blended and packaged in 60 gram packets. The blank had a gold concentration of less than 15ppb (0.015 g/tonne) gold.

Since a pulverized sample was used as a blank, it did not test two laboratory sample preparation processes that have significant potential for cross-contamination between samples: the jaw crushing and ring pulverizing stages. A pulverized blank would only check for contamination or sample mislabelling in the analytical side of the laboratory.

11.10.2.1 Acceptance Criteria for Routine Analyses – 2010-2011

CCIC/Treasury set 15 ppb (0.015 g/tonne) gold as the maximum acceptable value for the blanks. A blank sample that assayed greater than the maximum acceptable value is a failure.

11.10.2.2 Results of Routine Analyses – 2010-2011

The results from the Blanks were plotted against time with the maximum acceptable value as shown on the chart illustrated in **Error! Reference source not found.**11-9.

All blanks inserted into the sample shipments except two returned gold concentrations below the maximum acceptable value. One sample returned a value of 0.045 g/tonne Au and a second returned a value of 1.488 g/tonne Au (not shown in Figure 11-9) and is the likely result of a mislabelled standard, possible CDN-CM6.

In future sampling programs, Howe recommends that coarse field blanks be inserted in place of or in conjunction with pulverized blanks in order to test all potential sources of laboratory contamination.

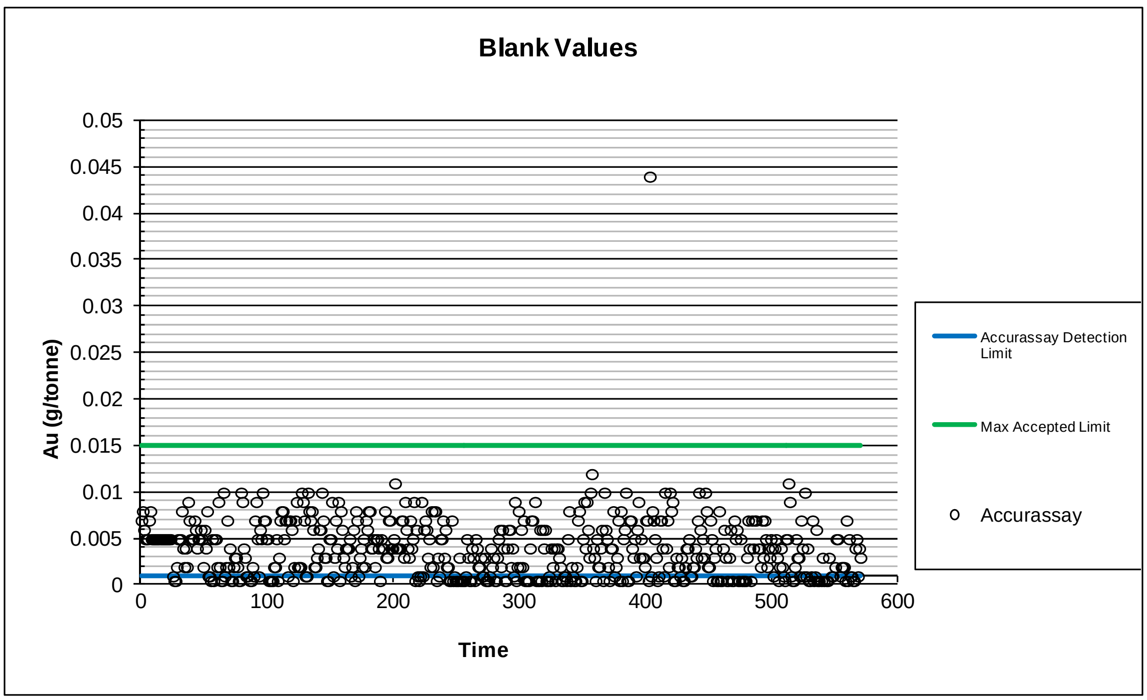


Figure 11-9. Gold analytical results *vs.* time - blank samples.

11.10.3 Precision

Precision is often monitored by the insertion of duplicate samples. The duplicates may be quarter core duplicates and/or preparation duplicates, split after the initial jaw-crushing phase to make two pulps. Treasury inserted quarter core duplicates into the sample stream each at a rate of 1 in

20 samples. In addition laboratories routinely analyse pulp duplicates, split after the pulverizing phase, as part of their internal quality control programs.

11.10.3.1 Quarter Core Duplicates

Generally, in a duplicate sampling program, quarter core duplicates are a compromise, as the best measure of precision would be to analyse the other half of the core, leaving no remaining core. Precision indicated by quarter core duplicate is generally poorer than indicated by half core duplicates. In a duplicate sampling program, the core duplicate analyses account for the largest portion of total error in the entire process, and as such provide the best indication of the precision of any individual analyses.

The Company submitted a total of 970 quarter core duplicate samples in the 2010 and 2011 drill programs. Original analysis data vs. the quarter core duplicate analysis is plotted in Figure 11-10. Any values that plot significantly away from the correlation line may indicate a potential nugget effect or, less likely sample preparation errors or analytical errors. Overall, the graph shows acceptable correlation between the original samples and quarter core duplicates, however relatively few of the samples gold values of greater than 1.0 g/tonne Au. It is difficult to make any meaningful analysis of potential nugget effect from so few higher-grade samples. Howe recommends additional quarter core duplicates be taken from the mineralized zone.

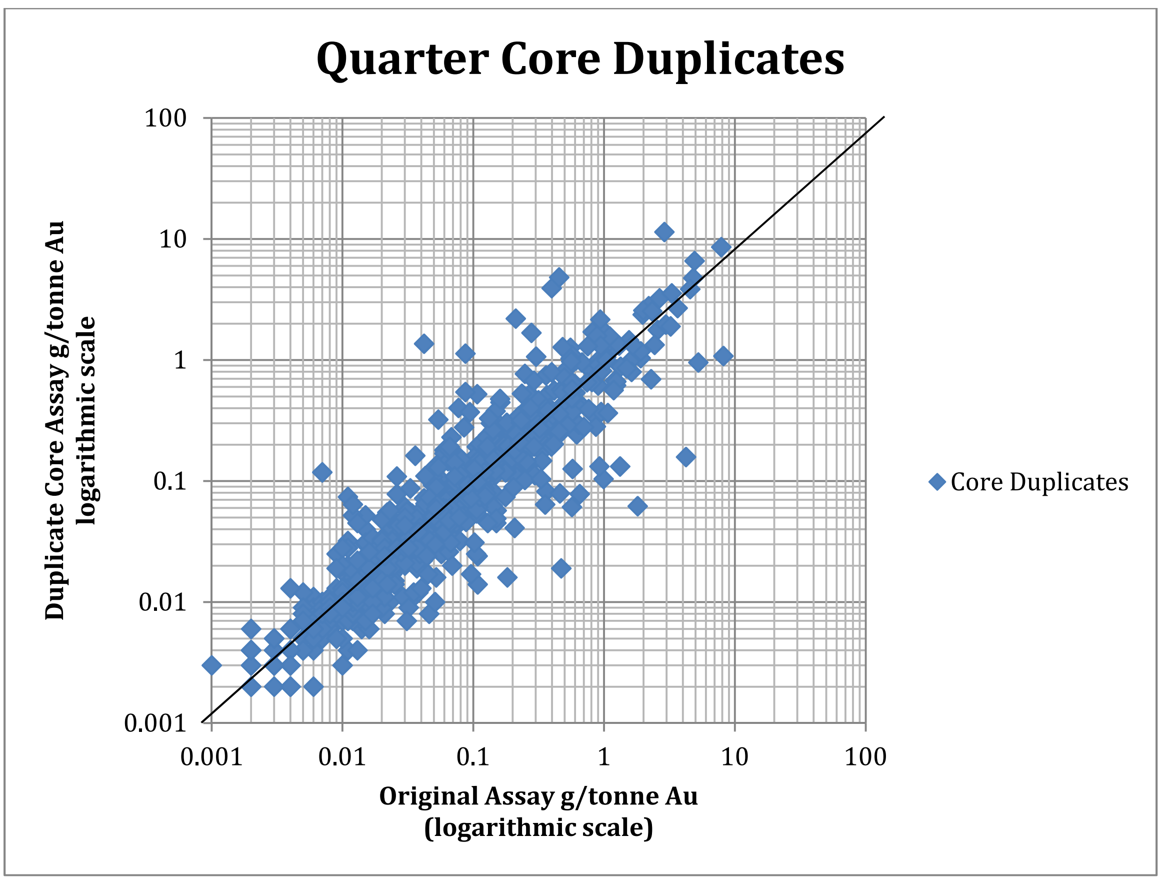


Figure 11-10. Plot of primary assays versus quarter core duplicate assays

12 DATA VERIFICATION

12.1 2008 HOWE SITE VISIT AND DUE DILIGENCE SAMPLING

Howe representative, Mr. Ian Trinder, completed a site visit to the Goliath Project during the period September 14th to 16th, 2008 as part of due diligence in the preparation of Howe's 2008 technical report. During the property visit, Mr. Trinder met with Dr. Scott Jobin-Bevans of Treasury and CCIC field personnel Rory Krocker, Amanda Tremblay and Terry Loney to examine the project area and discuss Treasury's exploration activities, methodologies, findings and interpretations. Mr. Trinder conducted a review of available data at Treasury's field office in Dryden, Ontario, and an inspection of surface outcrops and a recent trench at several areas of the Project area. The location of the reclaimed decline entrance and numerous drill collar locations were verified. While in the field, the diamond drill rig was inspected as it was drilling hole TL08-54. The condition of the historic Teck-Corona drill core was also checked at the Railside storage yard in Dryden. Selected drill core from Treasury's drill holes was examined at its secure core logging and storage facility in Dryden.

Howe collected six samples of mineralized diamond drill core from Treasury's 2008 diamond drill core. The samples consisted of quarter-core that was sawn under Howe supervision from the half-core archive that remained in core boxes at Treasury's core storage facility in Dryden. Howe sealed the sample bags with ladder lock ties and maintained possession of the samples until their delivery to SGS Laboratories in Toronto, Ontario. SGS-Toronto as a reputable, ISO/IEC17025 accredited laboratory qualified for the material analysed. SGS quality control procedures are method specific and include duplicate samples, blanks, replicates, reagent / instrument blanks for the individual methods.

The samples were prepared using SGS sample preparation package PRP89, which consists of conventional drying if required, in 105°C ovens; crushing; splitting and; pulverizing. After drying, the sample was passed through a primary oscillating jaw crusher producing material of 75% passing a 2mm screen. A 250-gram sub-sample was split from the crushed material using a stainless steel riffle splitter. This split was then ground to 85% passing 75 microns or better using a ring pulveriser.

The verification samples were analysed for gold and silver plus 40 elements, using SGS analytical codes FAI313, AAS21E and ICP40B as outlined in Table 12-1. Over limit gold and silver were analysed using FAG303 and FAG323 respectively.

Table 12-1. Verification samples – SGS analytical methods.

Method code	Description	Lower Detection Limit
FAI313	Au fire assay; ICP-AES finish, 30 g nominal sample weight.	>1ppb <10000ppb Au
FAG303	Au fire assay; gravimetric finish, 30 g nominal sample weight.	>0.03g/t Au
AAS21E	Ag – three acid digest, AAS finish	>0.3 g/t <300g/t Ag
FAG323	Ag fire assay, gravimetric finish, 30 g nominal sample weight.	>3g/t Ag
ICP40B	4 Acid digest (HCl, HNO_3, HF, $HClO_4$) and ICP-AES finish 32 elements – Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, K, La, Li, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Sc, Sn, Sr, Ti, V, W, Y, Zn, Zr	

The duplicate samples provide an independent confirmation of the presence of significant gold, silver and base metals at the Thunder Lake Deposit (Table 12-2). The data are too limited however, to make a meaningful comparison of Howe's duplicate sample analytical results with the Treasury's original analytical results. Howe notes however, that the variation between original half core and quarter core duplicate assay results are reasonable and are typical for gold exploration projects.

Table 12-2. Howe ¼-core drillhole duplicates *vs*. original samples.

DDH	From (m)	To (m)	Length (m)	Howe Sample #	Au (ppm)	Ag (ppm)	Cu (ppm)	Pb (ppm)	Zn (ppm)	Treasury Sample #	Au (ppm)	Ag (ppm)	Cu (ppm)	Pb (ppm)	Zn (ppm)
TL0801	73.00	74.00	1.00	ACA001	0.624	6	29.7	56	216	383572	1.484	96.5	96.5	202	428.5
TL0802	126.00	127.00	1.00	ACA002	0.651	3	42.6	79	203	383939	0.849	0.5	30	115	173
TL0803	65.00	65.45	0.45	ACA003	20.000	366	395	3770	>10000	384075	16.262	184.6	356	3695	17969
TL0804	114.50	115.50	1.00	ACA004	2.260	53.1	54.8	360	1060	384448	2.535	84.5	46	503	819
TL0830	35.00	35.50	0.50	ACA005	1.920	10.8	167.0	483	726	642054	0.874	11.3	101	294	464
TL0836A	174.00	175.00	1.00	ACA006	0.531	3	32	44	72.1	643648	2.113	5.3	32	94	81.5

12.2 2011 HOWE SITE VISIT

Mr. Roy visited the Project during the period November 25th to 27th, 2011, as part of due diligence in the preparation of this Report. During the property visit, Mr. Roy met with Treasury representatives, Rory Krocker and Ash Martin, to examine the project area and discuss Treasury's exploration activities, methodologies, findings, and interpretations. Mr. Roy conducted a review of available data at Treasury's field office in Dryden, Ontario, and an inspection of several areas of the Project. Selected drill core from Treasury's drill holes was examined at its secure core logging and storage facility in Dryden.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 BULK SAMPLES (TECK AND CORONA- 1998)

Four (4) bulk samples from the Main Zone, totalling 2,375 tonnes of material (all drift, slash and TDB rounds) and grading >3.0 g/tonne Au, were collected from various areas of the underground workings between May 15 and September 15, 1998 (Page et al., 1999b). A total of 1,737 tonnes of material was collected from the No. 1 Shoot (884 tonnes A-East low-grade; 450 tonnes A-East high-grade and; 403 tonnes A-East TDB) and 638 tonnes of material from was collected from the No. 2 Shoot (B West Zone). Face sample data indicated that two of the bulk samples were relatively low in grade (3.0 to 6.0 g/tonne Au) while the other two samples were of higher grade (>20 g/tonne Au). One of the two higher grade samples was derived from a small test-mining run of 400 tonnes and this is referred to as the "take down back" or "TDB" sample. The bulk samples were processed through a crushing plant and reduced in volume through a sampling tower to a total of 384 kg. The representative sample tower splits were shipped to Lakefield Research Ltd., Lakefield, Ontario where the samples were further processed and analysed for gold concentration (Page et al., 1999b). Approximately 2,336 tonnes of the remaining material was transported to and processed at the Stock Mine mill of St. Andrew Goldfields Ltd., Timmins, Ontario.

13.1.1 Low grade bulk sample

The two low-grade bulk samples were obtained from the B-West and A-East drifts (Page et al., 1999b). The lowest grade B-West sample showed good correlation between the face sample calculated grade of ~4.5 g/tonne Au and the bulk sample grade of 3.6 g/tonne Au. This represents a percentage decrease in grade of about 15-20% and an absolute decrease of 0.9 g/tonne Au. Page et al. (1999) suggested that the fairly uniform rock comprising this bulk sample was not greatly influenced by coarse gold. The A-East low-grade bulk sample yielded an increase in gold grade of about 20-25%, from 5.9 to 6.4 g/tonne Au in the face samples to 7.5 g/tonne Au in the bulk sample, representing an absolute increase of 1.1 to 1.6 g/tonne Au. Overall, this sampling of the low-grade material established approximately ±20% accuracy in calculated face sample grades versus actual recovered grades from the bulk sample.

13.1.2 High grade bulk sample

The A-East high-grade and "take-down-back" samples were derived from the No. 1 Shoot and essentially the same mineralized zone. The average (mean) face grade calculated for the two high grade samples was ~27.8 g/tonne Au with a range of between 22.7 to 35.1 g/tonne Au (Page et al., 1999b). The bulk sample grades of the high-grade A-East and TDB decreased to 16.8 g/tonne and 12.7 g/tonne Au respectively, representing significant percentage decreases in grade (40-50%) and absolute gold content. These decreases are significantly more than the ±20% variation that was defined by the low-grade bulk sample results. Page et al. (1999) surmised that the individual high gold assays from face samples were due to coarse gold nugget effects and resulted in an overestimate of anticipated gold grade in the bulk samples. Nugget effect was apparently not a significant factor in the large bulk samples.

13.1.3 Discussion of results

Prior to executing the underground exploration and bulk sampling program, estimates of expected tonnage and grade to be extracted in the bulk sample were calculated from drill hole data (Page et al., 1999b). The drill hole based estimate was approximately 3,900 tonnes grading 15.2 g/tonne Au, which contrasted with the 3 bulk samples (excluding 400 tonne TDB) that yielded about 1,950 tonnes grading 8.3 g/tonne Au. This represents a decrease of about 50% in tonnage and about 45% in grade between the expected drill hole estimate and the actual recovered bulk sample; the contained gold in the bulk sample was therefore less than 30% of that expected (Page et al., 1999b).

13.2 HISTORIC METALLURGICAL TESTING/RECOVERIES (TECK AND CORONA)

The original bulk sample of 2,375 tonnes had an estimated overall grade of 9.07 g/tonne Au or 692 contained ounces Au (Page et al., 1999b). Hogg (2002) reported that the recovered grade from the approximately 2,336 tonne bulk sample, processed through the Stock Mine mill of St. Andrew Goldfields Ltd. in 1999, was 5.63 g/tonne Au (0.164 opt Au) and 15.28 g/tonne Ag (0.446 opt Ag).

Teck conducted limited preliminary metallurgical test work, consisting of one gravity separation test, one flotation test, and one cyanidation test on a composite sample of 24 kg from the No. 1 Shoot at Lakefield Research in Ontario. No optimisation work was carried out. Metallurgical results obtained indicated that cyanidation achieved the best recoveries for gold at 98.7% (Corona, 2001; Hogg, 2002). Gravity and flotation resulted in recoveries of 97.3% Au and gravity alone recovered 69.1% Au (Corona, 2001; Hogg, 2002). Final gold recovery was calculated at 96.85% and silver recoveries were approximately 38% (Corona, 1999 and 2001).

Howe notes that the head grade of the composite sample was 25 g/tonne – nearly an order of magnitude greater than the average expected head grade of surface and underground sources, therefore the sample cannot be considered representative of the overall deposit. From experience, recovery values for gold deposits of this type decrease with decreasing head grades. No microscope work, that would give an understanding of the nature of the gold mineralization, was carried out. It is Howe's opinion that because only one test was carried out on a non-representative sample, the historic Teck metallurgical test work is of limited value. Further test work is therefore recommended.

13.3 METALLURGICAL TESTING 2011

Treasury retained G&T Metallurgical Services Limited (G&T) of 2957 Bowers Place, Kamloops, British Columbia V1S 1W5 to conduct initial metallurgical test work to follow up on the historical work performed by Teck Exploration Ltd. in 1998. Testing commenced on March 11, 2011 and concluded the week of May 16, 2011.

13.3.1 Samples and Mineralogy

G&T completed a preliminary metallurgical test program on a master composite sample made up from a shipment of thirty individual half-diamond drill core samples from the Goliath Gold Project with total weight of approximately 59 kilograms (as listed in Table 13-1). The composite sample tested had a measured gold and silver feed grade of about 3.5 and 25 g/tonne, respectively (Table 13-2). Minor concentrations, all below 0.1 percent, of copper, lead, and zinc were also present in the sample.

Table 13-1. Drill core samples.

Sample No.	Weight Kg	Sample No.	Weight Kg
383564	2.4	704102	0.8
383565	2.3	704103	0.3
383566	2.6	704395	1.5
383567	2.3	704397	1.5
383568	2.7	704398	1.2
384448	1.2	704399	3.0
384449	1.5	704401	1.4
384451	1.3	980282	3.7
643647	2.5	980283	2.2
643648	1.1	980284	2.8
643649	2.5	980285	2.3
643651	2.2	980287	2.7
704098	0.9	980288	2.8
704099	0.4	980781	2.5
704101	0.7	980782	3.3

Chemical analysis of the composite is shown in Table 13-2.

Table 13-2. Composite sample analyses.

Elements	Value
Au g/t	3.4
Ag g/t	25
S(T) %	1.4
Cu %	0.012
Pb %	0.04
Zn %	0.08
Fe, %	1.3

The total sulphide content accounted for 2.1% of the mass, with the primary sulphides being pyrite and lesser pyrrhotite No silver minerals were identified. 56% of the mass was quartz, with micas and feldspars representing 22% and 17% respectively.

13.3.2 Grindability

A Bond ball mill index was determined for the composite and returned a value of 11.1 kWh/tonne (10.9 kWh/T) at a screen size of 106 micron, indicating a relatively soft material.

13.3.3 Gravity Concentration

A series of gravity separation tests was completed on the composite sample to produce gravity tailing products for downstream testing. Each test was carried out using 4 kg of feed sample at grinds ranging from 68 to 144 microns. Gravity recovery of gold ranged from 23% to 43% in 0.2 % to 0.6% of feed weight, indicating moderate potential for gravity recovery of a substantial fraction of the gold. Silver recovery to the gravity concentrate was poor.

13.3.4 Cyanidation

Cyanidation tests with and without pre-concentration by flotation have been conducted on gravity tailings and, more recently, on whole ore. Primary grind had a minor effect on gold recovery and a grind of 105 microns was considered adequate. The behaviour of silver was somewhat unique in that overall recovery by flotation followed by cyanidation of the concentrate was substantially higher than by cyanidation alone. However, due to higher gold recovery with direct cyanidation, this route appears to be the most economic.

Gravity plus flotation test results are summarized in Table 13-3.

Table 13-3. Gravity plus Flotation.

Test	Cum Wt	Weight	Ag	Au	Distribution	
7A	**g**	**%**	**g/t**	**g/t**	**Ag,%**	**Au,%**
Gravity concentrate	13.6	0.34	696	497	10.0	46.4
GC+R1	133.8	3.36	612	103	86.5	94.2
GC+R1-R2	154.2	3.88	559	91	91.0	96.4
GC+R1-R3	165.0	4.15	529	86	92.2	97.0
GC+R1-R4	171.7	4.32	511	83	92.76	97.39

Test	Cum Wt	Cum Wt	Ag	Au	Distribution	
10A	**g**	**%**	**g/t**	**g/t**	**Ag,%**	**Au,%**
Gravity concentrate	24.6	0.31	630	439	7.5	35.7
GC+R1	417.3	5.23	457	67	92.0	92.5

Test 10A concentrate was leached for 48 hours and returned overall recoveries of 88.2% for both gold and silver. Cyanide concentration was 2 g/L and cyanide and lime consumptions were 2.9 kg/t and 1.2 kg/t respectively.

Gravity concentration with 48 hour cyanidation of the gravity tail is summarized in Table 13-4.

Table 13-4. Gravity plus Cyanidation.

Grind	NaCN	Gravity	Recovery, % Gravity		Recovery, % Grav + CN		Reagents, kg/t	
K80, µ	**g/L**	**Wt %**	**Au**	**Ag**	**Au**	**Ag**	**NaCN**	**CaO**
105	2.0	1.1	71	30	97	82	0.7	0.7
105	2.0	0.1	26	6	97	77	0.7	0.5
125	2.0	0.2	23	5	94	72	0.9	0.4
68	2.0	0.2	23	5	96	78	1.6	0.4
144	2.0	0.6	43	8	96	71	0.6	0.8
105	1.0	0.1	26	6	97	65	0.3	1.0
105	0.5	0.1	26	6	96	69	0.2	0.6

13.4 METALLURGICAL TESTING 2012

Results of two tests from a June 2012 metallurgical program on a sample assaying 2.15 g/tonne gold confirm the above gold extractions. One test was run with gravity concentration preceding cyanidation and on without gravity. Forty-eight hour recoveries at 1 g/L NaCN were 95.6% and 95.3 % respectively, indicating a small improvement with the use of gravity. The use of gravity also improves leach kinetics, with extraction essentially complete after 24 - 36 hours.

13.5 HOWE COMMENT ON 2011 AND 2012 METALLURGICAL TESTING

The results of the 2011 and 2012 metallurgical test programs are the basis for the current PEA flowsheet selection and estimates.

14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

During September-November, 2011, Howe carried out a resource estimate for the Goliath deposit using historical drilling and current drilling. Treasury was responsible for the current drilling that was carried out during 2010 and 2011. The resource estimate includes holes up to Hole TL11228, drilled during 2011.

This resource estimate was prepared by Doug Roy, M.A.Sc, P.Eng, Associate Mining Engineer with Howe. Micromine software (Version 12.0.5) was used to facilitate the resource estimating process.

The resource estimate was prepared in accordance with CIM Standards on Mineral Resources and Reserves[2] where:

- A *Measured Mineral Resource*, as defined by the CIM Standing Committee is "that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity."
- An *Indicated Mineral Resource* as defined by the CIM Standing Committee is "that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonable assumed." and,
- An *Inferred Mineral Resource* as defined by the CIM Standing Committee is "that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, working and drill holes."

Classification, or assigning a level of confidence to Mineral Resources, has been undertaken in strict adherence to the CIM Standards on Mineral Resources and Reserves.

This report quotes estimates for mineral resources only. There are no mineral reserves prepared or reported in this technical report.

Howe is not aware of any known environmental, permitting, legal, title, taxation, socio-political, marketing, or other issues that may materially affect this Resource Estimate. Treasury is

[2] CIM Definition Standards adopted November 27, 2010.

conducting ongoing community consultations including discussions with the local First Nation communities. The effect of these discussions on future access, title or the right or ability to perform work on the Project area is not known at this time.

14.2 DATA SOURCES

For resource estimation, Treasury provided several forms of digital data:

1. Digital drill hole databases in Microsoft Excel format that contained collar surveys, down-hole surveys, geological logs, and assays for holes up to, and including Hole TL11228.
2. A digital spreadsheet containing results of specific gravity ("SG") measurements.
3. A report on mineral processing test work that was carried out in 2011 (Folinsbee and Johnston, 2011).

14.2.1 Additional Drilling Data

The previous mineral resource estimate included holes up to Hole TL0986, drilled during 2009. The current data included assays for holes up to TL11228. Therefore, this mineral resource is current up to Hole TL11228.

The supplied data was imported to Micromine software. The supplied data files were imported as indicated in Table 14-1:

Table 14-1. Existing and supplied data.

Existing Micromine Database File	Existing Data Up To	Description	"New" Supplied File	New Data for Holes up To	Size (kB)	"New Data" Imported to Micromine File	Holes Imported To "Existing" Mircomine File[1.]
dh-Assay.dat	TL0986	Sample assays.	Assay.csv	TL11228	9,550	New Data Oct 17, 2011 - assay.dat	TL1087-TL11228
dh-Collar.dat	TL0986	Collar coordinates.	Collar.csv	TL11229	28	New Data Oct 17, 2011 - collars.dat	TL0801-TL11229
dh-Geology.dat	TL0986	Lithology.	Major.csv	TL11228	436	New Data Oct 17, 2011 - major.dat	TL1087-TL11228[2.]
(New File Made) dh-RQD.dat		Rock quality designation.	Rqd.csv	TL10112	575	New Data Oct 17, 2011 - rqd.dat	TL0805-TL10112[2.]
dh-Survey.dat	TL0986	Downhole surveys.	Survey.csv	TL11229	123	New Data Oct 17, 2011 - survey.dat	TL1087-TL11229

Notes:
1. In some cases, existing data was overwritten.
2. File contained data only up to this hole.

In the supplied assay file, the assay fields were in a format such as "au_gtp_alpm1."

- The first part was the element that was assayed.
- The middle part was the unit in which the element is reported, i.e. ppm = parts per million, gpt = grams per tonne, and one exception of wt_per = weight percentage.
- The third part was the analytical technique used, i.e. icp = inductively coupled plasma, ALFA1 = Gold analysis-Fire Assay-Atomic Adsorption finish, ALMA1 = ICP MA digestion, ALPM1 = Pulp Metallic analysis, ALFA7 = Gold analysis-Fire Assay-Gravemtric finish, ALINAA1 is the Accurassay Multi-element exploration package.

Supplied assays were appended to the current assay data file "dh-Assay.dat". With a few exceptions, all samples had fire assay values (supplied data field "au_gpt_alfa1"). These were imported to the "Au-ppm" field in the existing assay file. Where there were pulp metallic assays in the supplied data, these "overwrote" the "Au-ppm" value in the existing assay file because these were considered to be slightly more accurate (greater sample volume). Refer to Section 14.6 for further discussion of this matter.

For the other elements in the assay file, these were imported to the corresponding fields in the existing data file. There were many new elements that were assayed by ICP in the newly supplied assay data file that had not been assayed previously. A very small proportion of samples were multi-element-assayed in this manner. Therefore, it was decided to not add all of the new elements to the dh-assay.dat file.

Some holes that were drilled failed for some reason and a second hole was drilled from the same collar location. Subsequent holes were given a letter suffix. For example, Hole TL11209 failed. So, a second hole was drilled, named TL11209a.

Three "new" lithology codes were introduced in the newest data:

"New" Code	Frequency	Comment
D	9	Dike
MD	33	Mafic Dike
UNK	1	Unknown - one lost sample at the end of failed hole TL11209

The supplied collar coordinates were in a UTM grid rather than site grid. The UTM coordinates were imported, and then converted to site grid.

In the drilling database, the hole naming convention is as follows:

	Before 2008	2008-2009	2010 after Hole TL1099, and 2011
Field Name	Hole	Hole	Hole
Field Type	Text	Text	Text
Number of Characters	5	6	7
Example	TL001	TL0801	TL10100

The new drilling data was imported to Micromine and the revised database files were validated.

Figure 14-1 shows the locations of the current (2010-2011) holes and previous holes (Treasury 2008-2009 holes and historic Teck holes).

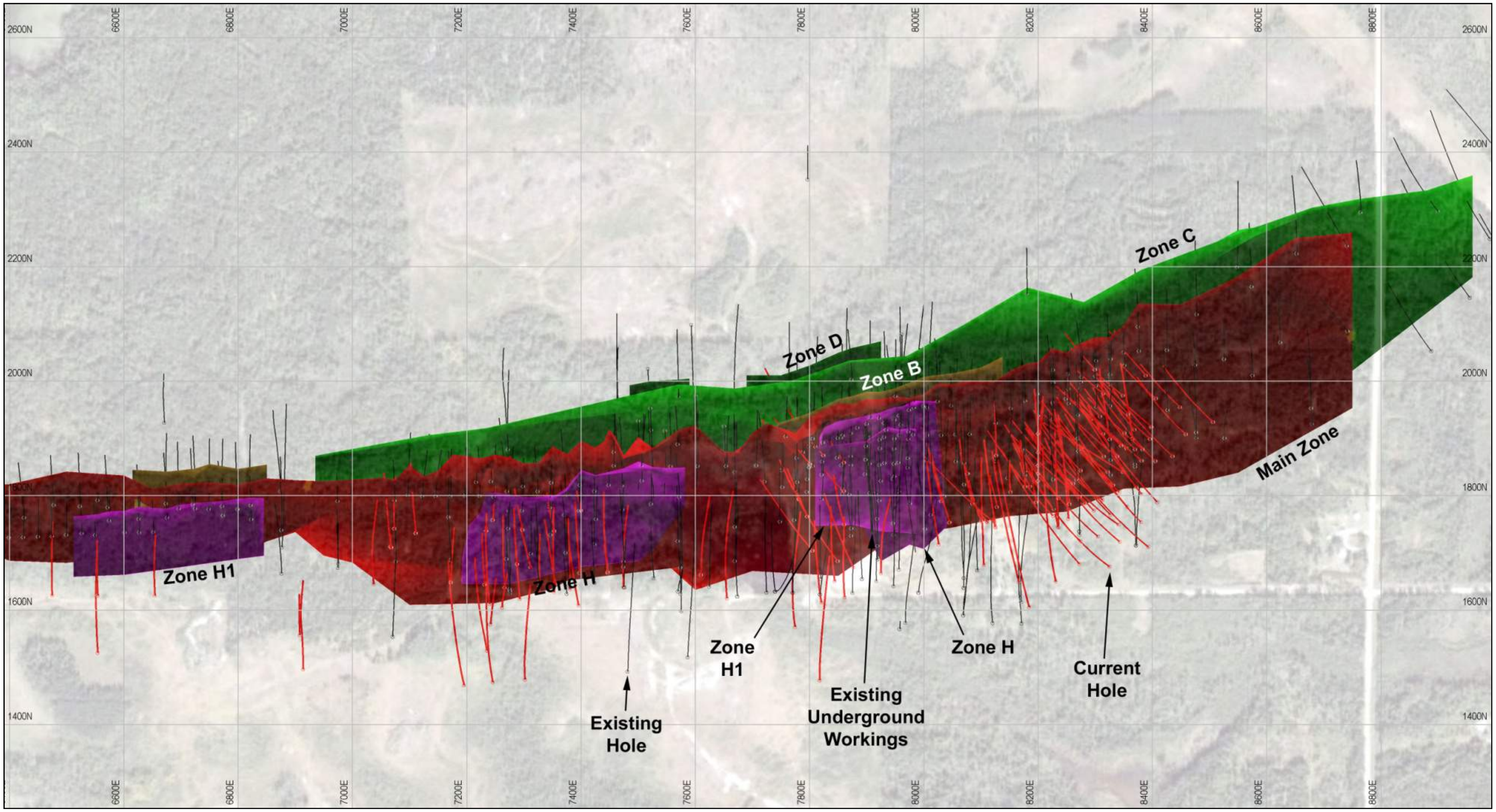


Figure 14-1. Plan view showing older and "current" drilling.

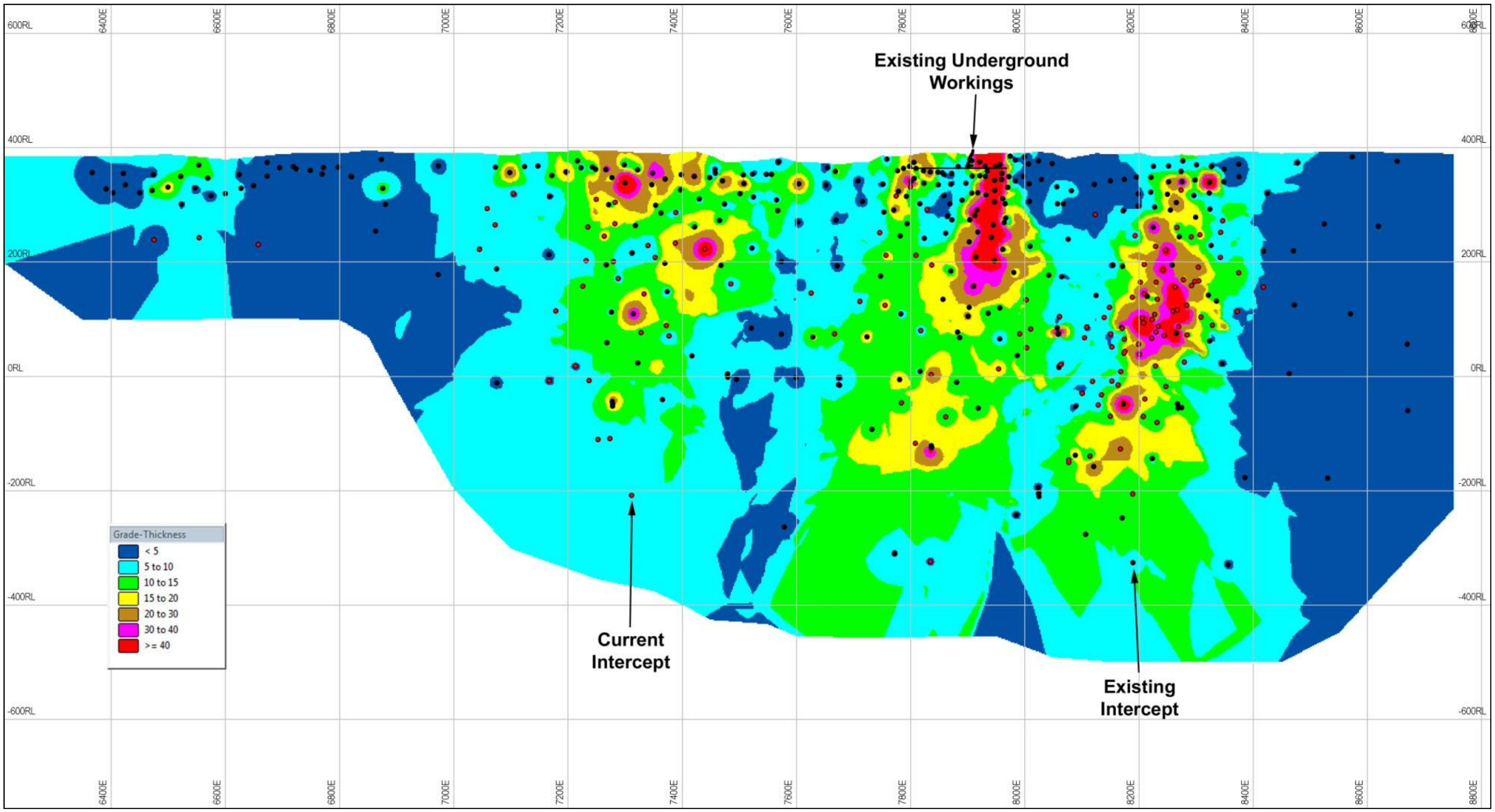


Figure 14-2. Longitudinal section of Main Zone (facing north) showing gram-metres (grade x true thickness), existing intercepts (black) and "current" intercepts (red).

14.3 SITE GRID

UTM coordinates were converted to site grid coordinates by subtracting 520,000 m from the UTM Easting and 5,510,000 from the UTM northing. There was no rotation (i.e.: simple translation).

14.4 GRADE COMPOSITING

To aid the zone interpretation process, the verified assay database "dh-Assay.dat" was grade-composited to highlight assay intervals that exceeded a 0.5 g/tonne cut-off grade over two metres (1 gram·metre).

14.5 MINERALIZED ZONE INTERPRETATION

Mineralized zones were outlined to enforce geological control during block modelling. The interpretations that Howe (2008 and 2010) made during the previous mineral resource estimates were modified slightly according to the following guidelines.

1. A cut-off grade of 0.5 g/tonne of gold was generally used. In other words, mineralized zones were outlined by following "gold-positive" samples. Cut-off grades are further discussed in Section 14.9.
2. The minimum horizontal zone width was approximately 2 metres.
3. Along strike, zones were extended halfway to the next, under-mineralized cross-section.
4. Zones were extended down-dip by a maximum of 150 metres beyond the last intercept.
5. Outlines were refined and smoothed using longitudinal sections of the zones.

Interpretations were accomplished by plotting and interpreting hard-copy cross- and longitudinal sections (refer to Table 14-2 for cross-section definitions; refer to Section 14.15.3 for longitudinal sections for each zone. Those interpretations were digitised and zone intercepts were tagged.

To refine that interpretation, the intercept intervals were manually adjusted within the assay file.

The zone intercepts were further refined using a grade-compositing technique, with the minimum composited grade equal to the cut-off grade (0.5 g/tonne). Zones were allowed to extend through "below cut-off" intercepts so long as there was a "geological reason" to do so.

Figure 14-3 to Figure 14-6 show several cross-sections through one of the richest parts of the deposit – the area around where the underground exploration work was carried out. Figure 14-7 shows a three-dimensional view of the interpreted zones.

The Main and B-Zones often veered towards each other and sometimes merged completely together.

Table 14-2. Cross-section definitions.

Easting	Away	Towards	Width
6380	50.0	12.5	62.5
6405	12.5	7.5	20.0
6420	7.5	15.0	22.5
6450	15.0	12.5	27.5
6475	12.5	12.5	25.0
6500	12.5	10.0	22.5
6520	10.0	15.0	25.0
6550	15.0	12.5	27.5
6575	12.5	12.5	25.0
6600	12.5	12.5	25.0
6625	12.5	12.5	25.0
6650	12.5	12.5	25.0
6675	12.5	10.0	22.5
6695	10.0	12.5	22.5
6720	12.5	15.0	27.5
6750	15.0	25.0	40.0
6800	65.0	10.0	75.0
6820	10.0	27.5	37.5
6875	27.5	50.0	77.5
6975	50.0	47.5	97.5
7070	47.5	15.0	62.5
7100	15.0	15.0	30.0
7130	15.0	10.0	25.0
7150	10.0	10.0	20.0
7170	10.0	15.0	25.0
7200	15.0	35.0	50.0
7220	10.0	10.0	20.0
7240	10.0	15.0	25.0
7270	35.0	25.0	60.0
7320	25.0	22.5	47.5
7365	22.5	27.5	50.0
7420	27.5	25.0	52.5
7470	25.0	22.5	47.5
7515	22.5	27.5	50.0
7570	27.5	22.5	50.0
7615	22.5	27.5	50.0
7670	27.5	25.0	52.5
7720	25.0	15.0	40.0
7750	15.0	12.5	27.5
7775	12.5	12.5	25.0
7800	12.5	12.5	25.0
7825	12.5	12.5	25.0
7850	12.5	12.5	25.0
7875	12.5	12.5	25.0
7900	12.5	25.0	37.5
7950	25.0	12.5	37.5
7975	12.5	12.5	25.0
8000	12.5	25.0	37.5
8050	25.0	15.0	40.0
8080	15.0	15.0	30.0
8110	15.0	30.0	45.0
8170	30.0	15.0	45.0
8200	15.0	25.0	40.0
8225	12.5	12.5	25.0
8250	25.0	12.5	37.5
8275	12.5	12.5	25.0
8300	12.5	12.5	25.0
8325	12.5	12.5	25.0
8350	12.5	12.5	25.0
8375	12.5	25.0	37.5
8425	25.0	25.0	50.0
8475	25.0	37.5	62.5
8550	37.5	45.0	82.5
8640	45.0	50.0	95.0

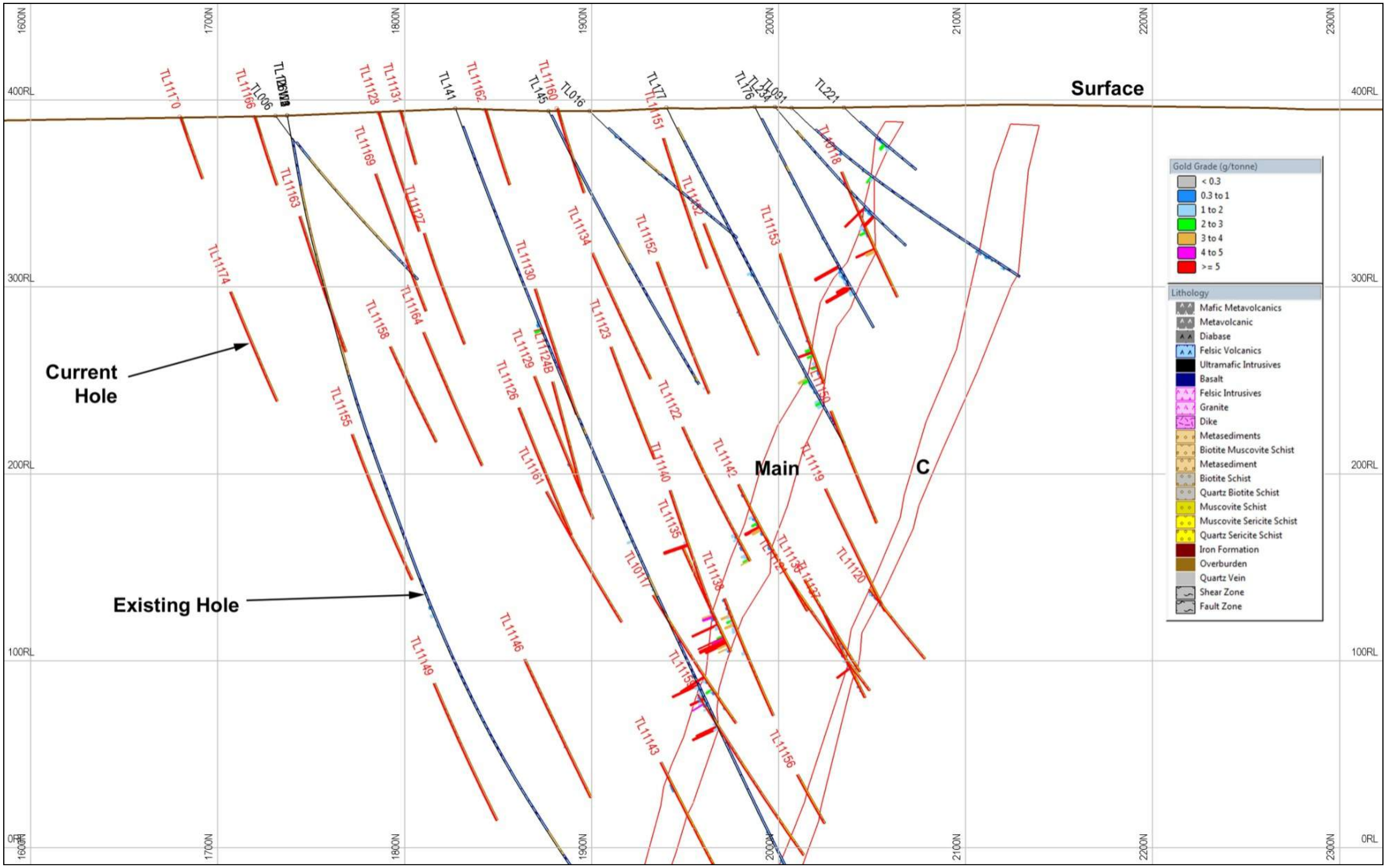


Figure 14-3. Cross-section 8275 m East, facing west.

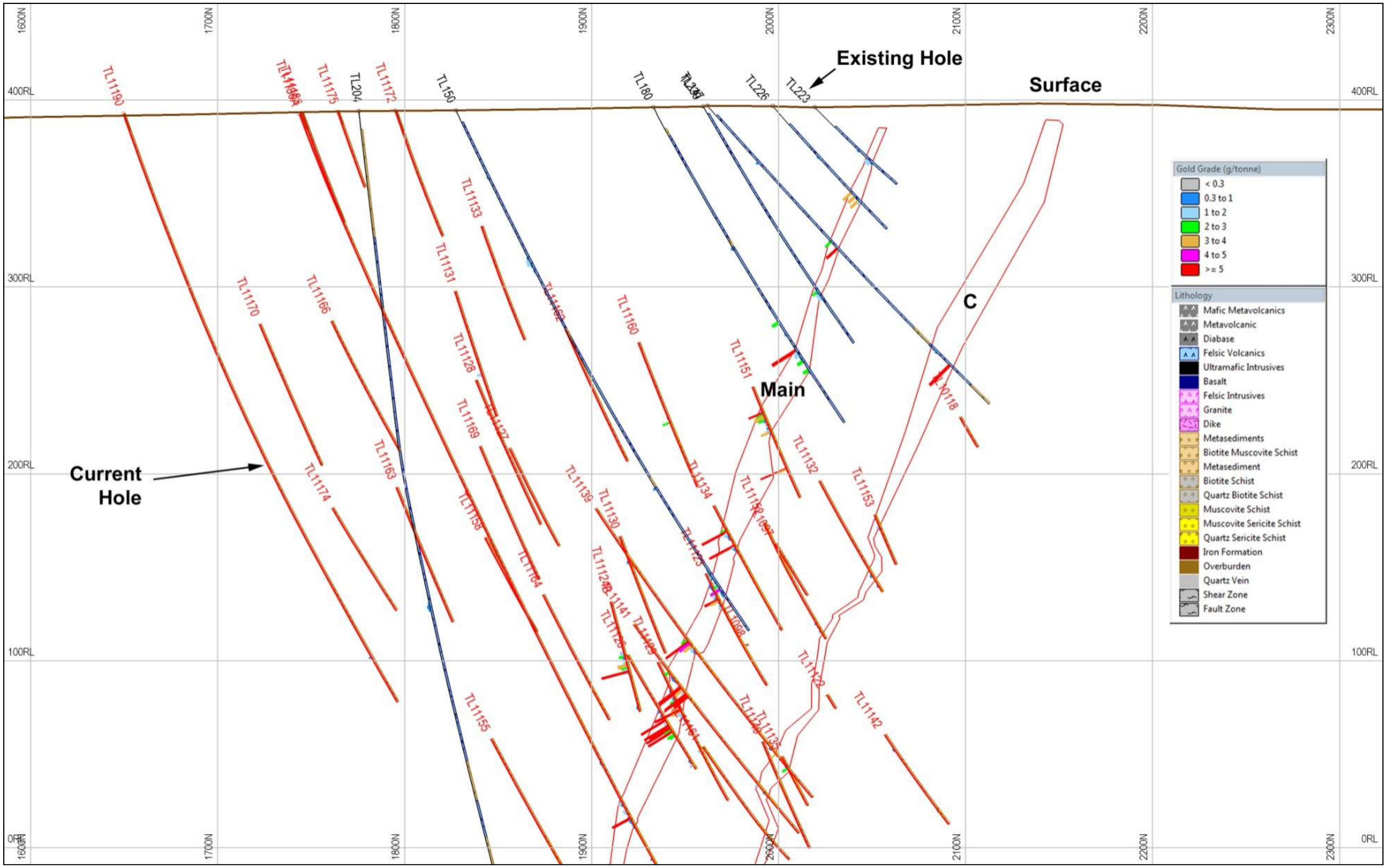


Figure 14-4. Cross-section 8225 m East - facing west.

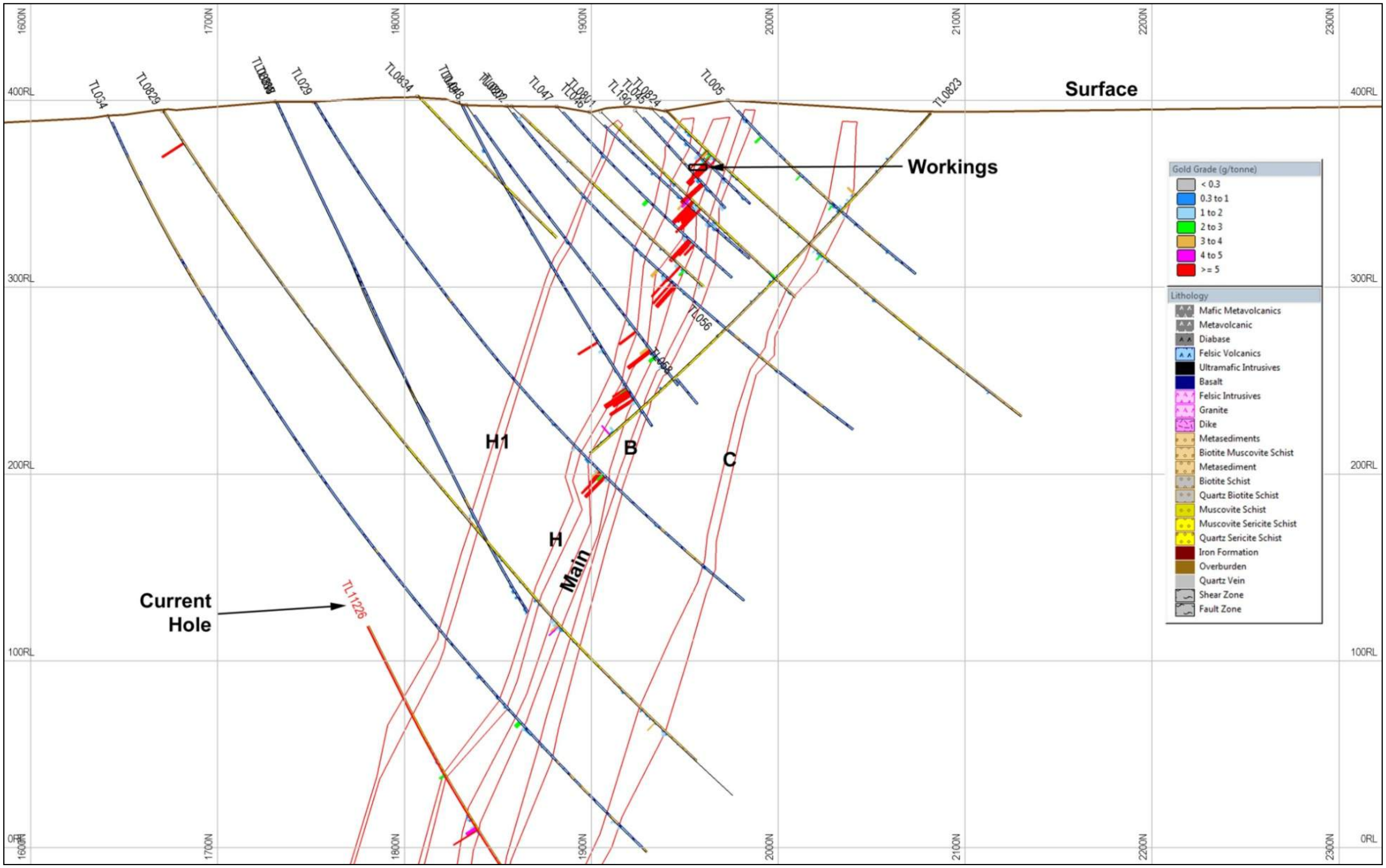


Figure 14-5. Cross-section 7950 m East – facing west.

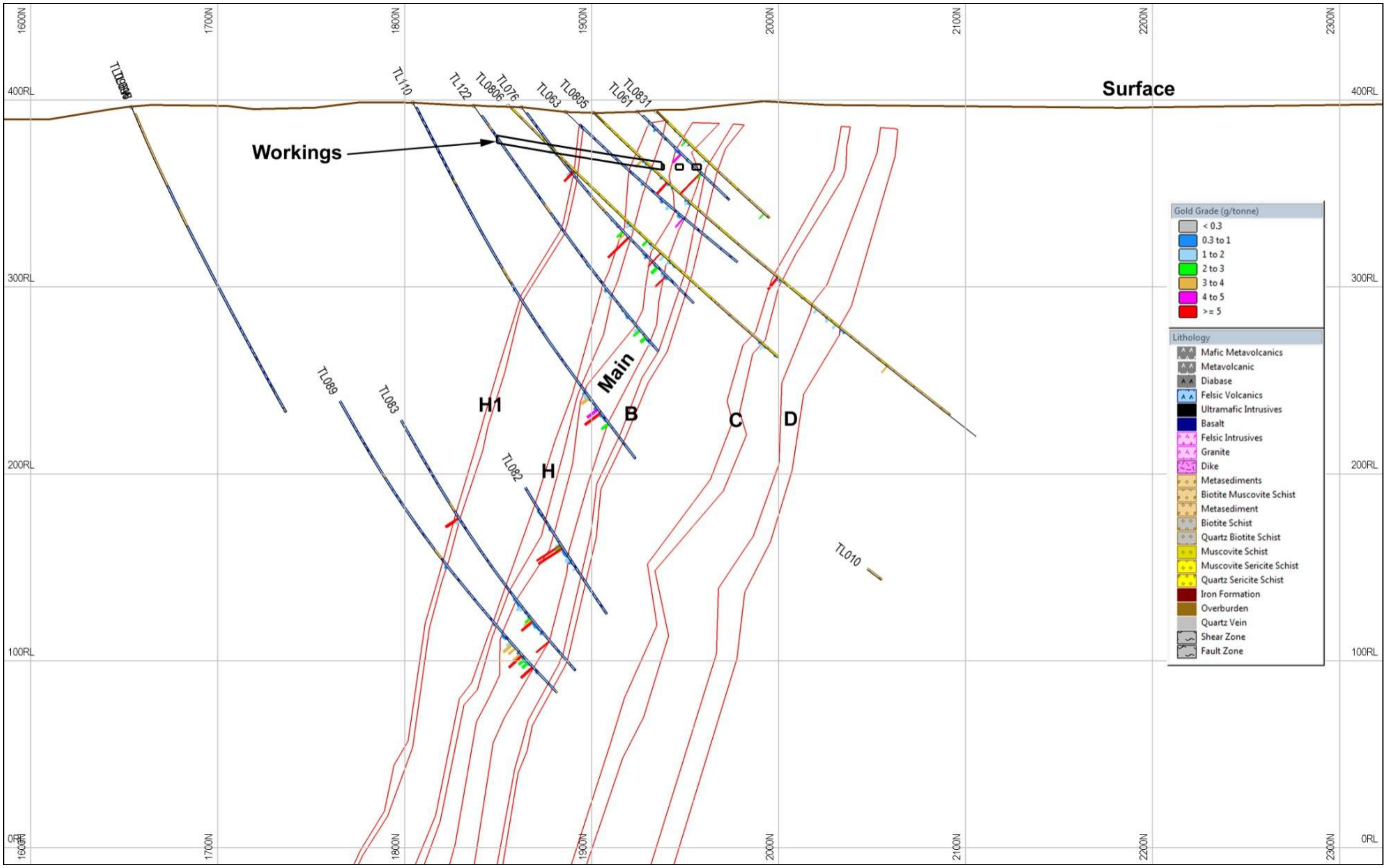


Figure 14-6. Cross-section 7900 m East – facing west.

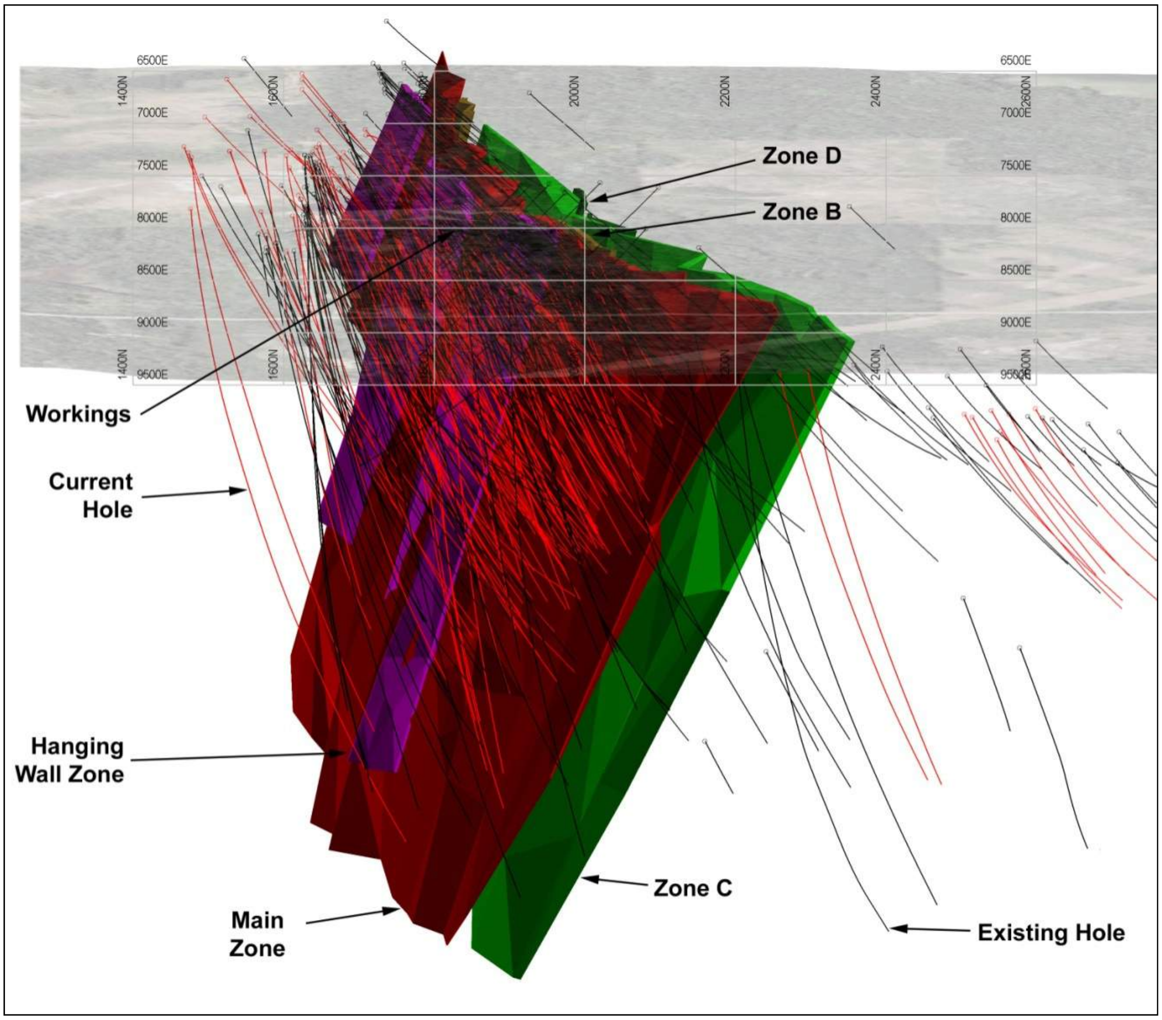


Figure 14-7. 3-D view of outlined zones, facing west.

14.6 STATISTICS

A number of samples (267) were assayed using both fire assay and pulp metallics. The correlation between the two methods was fairly good with a correlation coefficient of 0.9 (refer to Figure 14-8). Meaning, fire assay tended to give slightly higher grades than pulp metallics. In the author's opinion, the difference was acceptable. For conservatism, the pulp metallics result was used over the fire assay result.

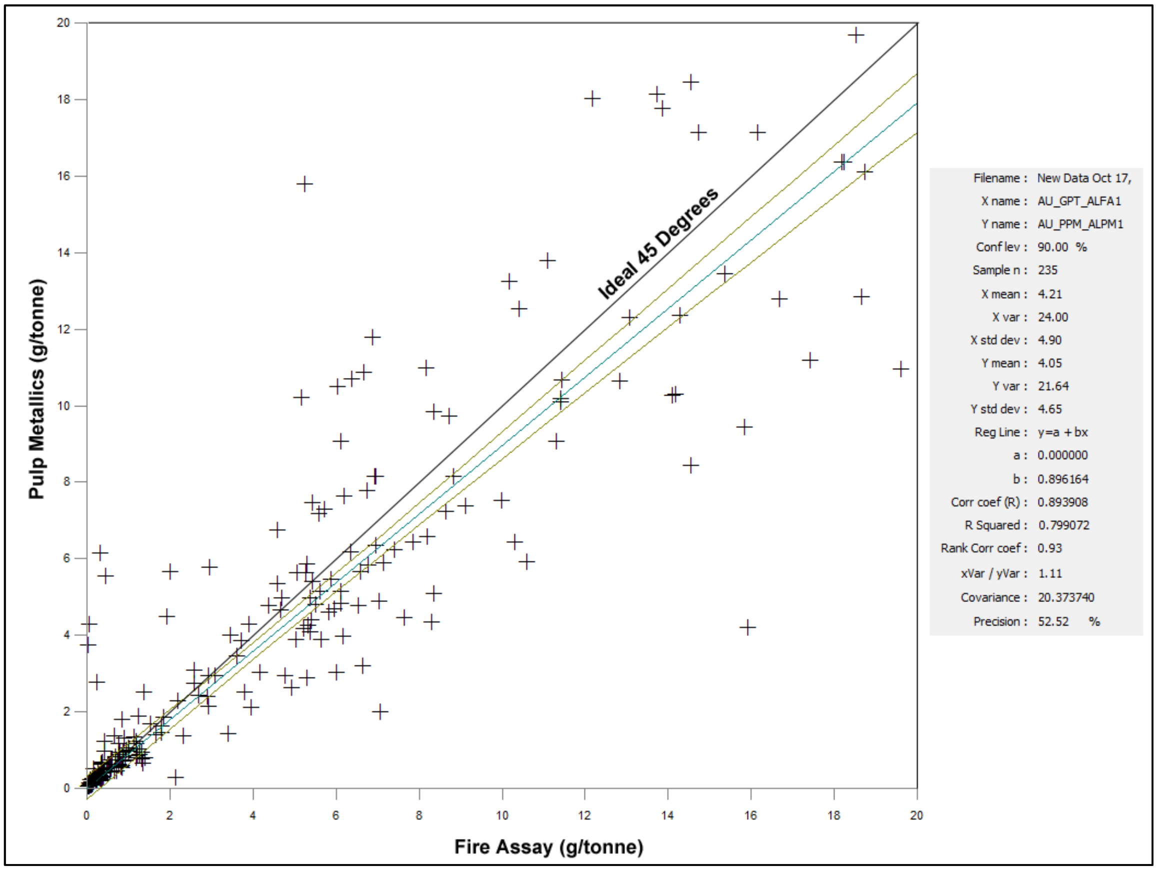
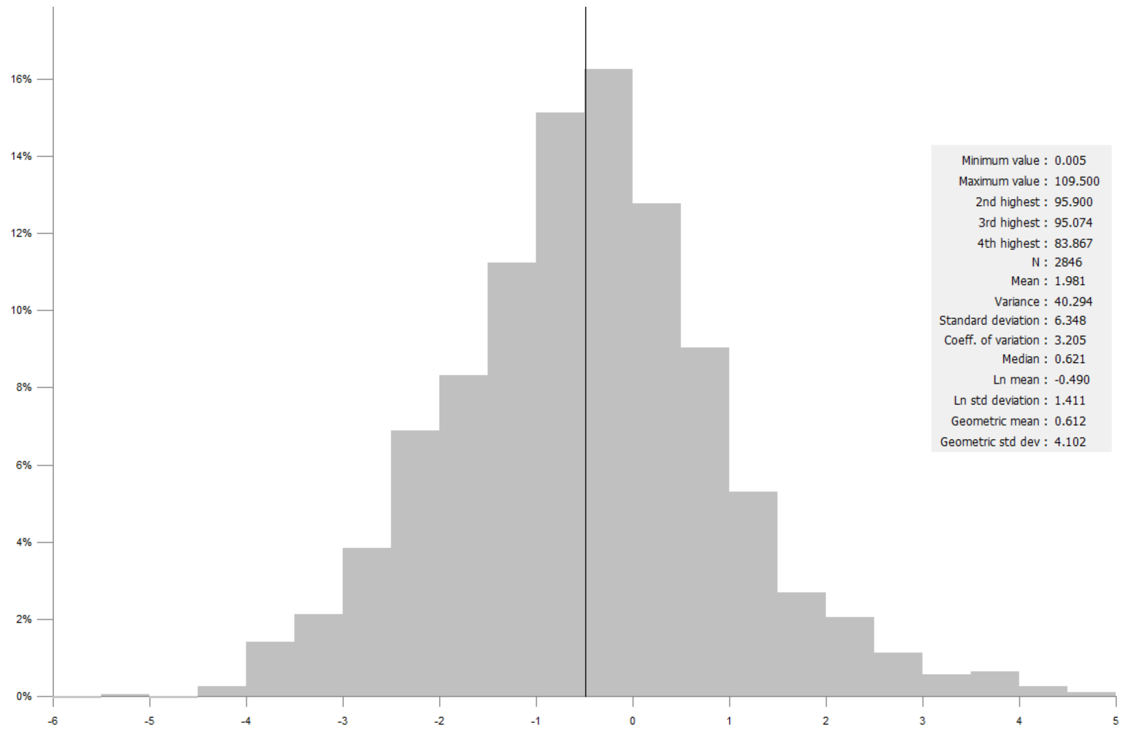


Figure 14-8. Scattergram: pulp metallics *vs.* fire assay.

Statistics were calculated for regularised (over 1.5 metre intervals) samples within the main zone (Figure 14-9). The average grade for the higher-grade domain was 2.0 g/tonne while the average grade for the lower grade domain was less than half that value at 0.9 g/tonne.

<u>Histogram - Main Zone, Higher Grade Domain</u>



<u>Histogram - Main Zone, Lower Grade Domain</u>

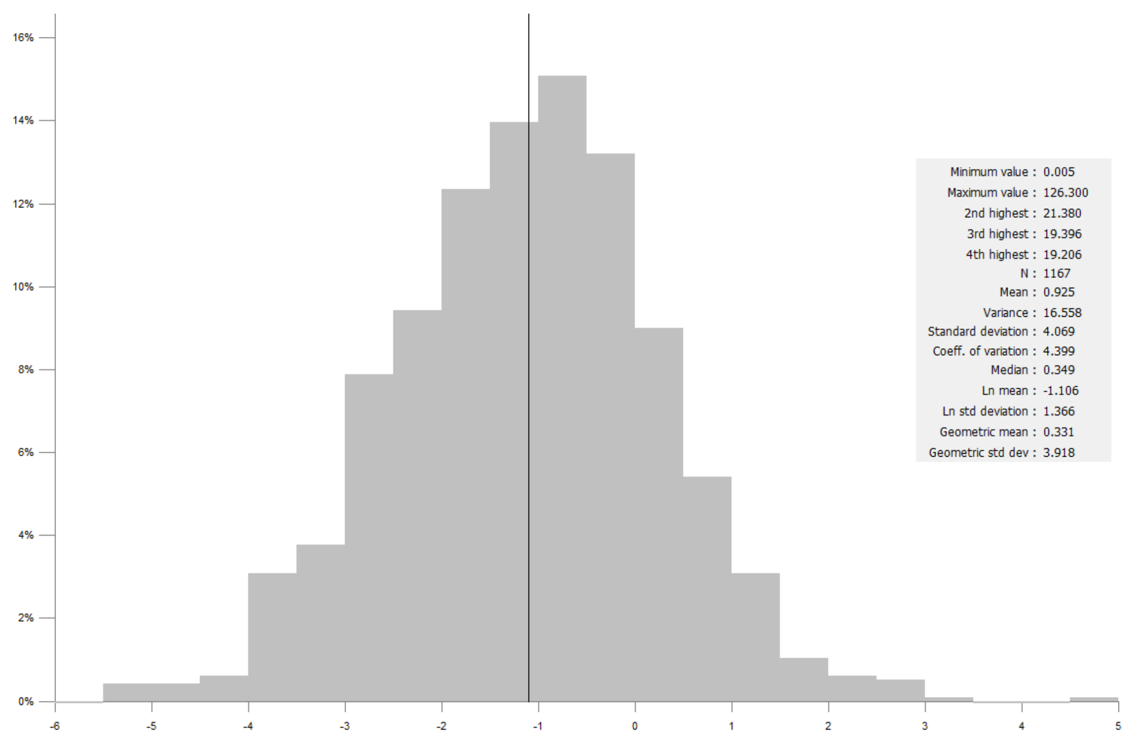


Figure 14-9. Statistics of regularised (1.5 metre), natural log transformed gold assays [Ln (g/tonne)] within the main zone.

The cumulative normal probability versus grade was plotted for each zone (1.5 metre regularised samples) (refer to Figure 14-10). For the main zone, which had (by far) the largest sample population, the inflection point is approximately 4 g/tonne. This *could* indicate two sample populations. Indeed, higher grade and lower grade domains are outlined (refer to Section 14.7.1).

The top end (higher grade end) of the main zone curve remained fairly linear - an indication of a lack of outliers.

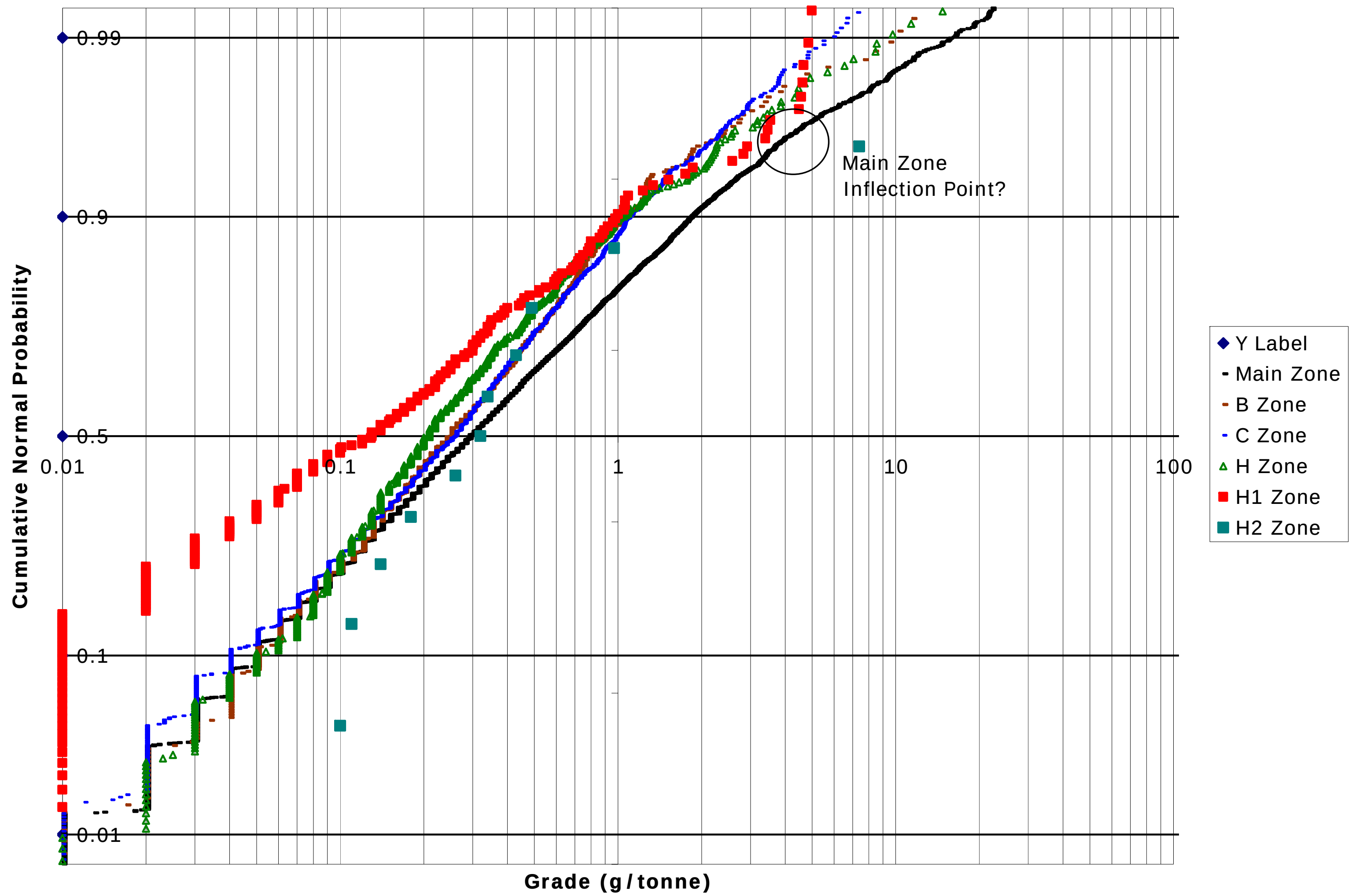


Figure 14-10. Cumulative normal probability for each zone.

14.7 VARIOGRAPHY

14.7.1 Main Zone Domains

A higher-grade domain is outlined in the Main Zone that follows the apparent "shoots". Assays within the shoots were tagged and variography was carried out on them.

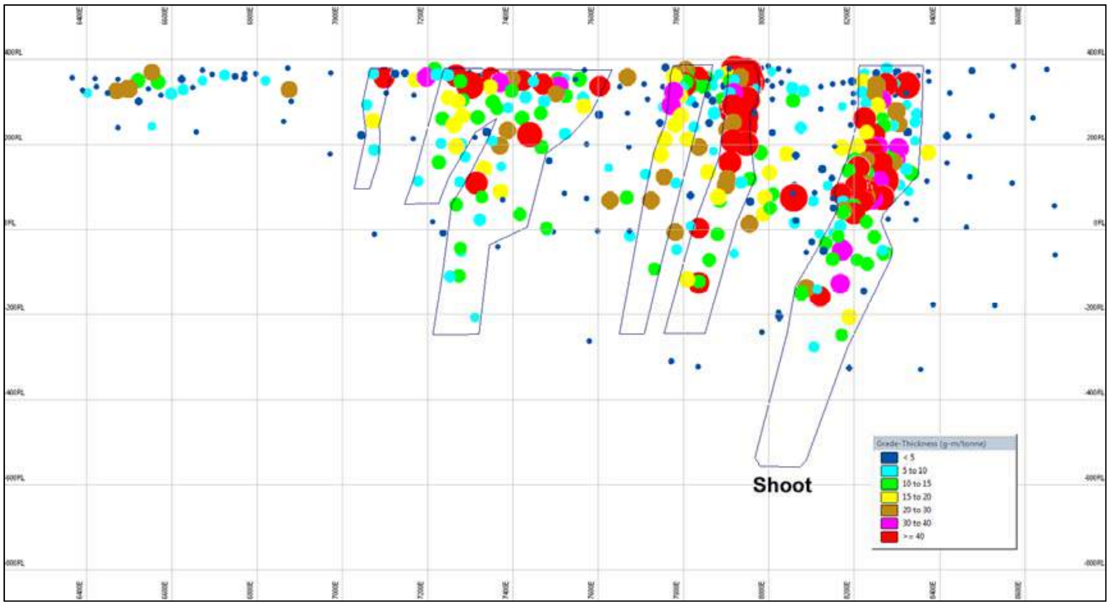


Figure 14-11. Higher-grade domains, Main Zone.

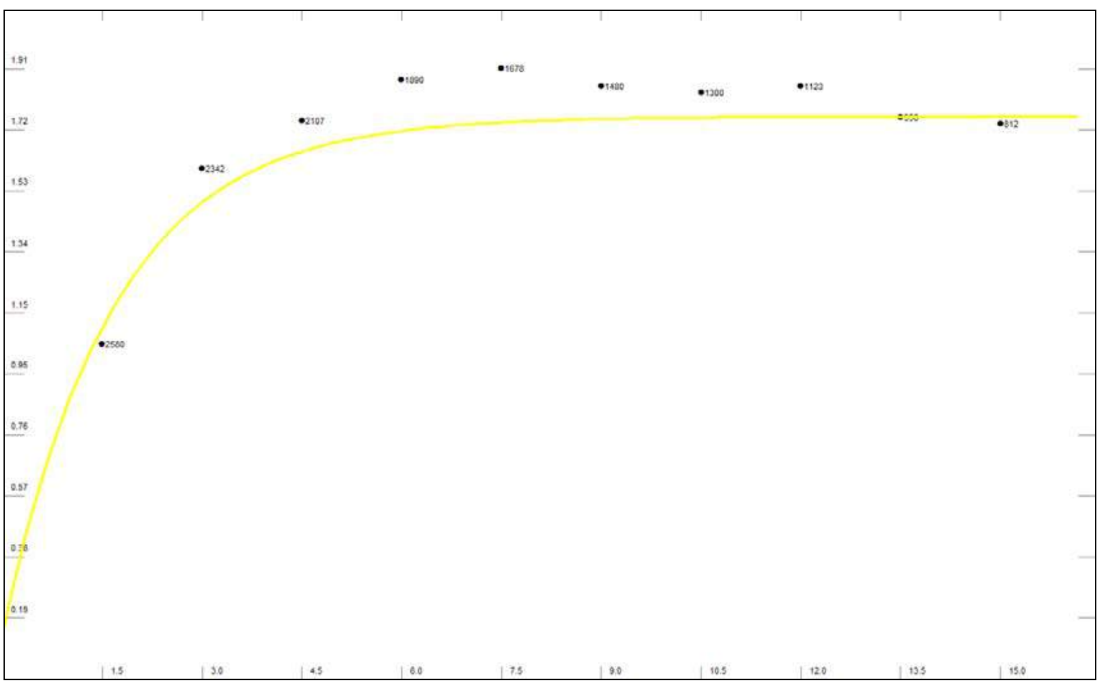


Figure 14-12. Higher-grade domain, downhole (nugget = 0.15, range approx. 5 m, partial sill = 1.61).

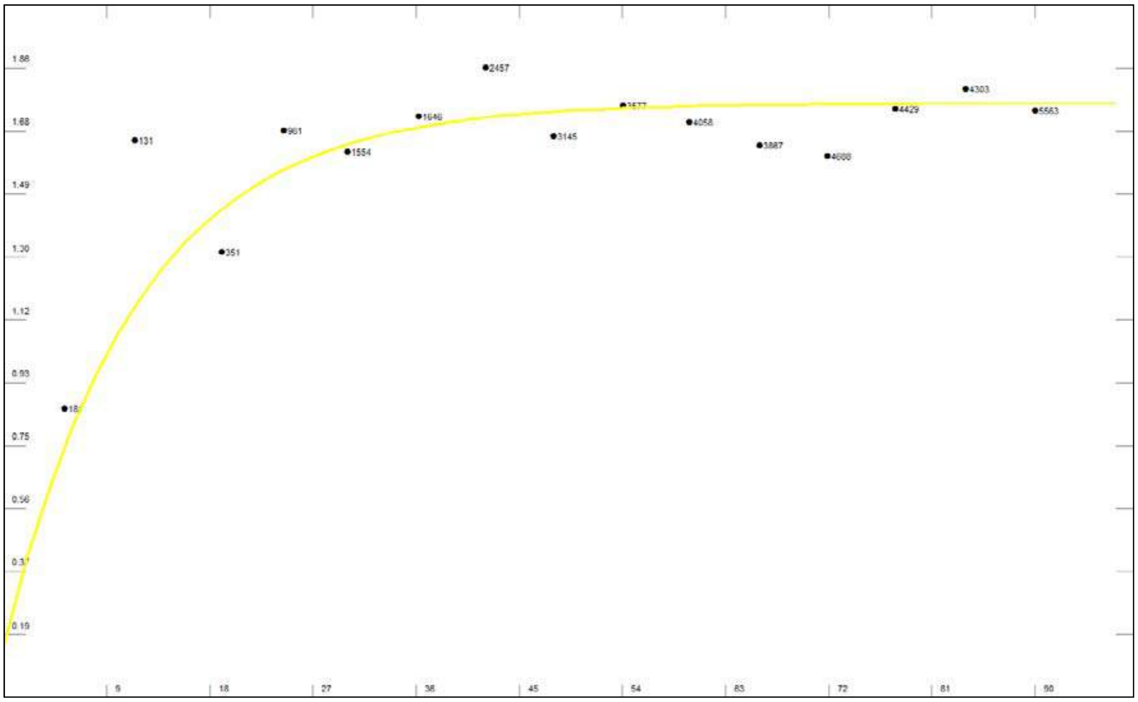


Figure 14-13. Higher-grade domain, along shoot (range = 35 m).

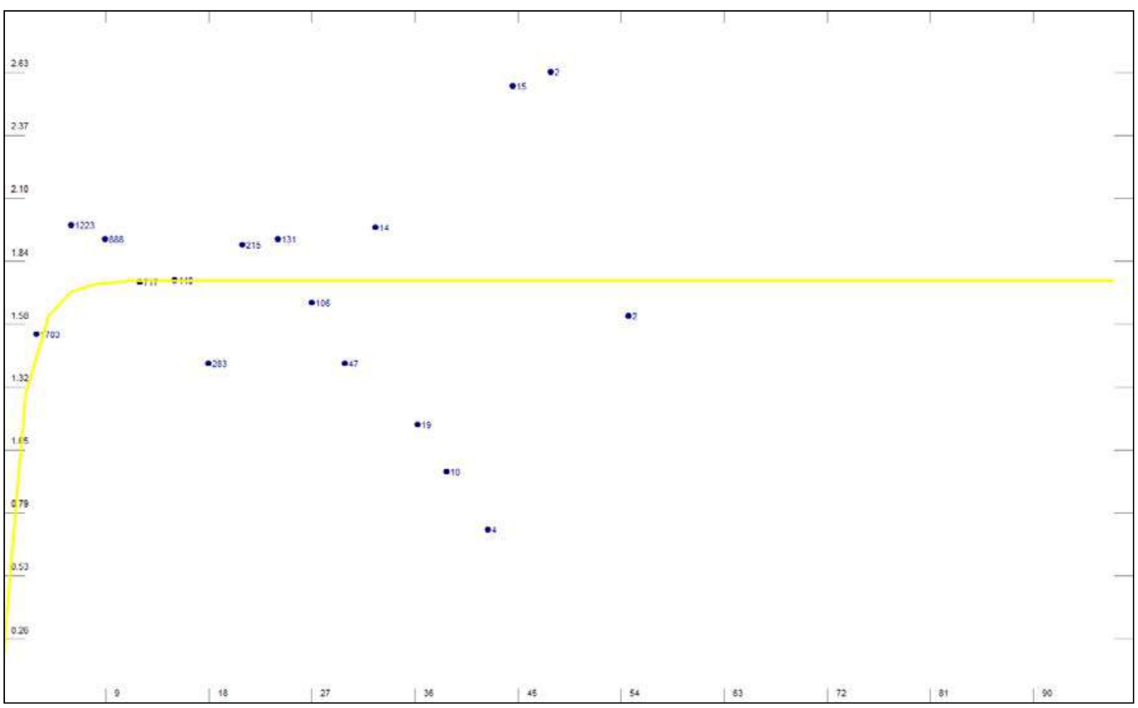


Figure 14-13. Higher-grade domain, across-shoot (very poor quality, range approx. 5 m).

Variography was also carried out for assays that were outside the shoots (i.e.: lower grade domain). The results were similar enough that the same variogram parameters were used for both domains.

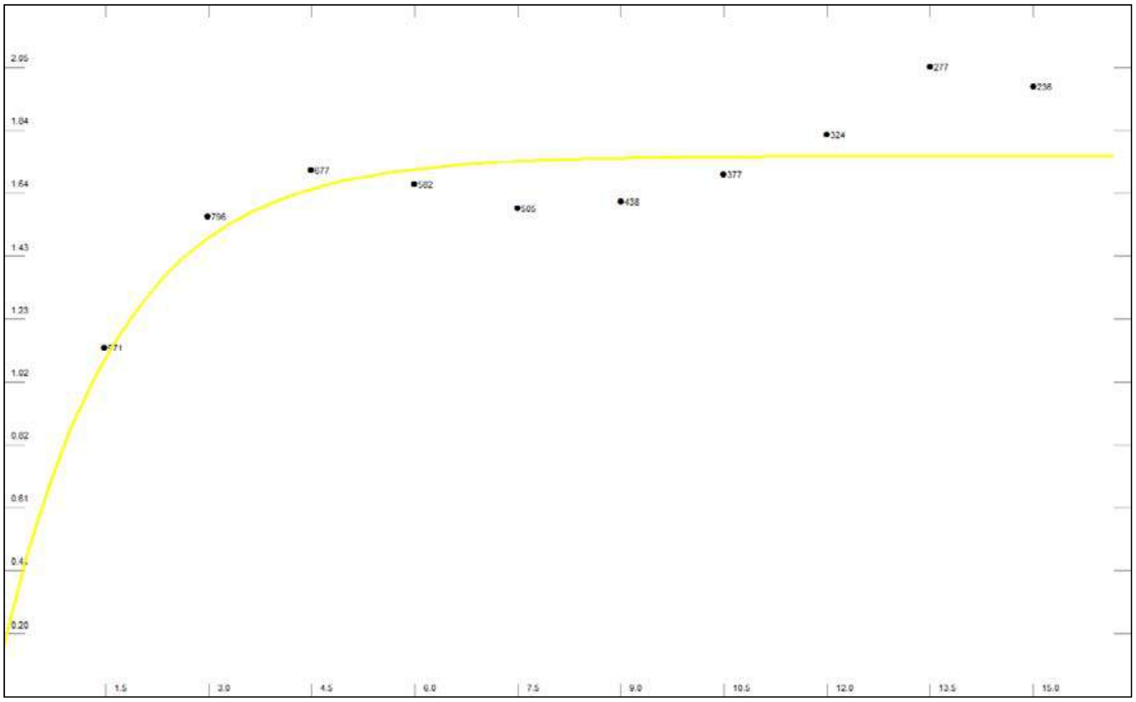


Figure 14-14. Lower-grade domain, downhole (nugget 0.15, partial sill 1.61, range 5 m).

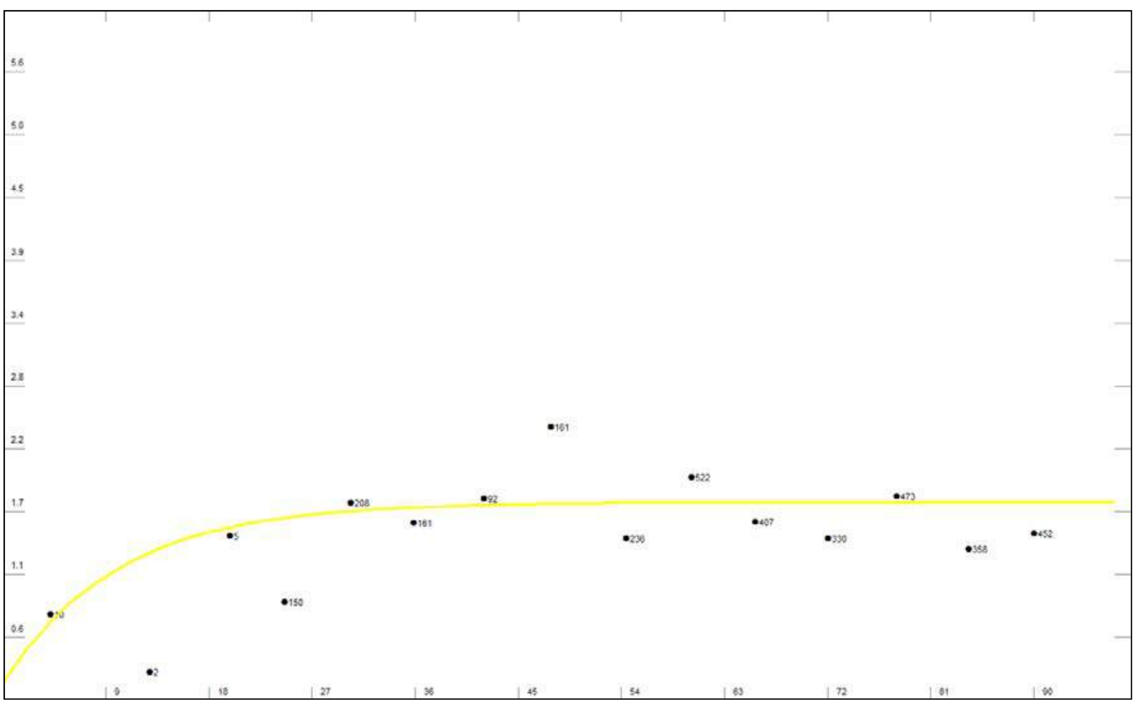


Figure 14-15. Lower-grade domain, down-shoot (range = 30 m).

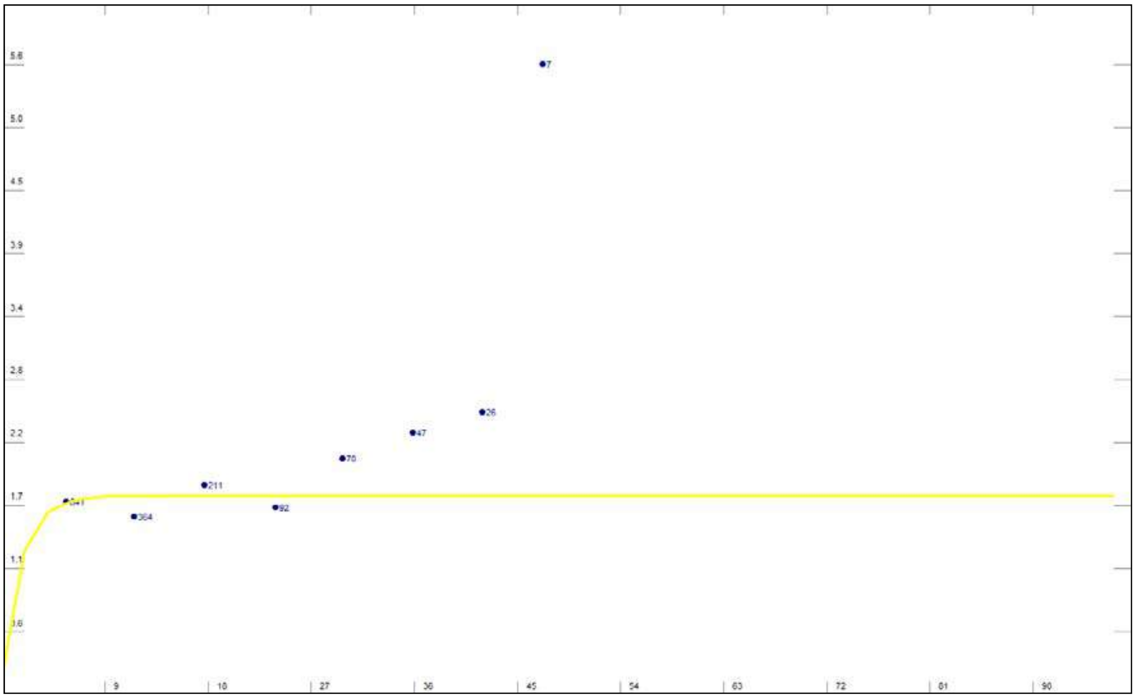


Figure 14-16. Lower-grade domain, across-shoot (very poor quality, range approx. 5 m).

Table 14-3. Variography results for the Main Zone, higher-grade domain.

Direction	Azimuth	Plunge	Data	Model Type	Model Range (m)	Nugget $[Ln(g/t)]^2$	Partial Sill $[Ln(g/t)]^2$	Fit
Normal to Plane of Mineralization (Down-hole)	200	-10 (Up)	1.5 metre Regularised	Exponential	5	0.15	1.61	Very Good
Down-Trend	200	80 (Down)	1.5 metre Regularised	Exponential	35	0.15	1.61	Very Good
Along Strike	290	0	1.5 metre Regularised	Exponential	5	0.15	1.61	Poor

14.7.2 Silver Variography

Variography on silver assays was carried out. Because silver is a minor by-product metal, only downhole and omni-directional semi-variograms were constructed. The quality of each was very good.

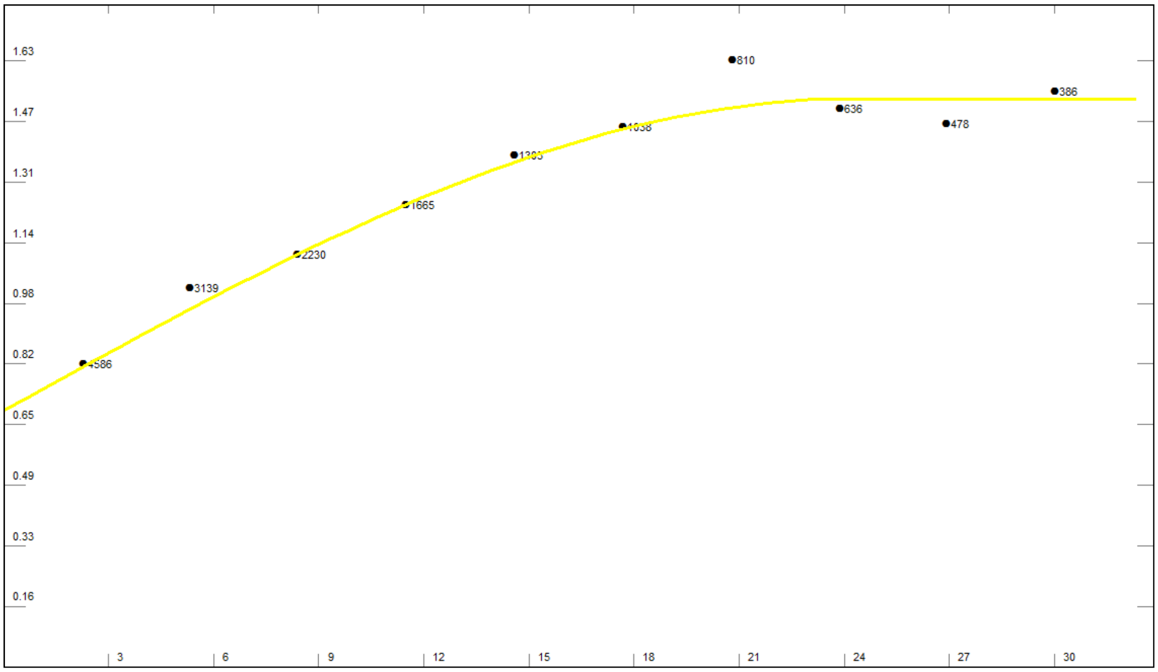


Figure 14-17. Downhole semi-variogram, silver (nugget =0.69).

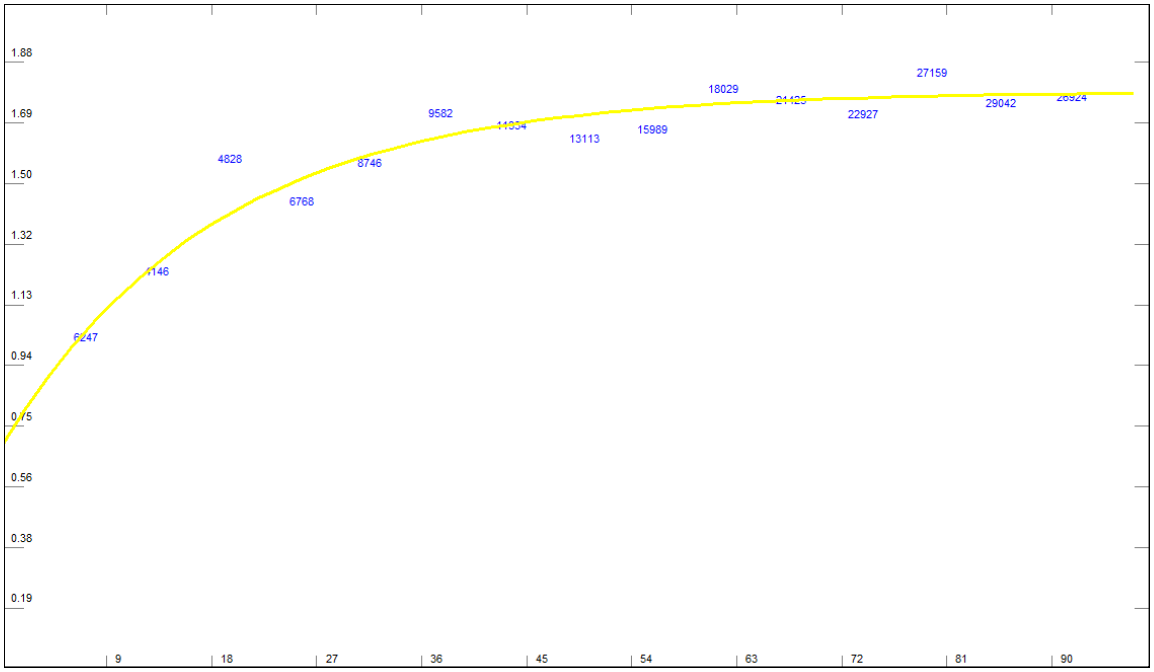


Figure 14-18. Omni-directional semi-variogram, silver (exponential model, range = 55 m, partial sill = 1.10).

14.8 METAL PRICES

A base case value of US$1,375 per ounce, approximately equal to the three-year trailing average, was used for economic modelling.

The three-year trailing average silver price was approximately US$ 26 per ounce.

Higher and lower metal prices were explored using price sensitivity analyses.

14.9 CUT-OFF GRADES

14.9.1 Zone Interpretation

In agreement with the 2010 mineral resource estimate, the chosen cut-off grade for mineralized zone interpretation is 0.5 g/tonne of gold. Considering a typical mining recovery of 95%, a typical overall processing recovery (milling) of 95%, a typical smelter return of 98% and a gold price of US$ 1500 per ounce, rock with that grade would have a revenue of US$ 21. This is a reasonable hybrid value for surface and underground zone interpretation.

14.9.2 Surface Resources

The chosen "block cut-off" grade for defining surface resources (those that would most likely be exploited using surface mining methods) is 0.3 g/tonne. Considering the same parameters as above, rock with that grade would have revenue of approximately US$14 per tonne.

For the 2010 resource report, the limit between "surface resources" and "underground resources" was set at an elevation of 290 metres – approximately 100 metres depth. At that point in time, preliminary optimum pits reached a depth of up to 120 metres. With the increase gold price from 2010, the author felt it prudent to increase the depth cut-off for surface resources to approximately 150 metres (an elevation of 240 metres).

14.9.3 Underground Resources

The chosen "block cut-off" grade for 'underground' resources (those that would most likely be exploited using underground mining methods) is 1.5 g/tonne of gold. Considering the same parameters as above, rock with that grade would have revenue of approximately US$64 per tonne.

14.10 SILVER EQUIVALENCY TO GOLD

Silver could be a by-product metal.

To determine silver's equivalency to gold, the relative prices and processing recovery factors were considered.

At the time of report writing, gold's and silver's relative prices were approximately US$1,500 per ounce and US$35 per ounce, respectively.

Mineral processing test work revealed that typical mill recovery rates for gold and silver were 95% and 72%, respectively.

Considering those factors, 1 gram of gold is equivalent to 57 grams of silver.

$$\text{Eq Grams Gold} = \frac{1 \quad \text{g silver}}{\text{tonne}} \times \frac{35 \quad \text{\$ per ounce silver}}{1500 \quad \text{\$ per ounce gold}} \times \frac{72\% \quad \text{silver recovery}}{95\% \quad \text{gold recovery}}$$

$$= 0.018$$

IE: 1 g silver = 0.018 g gold.
OR, 1 g gold = 57 g silver.

14.11 SPECIFIC GRAVITY

Treasury provided a spreadsheet describing the specific gravity ("SG") measurements for 194 samples. Those samples were imported to Micromine and the samples that were within the mineralized zones were tagged with a zone code. The Main Zone's SG was 2.75, equal to the overall average SG (refer to Table 14-4). That value (2.75) was used as a global average SG for mineral resource estimation.

Table 14-4. Specific gravity measurements.

Zone	Number of Samples	Average SG	Min SG	Max SG
Main	21	2.75	2.68	2.82
B	12	2.78	2.71	2.88
C	12	2.74	2.60	2.82
D	1	2.81	2.81	2.81
Waste	148	2.75	2.59	3.08
Total	**194**	**2.75**	**2.59**	**3.08**

14.12 TOP-CUT GRADE

A top-cut value is normally chosen to prevent the overestimation of block grades by a small number of very high assays or *outliers*.

The cumulative normal probability curves (refer to Figure 14-10) did not reveal the presence of any outliers that could cause an overestimation of block grades.

In the Main Zone, there were 53 samples (out of 5,346 Main Zone samples) that were greater than an ounce per tonne (31.1 g/tonne) and 26 samples greater than two ounces per tonne (62.2 g/tonne).

Because there were relatively few higher-grade samples and no indication, from the cumulative normal probability curve, of the presence of outliers, it was felt that an arbitrary top-cut was not necessary. No top-cut was applied because, in the author's opinion, a top-cut would not affect the global estimate.

14.13 BLOCK MODELING

Blank block models were created with the parameters that were reported in Table 14-5. A blank block model was created for each zone with the file name "Blocks Blank Zone X.dat", where "X" represented the zone name. The blocks were constrained by the mineralized zone wireframes.

The "parent" block size was 5x5x5 metres. That was considered to be the smallest size that could be practically sorted in a surface mining operation. The "smallest block size" is also known as a "selective mining unit," or "SMU."

There were two sub-blocks in the east and elevation (strike and dip, respectively) dimensions for a "geological resolution" of 2.5 metres. There were five sub-blocks in the north dimension (the thickness dimension) for a "geological resolution" of 1.0 metre.

Table 14-5. Block model parameters.

Direction	Model Origin (Grid, m)	Model Limit (Grid, m)	Model Extent (m)	Block Size (m)	Number of Blocks	Number of Sub-blocks
East	6000	9000	3000	5	601	2
North	1400	2800	1400	5	281	5
Elevation (RL)	-600	400	1000	5	200	2

As an artefact of wireframing, there were a few places where the Zone B wireframes overlapped slightly with the Main Zone wireframe. The Zone B blocks that were within the Main Zone wireframe were removed to avoid double-counting blocks.

14.14 GRADE ESTIMATION

In order to adequately represent block grades on a local scale, and also because of the often erratic gold grade distribution along the drilling intercepts, it was the author's opinion that the individual regularised sample grade values, rather than the average intercept grade values, would be more appropriate for use in the grade estimation process.

Ordinary block kriging, along with the semi-variogram parameters that were identified in Section 14.7, was the method that was selected for block grade estimation.

Blocks were discretised twice in all three dimensions. The grade estimation process was carried out separately for each of the zones. Also, for the Main Zone, the higher grade domain was estimated separately from the lower grade domain. A separate block model file was created for each zone and domain, named "Blocks Kriged X.dat", where "X" represented the zone name. The separate files were then "merged" into a single block model file named "Blocks IDS All Zones.dat". A description of that file's fields was reported in Table 14-7.

The grade estimation process was carried out in five "runs" in which the ellipse (really a sphere) radius increased with run. This limited the effect of far-away samples, even when the maximum number of samples had not been reached, when closer samples were available.

All blocks within the outlined mineralized zones were included in the Inferred mineral resource category. Indicated mineral resource blocks were identified using the procedure that was described in Section 14.16.

By far, gold is the most economically important metal. However, silver would likely be a small, but significant by-product. Gold and silver grades were estimated separately.

Table 14-6. Grade estimation parameters.

Parameter	Run 1	Run 2	Run 3	Run 4	Run 5
Min. Number of Holes	1	1	1	1	1
Min. Number of Samples	1	1	1	1	1
Max. Number of Samples	12	12	12	12	12
Search Ellipse Radius (m)*	20	35	70	140	400**

* Search ellipse was spherical in shape.

** The intention of choosing so large a “final run” radius was to “fill up” any remaining blocks that were within the interpreted inferred mineral resource wireframes.

Table 14-7. Block model fields.

Field	Description
East	Easting (Grid)
_East	Block Dimension, East Direction
North	Northing (Grid)
_North	Block Dimension, North Direction
RL	Reduced Level (Grid)
_RL	Block Dimension, North Direction
Zone	Outlined Zone
Domain	Higher or Lower grade domain (Main Zone only).
Resource Category	Resource category.
Au-ppm	Estimated Gold Grade (g/tonne)
Ag-ppm	Estimated Silver Grade (g/tonne)
Points	Number of Samples Used for Estimate
KR_Var	Kriging variance.
KR_StdErr	Kriging standard error.
NumHoles	Number of Holes Used for Estimate
Index	Unique Block ID

Table 14-8. Resulting merged block model files.

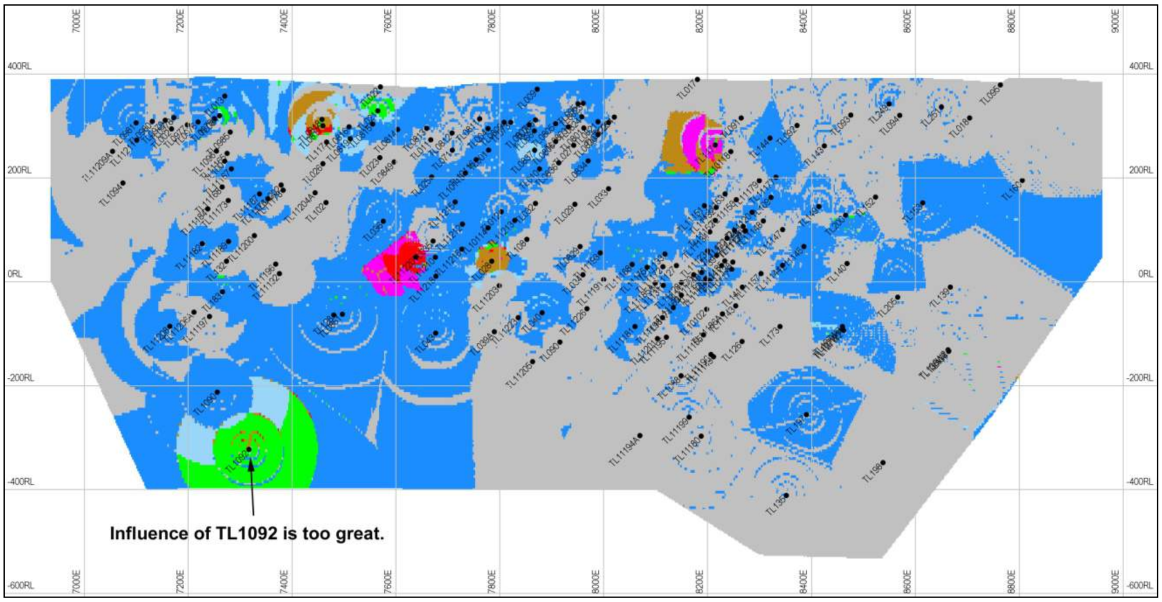
Zone	File
H1	Blocks -Kriged - Zone H1, Rad 400.DAT
H	Blocks -Kriged - Zone H, Rad 400.DAT
Main	Blocks -Kriged - Zone M, Dom LG, Rad 400.DAT (actually contains both higher grade and lower grade domains)
B	Blocks -Kriged - Zone B, Rad 400.DAT
C	Blocks -Kriged - Zone C, Rad 400.DAT
D	Blocks -Kriged - Zone D, Rad 400.DAT

14.15 Need to Limit Certain Holes

14.15.1 Gold

From a preliminary examination of the long sections, it was apparent that Hole TL1092 had too great an influence on Zone C. It was decided to limit the hole`s influence to 70 metres - twice the semi-variogram range of 35 metres.

Before Limiting:



After Limiting:

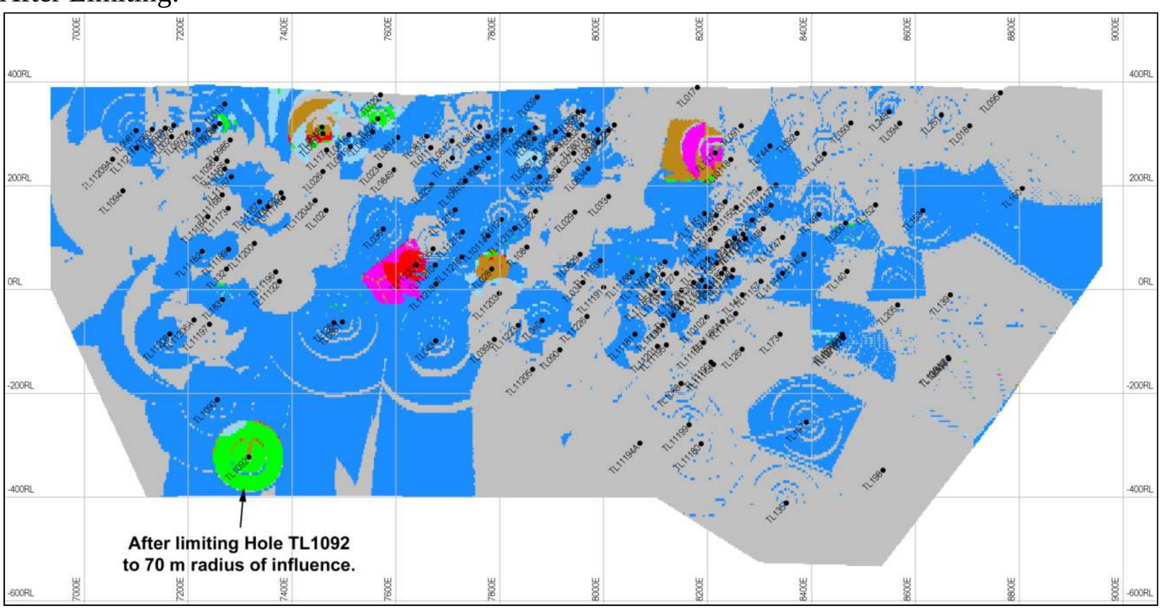


Figure 14-19. Long section of Zone C showing estimated gold grades, before (top) and after (bottom) limiting Hole TL1092`s radius of influence to 70 metres.

14.15.2 Silver

Examination of the physical distribution of block silver grade values in the Main Zone revealed that the zones of influence of two, high-grade silver intercepts were much too large (refer to Figure 14-20). Hole TL043 had a Main Zone silver intercept of 17 g/tonne of silver over 13 metres true width. Hole TL039A had a Main Zone silver intercept of 300 g/tonne of silver over 8.7 metres true width.

The author decided to limit the radius of influence of those two holes to 70 metres during the silver grade estimation process for the Main Zone.

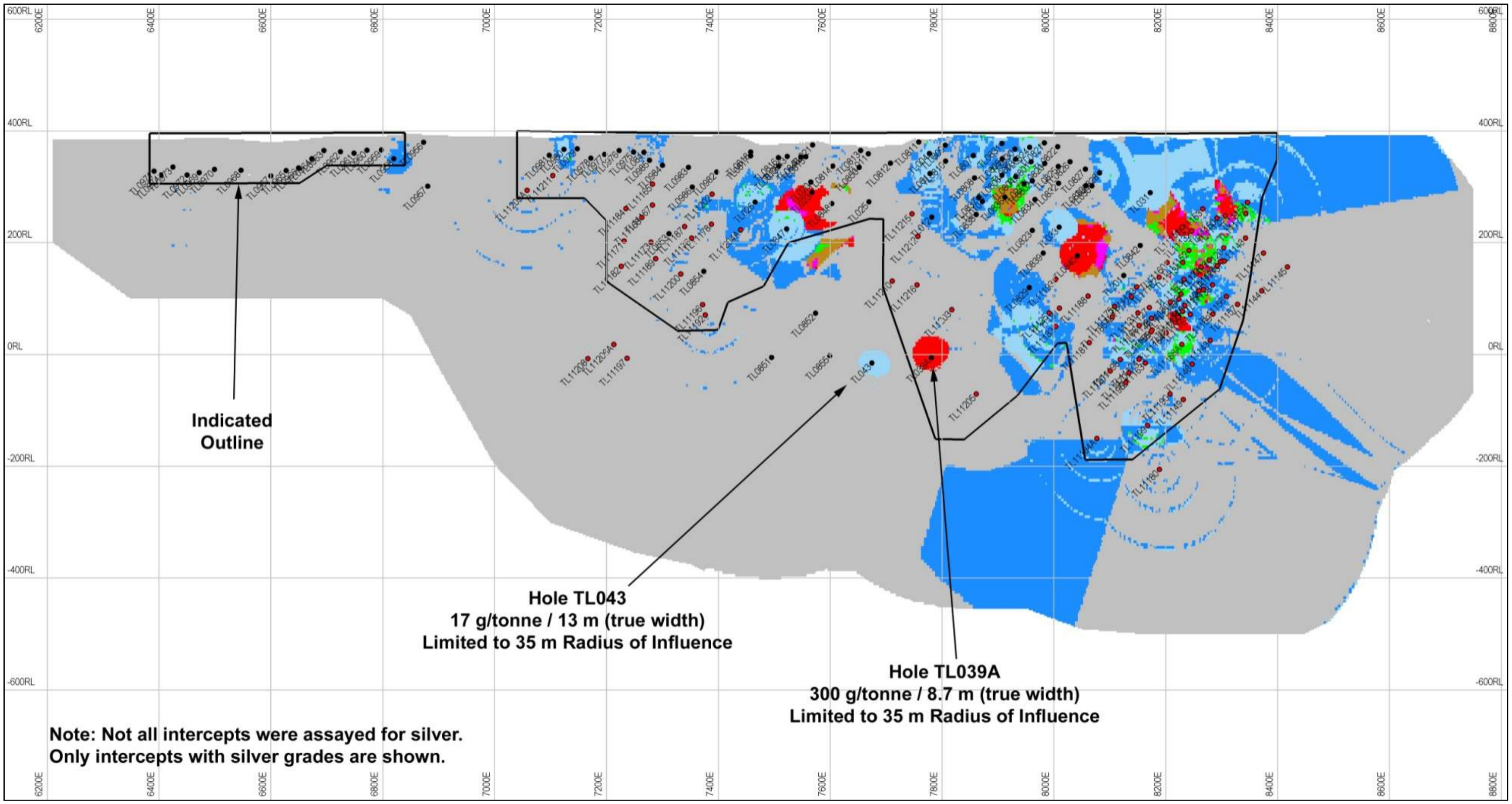
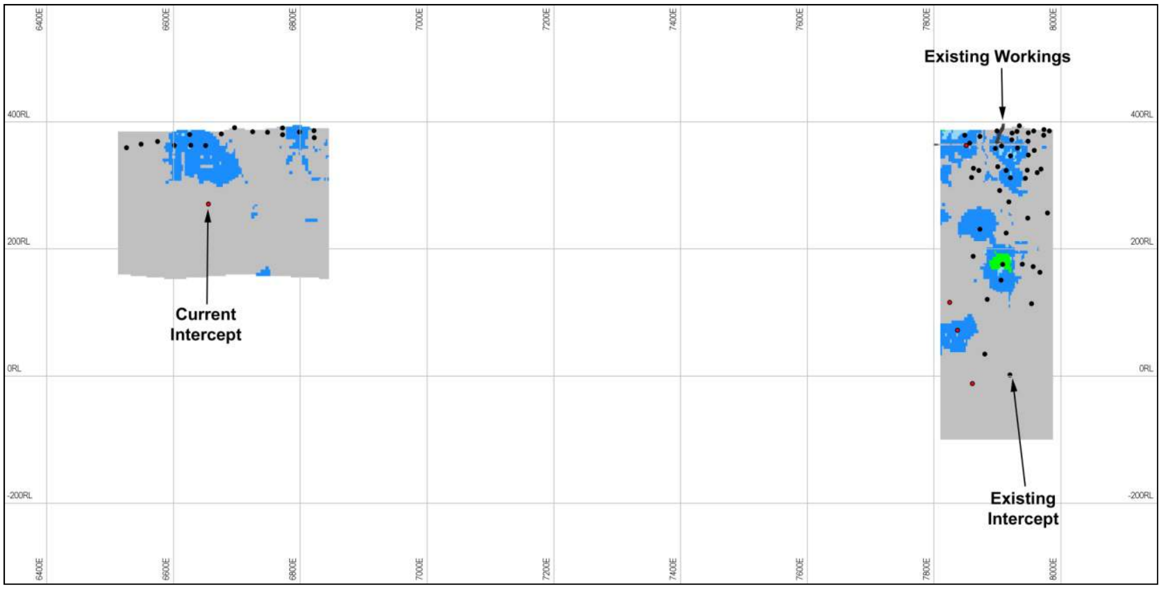


Figure 14-20. Longitudinal section of the Main Zone showing silver assay positions and block silver grade values (facing north, after limiting the influence of Holes TL043 and TL039A).

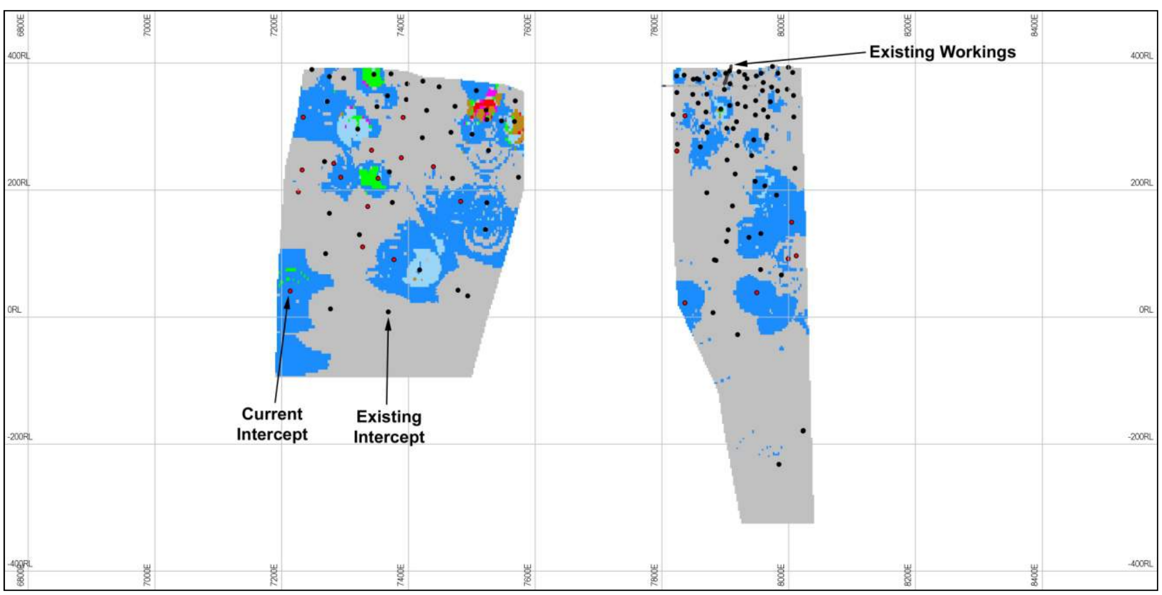
14.15.3 Longitudinal Sections Showing Gold Grades

From hanging wall to footwall, south to north, following are longitudinal sections showing the current mineral resource estimate.

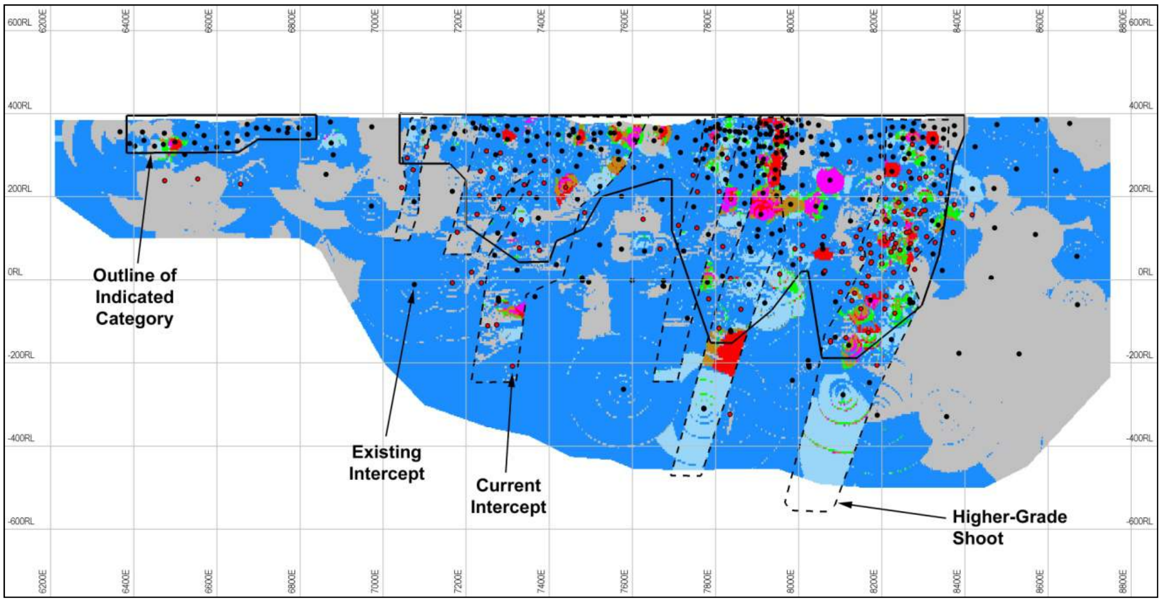
Zone H1:



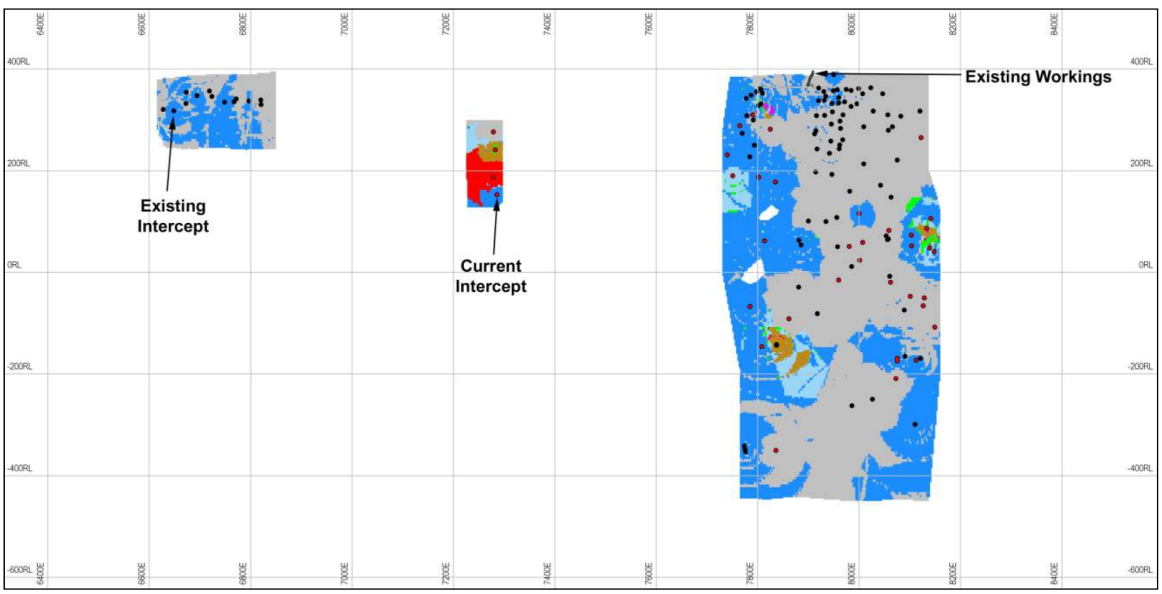
Zone H:



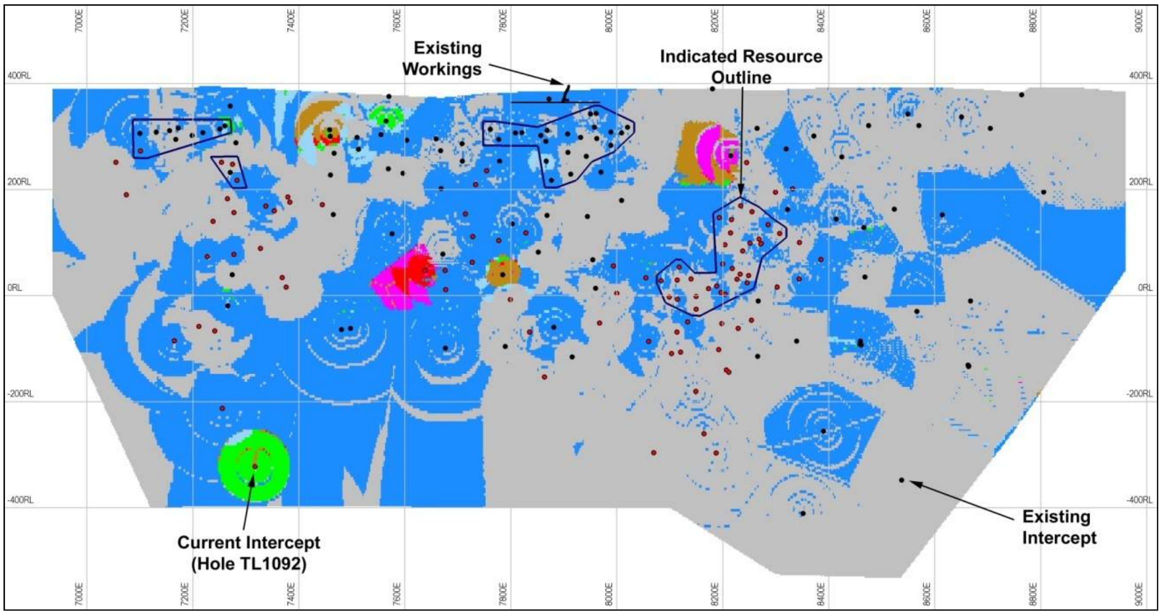
Main Zone:



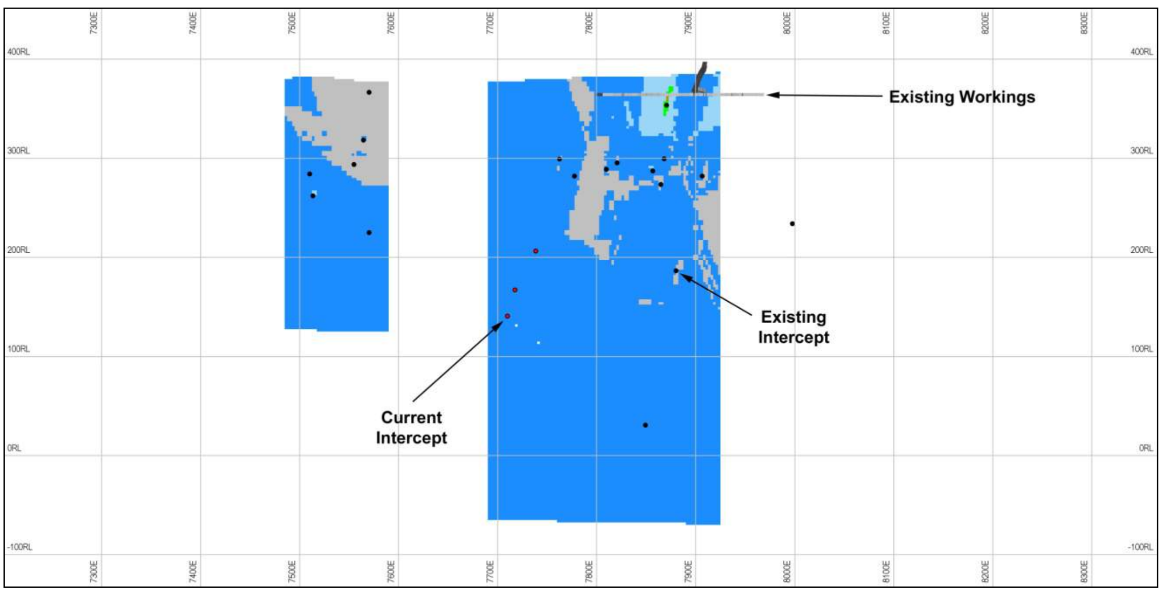
Zone B:



Zone C:


Existing Workings
Indicated Resource Outline
Current Intercept (Hole TL1092)
Existing Intercept

Zone D:


Existing Workings
Existing Intercept
Current Intercept

14.16 RESOURCE CLASSIFICATION PARAMETERS

Resource parameters were chosen based on a combination of variography results and the author's judgement. The degree of confidence in the reported resources was classified based on the validity and robustness of input data and the proximity of resource blocks to sample locations. Resources were reported, as required by NI 43-101, according to the CIM Standards on Minerals Resources and Reserves.

Rather than classifying resources using the search ellipse parameters (Table 14-6) Inferred resources were outlined graphically, on cross- and longitudinal sections using the process that was described in Section 14.5. In other words, all blocks that were within the outlined mineralized zones were considered to be (at least) Inferred.

The semi-variogram data for the main zone intercepts reached its ceiling value by approximately 45 metres lag. In other words, intercepts spaced 45 metres apart, or greater, are unrelated with respect to gold grade.

Because the Indicated resource category requires confidence in both geological and grade continuity, the intercept spacing would have to be less than the Main Zone range of 45 metres. In the author's opinion, a sample intercept spacing of 30-35 metres (up to approximately 80% of the variogram range) would be adequate for identifying Indicated resources in the Main Zone where the geological continuity has already been well established.

For the C-Zone, the range was approximately 30-35 metres (refer to Figure 14-22). Applying the same 80% factor to the C-Zone, the sample intercept spacing would have to be approximately 25 metres for outlining Indicated resources.

In the author's opinion, geological continuity has been well established for much of the Main Zone and parts of the C Zone. The other zones are less predictable and should stay entirely in the Inferred category, at least until more work indicates otherwise.

Indicated Resources were outlined graphically in the Main Zone on longitudinal sections within areas where the intercept spacing was approximately 35 metres or less in two dimensions. For the C-Zone, the maximum spacing (in two dimensions) for Indicated resources was 25 metres.

To aid the Indicated outlining process, longitudinal sections were made that showed 17.5-m-radius circles around the Main Zone intercepts and 12.5-m-radius circles around the C Zone intercepts (refer to Figure 14-23 and Figure 14-24, respectively). Outlines were drawn around areas where the circles were generally touching or very close to touching, in two dimensions.

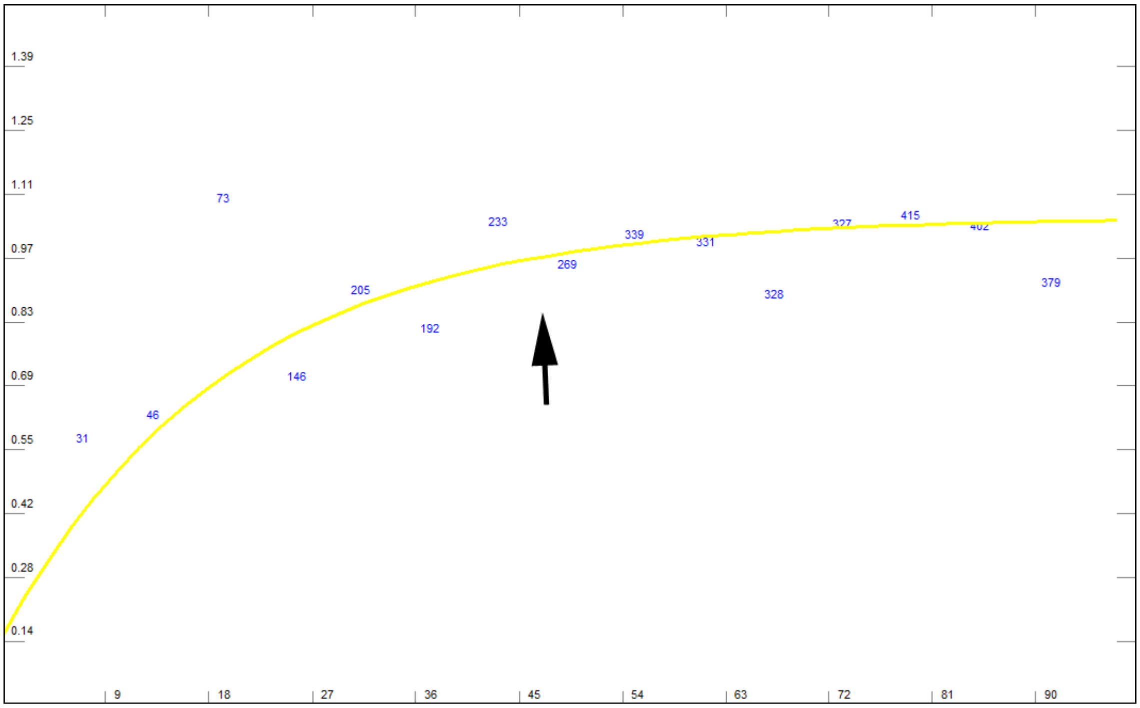


Figure 14-21. Semi-variogram of main zone intercepts (omni-directional). The arrow indicates the approximate range.

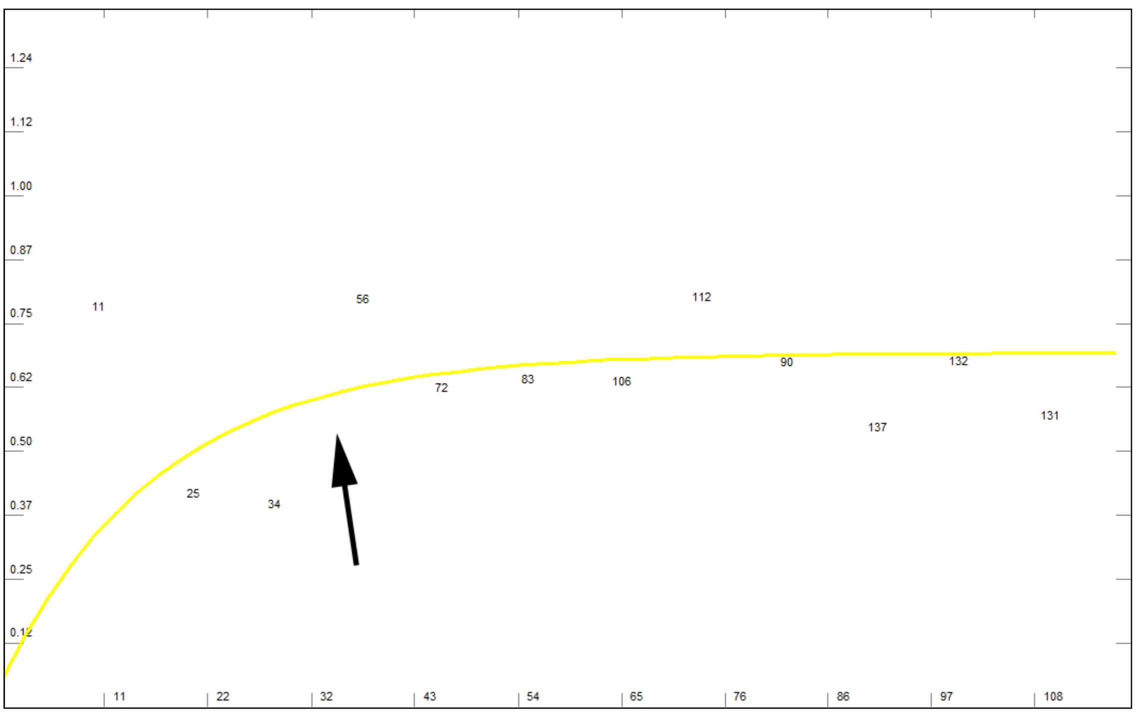


Figure 14-22. Semi-variogram of C-Zone intercepts (omni-directional).

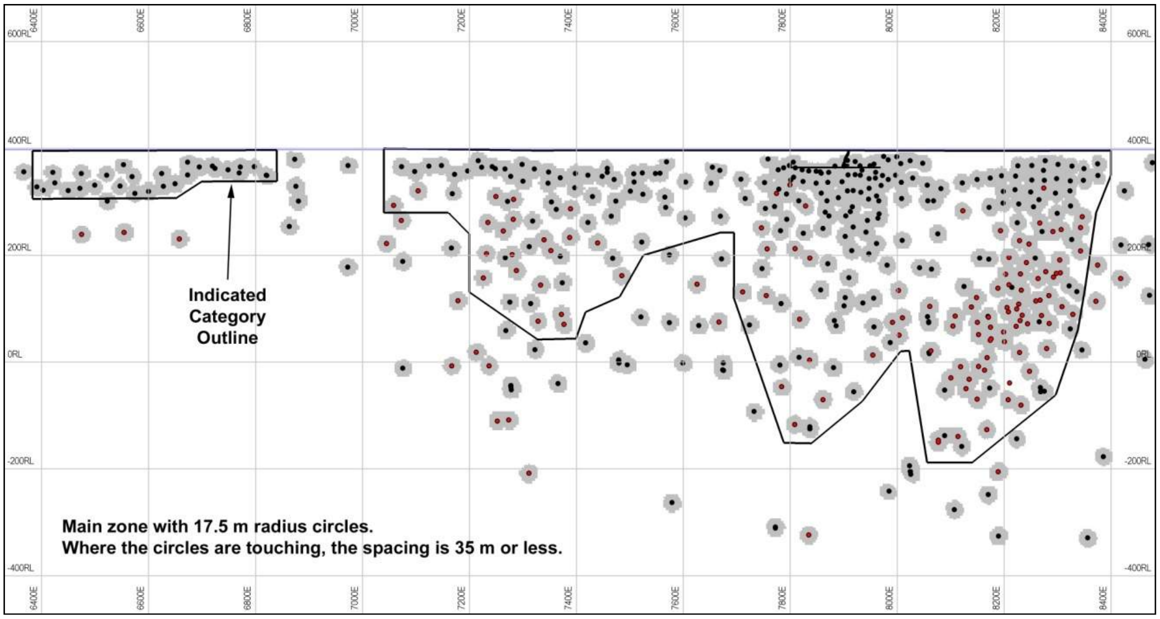


Figure 14-23. Outline of Indicated resources (black line) in the Main Zone.

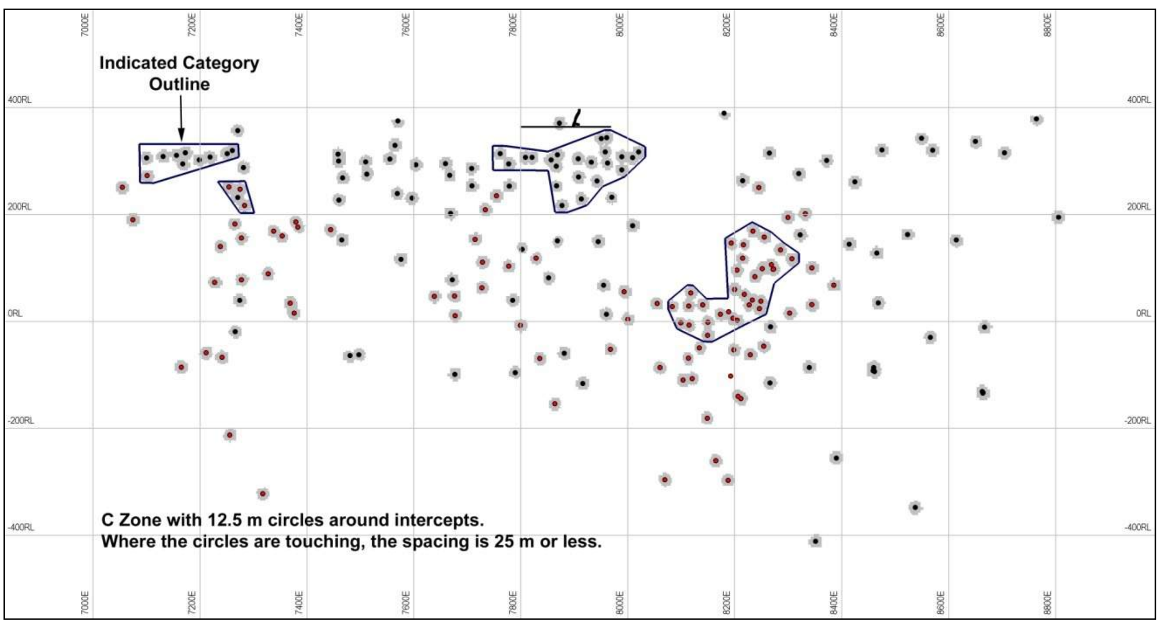


Figure 14-24. Outline of Indicated resources in the C Zone.

14.17 RESULTS

Resources were defined using a block cut-off grade of 0.3 g/tonne for surface resources (less than 150 metres deep) and 1.5 g/tonne for underground resources.

Non-diluted Indicated Mineral Resources (Surface plus Underground), located within the Main Zone and C-Zone, totalled 9.1 million tonnes with an average gold grade of

2.6 g/tonne and an average silver grade of 10.4 g/tonne, for 810,000 ounces of gold and gold equivalent.

Non-diluted Inferred Mineral Resources (Surface plus Underground), from all zones, totalled 15.9 million tonnes with an average gold grade of 1.7 g/tonne and an average silver grade of 3.9 g/tonne, for 900,000 ounces of gold and gold equivalent.

14.17.1 By-Product Base Metals

Lead, zinc and to a lesser extent, copper may be significant by-product metals. Should flotation be used as a mineral processing method, it is possible that by-product grades would be high enough to earn some smelter return.

Table 14-9. Summary of non-diluted mineral resources.

Category	Surface or Underground	Cut-Off Grade (g/tonne)	Tonnes	Gold Grade (g/tonne)	Silver Grade (g/tonne)	Gold Ounces	Silver Ounces	Gold Equivalent Ounces (of Silver)	Ounces Gold Plus Gold Equivalent
Indicated	Surface	0.30	6,002,000	1.8	7.1	326,000	1,257,000	22,000	348,000
Indicated	Underground	1.50	3,136,000	4.3	18.0	433,000	1,812,000	32,000	465,000
Total Indicated (Rounded)			**9,140,000**	**2.6**	**10.4**	**760,000**	**3,070,000**	**54,000**	**810,000**
Inferred	Surface	0.30	11,093,000	1.0	3.3	352,000	1,184,000	21,000	374,000
Inferred	Underground	1.50	4,789,000	3.3	5.2	514,000	807,000	14,000	528,000
Total Inferred (Rounded)			**15,900,000**	**1.7**	**3.9**	**870,000**	**1,990,000**	**35,000**	**900,000**

Table 14-10. Non-diluted mineral resources by zone.

Zone	Category	Surface or Underground	Cut-Off Grade (g/tonne)	Tonnes	Gold Grade (g/tonne)	Silver Grade (g/tonne)	Gold Ounces	Silver Ounces	Gold Equivalent Ounces (of Silver)	Ounces Gold Plus Gold Equivalent
Main	Indicated	Surface	0.30	5,314,000	1.8	7.1	308,000	1,213,000	21,000	329,000
Main	Indicated	Underground	1.50	3,127,000	4.3	18.0	432,000	1,810,000	32,000	464,000
C	Indicated	Surface	0.30	688,000	0.8	2.0	18,000	44,000	1,000	19,000
C	Indicated	Underground	1.50	9,000	2.0	6.3	600	1,800	30	1,000
Total Indicated (Rounded)				**9,140,000**	**2.6**	**10.4**	**760,000**	**3,070,000**	**54,000**	**810,000**
Main	Inferred	Surface	0.30	1,476,000	0.8	3.8	38,000	180,300	3,200	41,000
Main	Inferred	Underground	1.50	2,011,000	3.1	3.9	200,500	252,200	4,400	205,000
H1	Inferred	Surface	0.30	624,000	0.7	2.0	14,000	40,100	700	15,000
H1	Inferred	Underground	1.50	13,000	2.6	2.0	1,100	800	-	1,000
H	Inferred	Surface	0.30	917,000	1.2	2.8	35,400	82,600	1,400	37,000
H	Inferred	Underground	1.50	117,000	2.1	4.5	7,900	16,900	300	8,000
B	Inferred	Surface	0.30	1,112,000	0.7	4.5	25,000	160,900	2,800	28,000
B	Inferred	Underground	1.50	483,000	3.9	11.3	60,600	175,500	3,100	64,000
C	Inferred	Surface	0.30	5,934,000	1.1	3.2	209,900	610,600	10,700	221,000
C	Inferred	Underground	1.50	2,165,000	3.5	5.2	243,600	362,000	6,400	250,000
D	Inferred	Surface	0.30	1,030,000	0.9	3.3	29,800	109,300	1,900	32,000
D	Inferred	Underground	1.50	-	-	-	-	-	-	-
Total Inferred (Rounded)				**15,900,000**	**1.7**	**3.9**	**870,000**	**1,990,000**	**35,000**	**900,000**

Notes for Resource Estimate:

1. Cut-off grade for mineralized zone interpretation was 0.5 g/tonne.
2. Block cut-off grade for surface resources (less than 150 metres deep) was 0.3 g/tonne.
3. Block cut-off grade for underground resources (more than 150 metres deep) was 1.5 g/tonne.
4. Gold price was US$1,500 per troy ounce.
5. Zones extended up to 150 metres down-dip from last intercept. Along strike, zones extended halfway to the next cross-section.
6. Minimum width was 2 metres.
7. Non-diluted.
8. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
9. Resource estimate prepared by Doug Roy, M.A.Sc, P.Eng.
10. A specific gravity (bulk density) value of 2.75 was applied to all blocks (based on 194 samples).
11. Non-cut. Top-cut analysis of sample data suggested no top cut was needed because of the absence of high-grade outliers.
12. 1 ounce gold = 57 ounces silver. Silver equivalency parameters: Metallurgical recovery: Gold 95%, Silver 72%; Price: Gold $1500 per ounce, Silver $35 per ounce.

14.18 CROSS-VALIDATION OF RESULTS

Nearest neighbour estimation provides a good estimate of the global declustered mean. For each zone, block grades were estimated using nearest neighbour estimation and the results were compared with the kriged results (refer to Table 14-11).

The global declustered mean was slightly higher than the kriged average block grade. The largest difference could be seen in the Main Zone. This result is not alarming because simple averages- such as nearest neighbour- commonly overestimates the mean grade.

The author was satisfied with the cross-validation results.

Table 14-11. Results of nearest neighbour cross-validation.

		Kriging			Nearest Neighbour		
Zone	Cut-Off Grade (g/tonne)	Tonnes	Grade (g/tonne)	Ounces	Tonnes	Grade (g/tonne)	Ounces
H1	0	2,460,000	0.3	27,000	2,460,000	0.4	30,000
H	0	4,460,000	0.6	80,000	4,460,000	0.5	71,000
M	0	36,260,000	1.3	1,523,000	37,830,000	1.7	2,074,000
B	0	5,510,000	0.8	134,000	5,740,000	0.7	135,000
C	0	51,140,000	0.6	1,053,000	53,200,000	0.7	1,234,000
D	0	2,750,000	0.8	68,000	2,750,000	0.9	81,000
Total	**0**	**103,000,000**	**0.9**	**2,890,000**	**106,000,000**	**1.1**	**3,630,000**

14.19 COMPARISON WITH PREVIOUS MINERAL RESOURCE ESTIMATE

A comparison was made with the previous mineral resource estimate that Howe carried out in 2010. Major differences between the estimation methodologies are highlighted in Table 14-12.

The additional drilling caused a shift of some mineral resources that were in the Inferred category into the Indicated category. The net result was an increase in grade and gold content (by 490,000 ounces) for the Indicated category and a decrease in grade and gold content (by 60,000 ounces) for the Inferred category.

The same pattern was seen with silver, with an increase in silver content (by 2.3 million ounces) for the Indicated category and a decrease in silver content (by 0.9 million ounces) for the Inferred category.

The major causes behind the overall net increase in tonnes and metal content are:

- The significant number of new holes; and,
- The drop in block cut-off grades.

Table 14-12. Major differences between the current mineral resource estimation method and 2010's method.

Parameter	Current	2010
Grade Estimation Method	**Block Kriging**	**Inverse Distance Weighting**
Block Size (East x North x RL)	**5 x 5 x 5**	**5 x 5 x 5**
Samples	**1.5 metre Regularised**	**1.5 metre Regularised**
Main Zone Domains	**Higher Grade & Lower Grade**	**One Domain**
Indicated Category Outlined In	**Main and C Zones**	**Main Zone Only**

Table 14-13. Comparison with 2010 estimate.

Current Estimate

Category	Surface or Underground	Cut-Off Grade (g/tonne)	Tonnes	Gold Grade (g/tonne)	Silver Grade (g/tonne)	Gold Ounces	Silver Ounces
Indicated	Surface	0.30	6,002,000	1.8	7.1	326,000	1,257,000
Indicated	Underground	1.50	3,136,000	4.3	18.0	433,000	1,812,000
Total Indicated (Rounded)			**9,140,000**	**2.6**	**10.4**	**760,000**	**3,070,000**
Inferred	Surface	0.30	11,093,000	1.0	3.3	352,000	1,184,000
Inferred	Underground	1.50	4,789,000	3.3	5.2	514,000	807,000
Total Inferred (Rounded)			**15,900,000**	**1.7**	**3.9**	**870,000**	**1,990,000**

2010

Category	Surface or Underground	Cut-Off Grade (g/tonne)	Tonnes	Gold Grade (g/tonne)	Silver Grade (g/tonne)	Gold Ounces	Silver Ounces
Indicated	Surface	0.50	2,900,000	1.9	5.4	180,000	500,000
Indicated	Underground	2.00	490,000	5.7	13.8	90,000	220,000
Total Indicated (Rounded)			**3,400,000**	**2.5**	**6.6**	**270,000**	**720,000**
Inferred	Surface	0.50	5,400,000	1.1	2.5	190,000	430,000
Inferred	Underground	2.00	5,200,000	4.4	14.7	740,000	2,460,000
Total Inferred (Rounded)			**10,600,000**	**2.7**	**8.5**	**930,000**	**2,890,000**

Change From 2010

Category	Surface or Underground	Cut-Off Grade (g/tonne)	Tonnes	Gold Grade (g/tonne)	Silver Grade (g/tonne)	Gold Ounces	Silver Ounces
Indicated	Surface	-0.2	+3,102,000	-0.1	+1.7	+146,000	+757,000
Indicated	Underground	-0.5	+2,646,000	-1.4	+4.2	+343,000	+1,592,000
Total Indicated (Rounded)			+5,740,000	+0.1	+3.8	+490,000	+2,350,000
Inferred	Surface	-0.2	+5,693,000	-0.1	+0.8	+162,000	+754,000
Inferred	Underground	-0.5	-411,000	-1.1	-9.5	-226,000	-1,653,000
Total Inferred (Rounded)			+5,300,000	-1.0	-4.6	-60,000	-900,000

15 MINERAL RESERVE ESTIMATES

Treasury has not carried out any pre-feasibility or feasibility studies of the Project designed to convert the Mineral Resources previously described in this report to Mineral Reserves.

16 MINING METHODS

16.1 CAUTION TO THE READER

The reader is cautioned that this PEA uses Inferred Mineral Resources. NI 43-101 Part 2, Section 2.3(1)(b) and Companion Policy 43-101CP, Part 2, Section 2.3(1) Restricted Disclosure, prohibits the disclosure of the results of an economic analysis that includes or is based on inferred mineral resources, an historical estimate, or an exploration target.

"Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies." (CIM Definition Standards - For Mineral Resources and Mineral Reserves; Adopted by CIM Council on November 27, 2010)

Inferred Mineral Resources are based upon widely spaced samples and are speculative in nature. They may never be part of a mineral reserve.

Companion Policy 43-101CP, Part 2, Section 2.3(1), Restricted Disclosure states that "CIM considers the confidence in inferred mineral resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. The Instrument extends this prohibition to exploration targets because such targets are conceptual and have even less confidence than inferred mineral resources. The Instrument also extends the prohibition to historical estimates because they have not been demonstrated or verified to the standards required for mineral resources or mineral reserves and, therefore, cannot be used in an economic analysis suitable for public disclosure."

The Companion Policy 43-101CP, Part 2, Section 2.3(1), on the Use of Term "Ore" states: – *We consider the use of the word "ore" in the context of mineral resource estimates to be potentially misleading because "ore" implies technical feasibility and economic viability that should only be attributed to mineral reserves.*

However, under NI 43-101, Part 2, Section 2.3(3) and Companion Policy 43-101CP, Part 2 section 2.3(3), a Preliminary Economic Assessment is allowed to use inferred mineral resources and to carry out an economic assessment in order to inform investors of the potential of the property. Investors must be informed that the preliminary economic assessment is preliminary in nature, that it 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, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The next logical step is to follow up the PEA with a pre-feasibility study which requires validation of resources through closer spaced sampling and cost confirmation by obtaining and using detailed quotes from suppliers. A detailed knowledge of the physical conditions at the site and extensive confirmation testing to determine the optimum processing method is also required.

16.2 Introduction

A combined surface and underground operation is envisioned. Initially, mining will be by open pit methods. Mining from the pit will supply feed to the mill for 4 to 4½ years, while lower grade mill feed is stockpiled. The overall pit, mined in three distinct phases, will have a generally oval shape with its long axis oriented along the east-west strike of the deposit. Early in Year 2, underground development would begin with underground production commencing in Year 3 supplemented by the low-grade stockpile from surface mining. Underground mining will last for eight years.

Pre-production stripping of overburden and waste rock will take place during the final year of plant construction. The processing plant will then be fed from open pit and underground mining for 10½ years. The combined open pit and underground mining schedule is shown in Table 16-1.

Table 16-1. Combined open pit and underground mining schedule.

		'000 tonnes												
Location		**Pre-Prod.**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Total**
Central Pit	Mill feed, t		875	64										939
Western Pit	Mill feed, t			567	512									1,079
Eastern Pit	Mill feed, t			244	144	292	49							729
Sub-Total, Open Pit	Mill feed, t		875	875	656	292	49							2,747
Underground	Mill feed, t				219	583	583	583	583	583	583	583	226	4,526
Stockpile to Mill	Mill feed, t						243	292	292	292	292	292	63	1,766
Total Feed to Mill	Mill feed, t		875	875	875	875	875	875	875	875	875	875	289	9,039
Waste Stripping	t	1,800	11,740	10,300	9,480	7,500	1,210							42,030
Pit to Stockpile	Mill feed, t		767	509	386	88	15							1,766
Total Surface Material Moved	Tonnes	1,800	13,382	11,684	10,523	7,880	1,517	292	292	292	292	292	63	47,954

16.3 TARGETS

Treasury's targets for the proposed mining operation were:

- Capital costs of less than $100 million;
- A mill feed grade of 2 g/tonne or greater; and,
- A production rate should be 90,000-100,000 ounces per year, at least for the first couple of years.

Preliminary mine planning and scheduling were carried out with the aim of achieving those targets, or at least coming as close to the targets as possible.

16.4 SURFACE MINING

A series of nested pits were optimised using the parameters (Table 16-2).

Table 16-2. Pit optimisation parameters.

Item	Value
Exchange Rate	US$ 1.00 = C$ 1.02
Gold Price	Base Case US$ 1375 per Ounce For Nested Pits, $875-1625 per Ounce in $50 Increments
Silver Price	US$ 26 per Ounce
Mill Throughput	2,500 tonnes per day
Unconsolidated Overburden Stripping	$4 per Cubic Metre
Mining	$3.15 per tonne (Mineralized Rock) $3.00 per tonne (Waste Rock)
SG	2.75 (Rock) 2.0 (Soil)
Processing (Gravity / Cyanide)	$15.65 per tonne Milled
G&A	$2 per tonne Milled (Added to the Processing Cost During Pit Optimisation)
Maximum Slope Angle	50° (Avg., Including Haul Roads)
Dilution	15% at 0.20 g/tonne Au, 4.3 g/tonne Ag *
Mining Recovery	90%
Milling Recovery	95% Gold 70% Silver
Smelter Return	99%
Smelter Treatment Charge / Selling Cost	1% of Base Case Price: Gold: $14 per ounce Silver: $0.26 per ounce
Tailings Disposal	(Included in Milling Cost)
Waste Rock Reclamation	$0.25 per tonne

* Average grade of blocks below resource cut-off grades.

The results of nested pit optimisation are shown in Table 16-3.

The "US$1,175 pit shell" was selected for more detailed analysis partly because the present value of the operation steadily increases down to that pit depth (refer to Figure 16-1). If the pit were deepened from the US$1,175 shell, it would not improve the NPV. After a certain depth, the NPV decreases. In other words, going deeper than the US$1,175 shell would not improve the project's value.

Table 16-3. Results of nested pit optimisation.

Gold Price	**$ 875**	**$ 925**	**$ 975**	**$ 1,025**	**$ 1,075**	**$ 1,125**	**$ 1,175**
Diluted Mill Feed	2,546,000	2,756,000	2,892,000	2,975,000	3,243,000	3,903,000	**4,161,000**
Waste Rock (Incl Subgrade Zone Mat)	28,303,000	29,597,000	30,510,000	30,750,000	32,406,000	40,270,000	**42,232,000**
Waste Rock (Not Incl Zone Material)	26,235,000	27,501,000	28,403,000	28,676,000	30,349,000	37,967,000	39,909,000
Gold Sold (Ounces)	245,000	255,000	261,000	263,000	273,000	307,000	**317,000**
Avg In Situ Grade (g/tonne)	3.63	3.48	3.39	3.33	3.17	2.96	**2.87**
Silver Sold (Ounces)	365,000	394,000	412,000	421,000	449,000	576,000	**609,000**
Avg In Situ Grade (g/tonne)	5.40	5.39	5.36	5.33	5.22	5.56	**5.51**
Gold+Equiv Ounces Sold	250,000	260,000	267,000	269,000	279,000	315,000	**325,000**
Years of Milling	2.7	2.9	3.1	3.1	3.4	4.1	**4.4**
Gold+Equiv Ounces Sold per Year	93,000	89,000	87,000	85,000	81,000	76,000	**74,000**
Stripping Ratio ($tonnes_{waste}/tonnes_{feed}$)	11.1:1	10.7:1	10.5:1	10.3:1	10:1	10.3:1	**10.1:1**
Marginal Stripping Ratio		6:1	6.6:1	3.3:1	6.2:1	11.5:1	7.5:1
Yearly Revenue (Avg)	$ 127,314,815	$ 357,500,000	$ 367,125,000	$ 369,875,000	$ 383,625,000	$ 433,125,000	$ 446,875,000
Yearly Operating Costs (Avg)	$ 54,262,522	$ 155,207,588	$ 161,090,573	$ 163,658,249	$ 174,769,468	$ 214,472,117	$ 226,378,517
Yearly Gross Margin (Avg)	$ 73,052,293	$ 202,292,412	$ 206,034,427	$ 206,216,751	$ 208,855,532	$ 218,652,883	$ 220,496,483
$NPV_{10\%}$	$212,000,000	$414,800,000	$662,800,000	$663,300,000	$631,800,000	$823,000,000	$791,900,000

Gold Price	**$ 1,225**	**$ 1,275**	**$ 1,325**	**$ 1,375**	**$ 1,425**	**$ 1,475**	**$ 1,525**	**$ 1,575**	**$ 1,625**
Diluted Mill Feed	5,230,000	5,668,000	6,022,000	6,312,000	7,602,000	9,150,000	10,721,000	11,846,000	13,028,000
Waste Rock (Incl Subgrade Zone Mat)	55,506,000	58,268,000	58,831,000	60,936,000	77,566,000	93,102,000	109,994,000	123,451,000	138,755,000
Waste Rock (Not Incl Zone Material)	52,922,000	55,657,000	56,448,000	58,586,000	74,828,000	90,212,000	106,921,000	120,473,000	135,899,000
Gold Sold (Ounces)	371,000	385,000	392,000	401,000	458,000	515,000	569,000	609,000	651,000
Avg In Situ Grade (g/tonne)	2.67	2.56	2.45	2.39	2.27	2.12	2.00	1.94	1.88
Silver Sold (Ounces)	746,000	795,000	826,000	858,000	987,000	1,083,000	1,383,000	1,625,000	1,902,000
Avg In Situ Grade (g/tonne)	5.37	5.28	5.16	5.12	4.89	4.46	4.86	5.16	5.50
Gold+Equiv Ounces Sold	381,000	396,000	403,000	413,000	472,000	530,000	588,000	632,000	677,000
Years of Milling	5.5	6.0	6.4	6.7	8.0	9.7	11.3	12.5	13.8
Gold+Equiv Ounces Sold per Year	69,000	66,000	63,000	62,000	59,000	55,000	52,000	50,000	49,000
Stripping Ratio ($tonnes_{waste}/tonnes_{feed}$)	10.6:1	10.3:1	9.8:1	9.7:1	10.2:1	10.2:1	10.3:1	10.4:1	10.7:1
Marginal Stripping Ratio	12.2:1	6.2:1	2.2:1	7.4:1	12.6:1	9.9:1	10.6:1	12:1	13.1:1
Yearly Revenue (Avg)	$ 523,875,000	$ 544,500,000	$ 554,125,000	$ 567,875,000	$ 649,000,000	$ 728,750,000	$ 808,500,000	$ 869,000,000	$ 930,875,000
Yearly Operating Costs (Avg)	$ 292,413,276	$ 310,756,714	$ 320,180,954	$ 333,226,709	$ 414,914,464	$ 498,563,617	$ 587,108,110	$ 654,922,074	$ 729,976,283
Yearly Gross Margin (Avg)	$ 231,461,724	$ 233,743,286	$ 233,944,046	$ 234,648,291	$ 234,085,536	$ 230,186,383	$ 221,391,890	$ 214,077,926	$ 200,898,717
$NPV_{10\%}$	$938,900,000	$887,300,000	$1,073,600,000	$1,043,700,000	$1,117,800,000	$1,311,200,000	$1,433,900,000	$1,436,100,000	$1,388,800,000

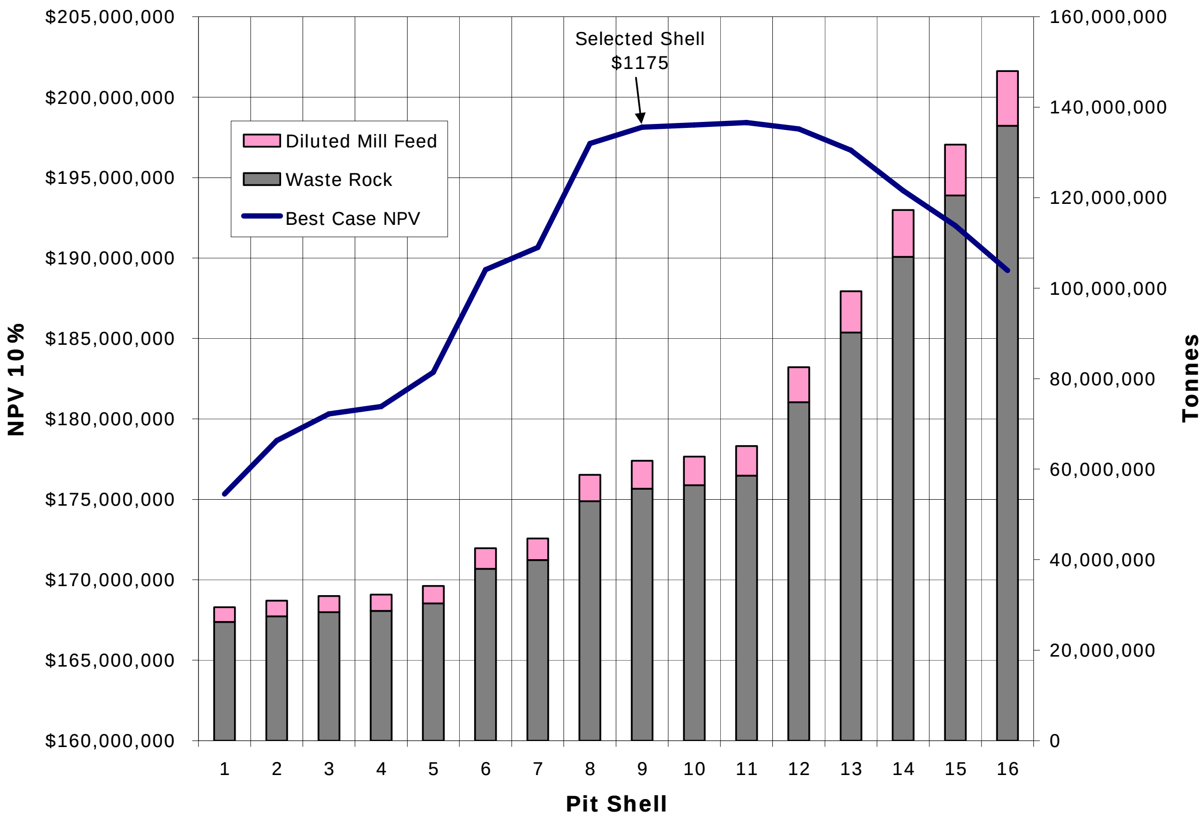


Figure 16-1. Net present value for nested pits.

16.5 Surface Mining and Scheduling

Various scheduling scenarios were attempted before deciding on the following schedule.

Milling would be carried out at the rate of 2,500 tonnes per day.

Pre-production would consist of stripping 1,800,000 tonnes of waste rock and mining 150,000 tonnes of mineralized rock to produce an initial 60 day mill stockpile.

Open pit mining will use conventional truck-and-shovel methods.

Mining would begin with the Central Pit (refer to Table 16-9). Around the end of Year 1, the Central Pit would be exhausted and would then be available for storing waste rock from open pit stripping at which time mining of the Western Pit would commence. Almost 90,000 ounces (gold + equivalent) would be produced in Year 1.

To meet Treasury's desired mill feed grade and yearly ounce production targets, lower grade material (between 0.5 g/tonne and 1.1 g/tonne) would be sent to a large low-grade stockpile. Rock with grades greater than 1.1 g/tonne would be sent directly to the mill stockpiles.

Because the Western Pit's average grade is slightly lower than the Central Pit's grade, the Eastern Pit (higher average grade) would be mined simultaneously with the Western Pit at a 30:70 ratio, respectively. The Western Pit would be exhausted in the Year 3 (and used for waste rock after mining is complete) with the Eastern Pit finishing at the start of Year 5.

After the end of active surface mining, rock from the low-grade stockpile would be fed into the mill at a rate of 830 tonnes per day to supplement underground production.

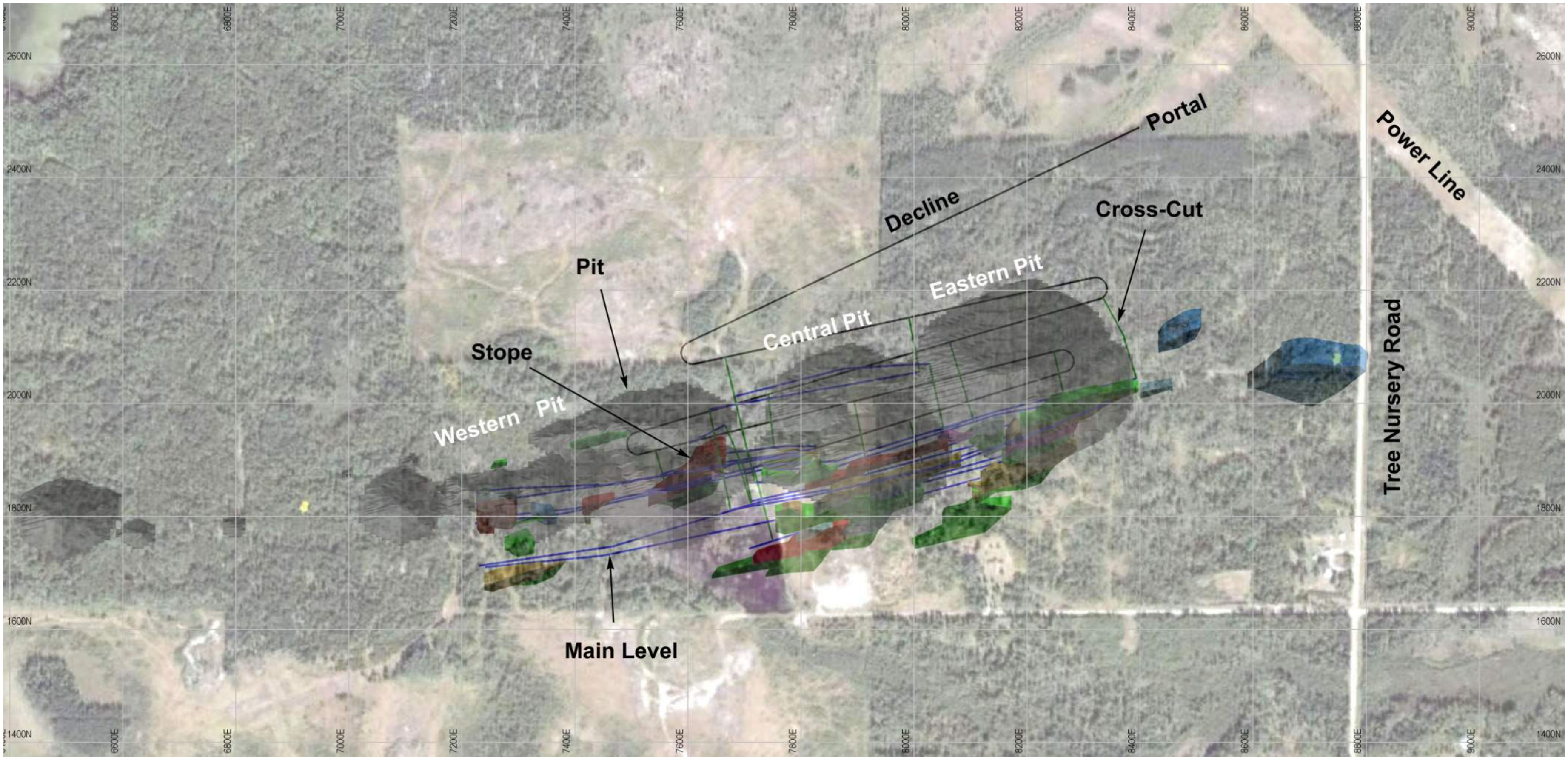


Figure 16-2. Plan view showing pits, underground development, and stopes.

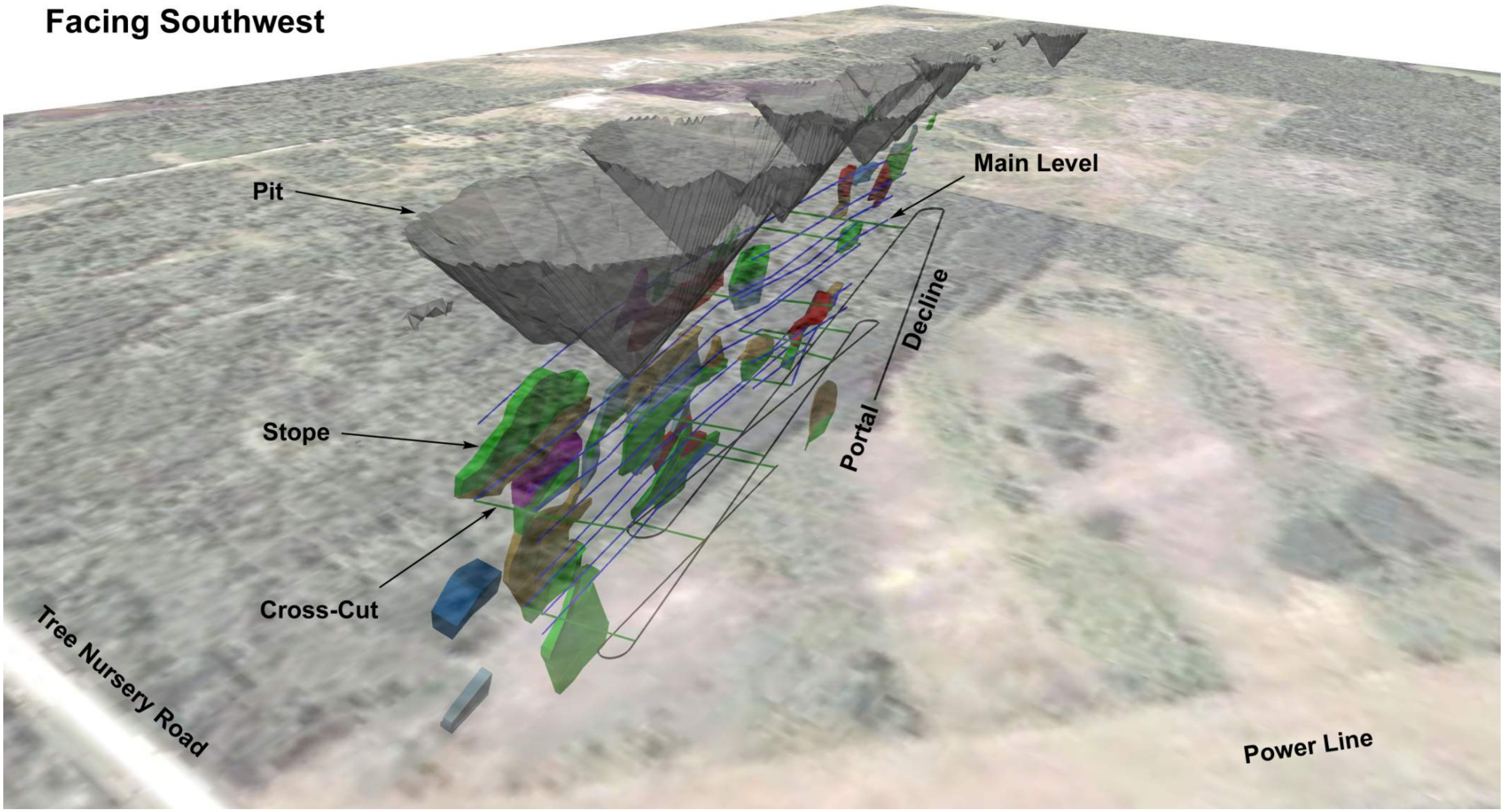


Figure 16-3. 3D view of the proposed pit and underground workings, facing southwest.

Table 16-4. Optimum pit resources.

							Non-Diluted Grade			Mill Feed Grade			In Situ Ounces		
Pit	Category	In Situ Grade	Non-Diluted Tonnes	Diluted, Recovered Tonnes	Years of Milling	NSR due to Others (Percent)	Gold	Silver	Gold + Eq Silver	Gold	Silver	Gold + Eq Silver	Gold	Silver	Gold + Eq Silver
Central	Low-Grade Stockpile	0.45-1.1	795,400	823,200	0.94	0.03	0.71	4.77	0.78	0.62	4.15	0.68	18,300	122,100	20,000
Central	Milled	1.1+	907,600	939,400	1.07	0.03	3.73	9.52	3.86	3.24	8.28	3.36	112,700	277,800	116,600
Central	Waste Rock		11,040,000												
Western	Low-Grade Stockpile	0.45-1.1	697,400	721,800	0.82	0.35	0.76	2.72	0.80	0.66	2.37	0.69	17,000	61,100	17,800
Western	Milled	1.1+	1,042,800	1,079,300	1.23	0.35	3.09	5.04	3.16	2.69	4.38	2.75	103,500	168,800	105,800
Western	Waste Rock		12,280,000												
Eastern	Low-Grade Stockpile	0.45-1.1	213,200	220,700	0.25	2.00	0.70	8.86	0.82	0.61	7.70	0.71	4,800	60,700	5,600
Eastern	Milled	1.1+	704,100	728,700	0.83	2.00	5.19	7.34	5.29	4.51	6.39	4.60	121,600	166,300	123,900
Eastern	Waste Rock		18,710,000												
Total	Low-Grade Stockpile	0.45-1.1	1,706,000	1,765,700	2.01		0.73	4.45	0.79	0.64	3.87	0.69	40,100	243,900	43,400
Total	Milled	1.1+	2,654,500	2,747,400	3.13		3.96	7.18	4.06	3.44	6.24	3.53	337,800	612,900	346,300
Total	Waste Rock		42,030,000												
Grand Total		0.45+	4,360,500	4,513,100	5.14		2.70	6.11	2.78	2.34	5.31	2.42	377,900	856,800	389,700

* Considers mining recovery & milling recovery.

16.6 SURFACE EQUIPMENT

Open pit mining will employ a hydraulic excavator loading dump trucks. A front-end-loader will provide back-up and be used for stockpile work, while a smaller front-end-loader will be used for general project duties and provide back-up to the larger loader. The main equipment fleet is shown in Table 16-5.

Table 16-5. Main open pit mining equipment.

Item	Description	Quantity
Production Drill	140 mm holes	1
Excavator	7 m^3	1
Wheel Loader	6 m^3	1
Haul Truck	55.3 t	4
Track Dozer	300 kW	1
Grader	14 foot	1
Water Truck	19,000 litres	1

16.7 UNDERGROUND MINING AND SCHEDULING

During the second year of open pit production, a decline ramp will be sunk to provide access for underground mining. Sufficient development, including main levels and a ventilation raise, will be completed in time for the underground mine to provide some of the mill feed during the third year. Underground production will be supplemented by recovery of material from the low-grade stockpile.

The underground mining method will be longhole stoping with delayed hydraulic backfill. The level interval is 45 metres vertically. The average stope width is 10.5 metres. Primary stopes will be 10 metres long and the backfill (classified mill tailings) will contain 5% Portland cement. Secondary stopes, 20 metres long, will be filled, but cement will not be required. This plan eliminates the need for rib pillars. The underground would be accessed via a portal and decline (refer to Figure 16-2 and Figure 16-3).

Stoping blocks were outlined at a cut-off grade of approximately 2.5 g/tonne (gold + equivalent). The majority of stopes were in the Main Zone, with other stopes in the B and C zones.

The underground level development was carried out assuming a level spacing of fifty vertical metres. Later, it was decided to change the spacing to 45 metres. Rather than re-design the preliminary development plan, the author (Mr. Roy) decided that using a "development factor" would be within the accuracy limits of this study. The factor was 50/45, or 1.11. In other words, the metres of development for a fifty metre spacing were multiplied by 1.11.

Refer to Table 16-9 for a summary of the underground production schedule and Table 16-7 for details.

Table 16-6. Underground stoping, by level.

Stopes					Non-Diluted, In-Situ Grade			Mill Feed Grade				In Situ Ounces	
Level	Zone	Non-Diluted Tonnes	Diluted, Recovered Tonnes	NSR Due to Others (%)	Gold	Silver	Gold + Equivalent	Gold	Silver	Gold + Equivalent	Gold Ounces	Silver Ounces	Gold Equivalent Ounces
200	B	35,900	37,200	0.56	8.69	3.52	8.74	7.56	3.06	7.60	10,000	4,100	10,100
200	M	108,100	111,900	-	6.93	4.96	7.00	6.02	4.31	6.08	24,100	17,200	24,300
200	M	22,400	23,200	1.59	4.50	40.98	5.07	3.91	35.63	4.41	3,200	29,500	3,700
200	M	57,300	59,300	-	3.00	6.24	3.09	2.61	5.43	2.68	5,500	11,500	5,700
200	M	117,500	121,600	1.84	2.61	24.77	2.95	2.27	21.54	2.57	9,800	93,600	11,100
Subtotal		341,200	353,200	0.80	4.79	14.21	5.00	4.17	12.36	4.34	52,600	155,900	54,900
100	B	34,300	35,500	0.51	11.95	6.47	12.04	10.39	5.63	10.47	13,200	7,100	13,300
100	B	6,100	6,300	0.80	5.09	3.16	5.13	4.42	2.74	4.46	1,000	600	1,000
100	M	249,300	258,000	0.21	5.30	12.51	5.48	4.61	10.88	4.76	42,500	100,300	43,900
100	M	42,300	43,800	-	3.93	3.24	3.98	3.42	2.82	3.46	5,400	4,400	5,400
100	M	353,500	365,900	1.80	3.24	13.15	3.42	2.82	11.43	2.97	36,800	149,400	38,900
Subtotal		685,500	709,500	1.04	4.49	11.88	4.65	3.90	10.33	4.05	98,900	261,800	102,500
0	B	34,300	35,500	1.50	2.79	50.16	3.49	2.42	43.61	3.03	3,100	55,300	3,800
0	C	182,300	188,700	-	7.33	10.82	7.48	6.37	9.41	6.50	42,900	63,400	43,800
0	C	171,100	177,100	-	5.81	11.68	5.97	5.05	10.16	5.19	31,900	64,300	32,800
0	C	28,500	29,500	-	3.59	13.24	3.77	3.12	11.51	3.28	3,300	12,100	3,500
0	C	74,000	76,600	-	2.93	8.46	3.04	2.54	7.36	2.65	7,000	20,100	7,200
0	M	35,600	36,800	-	3.53	235.04	6.80	3.07	204.38	5.91	4,000	268,800	7,800
0	M	236,000	244,300	1.83	4.74	18.90	5.01	4.13	16.44	4.35	36,000	143,400	38,000
0	M	268,100	277,500	1.95	2.55	12.04	2.71	2.21	10.47	2.36	21,900	103,800	23,400
0	M	374,200	387,300	0.69	3.81	5.28	3.88	3.31	4.59	3.37	45,800	63,600	46,700
Subtotal		1,404,100	1,453,300	0.90	4.34	17.60	4.58	3.77	15.31	3.99	195,900	794,800	207,000
-100	C	175,000	181,100	-	2.98	8.94	3.11	2.59	7.77	2.70	16,800	50,300	17,500
-100	M	152,300	157,600	1.82	3.58	4.93	3.64	3.11	4.28	3.17	17,500	24,100	17,800
-100	M	245,200	253,800	1.77	3.59	3.70	3.64	3.12	3.22	3.17	28,300	29,200	28,700
-100	M	457,800	473,800	0.07	2.11	48.81	2.79	1.84	42.44	2.43	31,100	718,500	41,100
-100	M	222,200	230,000	0.36	3.56	2.32	3.59	3.09	2.02	3.12	25,400	16,600	25,600
Subtotal		1,252,500	1,296,300	0.66	2.96	20.83	3.25	2.57	18.11	2.82	119,100	838,700	130,700
-200	B	32,200	33,300	-	2.96	3.18	3.00	2.57	2.77	2.61	3,100	3,300	3,100
-200	M	184,300	190,800	-	5.75	2.79	5.79	5.00	2.43	5.03	34,100	16,600	34,300
-200	M	123,100	127,400	-	5.01	2.49	5.04	4.36	2.16	4.39	19,800	9,800	20,000
-200	M	120,300	124,500	1.77	3.46	7.15	3.56	3.01	6.22	3.09	13,400	27,600	13,800
-200	B	57,700	59,700	-	2.72	3.08	2.77	2.37	2.68	2.40	5,100	5,700	5,100
-200	M	171,500	177,500	1.82	2.84	5.79	2.92	2.47	5.03	2.54	15,600	31,900	16,100
Subtotal		689,100	713,200	0.76	4.11	4.28	4.17	3.58	3.72	3.63	91,100	94,900	92,400
Subtotal to -200		**4,372,400**	**4,525,500**		**3.97**	**15.26**	**4.18**	**3.45**	**13.27**	**3.63**	**557,600**	**2,146,100**	**587,500**

* Considers mining recovery & milling recovery.

Table 16-7. Detailed underground production schedule.

Detailed Production Schedule - Underground

Level	Pre-Production	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Level 200														
Milled Tonnes				218,625	134,575									353,200
Ounces Recovered*				28,760	17,700									46,460
Feed Grade*				4.34	4.34									4.34
NSR due to Others (%)				0.80	0.80									0.80
Level 100														
Milled Tonnes					448,425	261,075								709,500
Ounces Recovered*					54,870	31,950								86,820
Feed Grade*					4.05	4.05								4.05
NSR due to Others (%)					1.04	1.04								1.04
Level 0														
Milled Tonnes						321,925	583,000	548,375						1,453,300
Ounces Recovered*						38,670	70,030	65,880						174,580
Feed Grade*						3.99	3.99	3.99						3.99
NSR due to Others (%)						0.90	0.90	0.90						0.90
Level -100														
Milled Tonnes								34,625	583,000	583,000	95,675			1,296,300
Ounces Recovered*								2,920	49,120	49,120	8,060			109,220
Feed Grade*								2.82	2.82	2.82	2.82			2.82
NSR due to Others (%)								0.66	0.66	0.66	0.66			0.66
Level -200														
Milled Tonnes											487,325	225,875		713,200
Ounces Recovered*											53,750	24,920		78,670
Feed Grade*											3.63	3.63		3.63
NSR due to Others (%)											0.76	0.76		0.76
Total Mill Feed														
Milled Tonnes				218,625	583,000	583,000	583,000	583,000	583,000	583,000	583,000	225,875		4,525,500
Ounces Recovered*				28,760	72,570	70,620	70,030	68,800	49,120	49,120	61,810	24,920		495,750
Feed Grade*				4.34	4.11	4.01	3.99	3.92	2.82	2.82	3.50	3.63		3.63
NSR due to Others (%)				0.80	0.98	0.96	0.90	0.89	0.66	0.66	0.75	0.76		0.82

* Considers mining recovery & milling recovery.

Table 16-8. Underground development schedule (factored to 45 m level spacing).

Level	Pre-Production	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Level 250														
Ramp Metres			1,000											
Level Metres			1,111											
Cross-Cut Metres			341											
Re-Mucks*			210											
Ventilation Raise			150											
Level 200														
Ramp Metres			333											
Level Metres			1,837											
Cross-Cut Metres			204											
Re-Mucks*			220											
Ventilation Raise			50											
Level 150														
Ramp Metres				333										
Level Metres				1,320										
Cross-Cut Metres				176										
Re-Mucks*				170										
Ventilation Raise				50										
Level 100														
Ramp Metres				333										
Level Metres				1,298										
Cross-Cut Metres				270										
Re-Mucks*				160										
Ventilation Raise				50										
Level 50														
Ramp Metres					333									
Level Metres					952									
Cross-Cut Metres					171									
Re-Mucks*					130									
Ventilation Raise					50									
Level 0														
Ramp Metres					333									
Level Metres					968									
Cross-Cut Metres					154									
Re-Mucks*					130									
Ventilation Raise					50									
Level -50														
Ramp Metres						333								
Level Metres						1,370								
Cross-Cut Metres						253								
Re-Mucks*						170								
Ventilation Raise						50								
Level -100														
Ramp Metres						333								
Level Metres						1,177								
Cross-Cut Metres						171								
Re-Mucks*						150								
Ventilation Raise						50								
Level -150														
Ramp Metres							333							
Level Metres							528							
Cross-Cut Metres							127							
Re-Mucks*							90							
Ventilation Raise							50							
Level -200														
Ramp Metres							333							
Level Metres							600							
Cross-Cut Metres							149							
Re-Mucks*							90							
Ventilation Raise							50							
Total														
Ramp Metres	-	-	1,333	667	667	667	667	-	-	-	-	-	-	4,000
Level Metres	-	-	2,948	2,618	1,920	2,547	1,128	-	-	-	-	-	-	11,160
Cross-Cut Metres	-	-	545	446	325	424	275	-	-	-	-	-	-	2,013
Re-Mucks*	-	-	430	330	260	320	180	-	-	-	-	-	-	1,520
Ventilation Raise	-	-	200	100	100	100	100	-	-	-	-	-	-	600

* Every 100 metres of ramp and level development, 10 metres long.

Table 16-9. Milling schedule.

Gold Price - US$1375/oz	Units	Total/ Average	Year -3	Year -2	Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
PRODUCTION																		
Central Pit																		
Waste Mined	kt	**11,040**	-	-	-	1,800	9,240	-	-	-	-	-	-	-	-	-	-	-
Ore Mined & Milled	kt	**939**	-	-	-	-	875	64	-	-	-	-	-	-	-	-	-	-
Gold Grade	g/t	**3.24**	-	-	-	-	3.24	3.24	-	-	-	-	-	-	-	-	-	-
Silver Grade	g/t	**8.28**	-	-	-	-	8.28	8.28	-	-	-	-	-	-	-	-	-	-
Ore Stockpiled	kt	**823**	-	-	-	-	767	56	-	-	-	-	-	-	-	-	-	-
Gold Grade	g/t	**0.62**	-	-	-	-	0.62	0.62	-	-	-	-	-	-	-	-	-	-
Silver Grade	g/t	**4.15**	-	-	-	-	4.15	4.15	-	-	-	-	-	-	-	-	-	-
Western Pit																		
Waste Mined	kt	**12,280**	-	-	-	-	1,500	5,000	5,780	-	-	-	-	-	-	-	-	-
Ore Mined & Milled	kt	**1,079**	-	-	-	-	-	567	512	-	-	-	-	-	-	-	-	-
Gold Grade	g/t	**2.69**	-	-	-	-	-	2.69	2.69	-	-	-	-	-	-	-	-	-
Silver Grade	g/t	**4.38**	-	-	-	-	-	4.38	4.38	-	-	-	-	-	-	-	-	-
Ore Stockpiled	kt	**722**	-	-	-	-	-	379	343	-	-	-	-	-	-	-	-	-
Gold Grade	g/t	**0.66**	-	-	-	-	-	0.66	0.66	-	-	-	-	-	-	-	-	-
Silver Grade	g/t	**2.37**	-	-	-	-	-	2.37	2.37	-	-	-	-	-	-	-	-	-
Eastern Pit																		
Waste Mined	kt	**18,710**	-	-	-	-	1,000	5,300	3,700	7,500	1,210	-	-	-	-	-	-	-
Ore Mined & Milled	kt	**729**	-	-	-	-	-	244	144	292	49	-	-	-	-	-	-	-
Gold Grade	g/t	**4.51**	-	-	-	-	-	4.51	4.51	4.51	4.51	-	-	-	-	-	-	-
Silver Grade	g/t	**6.39**	-	-	-	-	-	6.39	6.39	6.39	6.39	-	-	-	-	-	-	-
Ore Stockpiled	kt	**221**	-	-	-	-	-	74	44	88	15	-	-	-	-	-	-	-
Gold Grade	g/t	**0.61**	-	-	-	-	-	0.61	0.61	0.61	0.61	-	-	-	-	-	-	-
Silver Grade	g/t	**7.70**	-	-	-	-	-	7.70	7.70	7.70	7.70	-	-	-	-	-	-	-
Underground																		
Ore Mined & Milled	kt	**4,526**	-	-	-	-	-	-	219	583	583	583	583	583	583	583	226	-
Gold Grade	g/t	**3.45**	-	-	-	-	-	-	4.17	3.96	3.83	3.77	3.70	2.57	2.57	3.41	3.58	-
Silver Grade	g/t	**13.27**	-	-	-	-	-	-	12.36	10.80	13.08	15.31	15.47	18.11	18.11	6.08	3.72	-
Stockpile Feed			-	-	-													
Ore Reclaimed & Milled	kt	**1,766**	-	-	-	-	-	-	-	-	243	292	292	292	292	292	63	-
Gold Grade	g/t	**0.63**	-	-	-	-	-	-	-	-	0.63	0.63	0.63	0.63	0.63	0.63	0.63	-
Silver Grade	g/t	**3.87**	-	-	-	-	-	-	-	-	3.87	3.87	3.87	3.87	3.87	3.87	3.87	-
Totals																		
Waste Mined	kt	**42,030**	-	-	-	1,800	11,740	10,300	9,480	7,500	1,210	-	-	-	-	-	-	-
OP Ore Mined & Milled	kt	**2,747**	-	-	-	-	875	875	656	292	49	-	-	-	-	-	-	-
UG Ore Mined & Milled	kt	**4,526**	-	-	-	-	-	-	219	583	583	583	583	583	583	583	226	-
Ore Stockpiled	kt	**1,766**	-	-	-	-	767	509	386	88	15	-	-	-	-	-	-	-
Ore Milled	kt	**9,039**	-	-	-	-	875	875	875	875	875	875	875	875	875	875	289	-
Gold Grade	g/t	**2.87**	-	-	-	-	3.24	3.23	3.36	4.15	2.98	2.73	2.68	1.93	1.93	2.48	2.94	-
Silver Grade	g/t	**9.30**	-	-	-	-	8.28	5.22	6.70	9.32	10.15	11.49	11.60	13.36	13.36	5.34	3.76	-

16.8 Underground Equipment

Underground development will use 2-boom drill jumbos, load-haul-dump machines ("LHDs", "scooptrams"), and articulated dump trucks. Stoping will employ the same size LHDs and trucks, along with a mobile longhole drill rig. The main underground fleet is shown in Table 16-10.

Table 16-10. Main underground mining equipment.

Item	Description	Quantity
LHD	5 m^3	3
LHD	1.9 m^3	1
Articulated Dump Truck	36 ton	5
Longhole Drill		1
Jumbo	2-boom	2
Bolter jumbo		1
Grader	Low profile	1
Explosives truck		1
Scissor lift		1
Fuel truck		1
Lube truck		1

16.9 Mill Feed by Resource Category

The majority of proposed mill feed tonnes and metal are in the Indicated resource category (refer to Table 16-11).

Of the Main Zone blocks that lie within the mine plan, the vast majority is in the Indicated category (refer to Figure 16-4). Of the other zones, only Zone C has any resources in the Indicated category, though most of its Indicated resources are not within the mine plan. The other zones contain only Inferred mineral resources.

Table 16-11. Mill feed by resource category.

Area	Proportion of Tonnes	Proportion of Gold
Surface Pits*:		
Indicated	55%	58%
Inferred	45%	42%
Underground Stopes**:		
Indicated	62%	58%
Inferred	38%	42%
Overall*,**:		
Indicated	58%	58%
Inferred	42%	42%

* Above 0.5 g/tonne cut-off (only this would be sent to mill).
** "No cut-off." IE, all blocks within underground mine plan
(all would be sent to mill).

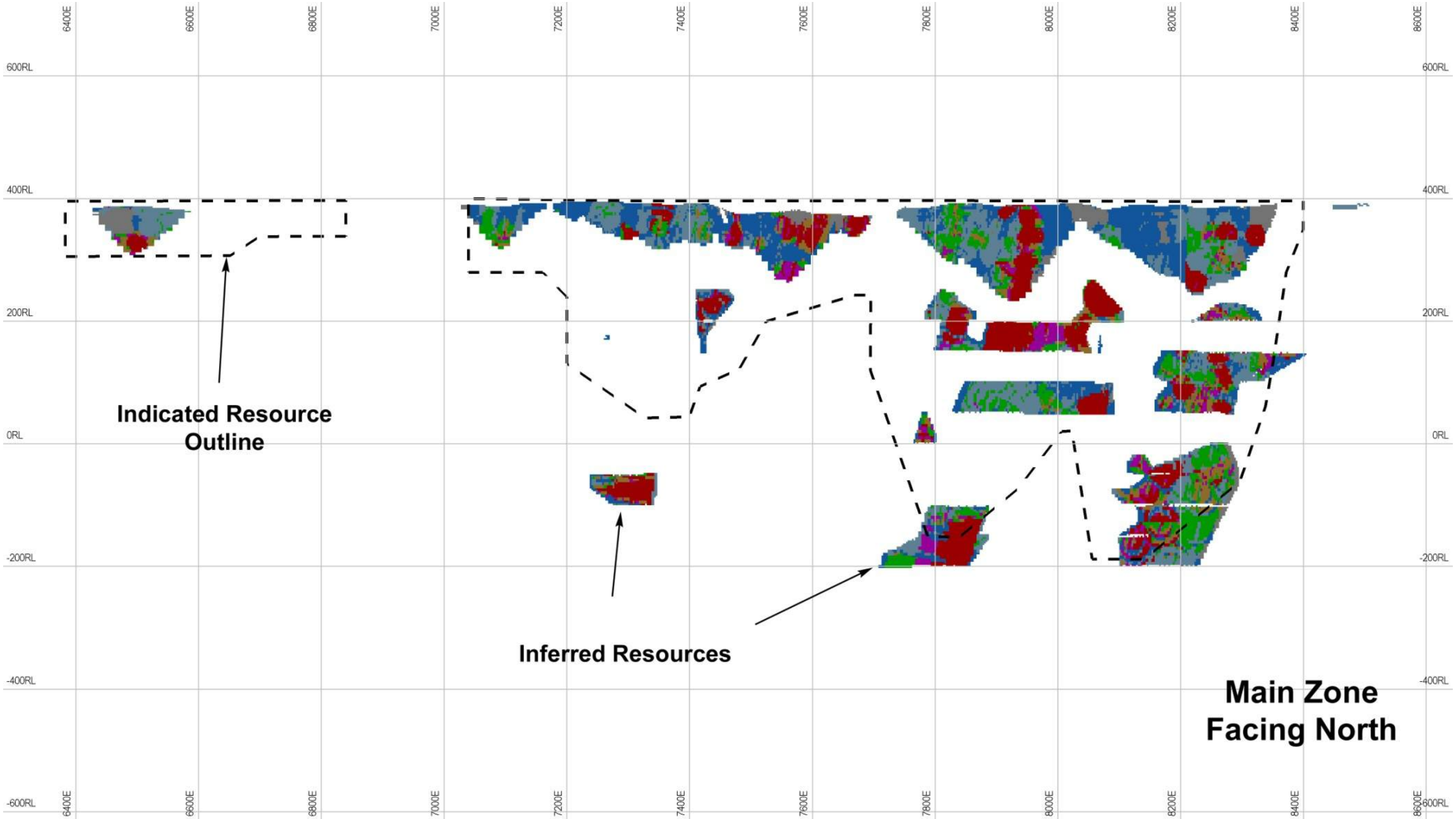


Figure 16-4. Main zone blocks in the mine plan showing resource category outlines.

16.10 NET SMELTER RETURN AGREEMENTS

The central part of the deposit is not subject to any net smelter return ("NSR") agreements with previous claim holders. However, various NSRs apply to surrounding claim blocks (Table 16-12).

Table 16-12. NSR agreements.

Area	Mass-Weighted Average NSR
Central Pit	0.03%
Western Pit	0.35%
Eastern Pit	2.00%
Average for Pits	0.57%
Underground	0.82%

17 RECOVERY METHODS

17.1 PROCESS SELECTION AND DESIGN PARAMETERS

The available metallurgical testwork indicates that the Goliath deposit is readily amenable to conventional processing and that gravity concentration followed by cyanidation can be used to obtain relatively high gold recovery.

For purposes of this PEA a flowsheet consisting of gravity concentration followed by cyanidation of the gravity tails via carbon-in-leach circuit (CIL) is selected. Selected design parameters for the study are shown in Table 17-1.

Table 17-1. Selected design parameters.

Area	Parameter	Value	Units
Grinding	Bond ball mill index	11.1	kWh/t
	Grind (K_{80})	105.0	microns
Gravity	Concentrate	0.1	wt %
Cyanidation	Gold recovery (overall)	95.0	%
	Silver recovery (overall)	70.0	%
	Total cyanidation time	32.0	h

17.2 PROCESS DESCRIPTION

Crushed mill feed is ground to a K_{80} of 105 microns in a two stage grinding circuit at a rate of 2,500 tonnes per day or 912,500 tonnes per annum (2,747 tonnes per day at 91% availability). A gravity recovery circuit is incorporated within the grinding circuit for recovery of free gold. The gravity concentrate is leached separately and the product directed to the main gold recovery circuit.

Ground product from the grinding circuit is fed to a CIL circuit for gold extraction. A conventional carbon elution circuit recovers gold that is smelted to yield a doré[3] product.

[3] A doré product is a semi-pure alloy of gold and silver created at the mine site and then transported to a refinery for further purification.

18 PROJECT INFRASTRUCTURE

The Goliath Project is located exceptionally close to current infrastructure. The project site is located approximately 2.5 kilometres from the Trans-Canada Highway (designated Highway 17) and is accessed by the municipally controlled Tree Nursery Road and Norman's Road. For this reason, minimal road upgrades will need to be considered for the project. Additional roads to be built will be limited and include "ring roads" to surround the open pit, waste storage and tailings storage facility. In general these roads could be considered for light duty use, such as by pickup trucks, for monitoring purposes. All processed gold (doré[4] bars) will be transported from site by truck using existing roads.

TransCanada Corporation operates a natural gas pipeline running parallel to the main highway. A small pipeline could be easily and cost effectively built for transport of natural gas to the project. This natural gas can be used for several purposes including heating of mine air in the wintertime. HydroOne operates both a 115 kV and a 230 kV electrical power supply that crosses the project site approximately 500 m from the proposed eastern pit limit. Initial contact with Hydro One has been made by Treasury, at which time HydroOne confirmed possible line capacity for Treasury to draw up to 10 MW of electrical power. Additional studies will be necessary to confirm this value. It is anticipated that an electrical substation will be built in close proximity to the transmission lines and subsequently to the processing plant infrastructure, which will lesson both associated costs and permitting requirements.

Treasury is the owner of a former Ministry of Natural Resources tree nursery and all associated office and warehouse infrastructure. The location of this area is approximately 2 kilometres to the north of the project site and will be sufficient for administration duties during both mine construction and operation. Some limited office space will be included in the plant and mobile equipment shop to house specific offices, such as a shop foreman or plant control room.

Treasury is investigating several options in regards to tailings storage. In general the terrain at the project site has little relief. Due to this, the project may require a "paddock" style tailings dam to contain tailings on all sides. However, several options are available that require little to no destruction of fish habitat, which may expedite the permitting process. An approximate volume of 10 million cubic metres will be required for the storage of tailings over the mine life. Treasury will be required to undertake an alternatives assessment of the tailings storage facilities during the Environmental Assessment process.

A waste rock storage area ("WRSA") will be built directly to the north of the proposed open pit. It will have a capacity of approximately 12.8 million m^3 or 26 million tonnes and have a footprint of 675,000 m^2. This will account for approximately 60% of the waste rock produced during mining. The additional 40% of waste rock removed will be backfilled to the open pit area. It should be noted that a portion of the footprint of the WRSA will also lie above the completed and filled open pit.

[4] A doré product is a semi-pure alloy of gold and silver created at the mine site and then transported to a refinery for further purification.

Geochemical testing is ongoing at this time to determine the Acid Rock Drainage (ARD) and neutralizing potential of the waste rock. Results of this study will help Treasury in the future design of waste dumps and any applicable surface water run-off control.

Design slopes have been set at 3:1 ratios to create a more natural appearance and total heights should be kept as low as possible to reduce the potential visual impact for neighbouring residents, in particular, the residents on the west side of Thunder Lake. Progressive reclamation should also commence as early as possible in the development of the waste rock storage area. This will not only be an advantage from an environmental perspective, but will also help to create a natural looking landscape.

In order to maintain a consistent Run-of-Mine (ROM) feed to the process plant, a low-grade stockpile will be required. The low-grade stockpile should have an approximate total volume of 900,000 m^3 or 1.8 million tonnes and an approximate footprint of 62,500 m^2 and should be located adjacent to the processing plant site for easy transport to the mill.

Up to three separate temporary stockpiles of varying grade will also be used to feed the process plant. These should be located in a radial form surrounding the primary crusher and will be used primarily to create a consistent plant feed but also act as temporary storage in the case of an unexpected mine stoppage. As noted, these stockpiles will be temporary in nature with constant replacement/turnover. They will collectively have a total capacity of approximately 15-30 days or 37,500 tonnes

19 MARKET STUDIES AND CONTRACTS

As of the date of this report, Treasury has not requisitioned any market studies or entered into any marketing contracts pertaining to the Goliath Project. Gold and silver are the primary expected products of the operation and are readily marketable.

20 ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL STUDIES AND PERMITTING

Treasury commissioned Environmental Base Line Studies using the services of Klohn Crippen Berger's ("KCB") Sudbury office. Fieldwork was completed in 2010 and 2011, and reports are currently being finalized. KCB's studies examined the health of the ecosystem by studying ground and surface water quality, sediment quality, fisheries, terrestrial resources and soil quality and include the following baseline components:

- Acid Rock Drainage Potential
- Aquatic Surveys, including Benthos and Fish
- Terrestrial and Vegetation study
- Soil and Sediment Quality study
- Hydrology and Surface Water study
- Hydrogeology Summary (desktop)
- Climate Summary

Additionally, RWDI Consultants completed the following in 2011:

- A Limited Scope Air Quality and Noise Baseline Study

Treasury has continued with additional environmental work for 2012, commissioning Thunderbay, Ontario, based DST Consulting to carry out studies on:

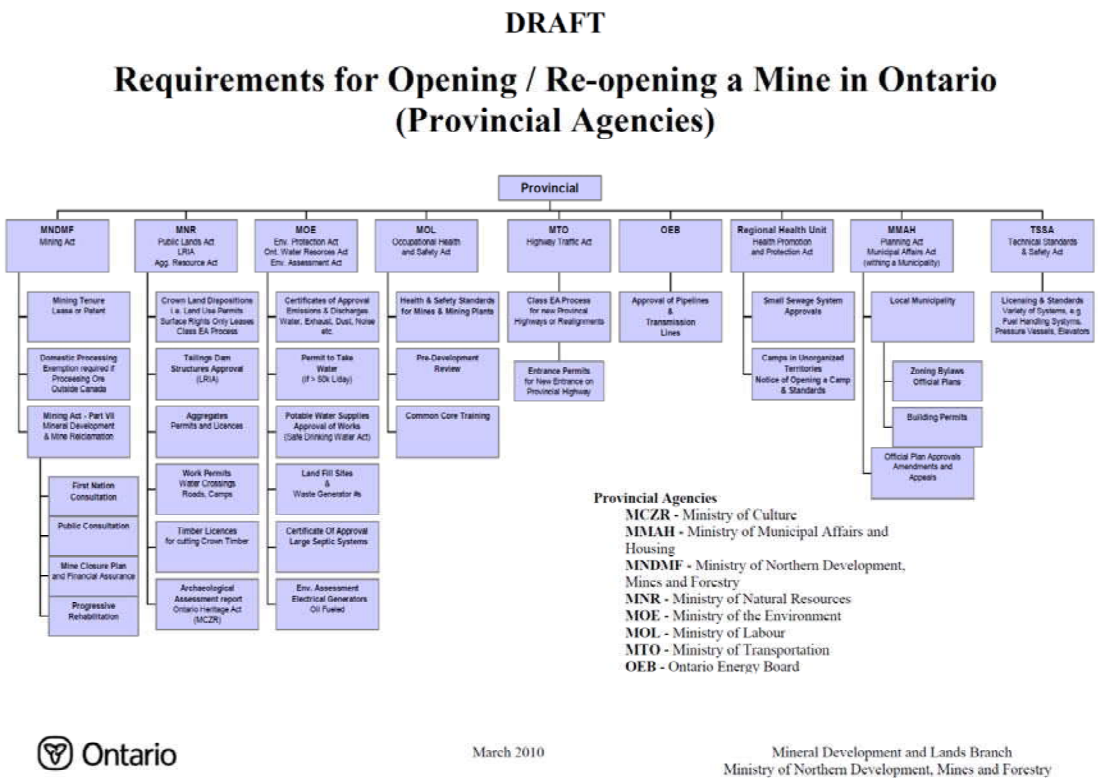
- Groundwater quantity and quality testing
- Monthly Surface Water Sampling
- A Stage One Archaeological Study (by Boreal Heritage) of the Goliath Gold Project Area is in the final stage (i.e. reporting)
- Bird, Bat and Fish (and habitat) Surveys
- Hydrologic monitoring
- Meteorological monitoring
- Large and small mammal surveys
- Sediment and Benthos monitoring

Completion of these studies and the development of the environmental baseline, along with ongoing community consultation and socio-economic studies, are key requirements for future government permitting of the Property leading to advanced exploration status with the Ontario Ministry of Northern Development and Mines.

Treasury warrants that it possesses all permits required to execute exploration activities it has undertaken to date on the property. Treasury is conducting ongoing community consultations including discussions with the local First Nation communities.

Howe includes as Figure 20-1, organizational charts that highlight various permitting and approvals requirements with respective provincial and federal agencies, which <u>may</u> need to be addressed in the permitting process. Specific permits and approvals to which a mineral project may be subject will depend on the specific features of the proposed project and its location in the social and environmental landscape. A more detailed summary of approvals processes for mining activities including provincial and federal laws, as well as local municipal bylaws that govern, or may govern, mineral development in Ontario is available in MNDF's *Practitioner's Guide to Planning for and Permitting a Mineral Development Project in Ontario*.

How the mine, or mines are developed and their tonnage throughputs will ultimately dictate their specific permitting requirements (e.g. Federal Environmental Assessment and/or Provincial), but generically any mine in production in Ontario would require a Permit to Take Water (PTTW) and a Section 53 Approval for an Industrial Sewage Works with supporting Receiving Water Assessment under the requirements of the *Ontario Water Resources Act*, and a Closure Plan under the provisions of the *Mining Act*.



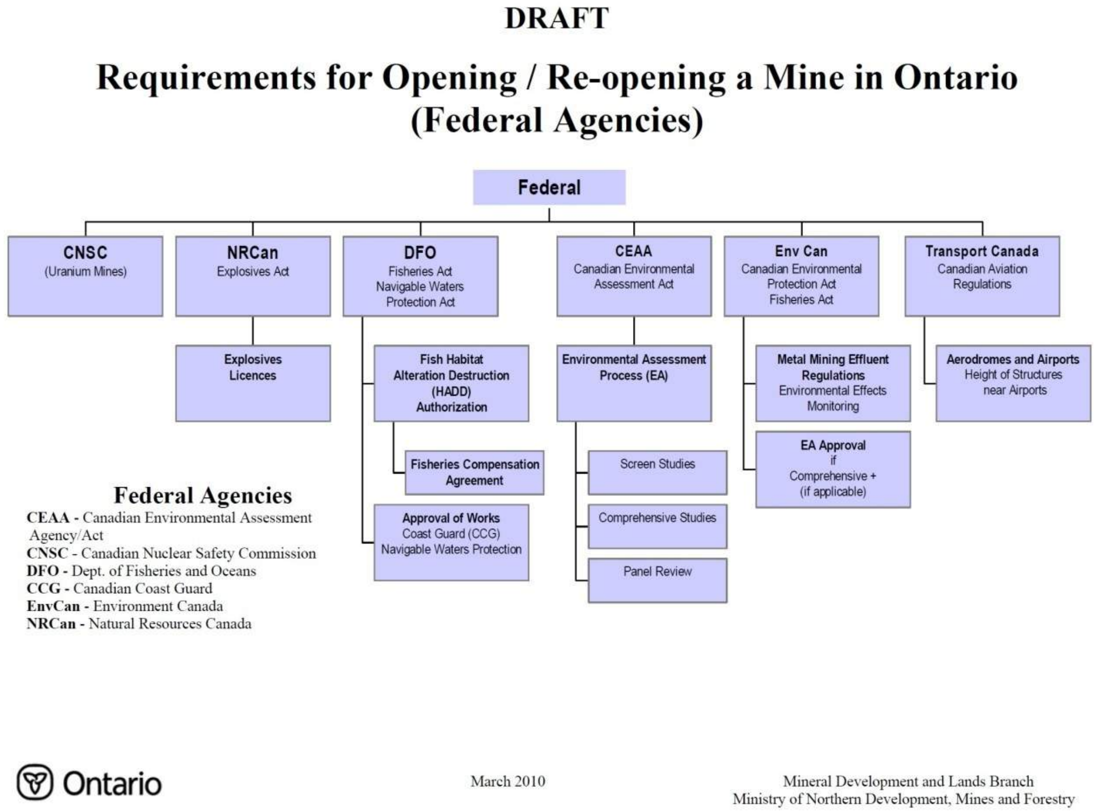


Figure 20-1. Overview Permitting and Approval Requirements that may Potentially Affect a Mine Development in Ontario – (http://www.mndm.gov.on.ca/mines/mg/mindev/permits_e.asp)

20.2 ANTICIPATED ENVIRONMENTAL COSTS DURING OPERATIONAL PHASE

Environmental costs during the operations phase of typical mining projects consist of compliance monitoring (as outlined in the EA and Operating Permits), effects monitoring (including Metal Mining Effluent Regulations requirements) and other monitoring and stewardship commitments made by the proponent or stipulated by approvals. For this scale of proposed operation a budget of $250,000 per annum may be appropriate. Approximately 50% of this relates to analytical costs and equipment requirements such as groundwater monitoring wells and surface water monitoring data loggers and 50% relates to staff and outside consulting for specialized services (benthic invertebrate, breeding bird surveys, etc.). A more detailed budget can be prepared after the EBS and permitting phases are complete because the costs related directly to the stipulations from the Province.

A reclamation bond will be required during mine operation. -Note that the bond value is typically returned in full after the reclamation program is completed and the company is “released” by the Province, so the net costs are only the interest costs on the bond.

20.3 ANTICIPATED ENVIRONMENTAL COSTS DURING CLOSURE PHASE

Costs relative to environmental considerations during the closure phase consist of the environmental monitoring costs that may post-date the completion of the physical reclamation (removal of buildings, revegetation of stockpiles, etc.). It is typical for these programs to run for

3-5 years after physical reclamation is completed and may be on the order of $50,000 to $100,000. Howe is not able to cost the physical reclamation program as the final mine design and permitted layout and reclamation approach will be developed later.

20.4 SOCIAL OR COMMUNITY IMPACT

20.4.1 Company Engagement/Consultation with Aboriginal Groups

Treasury understands that Aboriginal consultation represents not only a requirement in the permitting and EA process but also a great opportunity to engage and prepare the local Aboriginal communities as potential future employees and business associates. It is also important to understand any potential impacts the project may have on traditional uses of the land and areas of cultural significance. In this regard Treasury began the discussion and consultation process as early as 2008 and is presently continuing to reach out to the various local Aboriginal groups, which include, but are not limited to the Wabigoon Lake Ojibway Nation (WLON) and the Eagle Lake First Nation (ELFN).

Consultation and discussions have been ongoing with the WLON since an initial contact on June 2nd, 2008. WLON is the closest First Nation to the Goliath Project. Work is ongoing to initiate contracts for services to be provided using WLON where possible. For example, Treasury has requested a proposal for exploration drilling to be provided by WLON personnel. Treasury is very interested in continuing discussion with WLON and has requested the help of the Ministry of Northern Development and Mines to facilitate this process. WLON has expressed concern that the Project will take place on traditional lands and has requested access fees for drilling on these lands.

Consultation began with ELFN in August of 2011 and has been positive to date. ELFN has been very receptive to the development of training programs as well as various other initiatives to boost community involvement in industry. Recent consultation has been slow due to the election and commencement of the new Chief; however Treasury is very interested in continuing talks once the ELFN is ready and has communicated this desire.

In terms of continued consultation, Treasury would like to initiate a regular meeting schedule to keep the local First Nations informed and up to date with the most current information. It will also be of great benefit to hold general meetings for the public to attend and express and concerns they may have. Treasury is interested in starting these activities as soon as possible.

20.4.2 Company Engagement/Consultation with Stakeholders

In regards to consultation with stakeholders and groups other than First Nations, there are several key groups in which consultation must occur. For the purposes of the Project, three principle groups have been identified. These include government (federal, provincial, municipal), local communities (local community groups, land owners, natural resource users) and the public in general (interested individuals and non-governmental organizations).

To date, the general community has been positive about the project. Questions have been asked regarding several topics varying from expressing environmental concerns to inquiring as to how the municipality can assist Treasury in strengthening the local economy. Treasury has maintained a very open policy towards its relationship with the general community and will continue to do so. Answers to any questions will be fielded in a timely manner once sufficient information is available to properly address any concerns or comments.

In addition, it is of great benefit to engage the local municipal councils on a periodic basis to share information and updates on the project and answer questions. Treasury has found that presentations at local town council meetings have proven to be an effective method of distributing information to the community. Not only does the information get passed directly to town councillors, but the proceedings are also broadcast over Television to the surrounding area. It is imperative that information is made available to the general public in a clear and easy-to-understand fashion. Treasury will convey information through such means as newspaper articles, ads or pamphlets to community members at large.

Treasury recognizes that the Project will be of great benefit to the company and the surrounding communities and stakeholders. It is essential that the general public is aware of the Project scope, the benefits to the community, any risks that may be present and how the Project will be designed in order to mitigate these risks. As the project progresses, Treasury intends to sustain a good flow of communication to the general public by means of information sessions held at regular intervals or when additional new information warrants a public update.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

For the purposes of this PEA, expenses incurred before July 1, 2012 are considered to be sunk costs and are not included in this report's cash flow analysis. Estimated costs for ongoing environmental studies, ongoing permitting activity, feasibility study (including ongoing metallurgical testwork), and initial capital are C$93.8 million. Total life-of-mine capital costs are estimated to be C$200.5 million, as shown in Table 21-1 below.

Table 21-1. Capital cost estimate.

Item	C$ millions
Studies and permitting	2.4
Initial capital	91.4
Underground mine & backfill plant	99.5
Sustaining	8.2
Salvage	(3.0)
Closure	2.0
Total	**200.5**

In the present financial model, open pit mining equipment is purchased new and sold after being used for a little more than four years. Treasury will study the alternative of using a mining contractor, leasing, or purchasing used equipment. Considering the fact that used mining equipment can be purchased for reasonable prices, one or more of the alternatives (to purchasing new equipment) is likely to improve project economics.

21.2 OPERATING COSTS

Open pit mining costs are estimated to be C$3.15 per tonne for mill feed and C$3.00 per tonne for waste rock. The stripping ratio is 9.3, yielding an open pit mining cost of C$31.05 for each tonne of mill feed. Mill stockpile rehandling is estimated to cost C$0.80 per tonne.

Underground mining costs are estimated to average C$60 per tonne plus an additional $1.96 per overall tonne for the cement used in the primary stope backfill.

The processing cost estimate is C$15.81 per tonne of feed. A breakdown is shown in Table 21-2.

Table 21-2. Processing operating cost estimate.

Item	C$/t
Operating Labour	3.69
Power	2.06
Reagents	5.93
Operating Supplies	0.99
Maintenance Labour	1.35
Maintenance Supplies	1.03
Total	**15.06**
Contingency, at 5%	0.75
Total cost	**15.81**

General and administration costs are estimated to be C$1.8 million per year at full production.

22 ECONOMIC ANALYSIS

22.1 MODEL ASSUMPTIONS

An Excel spreadsheet was used to model and analyse the Net Cash Flow (NCF) of the Goliath Project. The model calculates the pre-tax and post-tax NCF as well as the Internal Rate of Return (IRR) and the Net Present Value (NPV) at various discount rates. The repayment period, the minimum gold price required to breakeven, and the IRRs at higher and lower metal prices and operating and capital costs are also calculated. Underlying assumptions and parameters used in Howe's model include:

- All units of measurement are metric unless otherwise stated.
- All dollars are Canadian Dollars unless otherwise stated.
- The gold (US$ 1,375 per troy oz) and silver (US$ 26.00 per troy oz) prices are based on the average London 2nd Fixing for the last three years as of June 30, 2012 (refer to Appendix B).
- The United States: Canadian exchange rate (C$1.02: US$1.00) is based on the three year trailing average as of June 30, 2012 (refer to Appendix B).
- The model has assumed a four year pre-production period. This allows for two years to complete environmental studies, permitting, a final feasibility study and the time to put financing in place. In the second two years, the model assumes that the company will build the processing plant, supporting infrastructure and strip 1.8 million tonnes of waste.
- The production rate is designed to supply 2,500 tonnes per day (tpd) or 875,000 tonnes per annum of feed to the mill. This generates an open pit life of 2 full years of production plus 3 partial years. In addition, the mine stockpiles 1,766,000 tonnes of lower grade material that is used to supplement the underground operation to satisfy mill feed requirements. The underground mine operates from year 3 to year 11 and produces a total of 4,526,000 tonnes of mineralized material. Thus the total mine life is 10.3 years
- 42,030,000 tonnes of waste are removed during the life of the open pit operation (including 1.8 million tonnes during development) for a waste: "ore" ratio of 9.3 (including stockpiled mill feed)
- The Production schedule includes waste and mineralized material tonnages and gold and silver grades for each production year as well by pit and underground.
- Mill recoveries are based on gravity concentration followed by cyanidation of the gravity tails via carbon-in-leach circuit (CIL) and are 95% and 70% for gold and silver respectively.
- Howe has estimated costs for gold and silver smelting and refining (including transportation and insurance) at $14.00 and $0.26 per ounce of gold and silver respectively produced by the proposed Goliath mill.
- There are a number of different royalties that apply to various areas of the Goliath property. These royalties are applied to the gold and silver revenues after deducting smelting and refining costs. The average royalty is 0.65% of Net Smelter Revenue (NSR) and at US$1,375 per oz for gold and $26.00 per oz for silver incurs a cost of $7.5 million over the life of the project.
- Capital costs have been developed by Howe and are shown in Section 21.
- Operating costs have been calculated by Howe and are shown in Section 21.
- The model calculates depreciation using the Units of Production (UOP) method. In this method the model calculated depreciation based on the amount of mineralized material milled each year.
- Working Capital is based on
    - Two weeks of precious metal inventory (at the NSR value).
    - Accounts Receivable as four weeks of metal production (at the NSR value).
    - Spare Parts and Supplies as $1.0 million.
    - Less: Accounts Payable as one half of four weeks of operating costs.

- The model calculates Federal and Ontario Corporate taxes and Ontario Mining Taxes. Basically, the Federal and Ontario Corporate taxes are based on net income as calculated for taxes.
- The Federal Income Tax base has been calculated as:
  - Earnings before Depreciation, Amortization and Taxes (EBITDA)
  - Less: Ontario Mining Taxes
  - Less: Capital Cost Allowance (CCA), i.e. depreciation where the two main forms are:
    - Class 41a, 100% Declining Balance (DB); applies to new mines.
    - Class 41b, 30% DB, most ongoing capital costs.
  - Less: Canadian Exploration Expenses (CEE), 100% DB; includes most pre-production exploration expenses plus waste stripping and mine excavations.
  - Less: Canadian Development Expense (CDE), 30% DB; resource acquisition costs as well as sinking mine shafts and major underground haulageways after coming into production.
  - Less: Interest Expense.
  - Equals Net Taxable Income.
  - Federal Corporate Tax is charged at 18% of Net Taxable Income.
  - Note that losses can currently be carried back three years and forward 20 years.
- Ontario Corporate Taxes are calculated on the same basis as Federal Corporate Taxes except:
  - There is a Ontario Resource Allowance Tax Credit equal to 25% of Net Corporate Tax.
  - The Ontario Corporate Tax Rate is 10% for mining operations.
- Ontario Mining Taxes are calculated as:
  - EBITDA.
  - Plus: Royalties payable to other stakeholders (except government royalties).
  - Less: Depreciation charged on New Mining Assets calculated on a Straight Line (SL) basis at 100%.
- Less: Depreciation on Ongoing Mining Assets calculated on a SL basis at 30%.
  - Less: Depreciation on Processing and Transportation Assets calculated on a SL basis at 15%.
  - Less: Depreciation Exploration and Development Expenses calculated on a DB basis at 100%.
  - Less: A Processing Allowance (PA) of 8% of processing and refining assets purchased and installed to date. The minimum PA is 15% of net income at this point with a maximum of 65% of net income at this point.
  - The first $10 million of net income at this point is tax free during the first three years of production.
  - The taxation rate is 10% of any net profits that exceed $500,000.
  - No deduction is allowed for interest expense or royalties paid to third parties.
  - Ontario Mining Tax is treated as a royalty rather than a tax as it is applied to the mine itself.

22.2 RESULTS

Results of the economic analysis are summarized in Table 22-1. The Goliath Project returns an IRR of 32.4% on a post-tax basis and 39.3% on a pre-tax basis. The respective payback periods are 2.8 years and 2.2 years after the start of production. The “break even” price of gold is US$930 per ounce post-tax and US$924 on a pre-tax basis where “break even” is the gold price required to produce a zero Net Cash Flow (i.e. all capital is paid back but no profit is incurred).

The project also generates a NCF of $249.8 million post-tax and $334.7 million pre-tax. At a 10% discount rate, the project’s NPVs are $83.5 million post-tax and $119.9 million pre-tax. A more detailed breakdown of the analysis is presented in Appendix B.

Table 22-1. Summary Net Cash Flow model and economic analysis.

PRODUCTION			
Preproduction Period	4.0	Years	
Open Pit Life	4.0 - 4.5	Years	
Underground Mine Life	7.8	Years	
Overall Mine Life	10.3	Years	
Preproduction Waste Stripping	1,800,000	tonnes	
Production Waste Stripping	40,230,000	tonnes	
Total OP Waste Mined	42,030,000	tonnes	(including 1.8 Mt preproduction)
OP Waste Ore Ratio	9.31:1.00		(including Stockpiled Ore)
OP Ore Mined & Milled	2,747,400	tonnes	
Gold Grade	3.36	g/t	
Silver Grade	6.24	g/t	
UG Ore Mined & Milled	4,525,500	tonnes	
Gold Grade	3.45	g/t	
Silver Grade	13.27	g/t	
OP Ore Stockpiled and Milled	1,765,700	tonnes	
Gold Grade	0.63	g/t	
Silver Grade	3.87	g/t	
Total Ore Mined & Milled	2,747,400	tonnes	
Gold Grade	2.87	g/t	
Silver Grade	9.30	g/t	
Mill Recoveries			
Gold	95%		
Silver	70%		
Precious Metal Production			
Gold	24,700	**kg**	
	793,000	**ozs**	
Silver	58,800	**kg**	
	1,892,000	**ozs**	
REVENUES			
Gold Prices	$1,375	US$/oz (3 Year avg at June 30, 2012)	
Silver Price	$26.00	US$/oz (3 Year avg at June 30, 2012)	
Exchange Rate	1.02	CAD$ per US$ (3 Year avg at June 30, 2012)	
Gold Revenue	$1,112,000,000		
Less: Smelting/Refining etc.	($11,000,000)		
Net Gold Revenue	$1,101,000,000		
Silver Revenue	$50,200,000		
Less: Smelting/Refining etc.	($500,000)		
Net Silver Revenue	$49,700,000		
Net Metal Revenue	$1,150,700,000		
Less: NSR Royalties			
Royalty Rate	0.652%		
Royalty Amount	$7,500,000		
Net Revenue after Royalties	$1,143,200,000		

Table 22-1. Summary Net Cash Flow model and economic analysis cont'd.

OPERATING COSTS	TOTAL	$/t	
Waste Stripping	$120,700,000	$3.00	per tonne waste mined
Open Pit Ore Mining	$14,200,000	$3.15	per tonne ore mined or stockpiled
Underground Mining	$281,600,000	$62.22	per tonne ore mined
Stockpile Rehandling	$1,400,000	$0.80	per tonne ore stockpiled
Processing	$142,900,000	$15.81	per tonne ore milled
G&A	$18,600,000	$2.06	per tonne ore milled
Total Operating Costs	$579,400,000	$64.10	per tonne ore milled
Operating Cost per Oz Gold	$668.44		after credit for silver
CAPITAL COSTS			
Preproduction	$102,600,000	$11.35	per tonne ore milled
Underground Development & Equip	$92,600,000	$20.46	per tonne UG ore mined
Sustaining	$6,200,000	$0.68	per tonne ore milled
Closer & Restorage (net of Salvage)	$950,000	$0.11	per tonne ore milled
Total Capital Costs	$200,450,000	$22.18	per tonne ore milled
Capital Costs per oz Gold Equivalent	$244.67		
CORPORATE & MINING TAXES			
Federal Corporate Tax	$51,500,000		
Ontario Corporate Tax	$33,500,000		
Ontario Mining Tax (Royalty)	$28,700,000		
Total Gov't Taxes and Royalties	$113,700,000		
EARNINGS			
EBITDA	$563,800,000		
Depreciation (Units of Production)	$199,100,000		
EBIT	$364,700,000		
	Pre-tax[1]	Post-tax	
NET CASH FLOW to PROJECT	$334,700,000	$249,800,000	
INTERNAL RATE OF RETURN	39.3%	32.4%	
NET PRESENT VALUES			
Discounted at 5.0%	$199,000,000	$144,300,000	
Discounted at 7.5%	$154,300,000	$109,900,000	
Discounted at 10.0%	$119,900,000	$83,500,000	
Discounted at 12.5%	$93,200,000	$63,200,000	
Discounted at 15.0%	$72,300,000	$47,300,000	
PAYBACK PERIOD	2.2 Years	2.8 Years	(from start of production)
Breakeven Gold Price	$923.72	$930.14	

1. Pre-tax excludes corporate taxes but includes the Ontario Mining Tax.
2. Some totals may not add due to rounding.

22.3 SENSITIVITY

Howe tested the sensitivity of the Goliath Project IRR to changes in metal prices, operating costs and capital costs. Metal prices and costs were varied up and down by 30%. As would be expected the IRR is more sensitive to changes in metal prices. The changes in operating and capital costs have approximately the same effect on the IRR. For instance, a drop in metal prices of 30%, leads to a post-tax IRR of 1.8% while an increase in metal prices of 30% raises the post-tax IRR to 54.9% (Figure 22-1). Similarly, an increase in operating costs of 30% drop in the post-tax IRR to 19.6% and a decrease in the operating costs of 30% raises the post-tax IRR to 43.6%.

Figure 22-1. Post-tax IRR sensitivity.

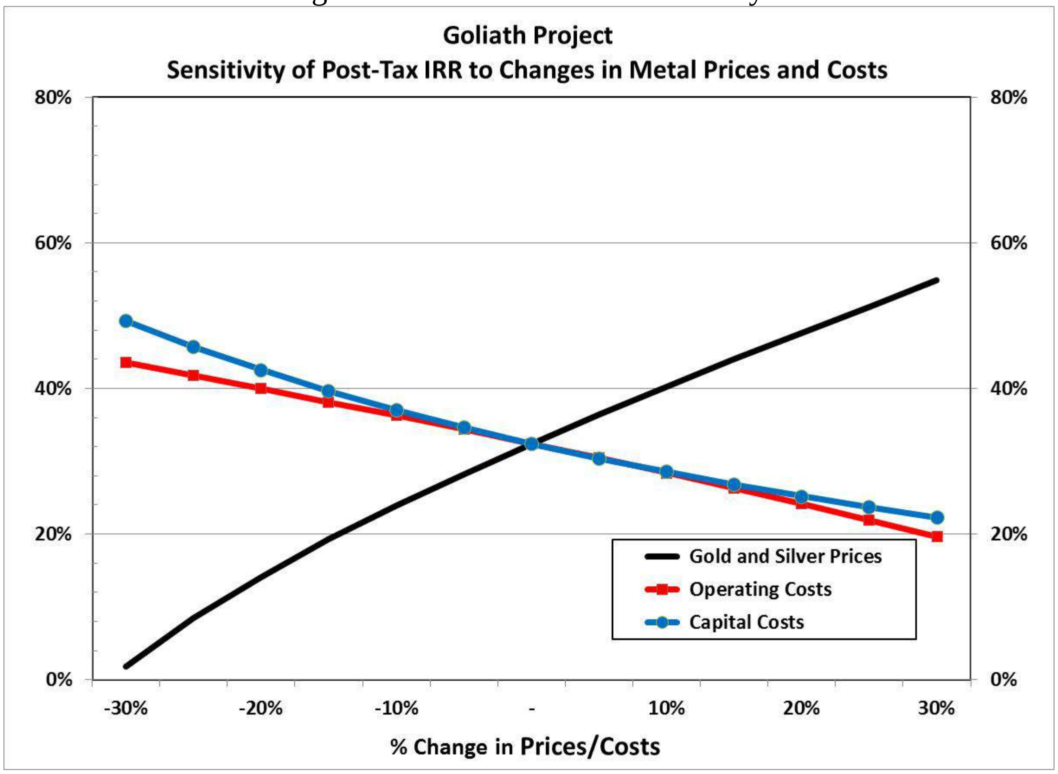
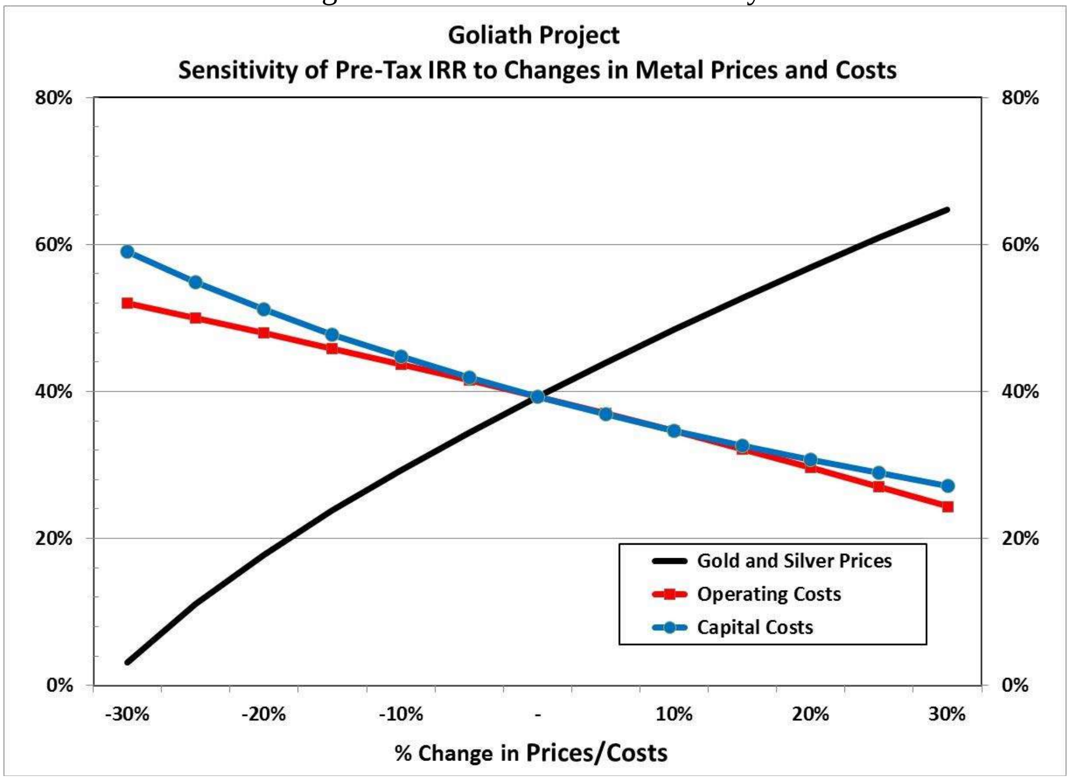


Figure 22-2. Pre-tax IRR sensitivity.



23 ADJACENT PROPERTIES

Howe is not aware of any other significant exploration programs or properties in the immediate area of the Goliath Project.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant information on the Goliath Project known to Howe that would make this Report more understandable or if undisclosed would make this Report misleading.

25 INTERPRETATION AND CONCLUSIONS

Howe has reviewed the Goliath Project data provided by Treasury, including the drilling database, and visited the site and reviewed sampling procedures and security. Howe believes the data presented by Treasury to be an accurate and reasonable representation of the Treasury project mineralization.

Work by Treasury at the Goliath Project has confirmed the grade of mineralization outlined by previous owners Teck and Corona, provided further detail on the nature of the mineralized zones and permitted the completion of an update to its 2010 NI 43-101 compliant Mineral Resource Estimate.

Drilling has outlined a series of nested, sub-vertical, relatively narrow zones to a maximum depth of approximately 800 metres and a strike length of approximately 2,300 metres. Mineral resource shells extend to a maximum depth of approximately 900 metres and a strike length of approximately 2,500 metres. A main zone, two hanging wall zones, and three footwall zones have been outlined. Higher-grade shoots are present in the main zone.

The near surface mineralization, down to a depth of 100-150 metres, would be amenable to surface mining methods. Underground mining methods would be more appropriate for the deeper mineralization.

The majority of the mineral resources are located in the Main Zone. Most of the Indicated mineral resources are located in the Main Zone, with a minor amount located in Zone C. The remainder were classified as Inferred.

There are no measured mineral resources, or mineral Reserves of any kind identified.

Resources are defined using a block cut-off grade of 0.3 g/tonne for surface resources (less than 150 metres deep) and 1.5 g/tonne for underground resources.

Non-diluted <u>Indicated</u> Mineral Resources (Surface plus Underground), located within the Main Zone and C-Zone, totalled 9.1 million tonnes with an average gold grade of 2.6 g/tonne and an average silver grade of 10.4 g/tonne, for 810,000 ounces of gold and gold equivalent.

Non-diluted <u>Inferred</u> Mineral Resources (Surface plus Underground), from all zones, totalled 15.9 million tonnes with an average gold grade of 1.7 g/tonne and an average silver grade of 3.9 g/tonne, for 900,000 ounces of gold and gold equivalent.

Howe's economic modelling and analysis of the Project reveals the Project could yield a post-tax IRR of 32.4% and a post-tax NPV, discounted at 7.5%, of C$109.9 million. In Howe's opinion the Goliath Project is a potentially very robust one and warrants Treasury's continued advancement of the Project towards an eventual pre-feasibility study.

26 RECOMMENDATIONS

To proceed with the assessment of the potential development of the Project, Howe recommends surface and underground bulk sampling, and pilot plant testing be undertaken.

For surface work, a portion of the Main Zone would be stripped-off. Geological mapping and sampling would be carried out. A bulk sample of at least 5,000 tonnes would be taken. The sample would be split down to 50-100 tonnes then shipped to a pilot plant laboratory facility.

For underground work, the existing exploration portal, decline, and underground workings could be rehabilitated and used as a starting point from which the B and C-Zones would eventually be accessed for bulk sampling purposes. As with the surface sample, this would be split down to 50-100 tonnes then shipped to a pilot plant laboratory facility.

In addition to the bulk samples, the lateral development and raising needed to collect the samples, plus any test stoping that would be carried out as well, would allow mining and processing parameters to be determined to a preliminary feasibility study level of accuracy (+/- 15-20%). Should the preliminary feasibility study yield positive results, mineral reserves can be identified for the Project. The grand total budgetary cost for this work is estimated to be in the order of C$3.2 million as tabled below.

Item	Description	Budgetary Cost	Totals
Surface Work:			
1	Strip main zone.	$ 25,000	
2	Mine 5,000 tonnes from surface.	$ 25,000	
3	Geological mapping and sampling.	$ 15,000	
4	Reclamation.	$ 10,000	
5	Sample shipping.	$ 20,000	
6	Blast hole assays.	$ 5,000	
7	Statistical and geostatistical work.	$ 10,000	
8	Pilot plant work.	$ 20,000	
9	Management and Supervision	$ 50,000	
10	Mineral processing analysis of results.	$ 10,000	
11	Contingency (20%)	$ 40,000	
Subtotal, Surface Work (Rounded)			$ 230,000
Underground Work			
12	Excavate portal.	$ 50,000	
13	Lateral Development (200 m)	$ 1,000,000	
14	Raising	$ 500,000	
15	Test stoping - 5,000 tonnes.	$ 500,000	
16	Geological mapping and sampling.	$ 50,000	
17	Assays	$ 10,000	
18	Statistical and geostatistical work.	$ 10,000	
19	Sample Shipping	$ 20,000	
20	Pilot plant work.	$ 20,000	
21	Management and Supervision	$ 50,000	
22	Mineral processing analysis of results.	$ 10,000	
23	Contingency (20%)	$ 440,000	
Subtotal, Underground Work (Rounded)			$ 2,700,000
Pre-Feasibility Study			$ 300,000
Grand Total (Rounded)			**$ 3,200,000**

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28 DATE AND SIGNATURE PAGE

This report titled "Preliminary Economic Analysis of the Goliath Gold Project, Kenora Mining Division, north-western Ontario, Canada" for Treasury Metals Inc. with an effective date of July 19, 2012 was prepared and signed by the following authors:

{SIGNED }
[*William Douglas Roy*]

Dated Aug. 20, 2012 at Halifax, Nova Scotia

William Douglas Roy, M.A.Sc., P.Eng.
Associate Mining Engineer
A.C.A. Howe International Limited

{SIGNED }
[*Ian D. Trinder]*

Dated Aug. 20, 2012 at Toronto, Ontario

Ian D. Trinder, M.Sc., P.Geo.
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29 CERTIFICATES OF QUALIFICATIONS

CERTIFICATE OF CO-AUTHOR: WILLIAM DOUGLAS ROY, M.A.Sc., P.ENG.

I, William Douglas Roy, M.A.Sc., P.Eng. (APENS), do hereby certify that:

1. I am an Associate Mining Engineer with A.C.A. Howe International Limited, whose office is located at 365 Bay Street, Suite 501, Toronto, Ontario, Canada.
2. I graduated with a Bachelor of Engineering ("B.Eng.") degree in Mining Engineering from the Technical University of Nova Scotia (now Dalhousie University) in 1997 and with a Master of Applied Science ("M.A.Sc.") degree in Mining Engineering from Dalhousie University in 2000.
3. I am a Professional Mining Engineer registered with the Association of Professional Engineers of Nova Scotia (Registered Professional Engineer, No. 7472).
4. I have worked as a mining engineer for fifteen years since graduating from university. This work has included the estimation of resources and reserves for precious metals, base metals and industrial minerals, as well as participation in pre-feasibility and feasibility studies.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43- 101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am co-author of the technical report titled: "Preliminary Economic Analysis of the Goliath Gold Project, Kenora Mining Division, north-western Ontario, Canada" for Treasury Metals Inc. dated July 19th, 2012, (the "Technical Report"). I am responsible for Section 14 and portions of Sections 1, 16, 18, 25, and 26. I visited the Goliath Project from November 25th to 27th, 2011.
7. I have read NI 43-101 and Form 43-101 F1. This Technical Report has been prepared in compliance with that Instrument and form.
8. I have had limited prior involvement with Treasury Metals Inc., their Principals or their shareholders. In 2008, I co-authored an independent Technical Report for the Thunder Lake mineral property titled "Report on the Goliath Project - Kenora Mining Division, north-western Ontario, Canada for Treasury Metals Inc." In 2010, I co-authored an independent Technical Report for the Thunder Lake mineral property titled "Technical Report and Preliminary Economic Assessment on the Goliath Gold Project - Kenora Mining Division, north-western Ontario, Canada for Treasury Metals Inc.**".** **A**nd in 2011 I co-authored an independent Technical Report for the Thunder Lake mineral property titled "Technical report and mineral resource update on the Goliath Gold Project - Kenora Mining Division, Northwestern Ontario, Canada for Treasury Metals Inc. A.C.A. Howe International Limited."
9. I am not aware of any material fact or material change with respect to the subject matter of this Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.
10. I am independent of the issuer, Treasury Metals Inc., applying all of the tests in Section 1.5 of NI 43-101 and Section 1.5 of NI 43-101 CP.
11. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the report not misleading.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes.

Effective Date: July 19th, 2012
DATED this 20th Day of August 2012.

{SIGNED and SEALED }
[*William D. Roy]*_____________
William Douglas Roy, M.A.Sc., P. Eng.
Associate Mining Engineer

CERTIFICATE OF CO-AUTHOR: IAN D. TRINDER, M.Sc., P.GEO.

I, Ian D. Trinder, M.Sc., P.Geo. (APGO, APEGM), do hereby certify that:

1. I reside at <personal information removed>
2. I am a employed as a Senior Geologist with the firm of A.C.A. Howe International Limited, Mining and Geological Consultants located at 365 Bay St., Suite 501, Toronto, Ontario, Canada. M5H 2V1.
3. I graduated with a degree in Bachelor of Science Honours, Geology, from the University of Manitoba in 1983 and a Master of Science, Geology, from the University of Western Ontario in 1989.
4. I am a Professional Geoscientist (P.Geo.) registered with the Association of Professional Engineers and Geoscientists of Manitoba (APEGM, No. 22924) and with the Association of Professional Geoscientists of Ontario (APGO, No. 452). I am a member of the Society of Economic Geologists and of the Prospectors and Developers Association of Canada.
5. I have over 25 years of direct experience with precious and base metals mineral exploration in Canada, USA and the Philippines including project evaluation and management. Additional experience includes the completion of various National Policy 2A and NI 43-101 technical reports for gold and base metal projects.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I am co-author of the technical report titled: "Preliminary Economic Analysis of the Goliath Gold Project, Kenora Mining Division, north-western Ontario, Canada" for Treasury Metals Inc. dated July 19th, 2012, (the "Technical Report"). I am responsible for Sections 1 to 13, 15, 19, 32 to 24 and 27 and portions of Sections 1, 20, 25, and 26 of the report. I visited the Goliath Project from September 14th to 16th, 2008.
8. I have had limited prior involvement with Treasury Metals Inc., their Principals or their shareholders. In 2008, I co-authored an independent Technical Report for the Thunder Lake mineral property titled "Report on the Goliath Project - Kenora Mining Division, north-western Ontario, Canada for Treasury Metals Inc." In 2010, I co-authored an independent Technical Report for the Thunder Lake mineral property titled "Technical Report and Preliminary Economic Assessment On The Goliath Gold Project - Kenora Mining Division, north-western Ontario, Canada for Treasury Metals Inc.". And in 2011, I co-authored an independent Technical Report for the Thunder Lake mineral property titled "Technical Report and Mineral Resource Update on the Goliath Gold Project, Kenora Mining Division, northwestern Ontario, Canada".
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, and the omission to disclose which would make the Technical Report misleading.
10. I am independent of the issuer, Treasury Metals Inc., applying all of the tests in Section 1.5 of NI 43-101 and Section 1.5 of NI 43-101 CP.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: July 19th, 2012
DATED this 20th Day of August 2012.

{SIGNED and SEALED}
[*Ian D. Trinder]*

____
Ian D. Trinder, M.Sc., P. Geo.
Senior Geologist

CERTIFICATE OF CO-AUTHOR: BRUCE BRADY, B.ENG., P.ENG.

I, Bruce Brady, B.Eng., P.Eng. (PEO, OIQ), do hereby certify that:

1. I reside at <personal information removed>
2. I am an Associate Mining Engineer with ACA Howe International Limited, whose office is located at 365 Bay Street, Suite 501, Toronto, Ontario, Canada.
3. I graduated with a Bachelor of Engineering ("B.Eng.") degree in Mining Engineering from McGill University in 1972.
4. I am a Professional Engineer registered with Professional Engineers Ontario (PEO) and the Quebec Order of Engineers (OIQ).
5. I have worked as a mining engineer for forty years since graduating from university and my experience includes mine operation, mine engineering, project evaluation, and feasibility studies.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I am co-author of the technical report titled: "Preliminary Economic Analysis of the Goliath Gold Project, Kenora Mining Division, north-western Ontario, Canada, for Treasury Metals Inc.", dated July 19th, 2012, (the "Technical Report"). I am responsible for Section 16 and portions of Sections 1, 18, 20, 21, 25, and 26 of the report. I have not visited the Goliath Project.
8. I have had no prior involvement with Treasury Metals Inc., their Principals, or their shareholders.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I am independent of the issuer, Treasury Metals Inc., applying all of the tests in Section 1.5 of NI 43-101 and Section 1.5 of NI 43-101 CP.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: July 19th, 2012
DATED this 20th Day of August 2012.

{SIGNED and SEALED }
[*Bruce Brady]*

____

Bruce Brady, P. Eng.
Associate Mining Engineer

CERTIFICATE OF CO-AUTHOR: GORDON WATTS, P.ENG.

I, Gordon Watts, B.A.Sc., P.Eng. (PEO) do hereby certify that:

1. I reside at <personal information removed> .
2. I am a Senior Associate Mineral Economist with ACA Howe International Limited, whose office is located at 365 Bay Street, Suite 501, Toronto, Ontario, Canada.
3. I graduated with a Bachelor of Applied Science ("B.A.Sc.") degree in Mining Engineering from the University of Toronto in 1969.
4. I am a Professional Engineer registered with Professional Engineers Ontario (PEO, number 49149016).
5. I have worked as a mining engineer for forty two years since graduating from university. My relevant experience includes:
   - The preparation of over 250 financial models during the past 28 years;
   - Skilled in tax modeling, tisk analysis and Monte Carlo simulations;
   - Constructed numerous mining cash flow models for mining consulting companies e.g. ACA Howe; Watts, Griffis, McOuat; Scott Wilson Roscos Postle Assoc.; MPH; Derry Michener Booth and Wahl.
   - Prepared reports on mineral properties throughout Canada, the United States of America, internationally.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I am co-author of the technical report titled: "Preliminary Economic Analysis of the Goliath Gold Project, Kenora Mining Division, north-western Ontario, Canada for Treasury Metals In**c.",** dated July 19th, 2012, (the "Technical Report"). I am responsible for Section 16 and portions of Sections 1, 18, 20, 21, 25, and 26 of the report. I have not visited the Goliath Project.
8. I have had no prior involvement with Treasury Metals Inc., their Principals, or their shareholders.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I am independent of the issuer, Treasury Metals Inc., applying all of the tests in Section 1.5 of NI 43-101 and Section 1.5 of NI 43-101 CP.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: July 19th, 2012
DATED this 20th Day of August 2012.

{SIGNED and SEALED }
[*Gordon Watts]*

____

Gordon Watts, P. Eng.
Senior Associate Mineral Economist

CERTIFICATE OF CO-AUTHOR: ALFRED HAYDEN, P.ENG.

I, Alfred S. Hayden, B.A.Sc., P.Eng. (PEO), do hereby certify that:

1. I am President of EHA Engineering Ltd , PO Box 2711, Postal Station "B", Richmond Hill Ontario.
2. I graduated from the University of British Columbia, Vancouver, B.C. in 1967 with a Bachelor of Applied Science in Metallurgical Engineering.
3. I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum and a Professional Engineer and Designated Consulting Engineer registered with Professional Engineers Ontario.
4. I have worked as a metallurgical engineer for a total of 45 years since my graduation from university.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am co-author of the technical report titled: "Preliminary Economic Analysis of the Goliath Gold Project, Kenora Mining Division, north-western Ontario, Canada" for Treasury Metals Inc.", dated July 19th, 2012, (the "Technical Report"). I am responsible for Sections 13 and 17; and portions of Sections 1, 21, 25, and 26 of the report. I have not visited the Project.
7. I have had no prior involvement with the property that is the subject of the Technical Report.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: July 19th, 2012
DATED this 20th Day of August 2012.

{SIGNED and SEALED }
[*Al Hayden]*

Al Hayden, P. Eng.
Associate Metallurgical Engineer

Appendix A
Lists of Unpatented and Patented Claims

Table 1-1. List of the unpatented (staked) mining claims, Goliath Project
Hartland and Zealand Townships, Ontario.

Township/Area	Claim Number	Claim Recording Date	Claim Due Date	Claim Units	Area (ha)	Status
HARTMAN	1144513	1991-Feb-26	2016-Feb-26	1	16	A
HARTMAN	1144514	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144515	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144516	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144517	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144518	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144519	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144520	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144521	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144522	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144523	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144524	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144525	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144526	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144527	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144528	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144529	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144530	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144531	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144532	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144533	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144534	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144535	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144536	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144537	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144538	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144539	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144540	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144541	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144542	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144543	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144544	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144545	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144546	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144547	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144548	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144549	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144550	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144551	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144552	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144553	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144554	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1144555	1991-Jan-26	2016-Jan-26	1	16	A
HARTMAN	1144556	1991-Feb-26	2015-Feb-26	1	16	A
HARTMAN	1210898	1996-Apr-02	2015-Apr-02	1	16	A
HARTMAN	1211082	1996-Apr-02	2014-Apr-02	4	64	A
HARTMAN	1247442	2007-Aug-21	2015-Aug-21	4	64	A
HARTMAN	3017886	2009-Jul-10	2015-Jul-10	4	64	A
HARTMAN	3017887	2009-Jul-10	2015-Jul-10	12	192	A
HARTMAN	3017888	2009-Jul-10	2015-Jul-10	1	16	A
HARTMAN	3017889	2009-Jul-10	2015-Jul-10	12	192	A
HARTMAN	3017890	2009-Jul-10	2017-Jul-10	8	128	A
HARTMAN	4211247	2007-Aug-21	2015-Aug-21	8	128	A

Township/Area	Claim Number	Claim Recording Date	Claim Due Date	Claim Units	Area (ha)	Status
HARTMAN	4211248	2007-Aug-21	2015-Aug-21	8	128	A
HARTMAN	4211249	2007-Aug-21	2015-Aug-21	8	128	A
HARTMAN	4211250	2007-Aug-21	2015-Aug-21	4	64	A
HARTMAN	4245003	2011-Feb-28	2016-Feb-28	4	64	A
HARTMAN	4245004	2011-Feb-28	2016-Feb-28	8	128	A
HARTMAN	4245005	2011-Feb-28	2016-Feb-28	8	128	A
ZEALAND	1106347	1989-Oct-13	2018-Oct-13	1	16	A
ZEALAND	1106348	1989-Oct-13	2018-Oct-13	1	16	A
ZEALAND	1106349	1989-Oct-13	2015-Oct-13	1	16	A
ZEALAND	1106350	1989-Oct-13	2015-Oct-13	1	16	A
ZEALAND	1106351	1989-Oct-13	2015-Oct-13	1	16	A
ZEALAND	1106352	1989-Oct-13	2015-Oct-13	1	16	A
ZEALAND	1119531	1989-Oct-26	2018-Oct-26	1	16	A
ZEALAND	1119532	1989-Oct-26	2018-Oct-26	1	16	A
ZEALAND	1119537	1989-Oct-26	2018-Oct-26	1	16	A
ZEALAND	1119538	1989-Oct-26	2018-Oct-26	1	16	A
ZEALAND	1119541	1989-Oct-26	2016-Oct-26	1	16	A
ZEALAND	1119542	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119543	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119544	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119545	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119546	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119547	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119548	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119549	1989-Oct-26	2018-Oct-26	1	16	A
ZEALAND	1119550	1989-Oct-26	2018-Oct-26	1	16	A
ZEALAND	1119551	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119552	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119553	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119554	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119555	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119556	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119557	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119558	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119559	1989-Oct-26	2018-Oct-26	1	16	A
ZEALAND	1119560	1989-Oct-26	2018-Oct-26	1	16	A
ZEALAND	1119561	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119562	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119563	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119564	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119565	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119566	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119567	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1119568	1989-Oct-26	2015-Oct-26	1	16	A
ZEALAND	1144557	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144558	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144559	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144560	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144561	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144562	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144563	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144564	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144565	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144566	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144567	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144568	1991-Feb-26	2015-Feb-26	1	16	A

Township/Area	Claim Number	Claim Recording Date	Claim Due Date	Claim Units	Area (ha)	Status
ZEALAND	1144569	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144570	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144573	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144574	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144575	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144576	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144577	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144578	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144579	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144580	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144581	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144582	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144583	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144584	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144585	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144586	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144587	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1144588	1991-Feb-26	2015-Feb-26	1	16	A
ZEALAND	1145300	1992-Jun-23	2014-Jun-23	4	64	A
ZEALAND	1145301	1992-Jun-23	2014-Jun-23	2	32	A
ZEALAND	3017934	2008-May-21	2015-May-21	4	64	A
ZEALAND	3017936	2008-May-21	2015-May-21	5	80	A
ZEALAND	3017937	2008-May-21	2015-May-21	9	144	A
ZEALAND	3017938	2008-May-26	2015-May-26	2	32	A
ZEALAND	3017939	2008-Jul-04	2015-Jul-04	6	96	A
ZEALAND	3017940	2008-Sep-10	2015-Sep-10	4	64	A
ZEALAND	3017941	2008-Oct-10	2015-Oct-10	4	64	A
ZEALAND	4211252	2007-Sep-06	2015-Sep-06	8	128	A
TOTAL	**137**			**254**	**4064**	

Notes: Source: Ontario Provincial Recording Office (MNDMF), July 19, 2012

Table 1-2. Patented land parcels (optioned and owned private lands)

TOWNSHIP	PARTY	PARCEL	LOT/CONCESSION	AREA (ha)	*RIGHTS
Zealand[1]	Lundmark	41941	N ½ Lot 6, Con III	66.57	MRO
Zealand[1]	Collins	17395	N ½ Lot 5, Con IV	66.4	MRO
Zealand[1]	Sheridan	21374	S.V. 200, Con III	16	M+SR
Zealand[1]	Johnson	15401	N ½ of S ½ Lot 5, Con IV	32	M+SR
Zealand[1]	Hudak	21609	N part of S ½ Lot 7, Con IV	31.56	M+SR
Zealand[1]	Fraser	15395	S ½ Lot 6, Con IV	65.96	MRO
Zealand[1]	Fraser	15395	S ½ Lot 6, Con IV	16.59	SRO
Zealand[1]	Betker	34461	W ½ of S ½ Lot 6, Con IV	32.78	SRO
Zealand[1]	LeClerc	34303	SE ¼ of S ½ Lot 6, Con IV	16.59	SRO
Zealand[2]	Delk	24724	SW ¼ of N ½ Lot 1, Con IV	16.23	M+SR
Zealand[2]	Davenport	19088	S ½ Lot 1, Con V	65.76	M+SR
Zealand[3]	Jones	41215	S part of Lot 8, Con IV	64.75	MRO
Hartman[2]	Nemeth	6556	S ½ Lot 10, Con IV	65.35	M+SR
Zealand[4]	Sterling	4822	Lot 7, Con III	78.4	M+SR
Zealand[4]	Medlee	21553	N Pt. Lot 8, Con III	31.1	MRO
Zealand[4]	Schultz	13492	Lot 7, Con III	57	M+SR
Zealand	Brisson	23R2434	Part of Broken Lot 9, Con IV	40.8711	SRO
Zealand	Tree Nursery	41807	Pts. 1-5 23R-9766	91.323	MR+SR
Zealand	Tree Nursery	41810	Pts. 1-2 23R-9937	26.169	SRO

[1]Thunder Lake West; [2]Thunder Lake East; [3]Jones Property, [4]Laramide Property *MRO=Mineral Rights only; SRO = Surface Rights only; M+SR=Mineral and Surface Rights

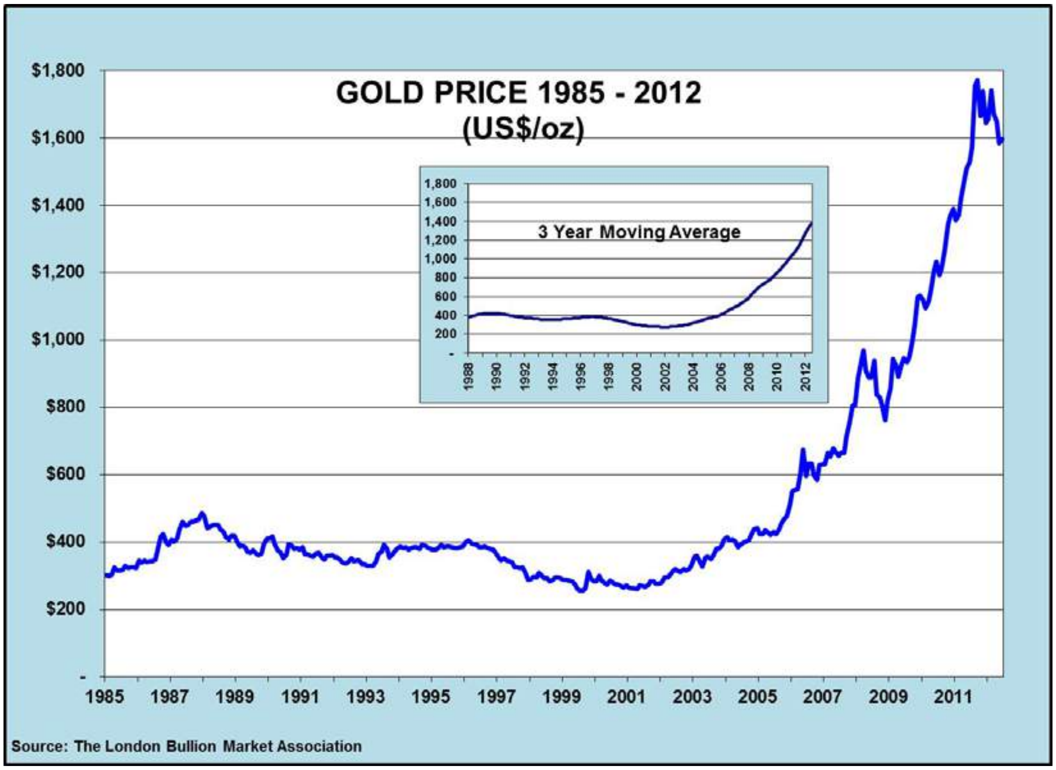
Appendix B:
Net Cash Flow Model

Detailed Net Cash Flow model and economic analysis

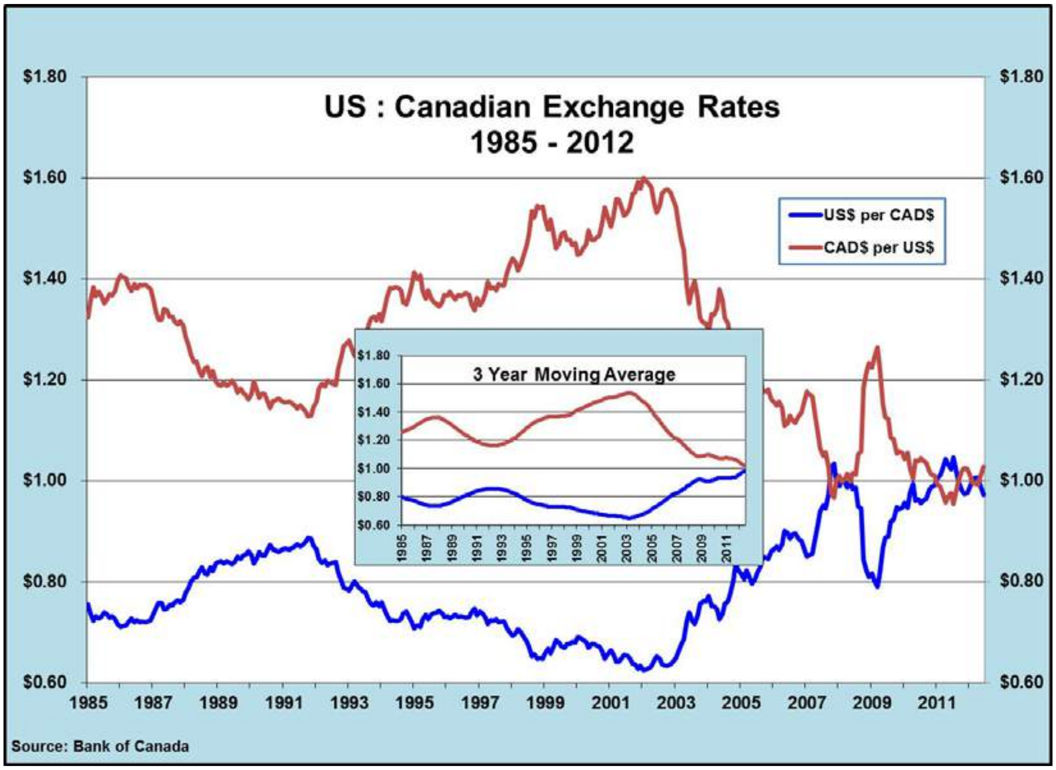
Gold Price - US$1375/oz	Units	Total/ Average	Year -3	Year -2	Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
PRODUCTION																		
Central Pit																		
Waste Mined	kt	11,040	-	-	-	1,800	9,240	-	-	-	-	-	-	-	-	-	-	-
Ore Mined & Milled	kt	939	-	-	-	-	875	64	-	-	-	-	-	-	-	-	-	-
Gold Grade	g/t	3.24	-	-	-	-	3.24	3.24	-	-	-	-	-	-	-	-	-	-
Silver Grade	g/t	8.28	-	-	-	-	8.28	8.28	-	-	-	-	-	-	-	-	-	-
Ore Stockpiled	kt	823	-	-	-	-	767	56	-	-	-	-	-	-	-	-	-	-
Gold Grade	g/t	0.62	-	-	-	-	0.62	0.62	-	-	-	-	-	-	-	-	-	-
Silver Grade	g/t	4.15	-	-	-	-	4.15	4.15	-	-	-	-	-	-	-	-	-	-
Western Pit																		
Waste Mined	kt	12,280	-	-	-	-	1,500	5,000	5,780	-	-	-	-	-	-	-	-	-
Ore Mined & Milled	kt	1,079	-	-	-	-	-	567	512	-	-	-	-	-	-	-	-	-
Gold Grade	g/t	2.69	-	-	-	-	-	2.69	2.69	-	-	-	-	-	-	-	-	-
Silver Grade	g/t	4.38	-	-	-	-	-	4.38	4.38	-	-	-	-	-	-	-	-	-
Ore Stockpiled	kt	722	-	-	-	-	-	379	343	-	-	-	-	-	-	-	-	-
Gold Grade	g/t	0.66	-	-	-	-	-	0.66	0.66	-	-	-	-	-	-	-	-	-
Silver Grade	g/t	2.37	-	-	-	-	-	2.37	2.37	-	-	-	-	-	-	-	-	-
Eastern Pit																		
Waste Mined	kt	18,710	-	-	-	-	1,000	5,300	3,700	7,500	1,210	-	-	-	-	-	-	-
Ore Mined & Milled	kt	729	-	-	-	-	-	244	144	292	49	-	-	-	-	-	-	-
Gold Grade	g/t	4.51	-	-	-	-	-	4.51	4.51	4.51	4.51	-	-	-	-	-	-	-
Silver Grade	g/t	6.39	-	-	-	-	-	6.39	6.39	6.39	6.39	-	-	-	-	-	-	-
Ore Stockpiled	kt	221	-	-	-	-	-	74	44	88	15	-	-	-	-	-	-	-
Gold Grade	g/t	0.61	-	-	-	-	-	0.61	0.61	0.61	0.61	-	-	-	-	-	-	-
Silver Grade	g/t	7.70	-	-	-	-	-	7.70	7.70	7.70	7.70	-	-	-	-	-	-	-
Underground																		
Ore Mined & Milled	kt	4,526	-	-	-	-	-	-	219	583	583	583	583	583	583	583	226	-
Gold Grade	g/t	3.45	-	-	-	-	-	-	4.17	3.96	3.83	3.77	3.70	2.57	2.57	3.41	3.58	-
Silver Grade	g/t	13.27	-	-	-	-	-	-	12.36	10.80	13.08	15.31	15.47	18.11	18.11	6.08	3.72	-
Stockpile Feed			-	-	-													
Ore Reclaimed & Milled	kt	1,766	-	-	-	-	-	-	-	-	243	292	292	292	292	292	63	-
Gold Grade	g/t	0.63	-	-	-	-	-	-	-	-	0.63	0.63	0.63	0.63	0.63	0.63	0.63	-
Silver Grade	g/t	3.87	-	-	-	-	-	-	-	-	3.87	3.87	3.87	3.87	3.87	3.87	3.87	-
Totals																		
Waste Mined	kt	42,030	-	-	-	1,800	11,740	10,300	9,480	7,500	1,210	-	-	-	-	-	-	-
OP Ore Mined & Milled	kt	2,747	-	-	-	-	875	875	656	292	49	-	-	-	-	-	-	-
UG Ore Mined & Milled	kt	4,526	-	-	-	-	-	-	219	583	583	583	583	583	583	583	226	-
Ore Stockpiled	kt	1,766	-	-	-	-	767	509	386	88	15	-	-	-	-	-	-	-
Ore Milled	kt	9,039	-	-	-	-	875	875	875	875	875	875	875	875	875	875	289	-
Gold Grade	g/t	2.87	-	-	-	-	3.24	3.23	3.36	4.15	2.98	2.73	2.68	1.93	1.93	2.48	2.94	-
Silver Grade	g/t	9.30	-	-	-	-	8.28	5.22	6.70	9.32	10.15	11.49	11.60	13.36	13.36	5.34	3.76	-
Mill Recoveries																		
Gold	%	95%	-	-	-	-	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	-
Silver	%	70%	-	-	-	-	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	-
Metal Production																		
Gold	kg	24,664	-	-	-	-	2,697	2,689	2,790	3,446	2,478	2,266	2,226	1,600	1,600	2,065	805	-
	ozs	793	-	-	-	-	87	86	90	111	80	73	72	51	51	66	26	-
Silver	kg	58,839	-	-	-	-	5,071	3,200	4,105	5,711	6,214	7,038	7,106	8,181	8,181	3,274	759	-
	ozs	1,892	-	-	-	-	163	103	132	184	200	226	228	263	263	105	24	-
REVENUES																		
Gold Revenue	kCAD$	1,112,127	-	-	-	-	121,611	121,244	125,814	155,407	111,749	102,171	100,389	72,164	72,164	93,115	36,299	-
Less: Smelting/Refining etc.	kCAD$	11,101	-	-	-	-	1,214	1,210	1,256	1,551	1,115	1,020	1,002	720	720	929	362	-
Net Gold Revenue	kCAD$	1,101,026	-	-	-	-	120,397	120,034	124,558	153,855	110,634	101,151	99,387	71,444	71,444	92,186	35,936	-
Silver Revenue	kCAD$	50,168	-	-	-	-	4,324	2,729	3,500	4,870	5,298	6,001	6,058	6,975	6,975	2,791	647	-
Less: Smelting/Refining etc.	kCAD$	492	-	-	-	-	42	27	34	48	52	59	59	68	68	27	6	-
Net Silver Revenue	kCAD$	49,676	-	-	-	-	4,281	2,702	3,466	4,822	5,246	5,942	5,999	6,907	6,907	2,764	641	-
Net Metal Revenue	kCAD$	1,150,702	-	-	-	-	124,678	122,736	128,024	158,677	115,880	107,093	105,386	78,350	78,350	94,950	36,577	-
NSR ROYALTY SCHEDULE																		
Central Pit																		
Royalty Rate	%	0.03%	-	-	-	-	0.03%	0.03%	-	-	-	-	-	-	-	-	-	-
Royalty Amount	kCAD$	39	-	-	-	-	37	3	-	-	-	-	-	-	-	-	-	-
Western Pit																		
Royalty Rate	%	0.03%	-	-	-	-	-	0.03%	0.03%	-	-	-	-	-	-	-	-	-
Royalty Amount	kCAD$	37	-	-	-	-	-	19	18	-	-	-	-	-	-	-	-	-
Eastern Pit																		
Royalty Rate	%	2.00%	-	-	-	-	-	2.00%	2.00%	-	-	-	-	-	-	-	-	-
Royalty Amount	kCAD$	1,483	-	-	-	-	-	932	551	-	-	-	-	-	-	-	-	-
Underground																		
Royalty Rate	%	0.82%	-	-	-	-	-	-	0.80%	0.98%	0.96%	0.90%	0.89%	0.66%	0.66%	0.75%	0.76%	-
Royalty Amount	kCAD$	5,737	-	-	-	-	-	-	315	980	937	872	843	450	450	631	260	-
Stockpile Feed																		
Royalty Rate	%	0.41%	-	-	-	-	-	-	-	-	0.41%	0.41%	0.41%	0.41%	0.41%	0.41%	0.41%	-
Royalty Amount	kCAD$	206	-	-	-	-	-	-	-	-	28	34	34	34	34	34	7	-
Total NSR Royalties	kCAD$	-																
Royalty Rate	%	0.65%	-	-	-	-	0.03%	0.78%	0.69%	0.62%	0.83%	0.85%	0.83%	0.62%	0.62%	0.70%	0.73%	-
Royalty Amount	kCAD$	7,502	-	-	-	-	37	954	883	980	965	906	877	485	485	665	267	-
Net Revenue after Royalties	kCAD$	1,143,200	-	-	-	-	124,642	121,782	127,141	157,698	114,915	106,188	104,509	77,866	77,866	94,284	36,310	-

Detailed Net Cash Flow model and economic analysis cont’d.

OPERATING COSTS																		
Waste Stripping	**kCAD$**	**120,690**	-	-	-	-	35,220	30,900	28,440	22,500	3,630	-	-	-	-	-	-	-
Open Pit Ore Mining	**kCAD$**	**14,216**	-	-	-	-	5,172	4,361	3,284	1,198	201	-	-	-	-	-	-	-
Underground Mining	**kCAD$**	**281,566**	-	-	-	-	-	-	13,327	35,539	36,122	36,122	36,122	36,705	36,705	36,705	14,221	-
Stockpile Rehandling	**kCAD$**	**1,413**	-	-	-	-	-	-	-	-	194	234	234	234	234	234	50	-
Processing	**kCAD$**	**142,900**	-	-	-	-	13,834	13,834	13,834	13,834	13,834	13,834	13,834	13,834	13,834	13,834	4,563	-
G&A	**kCAD$**	**18,594**	-	-	-	-	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	594	-
Total Operating Costs	**kCAD$**	**579,378**	-	-	-	-	56,025	50,895	60,685	74,871	55,781	51,989	51,989	52,572	52,572	52,572	19,427	-
EBITDA	**kCAD$**	**563,822**	-	-	-	-	68,616	70,887	66,456	82,827	59,134	54,199	52,520	25,294	25,294	41,712	16,883	-
Less: Depreciation	**kCAD$**	**199,078**	-	-	-	-	9,083	12,710	16,844	18,834	21,346	21,923	22,296	22,521	22,736	23,037	7,748	-
EBIT	**kCAD$**	**364,743**	-	-	-	-	59,533	58,178	49,612	63,993	37,787	32,275	30,224	2,773	2,558	18,676	9,135	-
Less: Federal Corp. Tax	**kCAD$**	**51,472**	-	-	-	-	-	4,629	7,274	9,391	6,046	5,844	5,846	2,511	2,777	5,179	1,975	-
Ontario Corp. Tax	**kCAD$**	**33,457**	-	-	-	-	-	3,009	4,728	6,104	3,930	3,799	3,800	1,632	1,805	3,367	1,284	-
Ontario Mining Tax	**kCAD$**	**28,662**	-	-	-	-	2,909	1,921	2,629	3,870	2,287	2,723	4,353	1,731	1,759	3,484	998	-
Net Income after Depr. & Taxes	**kCAD$**	**251,152**	-	-	-	-	56,624	48,619	34,981	44,628	25,526	19,910	16,225	3,101	3,783	6,645	4,878	-
POST TAX NET CASH FLOW to PROJECT																		
Net Income after Depr. & Taxes	**kCAD$**	**251,152**	-	-	-	-	56,624	48,619	34,981	44,628	25,526	19,910	16,225	3,101	3,783	6,645	4,878	-
Plus: Depreciation	**kCAD$**	**199,078**	-	-	-	-	9,083	12,710	16,844	18,834	21,346	21,923	22,296	22,521	22,736	23,037	7,748	-
Less: Capital Investment	**kCAD$**	**200,478**	1,675	675	27,456	62,753	1,267	33,836	34,440	14,583	15,905	3,075	1,614	750	500	400	150	1,400
Working Capital	**kCAD$**	**-**	-	-	-	-	13,231	27	234	2,991	4,204	868	197	3,142	-	1,915	5,461	4,473
Net Cash Flow to Project	**kCAD$**	**249,752**	1,675	675	27,456	62,753	51,209	27,520	17,152	45,888	35,171	39,626	37,104	21,812	18,453	27,367	17,937	3,073
Accum. NCF to Project	**kCAD$**	**249,752**	1,675	2,350	29,806	92,560	41,350	13,831	3,321	49,209	84,380	124,006	161,110	182,923	201,375	228,742	246,679	249,752
Internal Rate of Return	**%**	**32.4%**						**Payback Period**	**2.8 Years from start of Production**									
Net Present Value	**MCAD$**			**5.0%**	**$144.3**		**7.5%**	**$109.9**		**10.0%**	**$83.5**		**12.5%**	**$63.2**		**15.0%**	**$47.3**	
PRE TAX NET CASH FLOW to PROJECT																		
Post Tax NCF to Project	**kCAD$**	**249,752**	1,675	675	27,456	62,753	51,209	27,520	17,152	45,888	35,171	39,626	37,104	21,812	18,453	27,367	17,937	3,073
Plus: Federal Corporate Taxes	**kCAD$**	**51,472**	-	-	-	-	-	4,629	7,274	9,391	6,046	5,844	5,846	2,511	2,777	5,179	1,975	-
Ontario Corporate Taxes	**kCAD$**	**33,457**	-	-	-	-	-	3,009	4,728	6,104	3,930	3,799	3,800	1,632	1,805	3,367	1,284	-
Pre- Tax NCF to Project	**kCAD$**	**334,681**	1,675	675	27,456	62,753	51,209	35,157	29,154	61,383	45,146	49,269	46,751	25,955	23,035	35,913	21,196	3,073
Accum. Pre Tax NCF to Project	**kCAD$**	**334,681**	1,675	2,350	29,806	92,560	41,350	6,193	22,961	84,344	129,490	178,759	225,510	251,465	274,500	310,412	331,608	334,681
		-																
Internal Rate of Return	**%**	**39.3%**						Payback Period	**2.2 Years from start of Production**									
Net Present Value	**MCAD$**			**5.0%**	**$199.0**		**7.5%**	**$154.3**		**10.0%**	**$119.9**		**12.5%**	**$93.2**		**15.0%**	**$72.3**	


GOLD PRICE 1985 - 2012
(US$/oz)
3 Year Moving Average
Source: The London Bullion Market Association


SILVER PRICE 1985 - 2012
(US$/oz)
3 Year Moving Average
Source: The London Bullion Market Association


US : Canadian Exchange Rates
1985 - 2012
3 Year Moving Average
US$ per CAD$
CAD$ per US$
$1.80
$1.60
$1.40
$1.20
$1.00
$0.80
$0.60
1985
1987
1989
1991
1993
1995
1997
1999
2001
2003
2005
2007
2009
2011
Source: Bank of Canada